TECHNICAL REPORT AND MINERAL RESOURCE UPDATE

ON THE

GOLIATH GOLD PROJECT

KENORA MINING DIVISION
NORTHWESTERN ONTARIO, CANADA

for

TREASURY METALS INCORPORATED

Report No. 955

A.C.A. Howe International Limited
Toronto, Ontario, Canada

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

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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, MBA, B.Sc. (Geology), 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") and focuses on Howe's updated independent mineral resource estimate of the Thunder Lake mineralised zones within the Goliath Project.

1.1 PROPERTY LOCATION ACCESS AND DESCRIPTION

The Goliath Gold 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), 17 patented land parcels (763.9 hectares) and a private land parcel (101 hectares) as detailed in Appendix A. The total area of the claim group is approximately 4,929 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 17 patented land parcels. Treasury's 2008 drilling was confined to unpatented claims 1106348 and 1106347, and patented claims 21609, 34461 and 4822. Treasury's 2009 drilling was confined to unpatented claim 1106348 and patented claims 41215 and SV200. Treasury's 2010 and 2011 drilling was confined to unpatented claims 1106348 and 1106347, and patented claims 15395, 41215, 21553, 4822 and SV200. 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 mineralisation 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 mineralisation 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 MINERALISATION

The main zones of mineralisation (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 mineralisation 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 mineralisation extends over a strike length of at least 3,500 metres (Corona, 1998) and work by Treasury has shown this anomalous gold mineralisation and alteration to extend over a strike length of +5,000 metres.

The mineralised 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 mineralisation 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 mineralisation 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 mineralised 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 mineralisation is pervasive in the Main Zone, Hangingwall Zone and in the Footwall Zone, whereas high-grade gold mineralisation (>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 mineralisation.

The high-grade shoots are interpreted to be the result of tight folding of the mineralised 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, mineralised 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 exploration program comprised diamond drilling totalling 49,926.5 metres.

1.6 MINERAL RESOURCE ESTIMATE

During September-November, 2011, ACA Howe International Limited ("Howe") carried out a resource estimate for the Goliath deposit using historical drilling and current drilling. The resource estimate includes holes up to Hole TL11228, drilled during 2011. The mineral resource estimate was prepared by Doug Roy, M.A.Sc., P.Eng., Associate Mining Engineer with Howe.

Mineralised zones were outlined to enforce geological control during block modelling. The interpretations that ACA Howe (2008 and 2010) made during the previous mineral resource estimates were modified slightly in consideration of the current drilling.

A main zone, two hanging wall zones, and three footwall zones were outlined. Higher grade shoots were observed in the main zone. Therefore, the main zone was broken down into two domains - a higher grade and lower grade domain. 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.

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. Meaning, fire assay tended to give slightly higher grades than pulp metallics. For conservatism, the pulp metallics result was used over the fire assay result.

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.

Variography was carried out on regularised gold assays, with the following results for the main zone:

Direction	Azimuth	Plunge	Data	Model Type	Model Range (m)	Nugget $[Ln(g/tonne)]^2$	Partial Sill $[Ln(g/tonne)]^2$	Fit
Normal to Plane of Mineralisation (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

Variography was also carried out for silver, which could be a byproduct of gold production. The semi-variogram range was 55 metres. Considering relative metal prices and relative expected processing recovery values, one gram of gold was equal to 57 grams of silver.

Based on 46 samples from the mineralised zones, the average specific gravity ("SG") was calculated as 2.75.

Ordinary block kriging was used for estimating block grades. 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.

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.

Resource parameters were chosen based on a combination of variography results and the author's judgement. All blocks that were within the outlined mineralised zones were considered to be (at least) Inferred. 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.

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.

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 mineralised 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 1500 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.

The results of block kriging were cross-validated against a nearest-neighbour estimate. Though the global declustered mean was slightly higher than the kriged average block grade, the author was satisfied with the cross-validation results.

A comparison was made with the previous mineral resource estimate that ACA Howe carried out in 2010. 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 major causes behind the overall net increase in tonnes and metal content were:

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

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

1.7 ENVIRONMENTAL AND PERMITTING STATUS

Treasury has commissioned Environmental Base Line Studies using the services of Klohn Crippen Berger ("KCB"). Studies were initiated in the Fall of 2010 and have continued to the date of this report. These studies will examine the health of the ecosystem by studying ground and surface water quality, sediment quality, fisheries, terrestrial resources and soil quality. 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.

1.8 CONCLUSIONS AND RECOMMENDATIONS

In Howe's opinion, Treasury should continue work to advance the Project, by gathering information and undertaking studies with the view to eventually undertaking a 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. The overall objective of the work would be to determine mining and processing parameters to the preliminary feasibility level of accuracy (plus or minus 15-20%). Should the preliminary feasibility be positive, mineral reserves can be identified.

The grand total budgetary cost for this work, including a preliminary feasibility study, is estimated to be $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 updated independent mineral resource estimate of the Thunder Lake mineralised zones within the Goliath Project.

The Goliath Project, located in northwestern 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 48 Colonization Ave. North in the City of Dryden, Ontario. approximately 20 kilometres west of the Project 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 the update of Howe's NI 43-101 compliant 2010 independent mineral resource estimate of the Thunder Lake mineralised zones within the Goliath Project with recommendations to allow Treasury and current or potential partners to reach informed decisions. This Report was prepared by Mr. Doug Roy, M.A.Sc., P.Eng. Associate Mining Engineer with Howe and Mr. Ian D. Trinder, M.Sc. (Geology), P.Geo., Senior Geologist with Howe. Mr. Roy is a mining engineer with over ten years experience in the mining industry. He has participated in numerous projects and resource estimates for precious metals and base metals projects and has authored or co-authored numerous OSC-2A and NI 43-101 resource reports. Mr. Trinder has over 20 years experience in the mining industry with a background in international precious and base metals mineral exploration including project evaluation and management.

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 of Treasury and Caracle Creek International Consulting Inc. (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 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 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.

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. The current resource estimate update is based on assay data available to Howe as of that date and includes the previous data and an additional 144 new Treasury drill holes totaling approximately 60,000 metres taking 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 is collared northeast of the current resource area and therefore would not be included in the resource estimate update.

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 November 9, 2011.

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 = 34.2857 grams/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 (CAD$) unless otherwise noted. Zinc, copper, and lead prices are stated as US$ 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 northwestern 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), 17 patented land parcels (763.9 hectares) and a private land parcel (101 hectares) as detailed in Appendix A. The total area of the claim group is approximately 4,929 hectares (approximately 49 km^2) covering portions of Hartman and Zealand townships east of the City of Dryden, Kenora Mining Division (Figure 4-3). Treasury's 2008 drilling was confined to unpatented claims 1106348 and 1106347, and patented claims 21609, 34461 and 4822. Treasury's 2009 drilling was confined to unpatented claim 1106348 and patented claims 41215 and SV200. Treasury's 2010 and 2011 drilling was confined to unpatented claims 1106348 and 1106347, and patented claims 15395, 41215, 21553, 4822 and SV200. 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 (Dryden Tree Nursery) totaling 101 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 17 patented land parcels, totaling 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	CAD$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	CAD$50,000	January 1st	-	2.0
Delk[2]	24724	-		-	2.5
Davenport[2]	19088	-		-	2.0
--[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				$45,000***	
	TOTAL CAD$:	**$100,000**		**$45,000**	
	TOTAL US$:	**$3,500**			

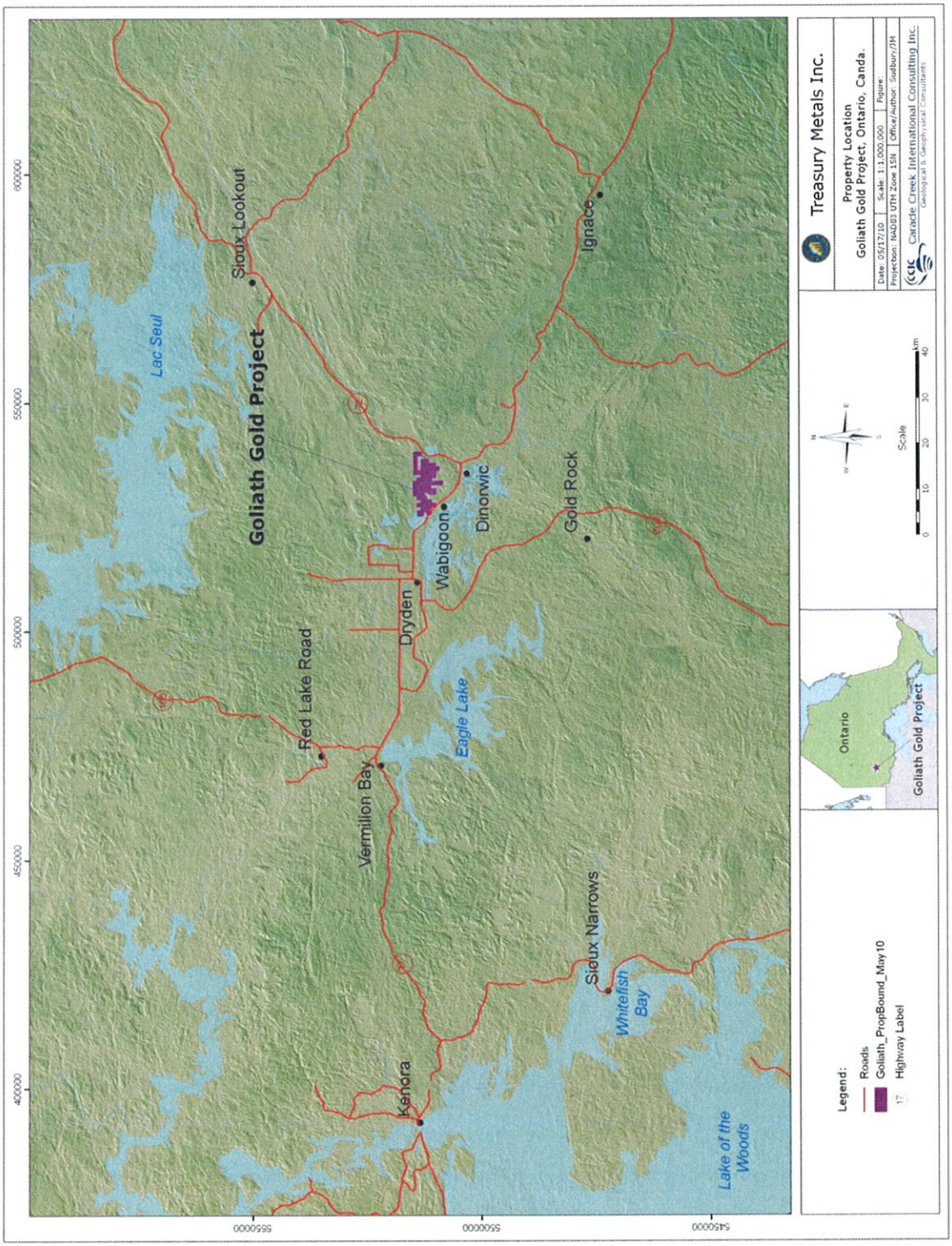
[1]Thunder Lake West; [2]Thunder Lake East; [3]Jones Property, [4]Laramide Property
*subject to withholding tax;
*** Option payments vary according to anniversary – See Table 4.3

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.

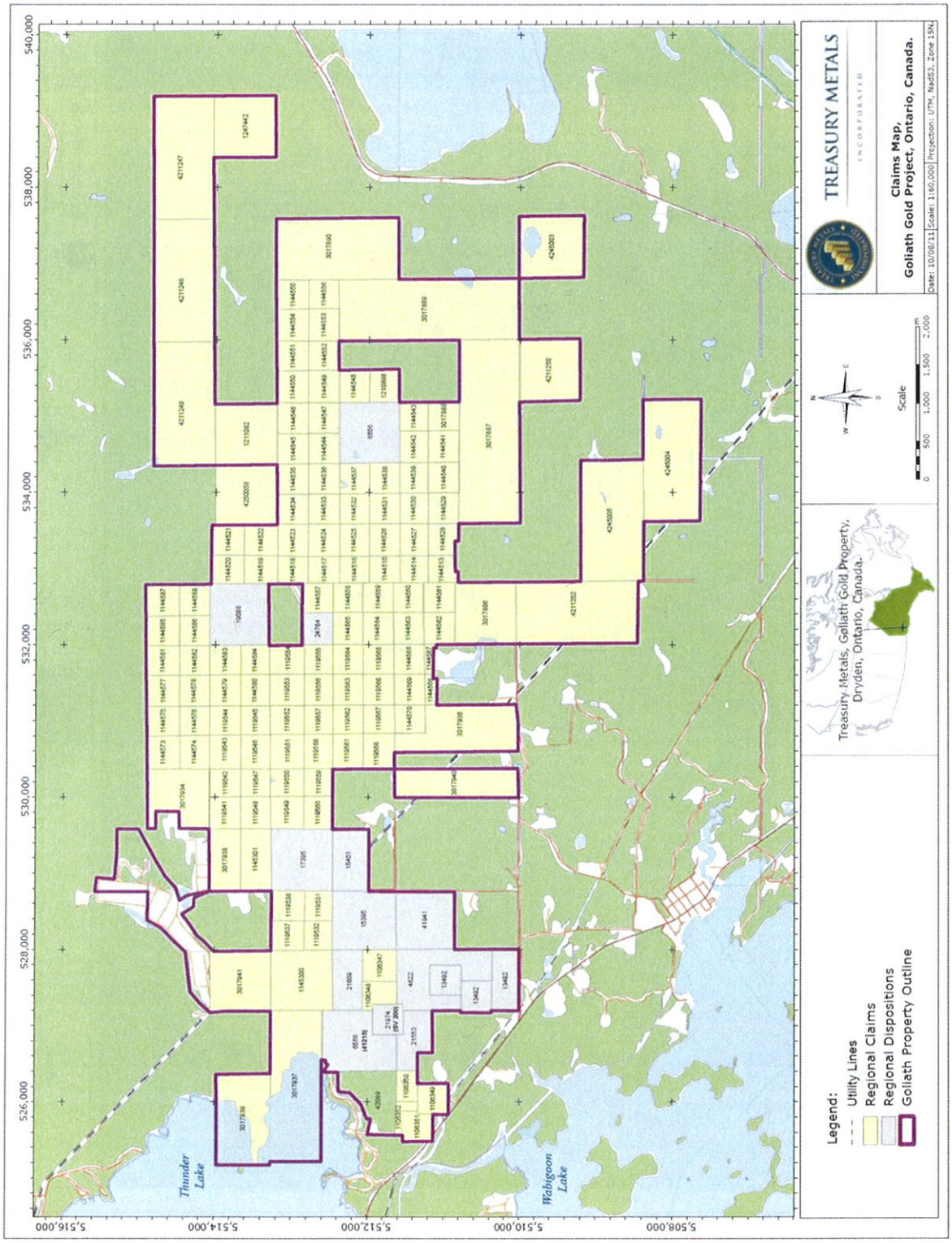
Source: CCIC, 2008

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 Company of 10% and 2.27%. All of the obligations to Teck and Corona have been met by Laramide and Treasury.

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 been 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 1776 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**	**1776**	

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 of 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 of 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 of 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 of 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 (101 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 Town 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 town 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. Hartman Township is characterized by swamps and a lack of outcrops 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 mineralisation 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 (Appendix B). 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 mineralisation 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 mineralisation 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 mineralisation 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 mineralised zone, lower in grade but thicker than the high-grade area, was partially defined. Drilling for the remainder of 1994 traced the high grade mineralised 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 mineralisation. 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 mineralised zone, confirming that the mineralisation 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 mineralisation; to determine the structural control of the high grade shoots by detailed underground mapping; and, to establish the true grade of gold mineralisation. 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 mineralisation 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 mineralised 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 mineralisation. A total of 220 metres of lateral drifting (3.0 metre by 3.0 metre cross section) was 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 mineralisation 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 mineralised regions;
- Nine (9) documented occurrences of coarse visible gold/electrum;
- Definition of the Main Zone No. 1 Shoot mineralisation, 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 mineralisation 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 (Appendix D). 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 mineralised 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

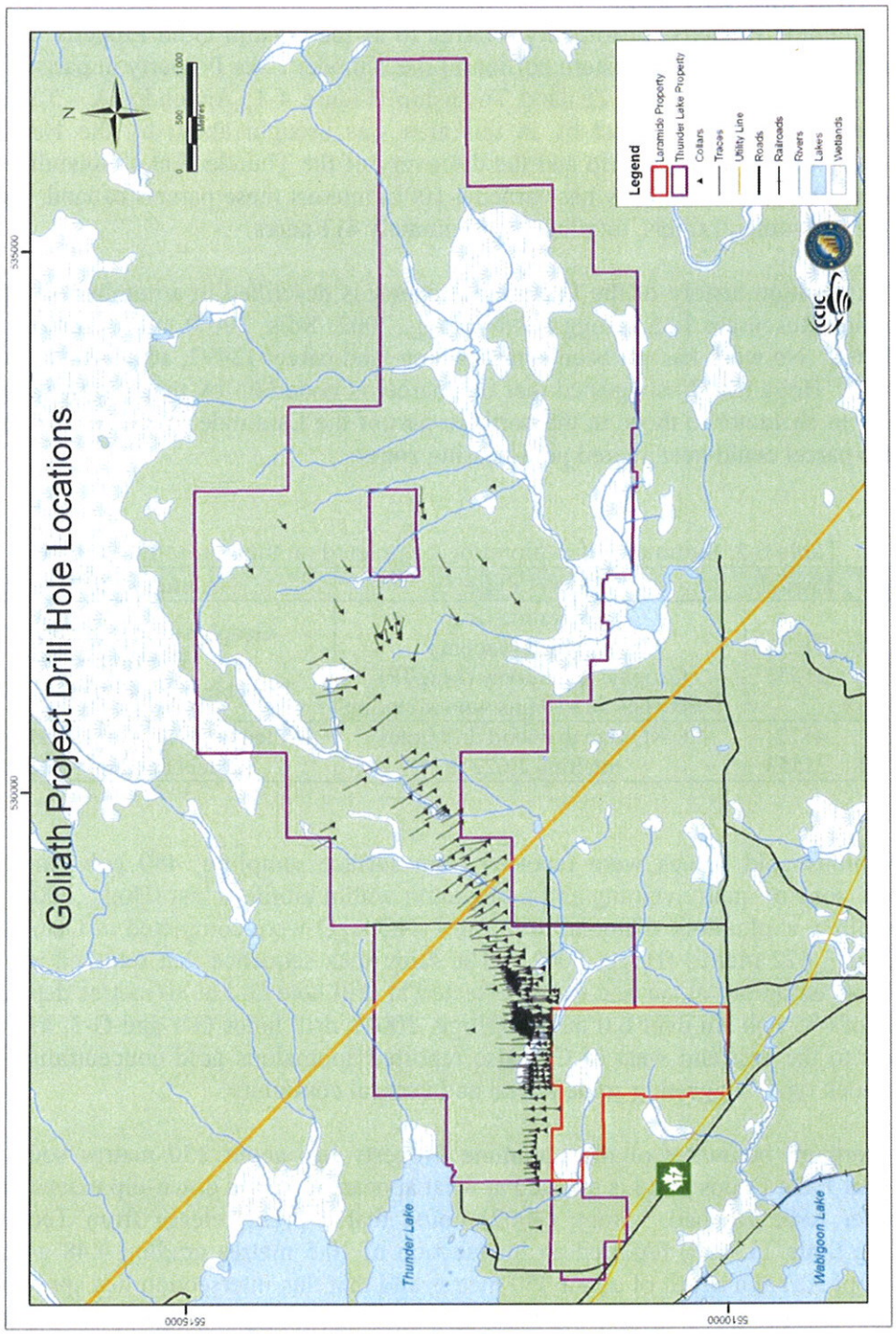


Figure 6-3. Location of drill collars and projections from Teck Cominco and Corona Gold joint venture (1989-1998).

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 mineralisation 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 mineralised 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).

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 ACA HOWE MINERAL RESOURCE ESTIMATES

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 MINERALISATION

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 km 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 Town 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.

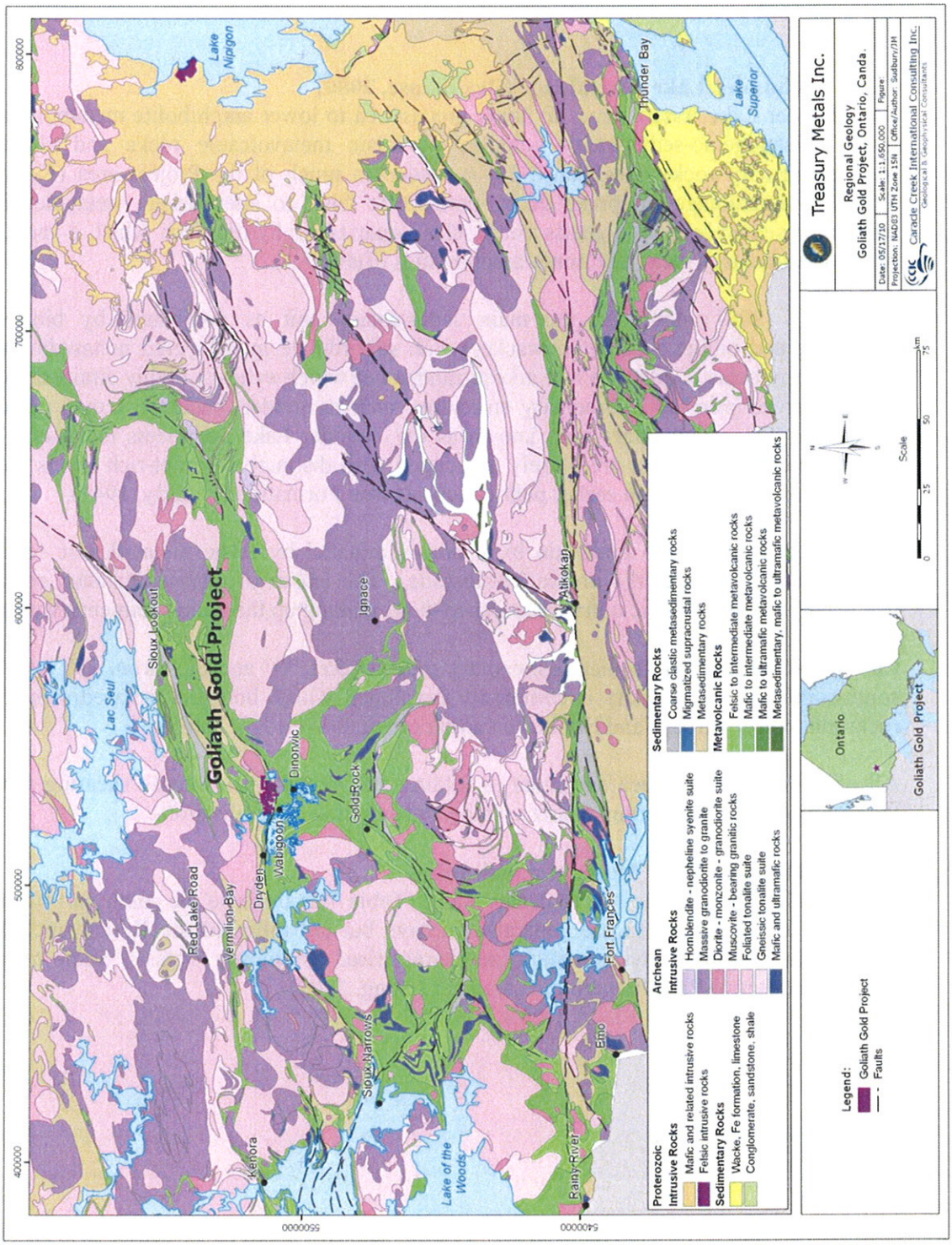


Figure 7-1. Location of the Goliath Project and regional geology of northwestern Ontario (from Percival and Easton, 2007a).

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 mineralisation.

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).

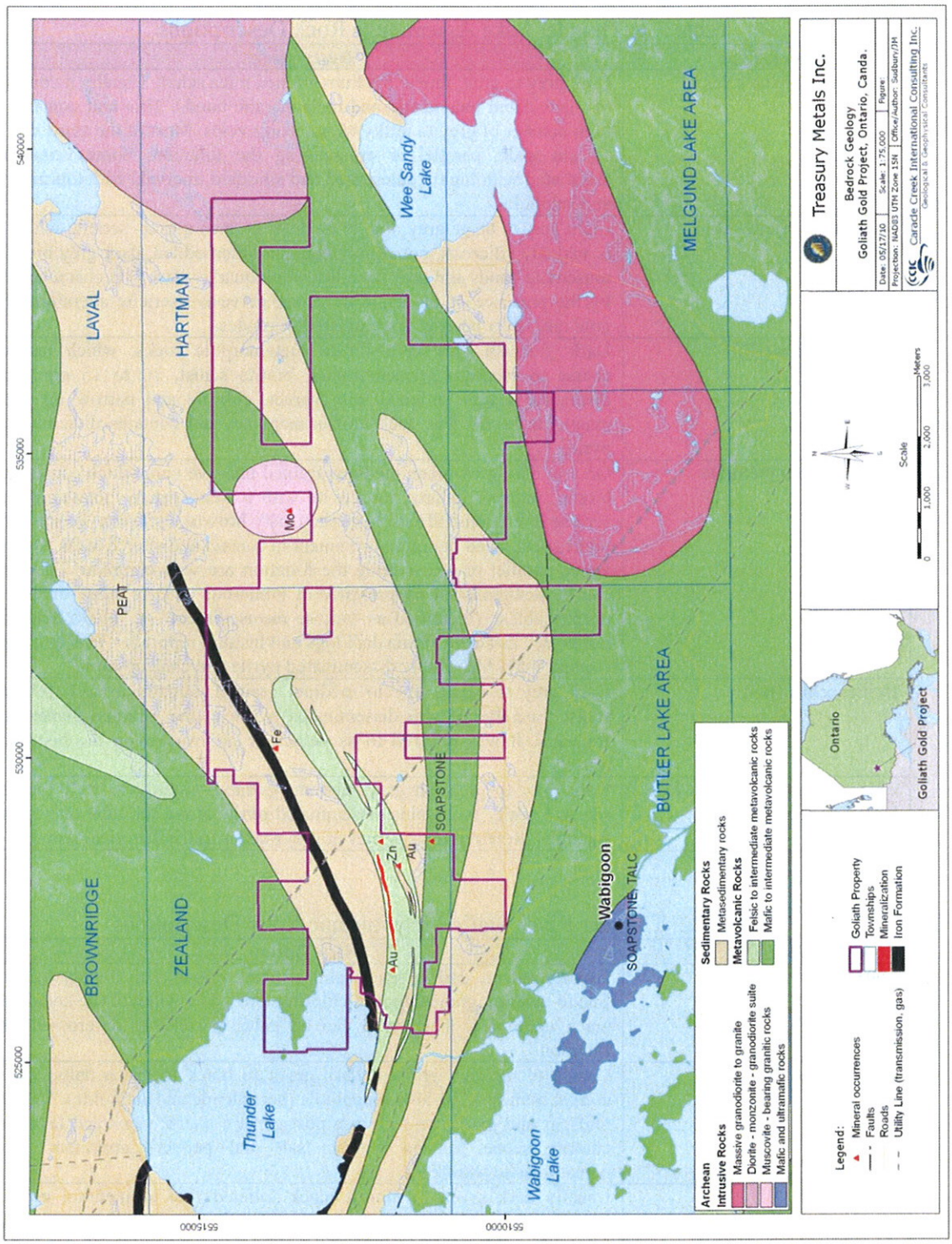


Figure 7-2. Bedrock geology in the area of the Goliath Project area, northwestern Ontario (after Beakhouse and Idziszek, 2006; Percival and Easton, 2007a).

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 mineralisation. 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 mineralisation was deposited during this event because the mineralisation 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)

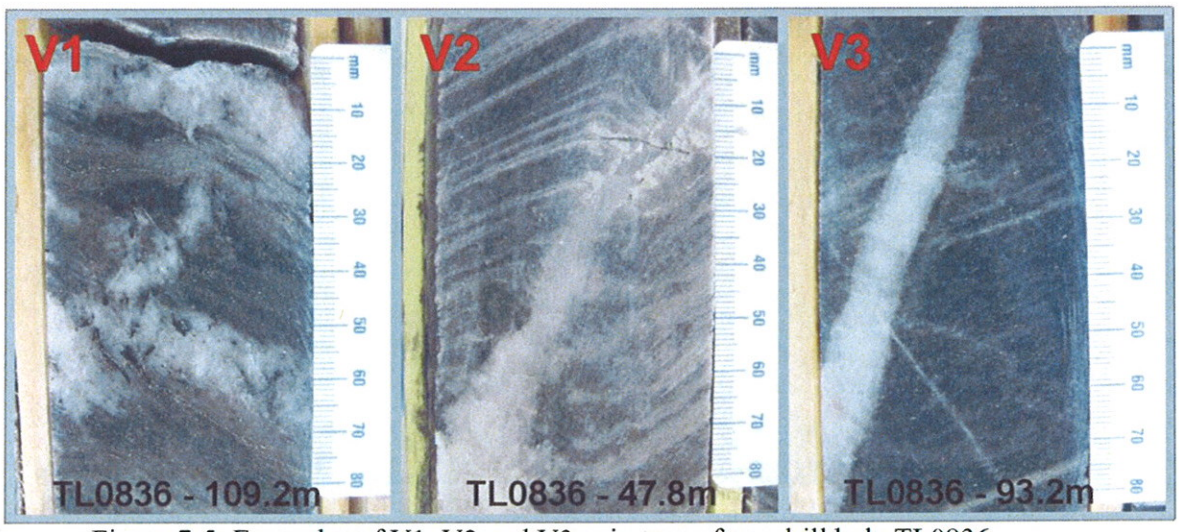


Figure 7-5. Examples of V1, V2 and V3 vein types from drill hole TL0836

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's in scale and affect the position of the felsic volcanic package that hosts mineralisation on the Goliath Project. Where F_2 fold axes and fold noses occur within the gold-silver mineralised 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 mineralisation 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 mineralisation block model, the F_2 fold axes are almost vertical and the intersections of the F2 fold axes and the mineralisation plunge steeply westward (**Error! Reference ource not found.**). 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 mineralisation. Determining where F_2 folds are likely to be located will identify the location of potential high-grade mineralisation. 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 mineralisation but the NW fault, appears to offset the mineralised zone. Wetherup (2008) demonstrated that high-grade mineralisation occurs along the steeply SW plunging intersections of F1-F2 fold axes and that these shoots are offset by the NW fault.

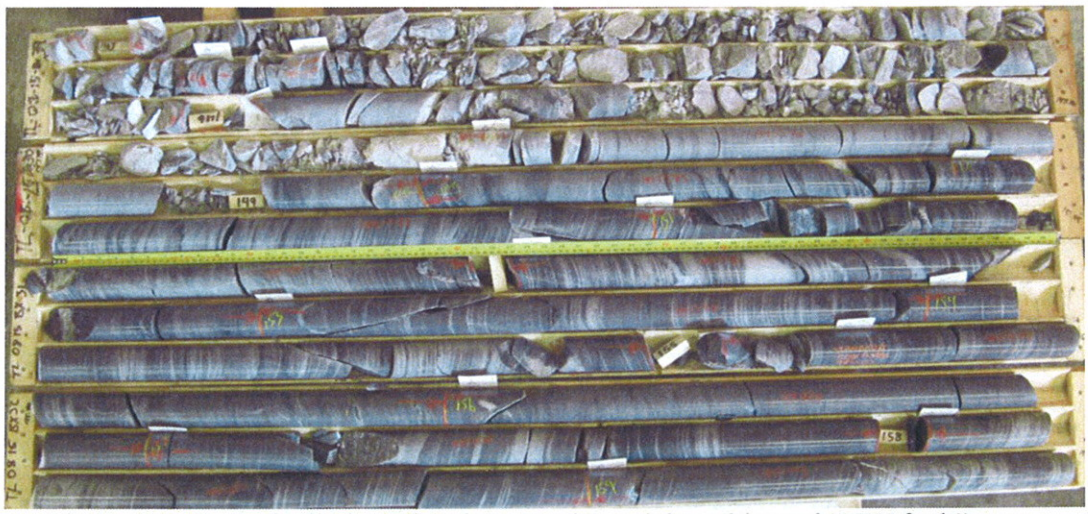


Figure 7-7. Tectonically brecciated muscovite sericite schist and "NW fault" zone intersected in the drill hole TL0815 at 148 m. The fault is filled in with white-greenish clay/gouge.

7.4 MINERALISATION

The main zones of mineralisation (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 mineralisation 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 mineralisation extends over a strike length of at least 3,500 metres (Corona, 1998) and work by Treasury has shown this anomalous gold mineralisation and alteration to extend over a strike length of +5,000 metres.

The mineralised 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 mineralisation 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 mineralisation for the Thunder Lake Deposit, including the Main Zone, Footwall and the Hangingwall Zone.

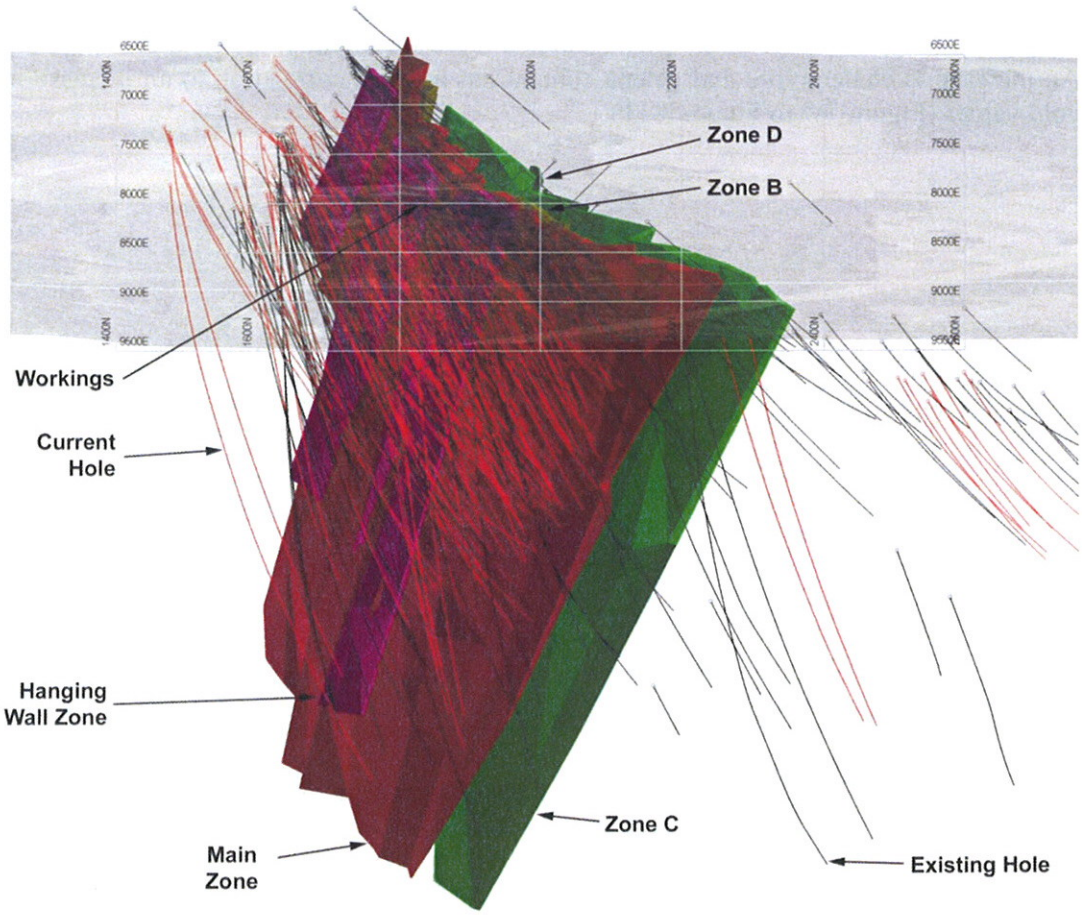


Figure 7-8. 3D view of interpreted mineralised zones of the Thunder Lake Deposit looking west -
Main Zone, Footwall Zone (B, C and D subzones) and, Hangingwall Zone

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 mineralisation 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 mineralised 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 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).

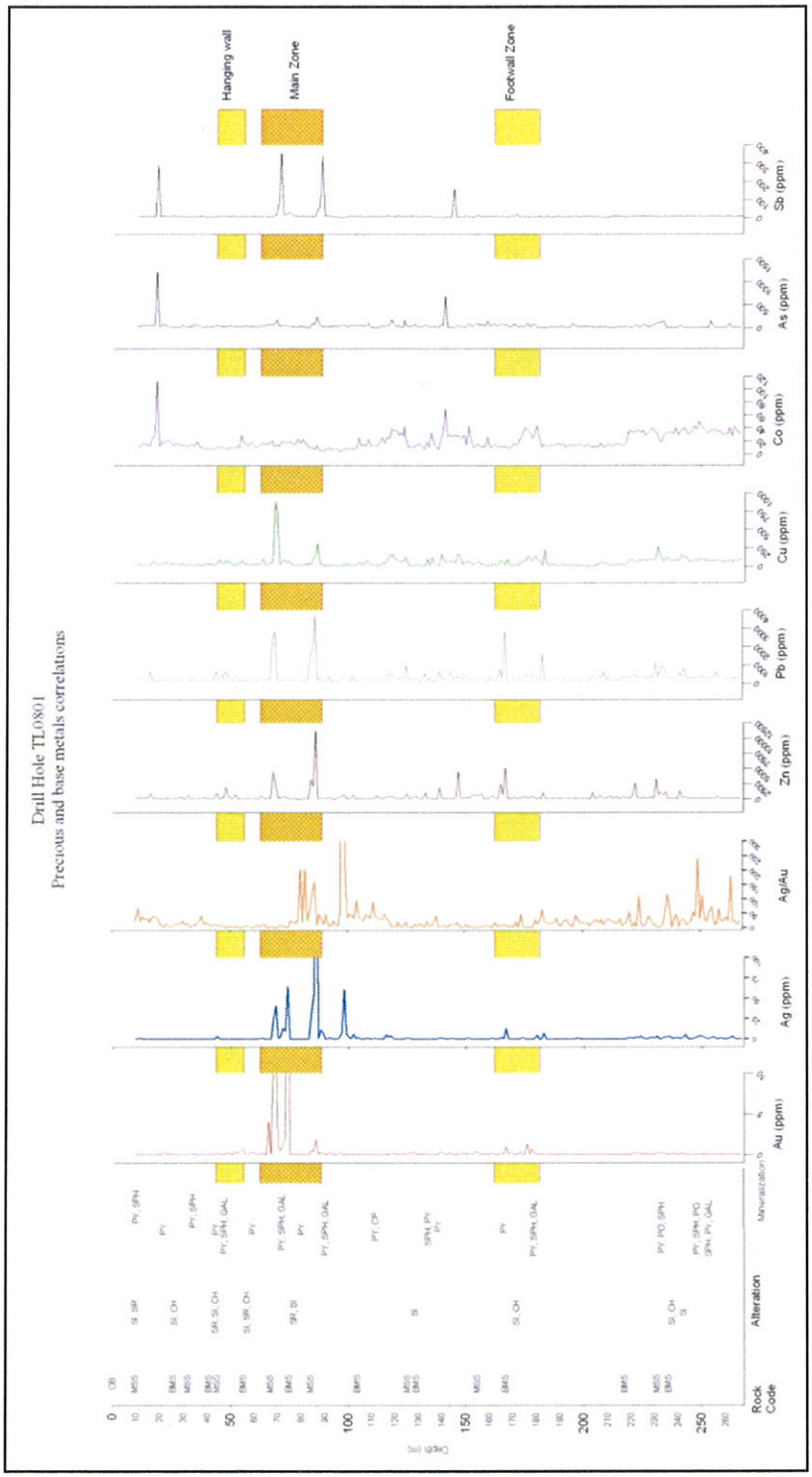


Figure 7-9. Correlation between precious and base metals in drill hole TL0801.

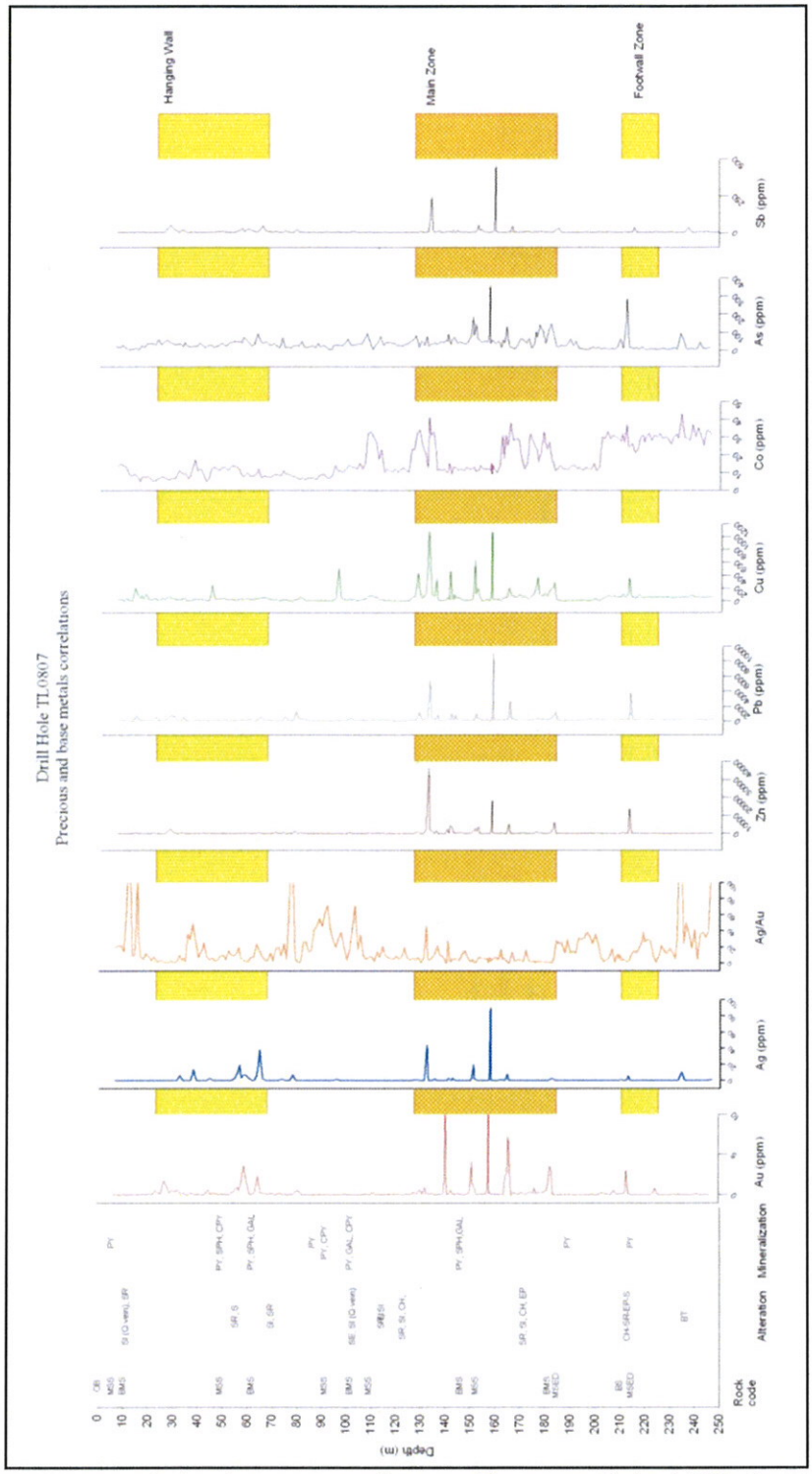


Figure 7-10. Correlation between precious and base metals in drill hole TL0807.

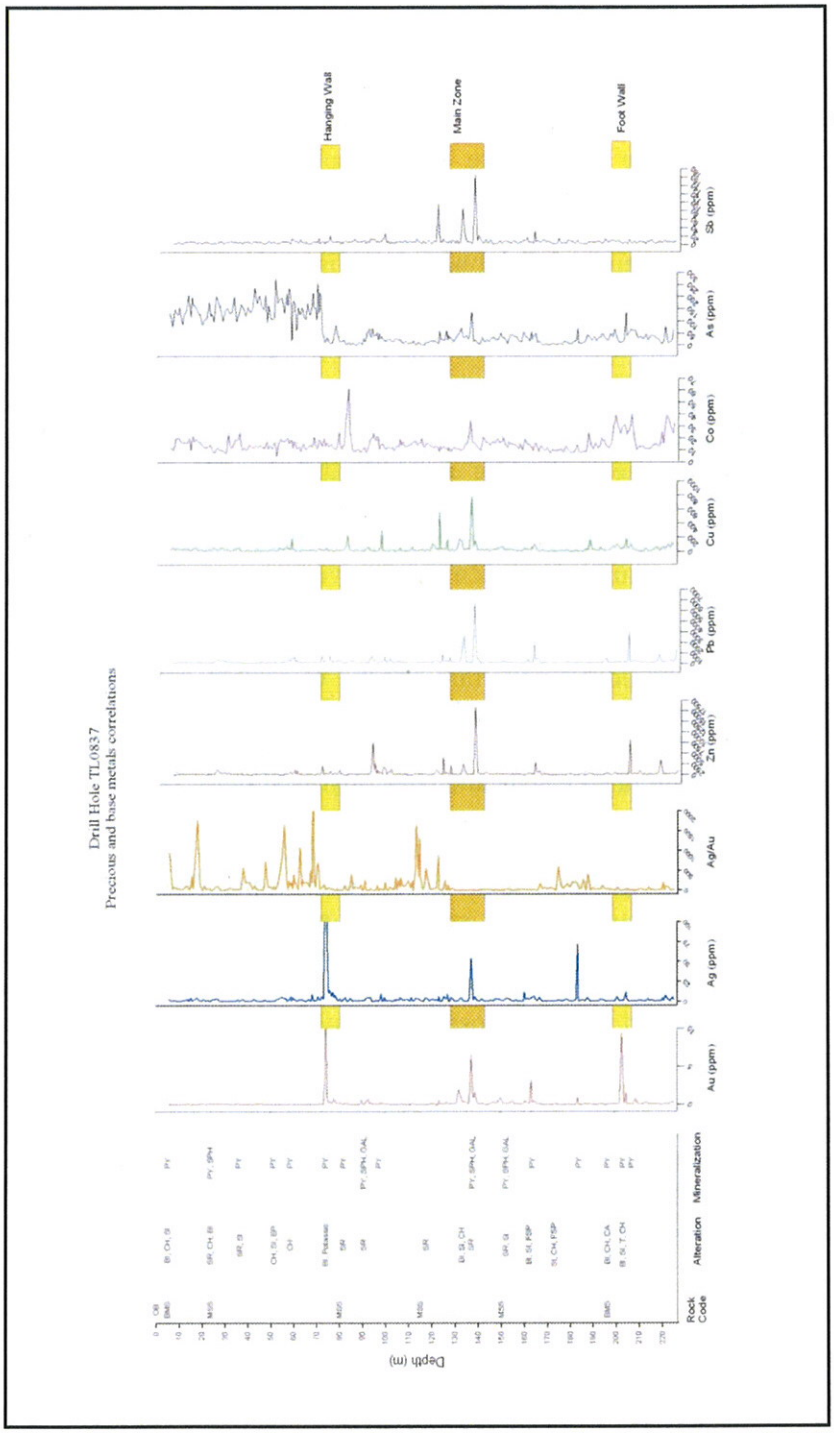


Figure 7-11. Correlation between precious and base metals in drill hole TL0837.

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 mineralisation event, more silver than the gold was contained in the hydrothermal solutions (ratio Ag/Au>>1), but during the epigenetic mineralisation 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 mineralisation 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 mineralisation 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 mineralisation is pervasive in the Main Zone, Hangingwall Zone and in the Footwall Zone, whereas high grade gold mineralisation (>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 mineralised 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 mineralised 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.

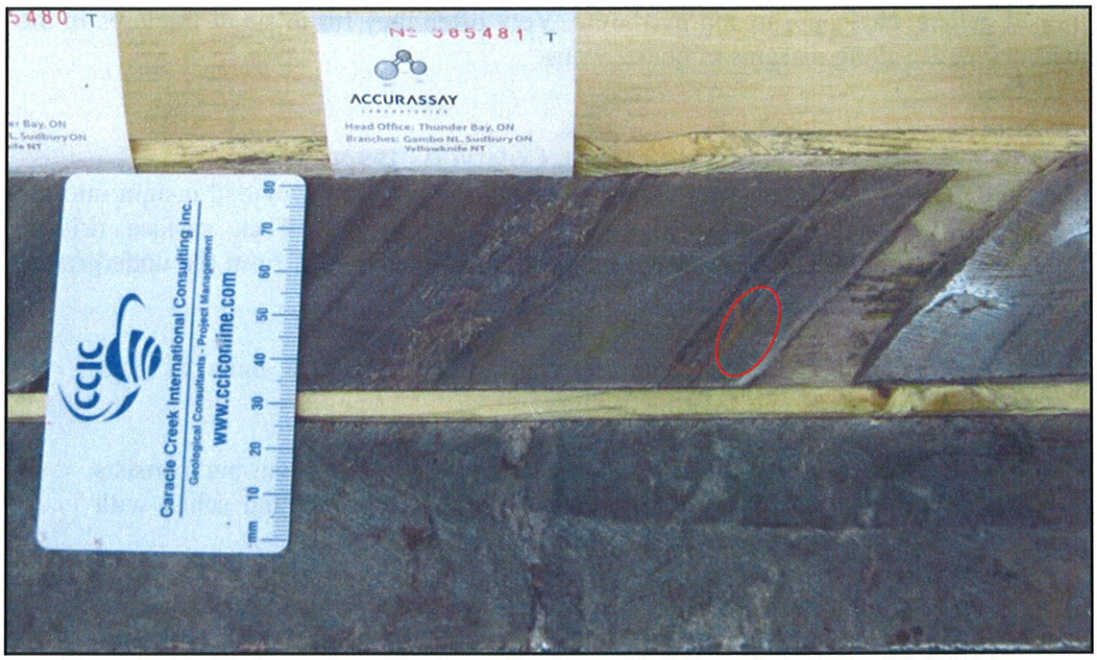


Figure 7-12. High-grade gold mineralisation with flakes of visible gold (VG) in a strongly altered section in felsic metavolcanic (biotite-muscovite schist) from the Main Zone (TL08-15, 50.8 m).

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 mineralised 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 mineralisation intersected during the historic Teck surface drilling programs. Page et al. (1999) reported the following observations from the underground program:

- More significant mineralised 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 mineralisation is localized in siliceous quartz-sericite schist containing disseminated sulphides, sulphide veins, and sulphide mineralised 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 mineralisation. 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 CCIC'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 mineralisation.

CCIC'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 mineralisation 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 mineralisation. 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

CCIC 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.

CCIC 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 mineralisation 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 mineralisation 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 mineralised 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 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.

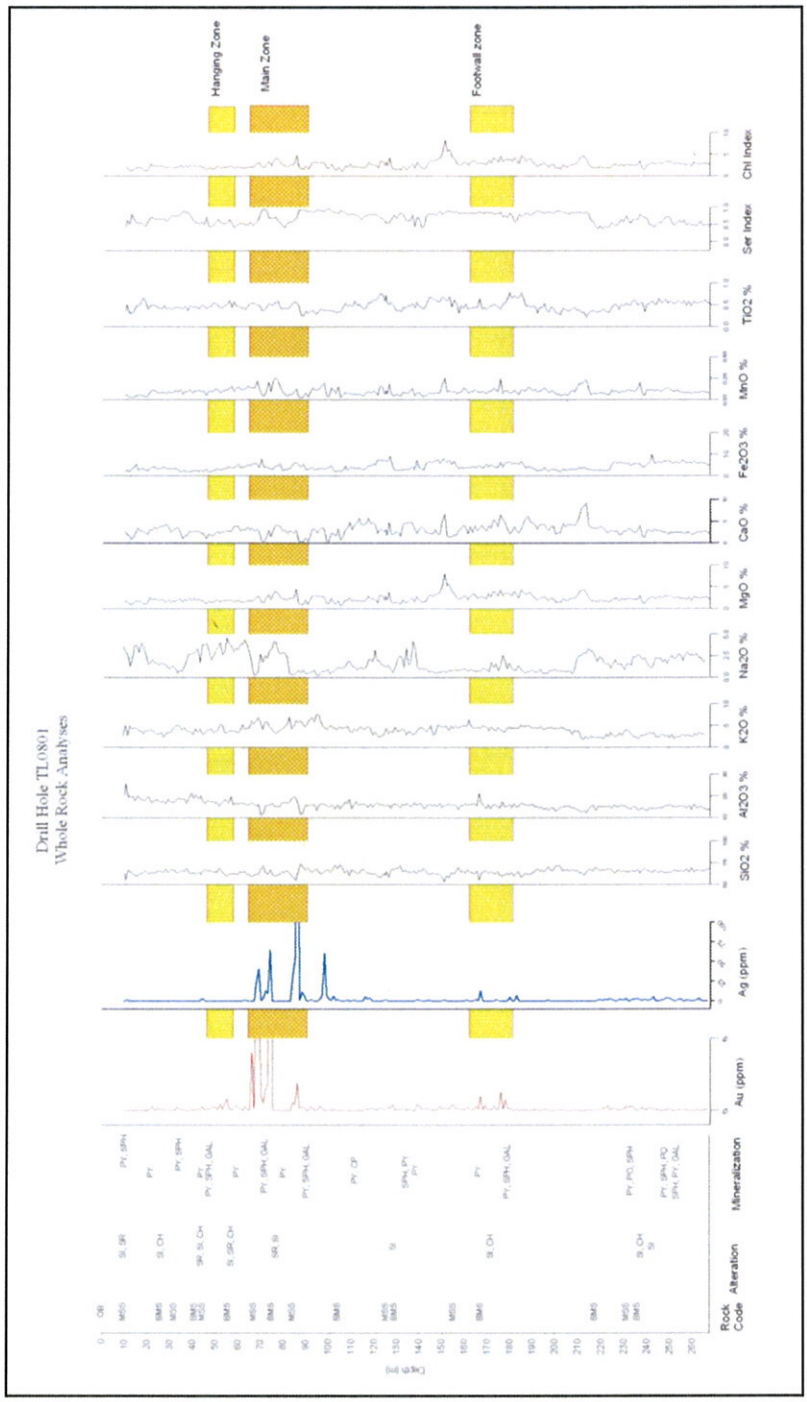


Figure 7-13. Whole rock analyses and the correlation with the Au-Ag mineralisation for TL0801.

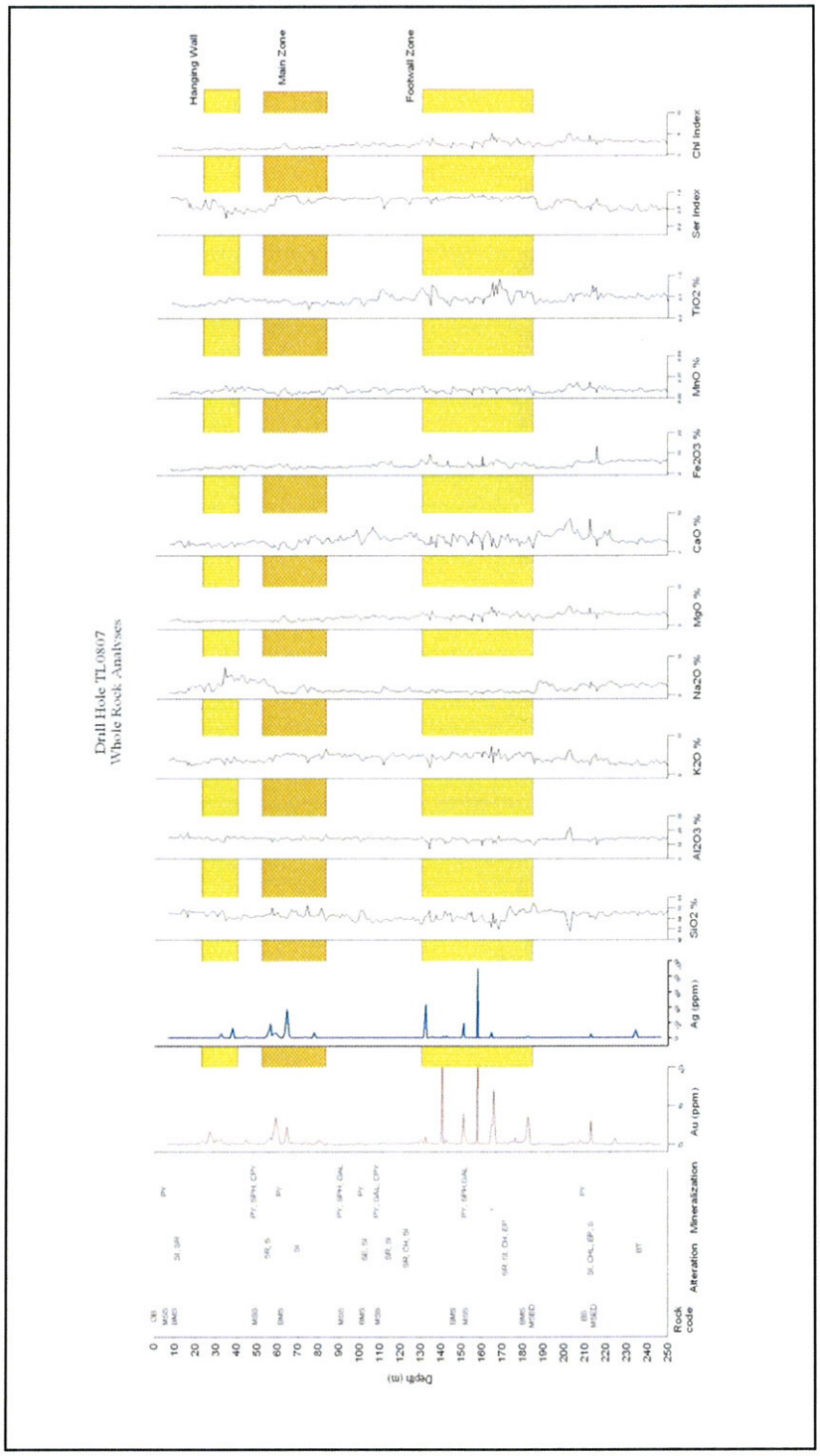


Figure 7-14. Whole rock analyses and the correlation with the Au-Ag mineralisation for TL0807.

8 DEPOSIT TYPES

The Thunder Lake Deposit was described by Teck-Corona (2001) as a shear-hosted mesothermal gold deposit with structurally controlled gold mineralisation 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 mineralisation is generally associated with highly elevated silver (locally >100 g/tonne), zinc, copper, and lead. The gold mineralisation 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 mineralisation 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 mineralisation of the Thunder Lake Deposit. Treasury notes that there is evidence for anomalous syngenetic gold (silver) mineralisation 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 mineralisation 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 mineralisation 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 mineralisation occurs within a zone of potassic altered in-situ brecciation at the margin of a mafic intrusive. Mineralisation also occurs in felsic dykes cutting the zone.

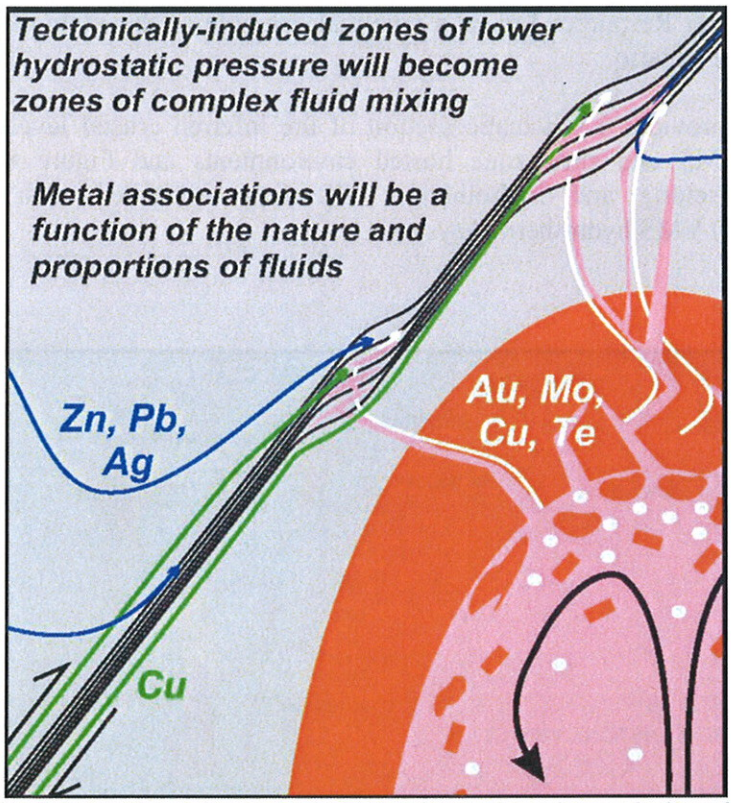


Figure 8-1. Idealized formation of magmatic hydrothermal Archaean lode gold deposit (after Burnham, 1979).

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 mineralisation (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 mineralising 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 mineralisation 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.

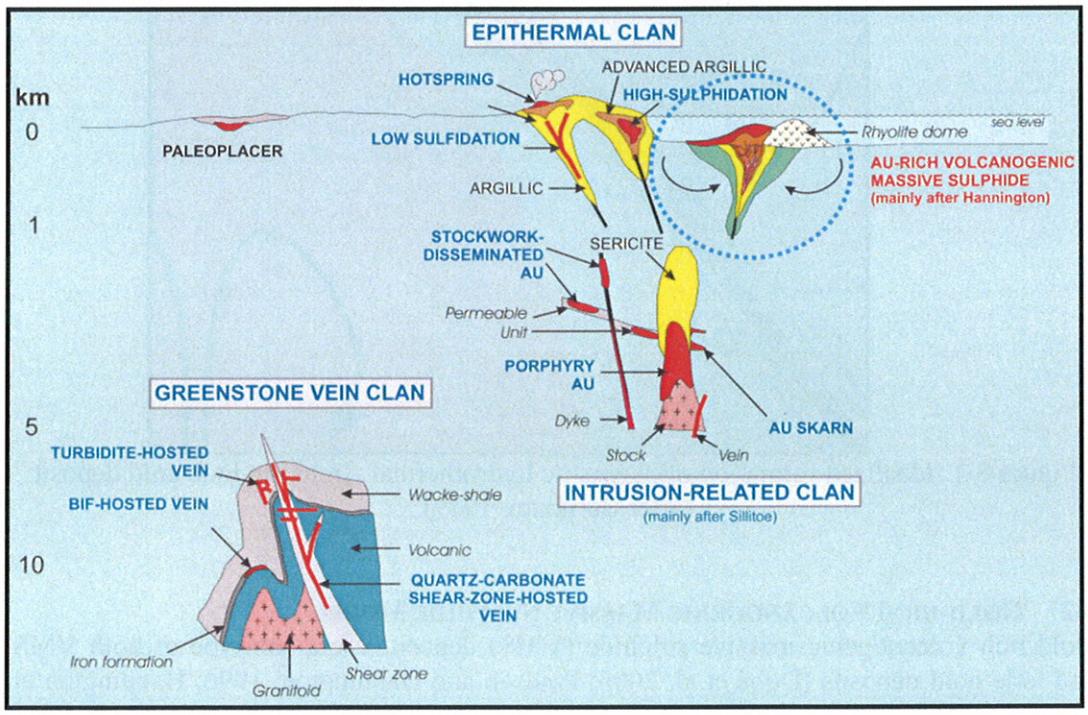


Figure 8-2. Various types of gold deposits and the inferred crustal levels of formation for gold-rich VMS deposits (Dubé et al., 2006).

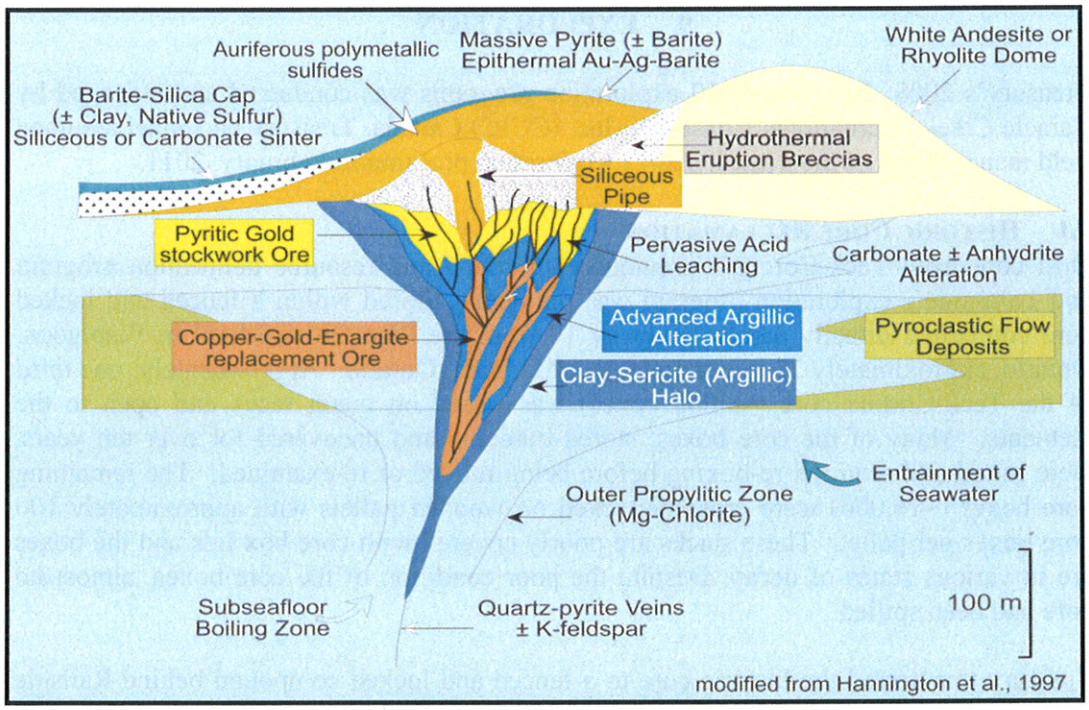


Figure 8-3. Geological setting and hydrothermal alteration associated with Au-rich high-sulphidation VMS hydrothermal systems (Dubé et al., 2006; Hannington et al., 1999).

9 EXPLORATION

Treasury's 2008, 2009 and 2010 exploration programs was conducted and managed by Caracle Creek International Consulting Inc. (CCIC) Canada. 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 to a fenced and locked compound behind Railside Sports and Marine in Dryden, Ontario. The cross-stacked core pallets were transported intact. Racked core was re-boxed if necessary and re-racked or cross-stacked in Dryden. It appears that with careful re-boxing, 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 1:5000 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.

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).

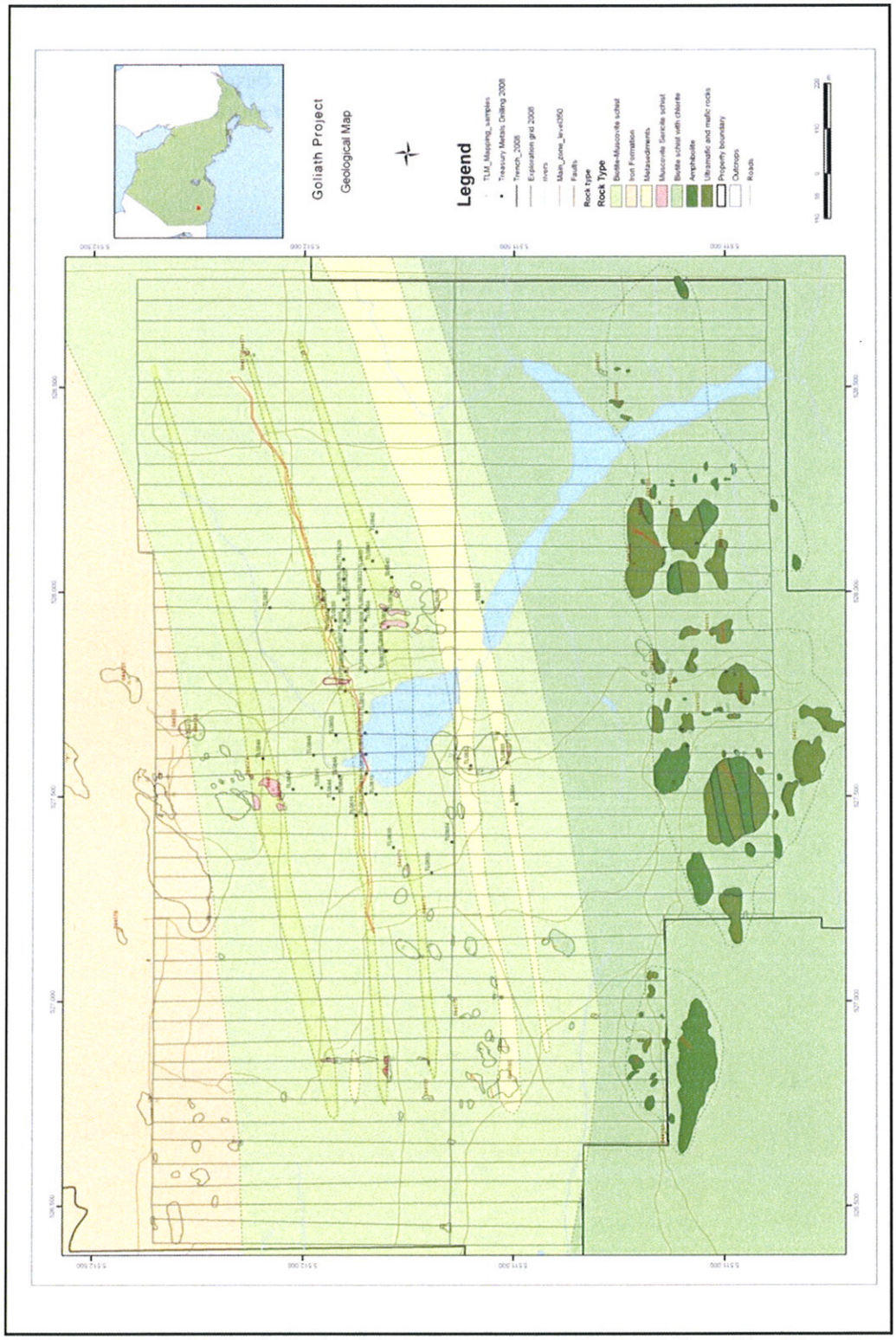


Figure 9-1. 2008 Geological Grid Map – Thunder Lake Deposit outlined in red.

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 mineralisation.

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 mineralised 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 mineralised 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.

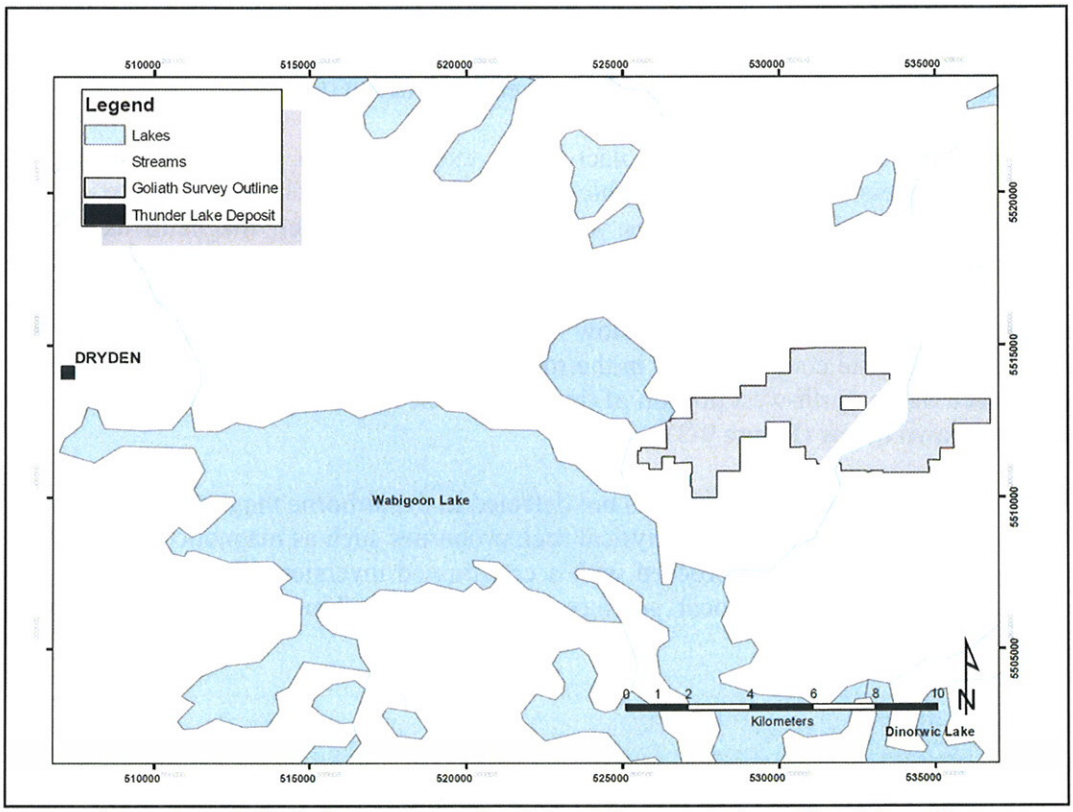


Figure 9-2. Goliath Survey Location Map. The survey is outlined in yellow.

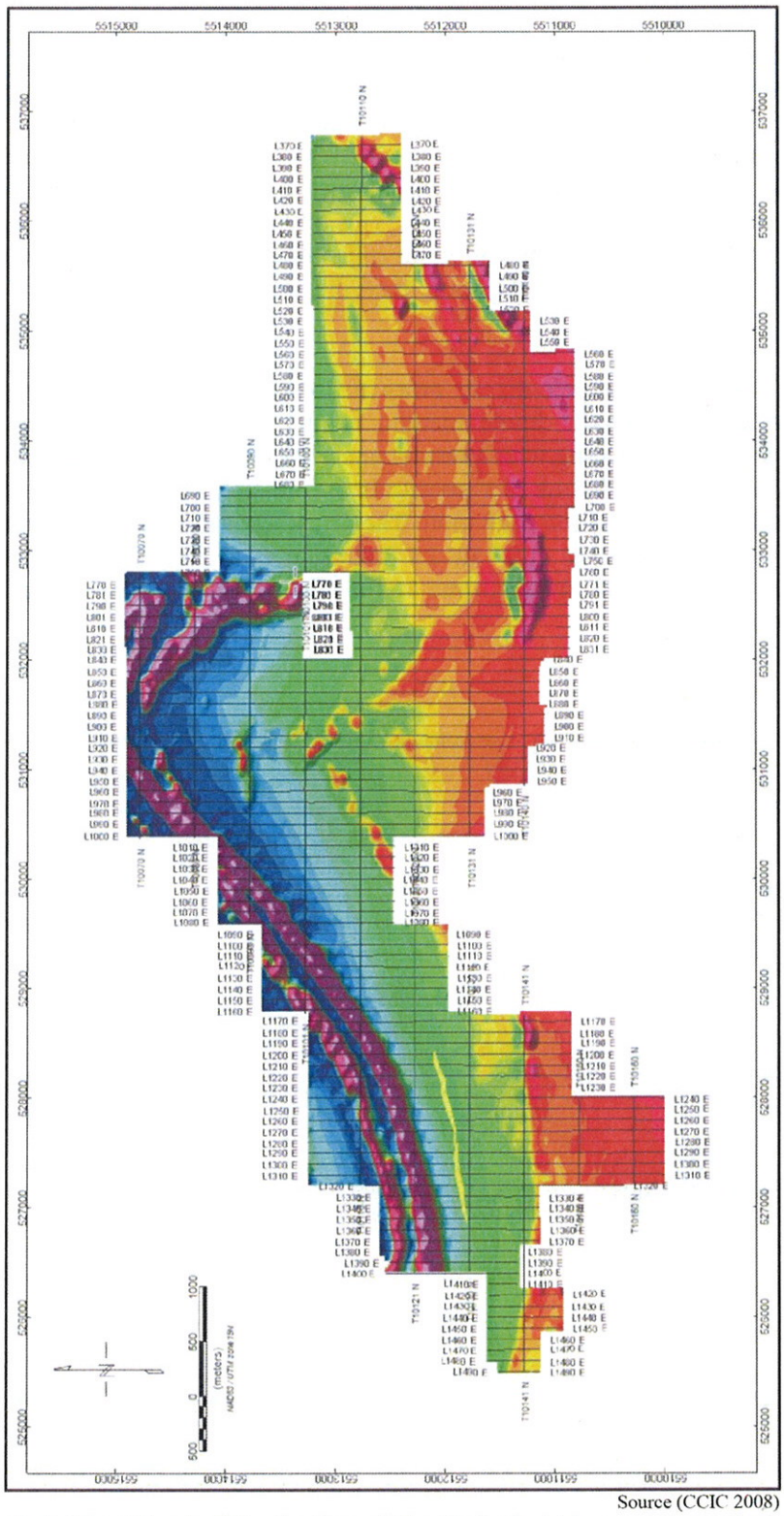
Table 9.2- Specifications of the Goliath Airborne Magnetic Survey.

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 mineralised 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.

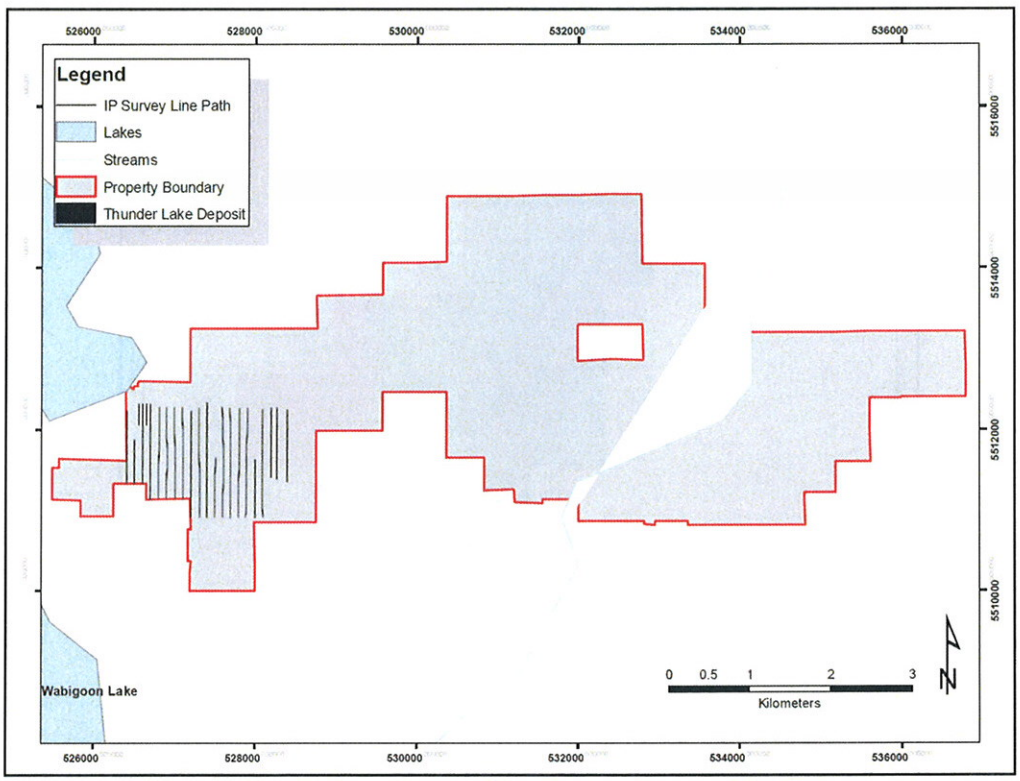


Figure 9-4. Location Map of IP survey on the Goliath Property.

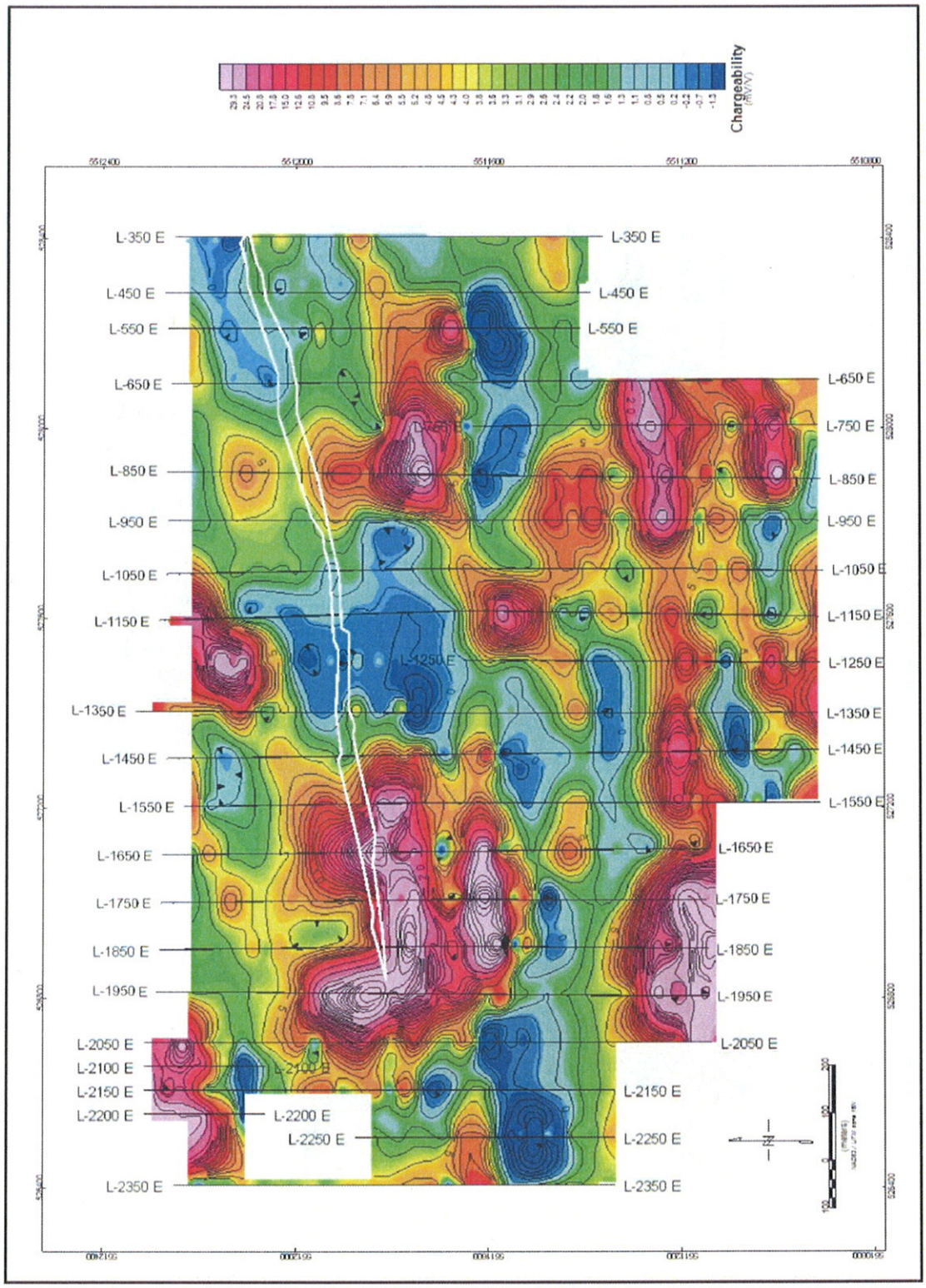


Figure 9-5. Chargeability (n:2) map.
Thunder Lake Deposit outlined in white (Source: CCIC 2008)

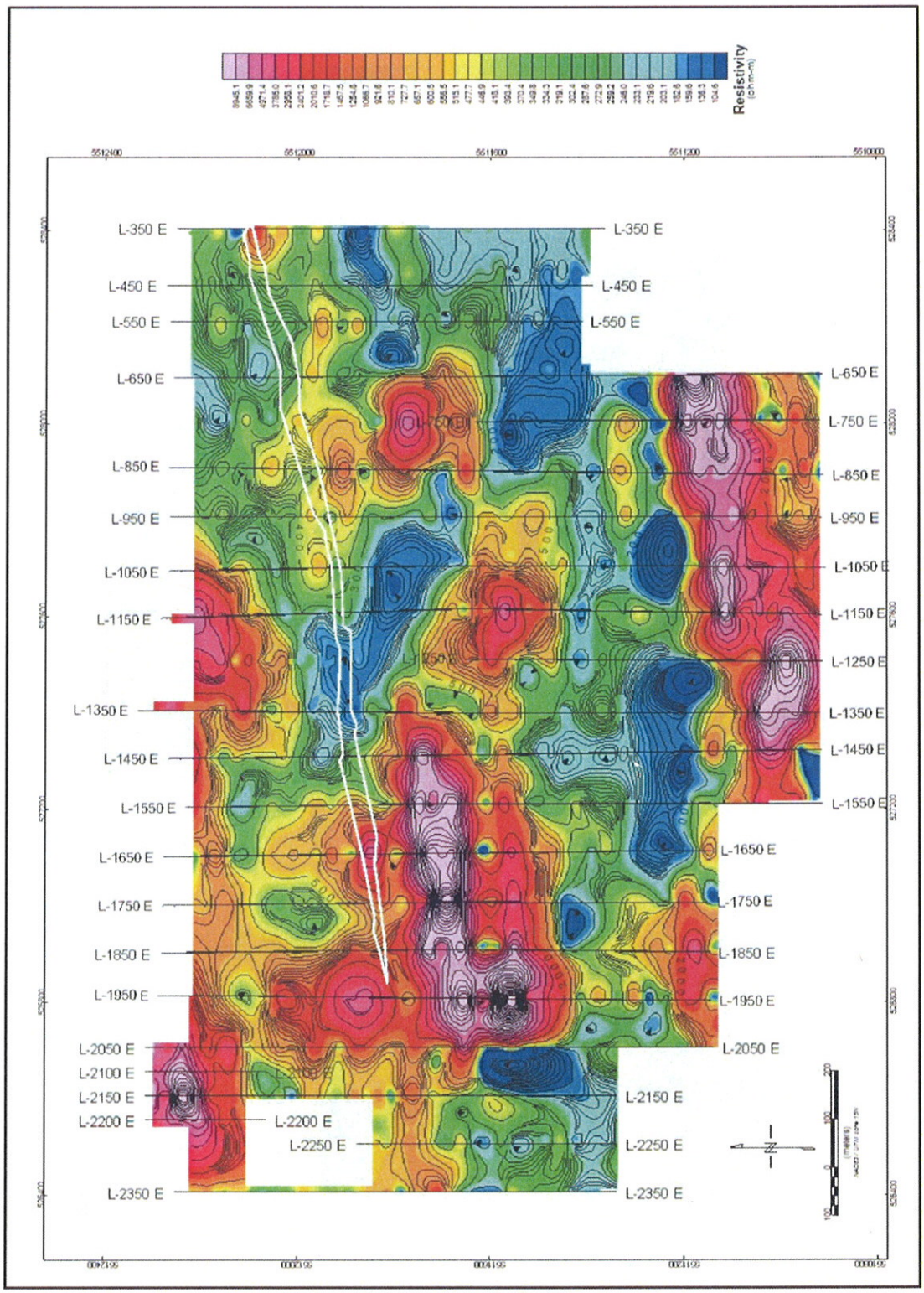


Figure 9-6. Resistivity (n:2) map.
Thunder Lake Deposit outlined in white (Source: CCIC 2008)

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 mineralisation and/or volume-percentage of conducting metallic sulphides may be low.

Four pseudosection products were generated by JVX and supplied as final products: chargeability (Figure 9-5), apparent resistivity (Figure 9-6), spectral MIP, and spectral tau. These products show a spatial coincidence between the north-west trending fault and low chargeability values. This appears to extend to the west-northwest (Figure 9-7). Treasury's and Teck's drilling indicates that alteration and gold mineralisation 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, 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 ore and host rocks including resistivity, time-domain IP, and chargeability and, to then to proceed with a constrained inversion of the IP data. This ould allow for a proper 3D integration of the data.

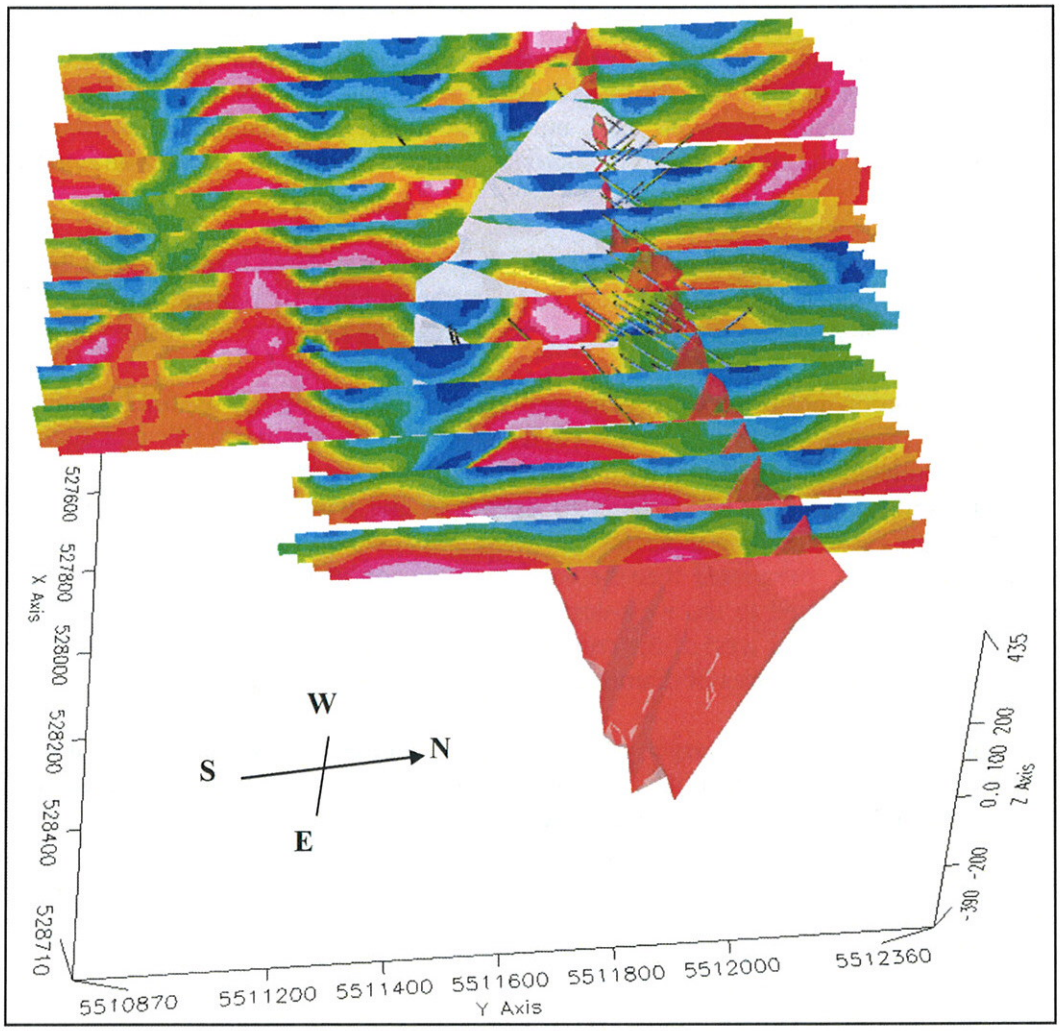


Figure 9-7. 3D view of Chargeability sections.
Fault (grey) possibly extends to west-northwest. Mineralised zone in red (Source: CCIC 2008)

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 mineralised 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).

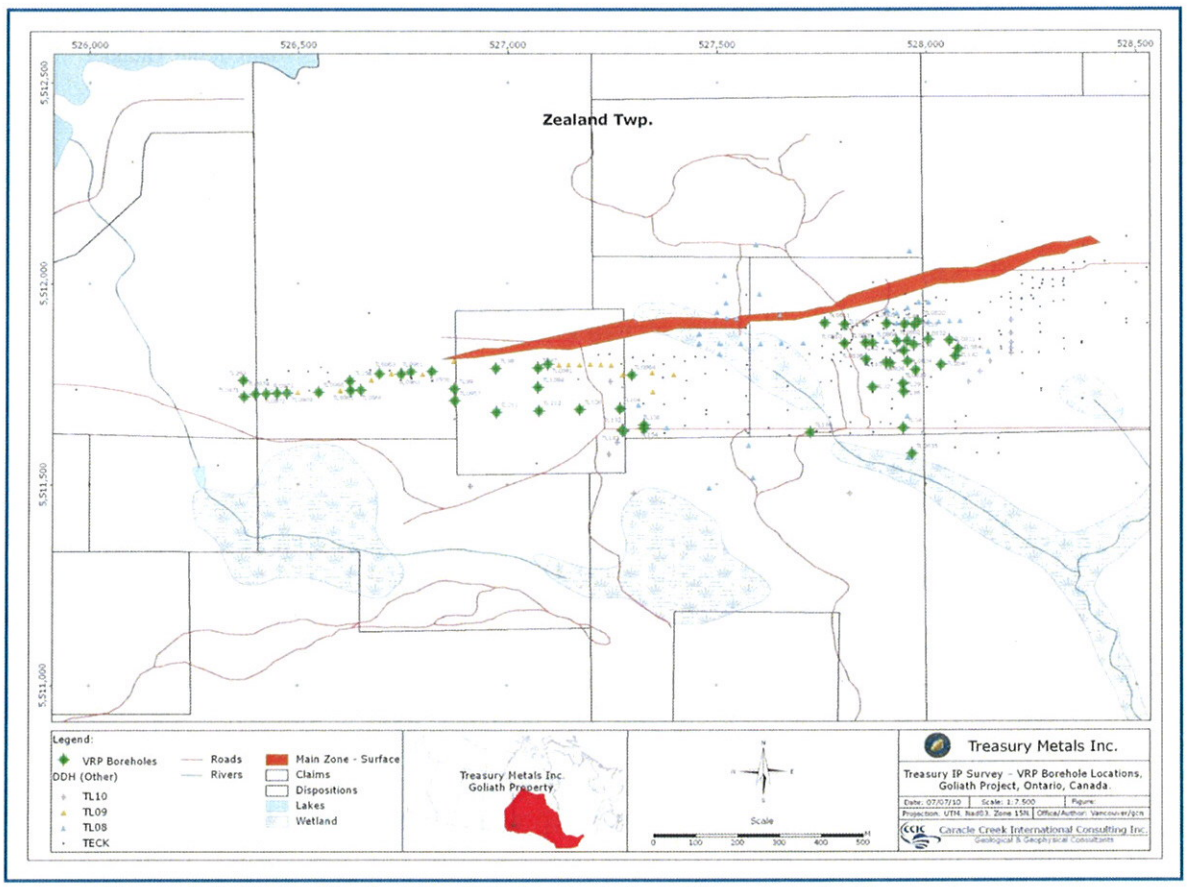


Figure 9-8. VRP (Vertical Resistivity Probe) and Tomography Locations

Resistivity/Chargeability Correlations

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

- Mineralised 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-mineralised zones
- Resistivity responses less than 5,000 Ω.m (3.7 log Ω.m) reflect mineralised 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-mineralised zones
- Chargeability responses greater than 50 mV/V reflect mineralised zones
- There is an overlap of resistivity and chargeability response between the mineralised and non-mineralised 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.

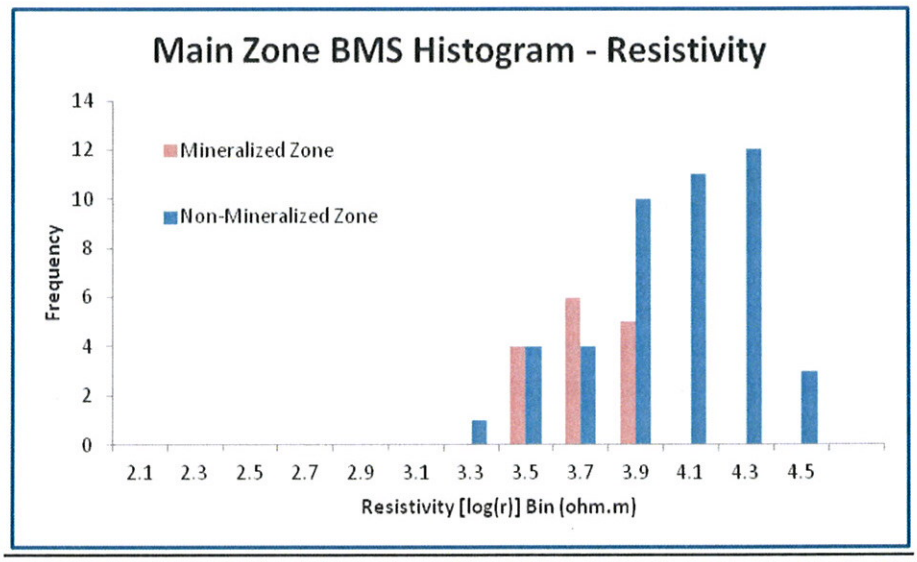


Figure 9-9: Mineralised vs Non-mineralised Resistivity Responses– Main Zone

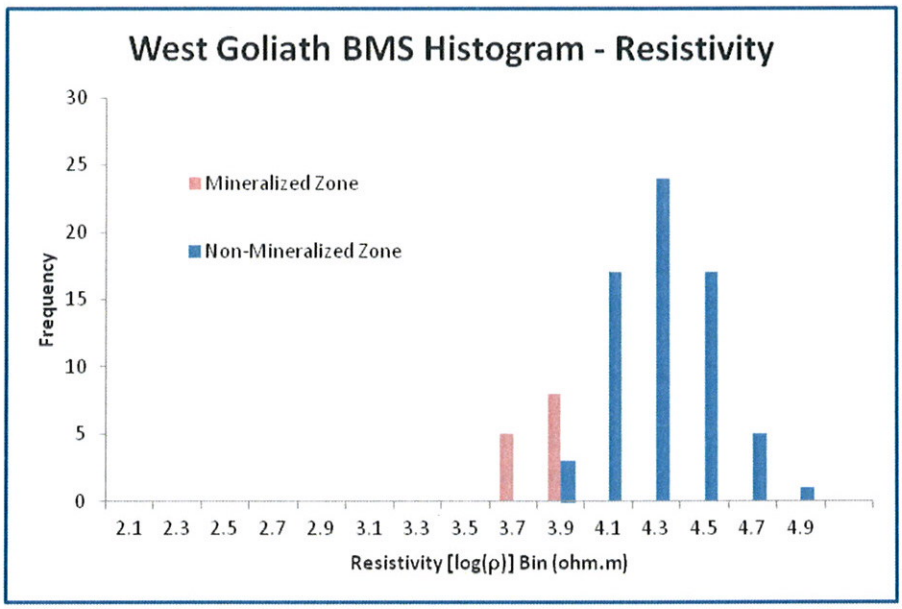


Figure 9-10: Mineralised vs Non-mineralised Resistivity Responses– West Goliath Ext.

Mineralisation Response

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

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

Anomaly Summary

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 mineralisation continuity.
- Within the West Goliath extensional exploration area:
  - VRP and tomography results were well correlated with known mineralisation 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

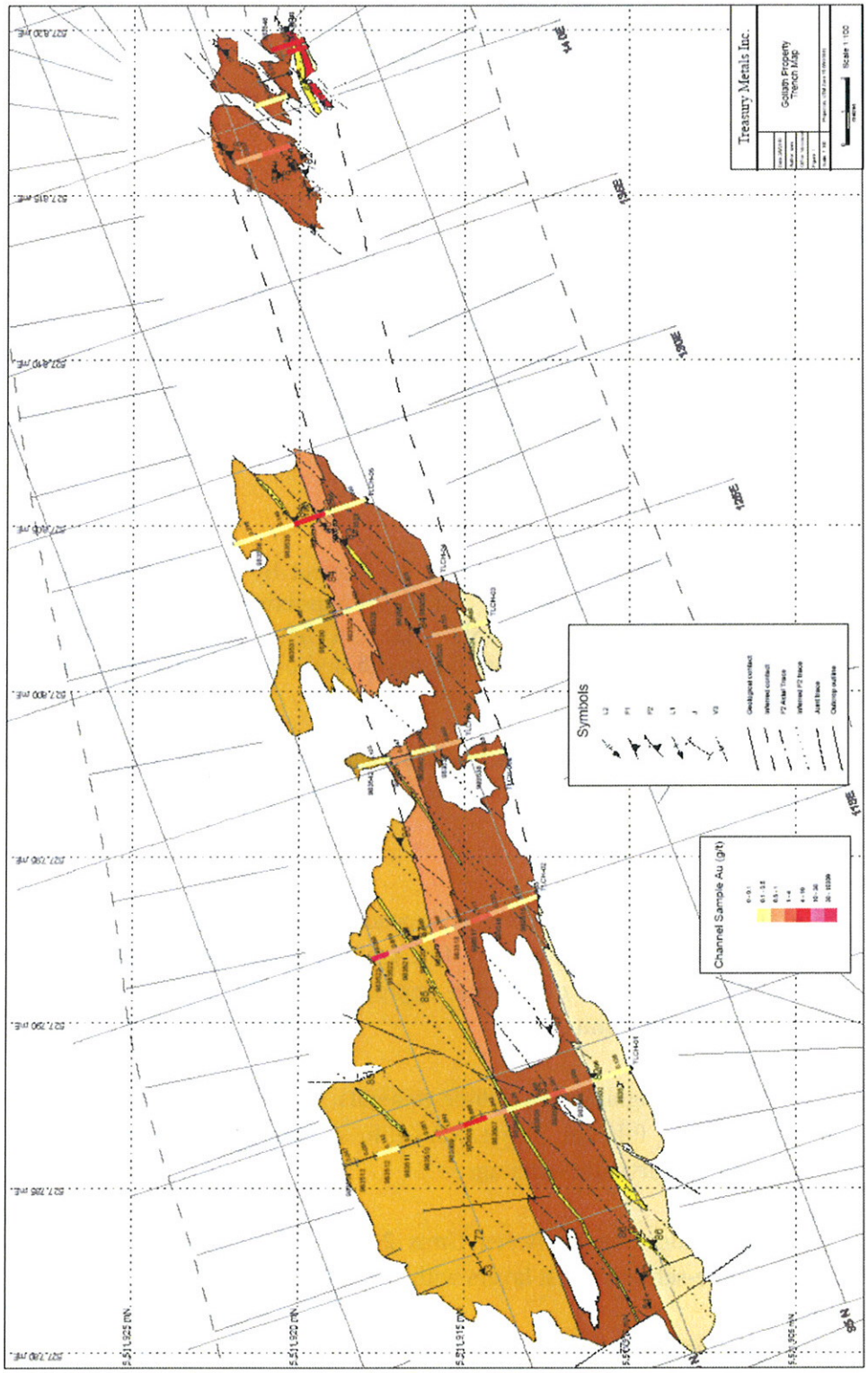


Figure 9-11: Geology and structural map of the 2010 Main Zone trench with gold assays

Through detailed structural mapping, key controls on the gold mineralisation 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°		
D_2	F_2	Closed (interlimb angle 60°) 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°		
D_3	J (?)	Brittle joints oriented ~162/81 and 032/82 ; possibly related to NW Fault	V_3	White un-deformed, planar cross-cutting 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/t Au with a high value of 0.48g/t Au over 0.3 m. Unit 2 averaged 1.06 g/t Au with a high value of 5.55 g/t Au over 1 m, which came from a zone of semi-massive sulphide. Unit 3 averaged 2.11 g/t Au with a high value of 7.09 g/t Au over 1 m. Samples from Unit 4 averaged 2.99 g/t with a high value of 49.06 g/t over 0.55 m. Overall, Units 2, 3 and 4 average 1.9 g/t 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 km. 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	1000m
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 (Furgo, 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 powerline 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.

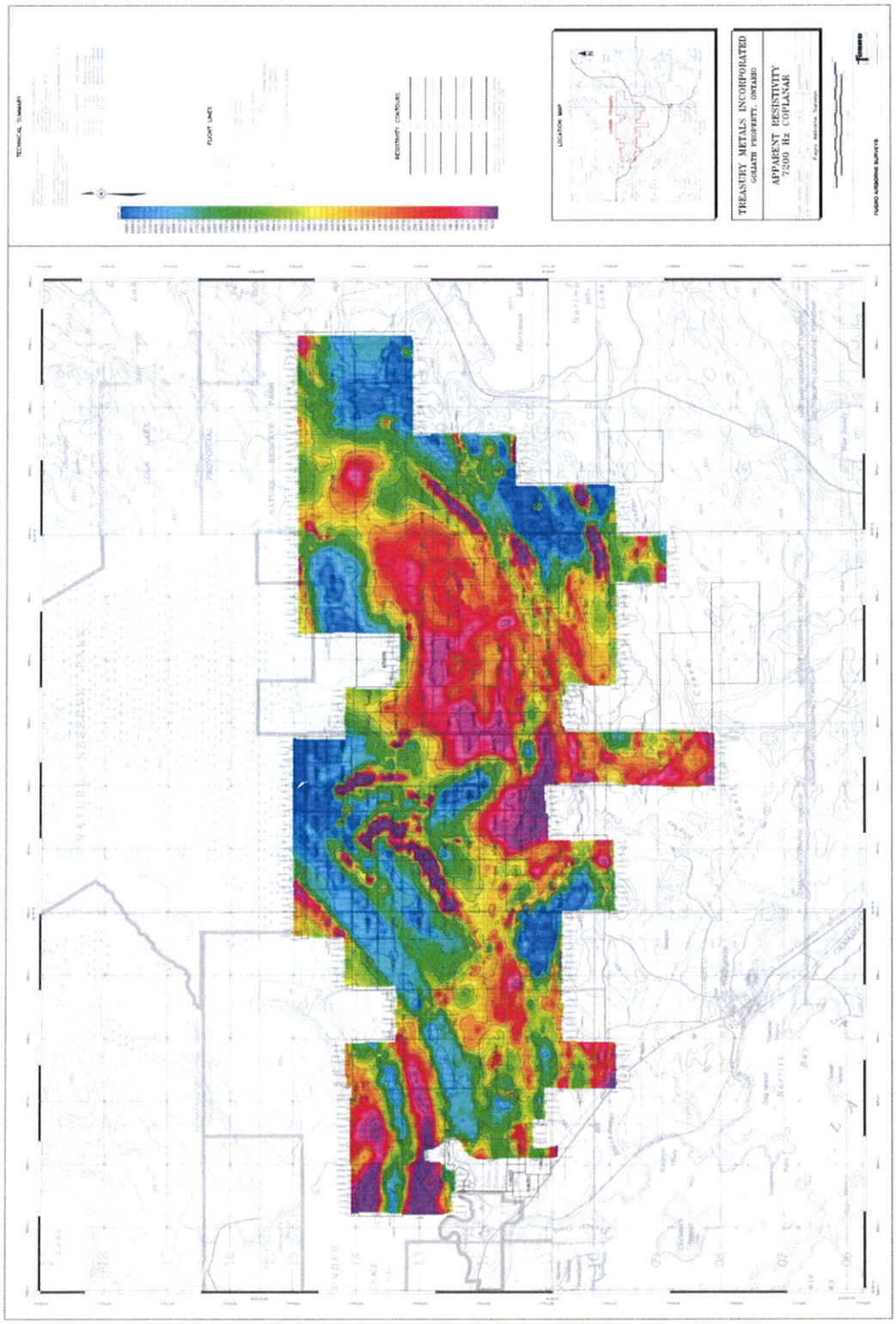


Figure 9-12: Apparent Resistivity 7200Hz Coplanar

9.12 2008-2011 DIAMOND DRILL PROGRAMS

Treasury has completed four 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, mineralisation 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 mineralisation.

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 ore shoots and prepare for the expansion of mineral resources. Results from Phase 2 drilling confirmed the alteration and gold mineralised 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, mineralisation 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

The Goliath Project drill hole database utilized in the 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 Drill-Holes 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 mineralisation 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 mineralised zone, which

strikes approximately east-west. The mineralised zone dips 72° to 78° toward the south thus drill holes drilled at –45 to –87 degrees dip did not intersect perpendicular to the mineralisation. True thicknesses of mineralisation intersected are perhaps 75% to 90% of the apparent thickness of the mineralised core intercepts in shallower dipping holes and perhaps as low as 25% of the apparent thickness of the mineralised core intercepts in the steeply dipping near vertical holes.

The updated Howe resource estimation presented in Section 14.0 includes holes TL0801 to TL11228 of the Treasury's 2008 to 2011 drill programs in addition to historic Teck drill holes and underground samples.

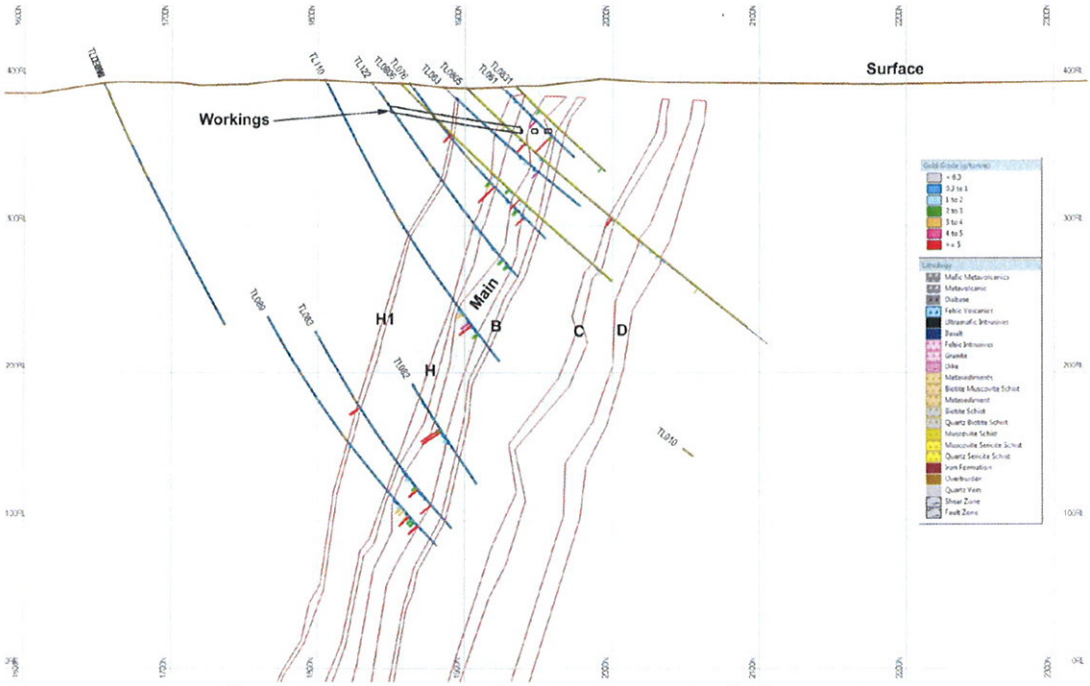


Figure 10-1: Cross-section 7900 m East – facing west.

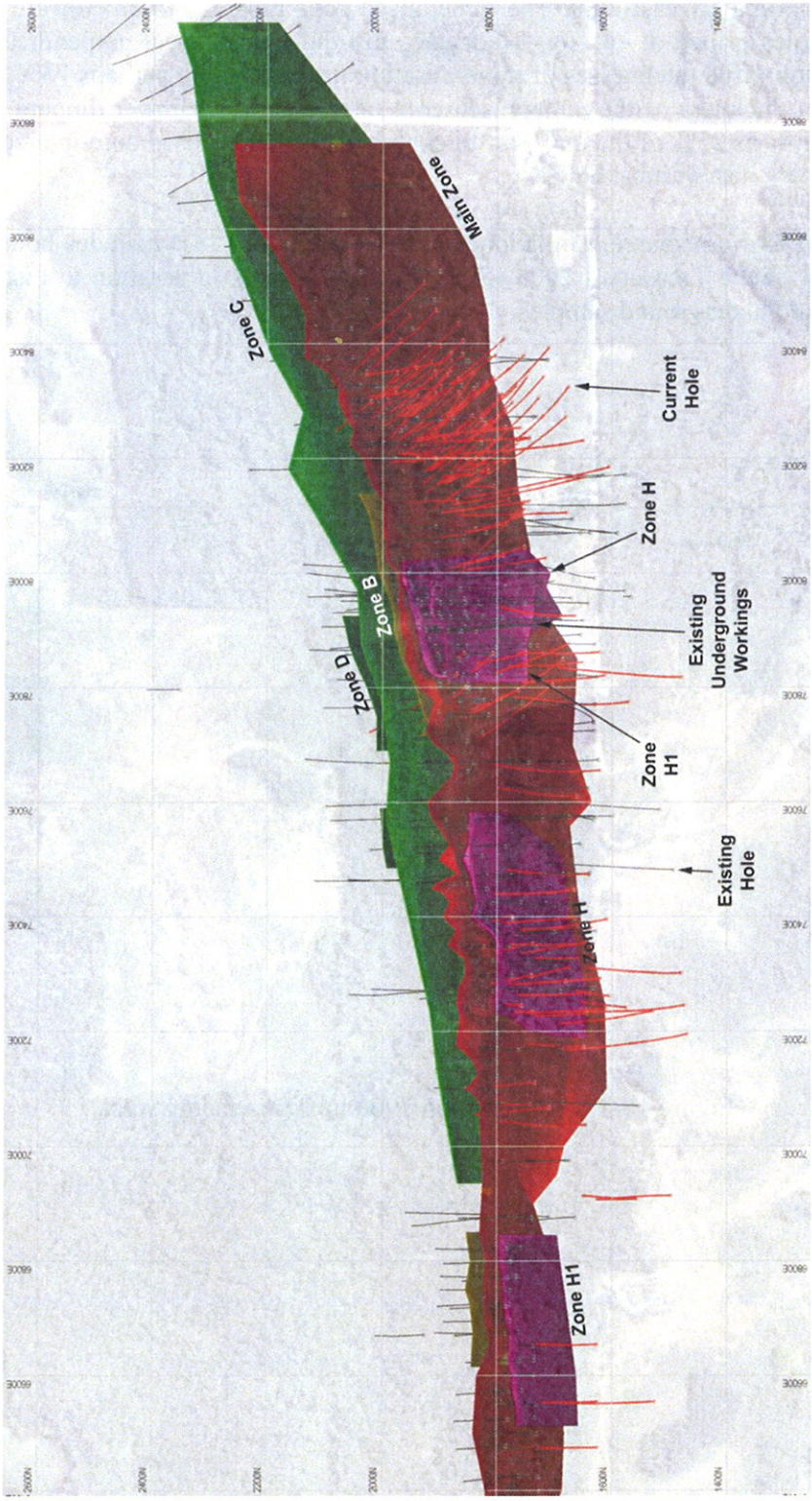


Figure 10-2: Plan view showing 2010-11 drill holes in red and older holes in black.

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 mineralisation 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 metres 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 mineralisation 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 A.C.A. 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 2010-2011 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 20th sample is a quarter core duplicate.

Table 11.1: Example of how CRMs, blanks and duplicates are inserted in 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

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 $\pm 3\sigma$ (standard deviation) and warning limits (WL) at accepted mean $\pm 2\sigma$. 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 (Figure 11-1 to Figure 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 mineralised zone returned values within STD±3SD (standard deviations). In sample batches where the standard failed within or near significant mineralisation (eg. a sample with greater than 0.5g/tonne Au), CCIC/Treasury staff elected to re-analyse the pulps from the preceeding 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 which is interpreted to be the result of a mislabelled higher grade standard, possible CDN-GS1D or CDN-CGS13.

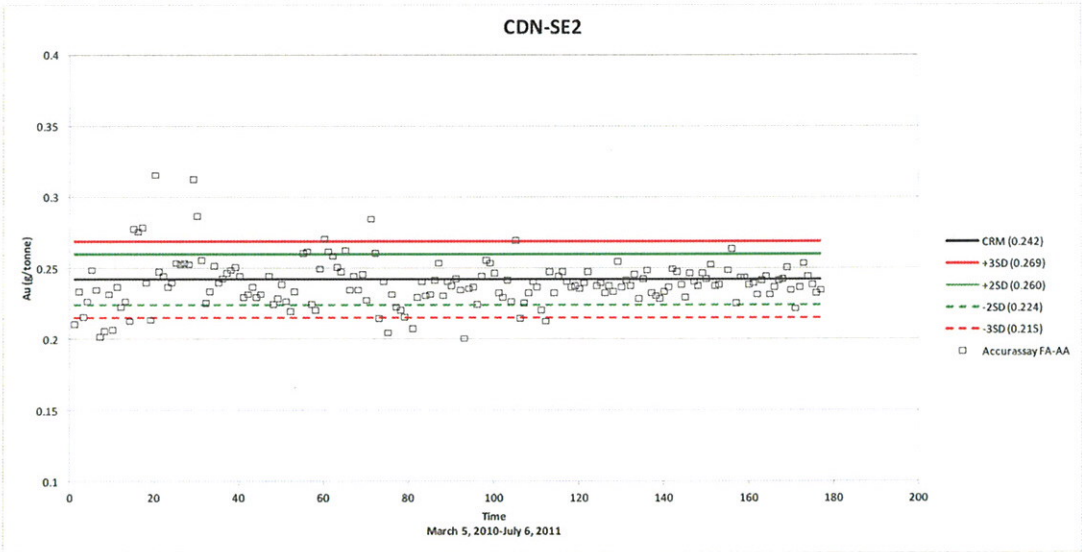


Figure 11-1: Standard CDN-SE2.

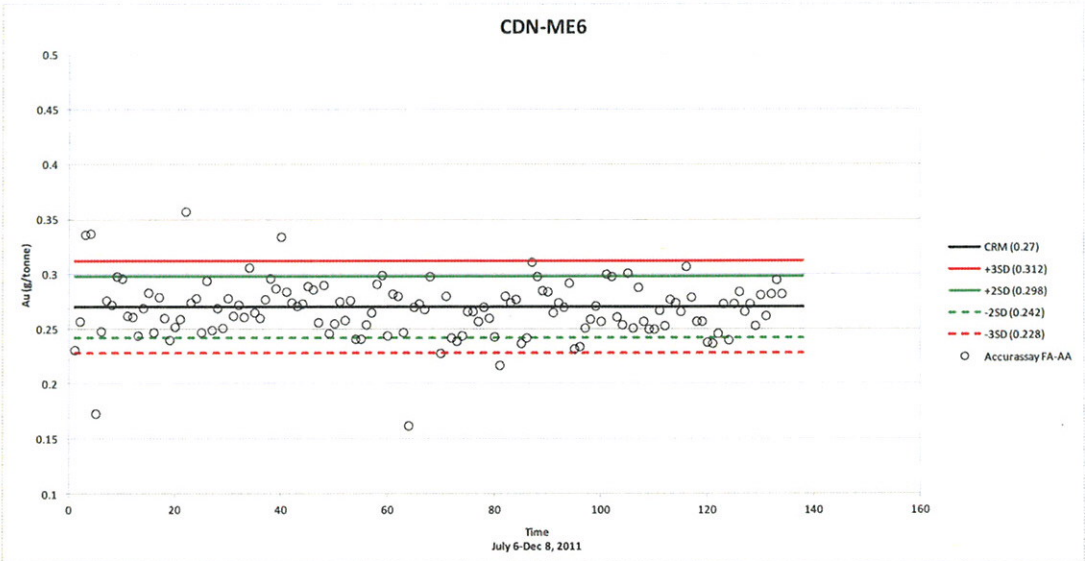


Figure 11-2: Standard CDN-ME6.

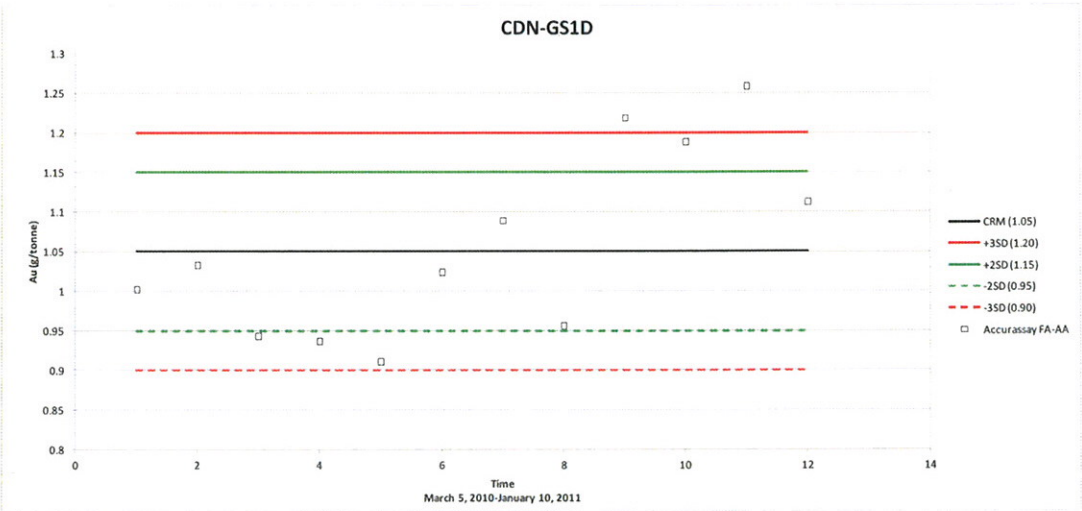


Figure 11-3: Standard CDN-GS1D.

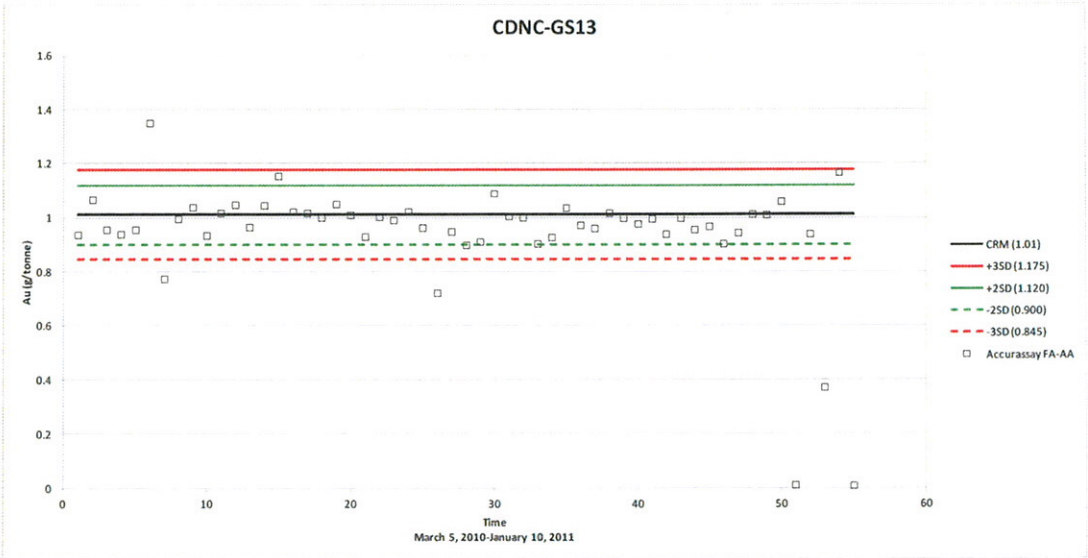


Figure 11-4: Standard CDN-CGS13

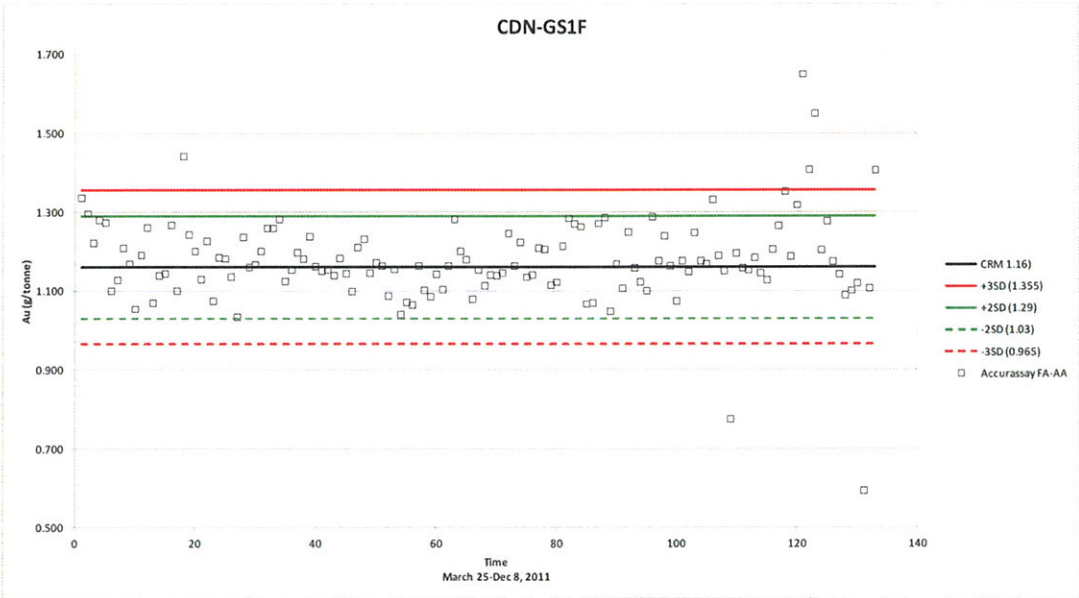


Figure 11-5: Standard CDNGS1F

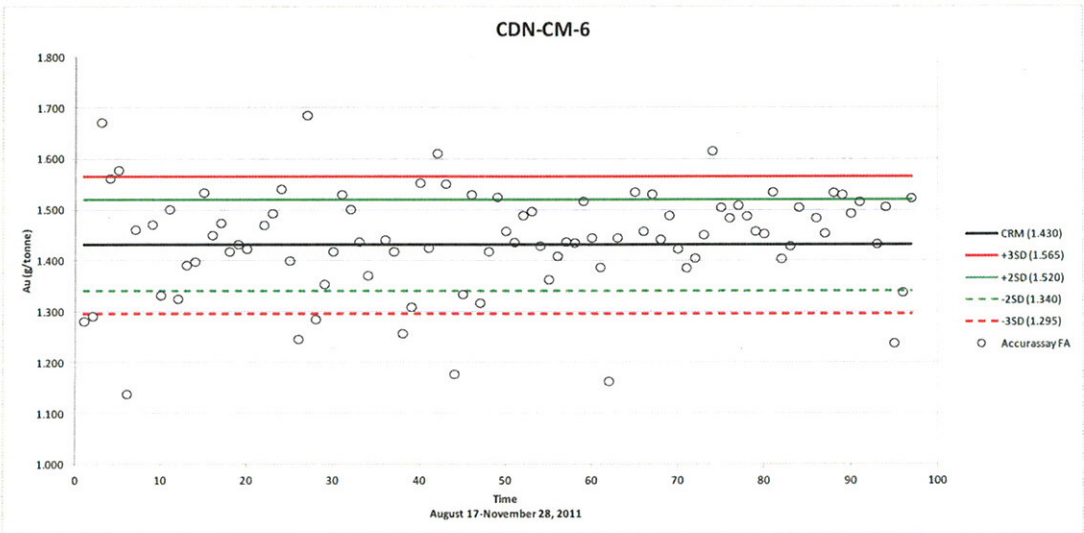


Figure 11-6: Standard CDN-CM6

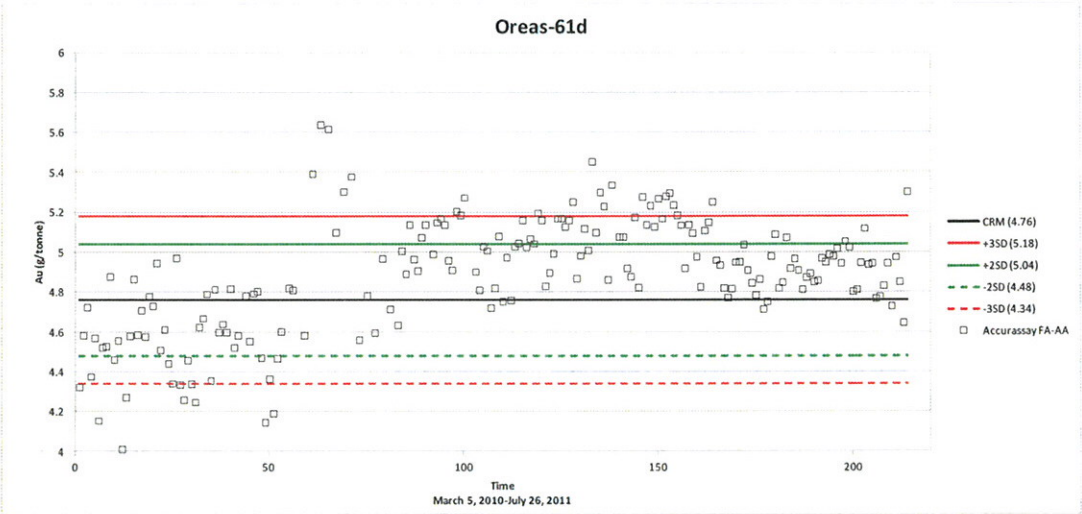


Figure 11-7: Standard Oreas-61D

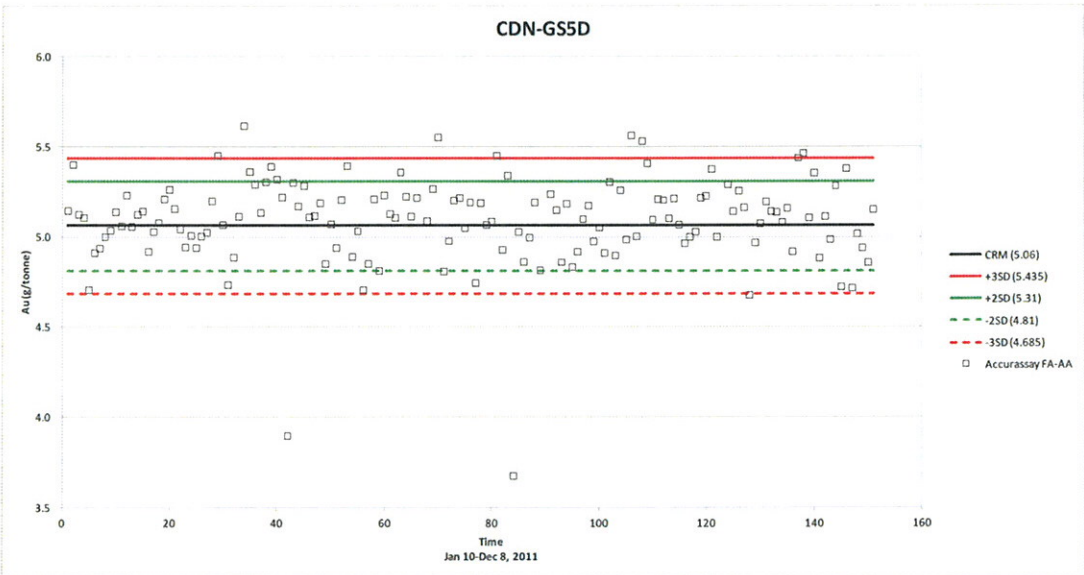


Figure 11-8: Standard CDN-GS5D

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 Figure 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.

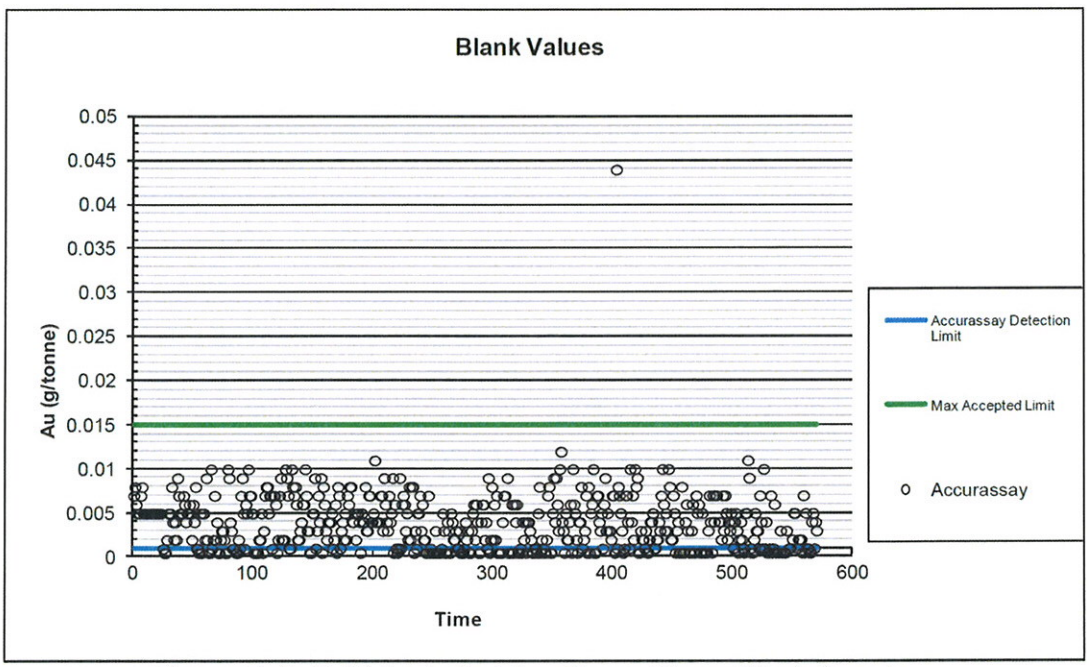


Figure 11-9. Gold Analytical Results vs. Time - Blank Samples.

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 mineralised zone.

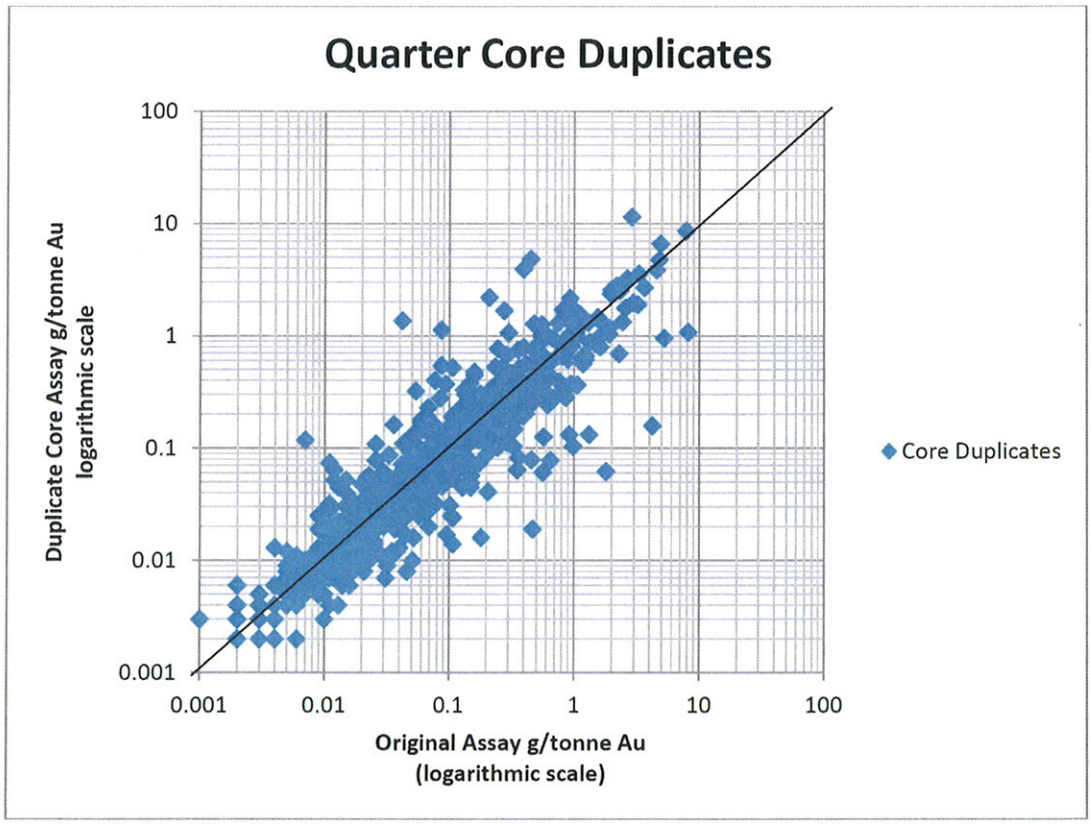


Figure 11-10: Plot of primary assays versus quarter core duplicate assays

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 Mr. 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 mineralised 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. Overlimit 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. ACA Howe ¼ Core Drill-Hole Duplicates *vs.* Original Samples

DDH	From (m)	To (m)	Length (m)	ACA 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 mineralised 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 mineralisation, 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.

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. The composite sample tested had a measured gold and silver feed grade of about 3.5 and 25 g/tonne,

respectively. Minor concentrations, all below 0.1 percent, of copper, lead, and zinc were also present in the sample.

The sulphide mineral content in the sample was about 2 percent. Most of the sulphide mineral was present as pyrite (about 90 percent) and pyrrhotite (about 10 percent). Traces of copper sulphides, galena, and sphalerite were also observed. The iron sulphide minerals were about 85 percent liberated, as estimated in two dimensions.

A single standard F.C. Bond ball mill work index test revealed the sample to be relatively soft. The Bond ball mill work index was 11.1 kWh/tonne.

Two alternate process flowsheets were investigated for processing the composite sample. Gravity concentration, followed by cyanidation of the gravity circuit tailing was one option. The other utilized gravity concentration, flotation of the gravity tailing, and cyanidation of the flotation concentrate.

The gravity/cyanidation flowsheet produced the best overall metallurgical performance with between 96 to 97 percent gold recovery to the combined gravity concentrate and 48 hour cyanidation liquor.

The primary grind sizing and target sodium cyanide oncentration did not impact significantly on gold extractions in the range tested for these variables. Some additional testing is required to confirm the optimal primary grind sizing. It may be possible to coarsen beyond 100μm K_{80}.

At a target NaCN concentration of 500 ppm, sodium cyanide consumption was 0.2 g/tonne, on the lower end of cyanide consumption for gold ores. Lime consumption was estimated at about 0.6 kg/tonne.

The gravity/flotation/cyanidation flowsheet produced lower gold recoveries, estimated at about 90 percent. Silver recoveries were higher using this flowsheet, mainly due to higher cyanidation extractions from the flotation concentrate.

Based on this testing, G&T concluded that the gravity/cyanidation flowsheet is the best flowsheet for processing the feed. G&T also recommended additional optimization work to confirm the primary grind sizing and consideration of testing on a suite of variability samples (G&T, 2011).

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[1] 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.

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

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

Hannington, M.D., Poulsen, K.H., Thompson, J.F.H., and Sillitoe, R.H. (1999) Volcanogenic gold in massive sulphide environment: Reviews in Economic Geology, v8, p. 325-356.

Hogg, G.M. (1996) A Report on the Zealand Twp. Gold Property of Laramide Resources Ltd., District of Kenora, Ontario, G.M. Hogg & Associates Ltd., July 9, 1996, 14p. (plus Appendix and Map).

Hogg, G.M. (2002) A Report on the Goliath Project of Laramide Resources Ltd., Zealand Twp., Ontario, G.M. Hogg & Associates Ltd., June 17, 2002, 26p. (plus Appendix and Map).

Huston, D.L. (2000) Gold in volcanic-hosted massive sulphide deposits; distribution, genesis, and exploration, In Hagemann, S.G. ed., Gold in 2000: Reviews in Economic Geology, v13, p.401-426.

Inmet Mining Corporation (2003) Press Release, www.inmet-mining.com dated March 26, 2003.

Inmet Mining Corporation (2007) Annual Review, 37 p.

JVX Ref: 8-32 (October, 2008). Report on a Spectral IP/Resistivity Survey Goliath Project, Thunder Lake Property Dryden/Wabigoon Area, Ontario – Treasury Metals Inc.

JVX Ref: P-620 (March 24th, 2008). Proposal for IP/Resistivity Surveys and Magnetometer Surveys on the Goliath Project. Dryden/Wabigoon Area, Northwestern Ontario. Contract between Treasury Metals Inc and JVX Ltd. Contact: Blaine Webster.

Kirkham, R.V., and Sinclair, W.D., 1995. Porphyry copper, gold, molybdenum, tungsten, tin, silver, in Eckstrand, O.R., Sinclair, W.D., and Thorpe, R.I., eds., Geology of Canadian Mineral Deposit Types: Geological Survey of Canada, Geology of Canada, no. 8, p. 421-446.

Laramide Resources Ltd. (2007a) 2006 Annual Report, www.laramide.com.

Laramide Resources Ltd. (2007b) Press Release, www.laramide.com dated September 24, 2007.

Laramide Resources Ltd. (2007c) Press Release, www.laramide.com dated October 4, 2007.

MNDM (1999) West Thunder Lake Project – Dryden; Ministry of Northern Development and Mines, Mine Rehabilitation, Examples of Rehabilitated Sites online at http://www.mndm.gov.on.ca/mndm/mines/mg/rehab/thunderlake_e.asp (November 14, 2007).

Ontario Geological Survey (1991) Brownridge-Zealand Townships Area (NTS 52F/15 SE), Kenora Mining Division, Geological Data Inventory Folio 579 (GDIF 579); Resident Geologist's office, Kenora.

Page, R. (1993) Report on the 1993 Exploration Program on the Thunder Lake East Property (Cameco Option) Dryden Area, Ontario; Report No. 1237NB, Teck Exploration Ltd., 8p.

Page, R. (1994) Report on the Winter 1994 Exploration Program, Thunder Lake West Project, Zealand Township, Ontario (NTS 52 F/15), Report No. 1249NB, Teck Exploration Ltd., 19p.

Page, R. (1995) Report on the 1994 Exploration Program, Thunder Lake West Project, Zealand Township, Ontario, Part 1 (NTS 52 F/15), Report No. 1263NB, Teck Exploration Ltd., 41p.

Page, R. and Waqué, P. (1998) Report on the 1997 Exploration Program on the Thunder Lake West Project, Zealand Township, northwestern Ontario, Part 1. (NTS 53 F/15); Report No. 1303NB, Teck Exploration Ltd..

Page, R., Waqué, P. and Galway, C. (1999a) Report on the 1998 Drilling Program on the Thunder Lake West Project, Zealand Township, Northwestern Ontario (NTS 52F/15); Report No. 1319NB, Teck Exploration Ltd.

Page, R., Stewart, R., Waqué, P. and Galway, C. (1999b) Report on the 1998 Underground Exploration and Bulk Sampling Program, Thunder Lake West Project, Zealand Township, Northwestern Ontario (NTS 52F/15); Report No. 1321NB, Teck Exploration Ltd.

Percival, J.A. and Easton, R.M. (2007a) Geology of the Canadian Shield in Ontario: An Update; Ontario Geological Survey, Open File Report 6196, Geological Survey of Canada, Open File 5511, Ontario Power Generation, Report 06819-REP-01200-10158-R00, 65p.

Percival, J.A. and Easton, R.M. (2007b) Geology of the Canadian Shield in Ontario: An Update; Ontario Geological Survey, Miscellaneous Release—Data 216 (MRD216).

Poulsen, K.H., and Hannington, M.D. (1996) Volcanic-associated massive sulphide gold, In Eckstrand, O.R., Sinclair, W.D., and Thorpe, R.I., eds., Geology of Canadian mineral deposit types: Geology of Canada, v8, p.183-196.

Poulsen, K.H., Robert, F. and Dubé; B. (2000) Geological Classification of Canadian Gold Deposits: Geological Survey of Canada Bulletin 540, 106p.

Price Waterhouse Coopers. (2009) Gold Medal Performance, 2009 gold price survey report

Rainsford, D., Muir, T. (2006) Ontario Geophysics Overview. Poster presented at Ontario Exploration and Geoscience Symposium Dec 12-13, 2006. Ontario Geological Survey.

Ravnaas, C., Raoul, A. and McDonald, J. (2007) Report of Activities 2006, Resident Geologist Program, Red Lake Regional Geologist Report: Kenora District; In Report of Activities 2006, Resident Geologist Program, Red Lake Regional Geologist Report: Red Lake and Kenora Districts, Ontario Geological Survey, Open File Report 6200, p.25-29.

Roed, M.A. (1980) Northern Ontario Engineering Geology Terrain Study 22. Wabigoon Lake Area with Map 5059, NOEGTS 22, Ministry of Northern Development and Mines.

Roy, W.D. and Trinder, I.D. (2008) Report on the Goliath Project - Kenora Mining Division, Northwestern Ontario, Canada for Treasury Metals Incorporated. A.C.A. Howe International Limited

Roy, W.D., Hannon, P.J., Thorton, E.W.J., and Trinder, I.D. (2010) Technical Report and Preliminary Economic Assessment on the Goliath Gold Project – Kenora Mining Division, Northwestern Ontario, Canada for Treasury Metals Incorporated. A.C.A. Howe International Limited

Saeki, Y., and Date, J., 1980, Computer applications to the alteration data of the footwall dacite lava at the Ezuri kuroko deposits, Akita Prefecture: Mining Geology, 30, 4, 241-250.

Satterly, J. (1941) Geology of the Dryden-Wabigoon area; Ontario Department of Mines, Annual Report, v.50, pt.2, accompanied by Map 50e, scale 1:63 360.

Sills, T.K. (2007) Laramide Resources Ltd., Acquisition Opportunity: Comments on the Thunder Lake Property of Teck, Unpublished report by OreVal Consulting Services Ltd., 42p.

Stewart, R. (1995) Report on the Winter 1995 Exploration Program, Thunder Lake West Program, Zealand Township, Ontario, Report 1264NB, Teck Exploration Ltd., 11p.

Stewart, R. (1996) Report on the Winter 1995-96 Exploration Program, Thunder Lake West Project, Zealand Township, Ontario (NTS 52/F15); Report 1276NB, Teck Exploration Ltd., 16p.

Stewart, R., Page, R., and Waqué, P. (1997) Report on the Summer/Fall 1996 Exploration Program, Thunder Lake West Project, Zealand Township, Ontario (NTS 52F/'15); Report No. 1282NB, Teck Exploration Ltd.

Wetherup, S, Kelso, I. (February 1st, 2008) Independent Technical Report: Thunder Lake Property, Goliath Project – Treasury Metals Inc. Caracle Creek International Consulting – Canada.

Wetherup, S. (April 5th, 2008) Structural Summary Report – Thunder Lake Property. Caracle Creek International Consulting – Canada (Abbotsford).

28 DATE AND SIGNATURE PAGE

This report titled "Technical Report and Mineral Resource Update on the Goliath Project - Kenora Mining Division, Northwestern Ontario, Canada" for Treasury Metals Incorporated with an effective date of November 9, 2011, was prepared and signed by the following authors:

{SIGNED}
[*William D. Roy*]

Dated Dec. 23, 2011 at Halifax, Nova Scotia

William D. Roy, M.A.Sc., P.Eng.
Associate Consultant Engineer
A.C.A. Howe International Limited

{SIGNED}
[*Ian D. Trinder]*

Dated Dec. 23, 2011 at Toronto, Ontario

Ian D. Trinder, M.Sc., P.Geo.
Associate Geologist
A.C.A. Howe International Limited

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., do hereby certify that:

1) I am an Associate Mining Engineer with ACA 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). I am a member of the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") and of the Prospectors and Developers Association of Canada ("PDAC").

4) I have worked as a mining engineer for more than twelve 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: "Technical Report and Mineral Resource Update On The Goliath Gold Project - Kenora Mining Division, Northwestern Ontario, Canada" for Treasury Metals Incorporated dated November 9th, 2011, (the "Technical Report"). I am responsible for Section 17: Mineral Resources and portions of Sections 1: Executive Summary; Section 25: Interpretations and Conclusions and; Section 26: Recommendations. 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 Incorporated, 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, Northwestern Ontario, Canada for Treasury Metals Incorporated." And 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, Northwestern Ontario, Canada for Treasury Metals Incorporated"

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 Incorporated, 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: November 9th, 2011
DATED this 23rd Day of December 2011.

{SIGNED and SEALED }
[William D. Roy]
William Douglas Roy, M.A.Sc., P. Eng.
Associate Mining Engineer
ACA Howe International Limited

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

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

1. I reside at 4185 Taffey Crescent, Mississauga, Ontario, L5L 2A6.
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 20 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: "Technical Report and Mineral Resource Update on the Goliath Gold Project - Kenora Mining Division, Northwestern Ontario, Canada" for Treasury Metals Incorporated dated November 9th, 2011, (the "Technical Report"). I am responsible for Sections 2 to 13; 15 to 24 and 27 and portions of Sections 1, 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 Incorporated, 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, Northwestern Ontario, Canada for Treasury Metals Incorporated." And 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, Northwestern Ontario, Canada for Treasury Metals Incorporated"
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 Incorporated, 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: November 9th, 2011
DATED this 23rd Day of December 2011.

{SIGNED and SEALED }
[Ian D. Trinder]

Ian D. Trinder, M.Sc., P. Geo.
Senior Geologist
ACA Howe International Limited

Appendix 1
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	2015-Feb-26	1	16	A
HARTMAN	1144514	1991-Feb-26	2014-Feb-26	1	16	A
HARTMAN	1144515	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144516	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144517	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144518	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144519	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144520	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144521	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144522	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144523	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144524	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144525	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144526	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144527	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144528	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144529	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144530	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144531	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144532	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144533	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144534	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144535	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144536	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144537	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144538	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144539	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144540	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144541	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144542	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144543	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144544	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144545	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144546	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144547	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144548	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144549	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144550	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144551	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144552	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144553	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144554	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1144555	1991-Jan-26	2013-Jan-26	1	16	A
HARTMAN	1144556	1991-Feb-26	2012-Feb-26	1	16	A
HARTMAN	1210898	1996-Apr-02	2012-Apr-02	1	16	A
HARTMAN	1211082	1996-Apr-02	2012-Apr-02	4	64	A
HARTMAN	1247442	2007-Aug-21	2013-Aug-21	4	64	A
HARTMAN	3017886	2009-Jul-10	2014-Jul-10	4	64	A
HARTMAN	3017887	2009-Jul-10	2014-Jul-10	12	192	A
HARTMAN	3017888	2009-Jul-10	2014-Jul-10	1	16	A
HARTMAN	3017889	2009-Jul-10	2014-Jul-10	12	192	A
HARTMAN	3017890	2009-Jul-10	2013-Jul-10	8	128	A
HARTMAN	4211247	2007-Aug-21	2013-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	2013-Aug-21	8	128	A
HARTMAN	4211249	2007-Aug-21	2013-Aug-21	8	128	A
HARTMAN	4211250	2007-Aug-21	2013-Aug-21	4	64	A
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
ZEALAND	1106347	1989-Oct-13	2015-Oct-13	1	16	A
ZEALAND	1106348	1989-Oct-13	2015-Oct-13	1	16	A
ZEALAND	1106349	1989-Oct-13	2012-Oct-13	1	16	A
ZEALAND	1106350	1989-Oct-13	2012-Oct-13	1	16	A
ZEALAND	1106351	1989-Oct-13	2012-Oct-13	1	16	A
ZEALAND	1106352	1989-Oct-13	2012-Oct-13	1	16	A
ZEALAND	1119531	1989-Oct-26	2016-Oct-26	1	16	A
ZEALAND	1119532	1989-Oct-26	2016-Oct-26	1	16	A
ZEALAND	1119537	1989-Oct-26	2016-Oct-26	1	16	A
ZEALAND	1119538	1989-Oct-26	2016-Oct-26	1	16	A
ZEALAND	1119541	1989-Oct-26	2013-Oct-26	1	16	A
ZEALAND	1119542	1989-Oct-26	2012-Oct-26	1	16	A
ZEALAND	1119543	1989-Oct-26	2012-Oct-26	1	16	A
ZEALAND	1119544	1989-Oct-26	2012-Oct-26	1	16	A
ZEALAND	1119545	1989-Oct-26	2012-Oct-26	1	16	A
ZEALAND	1119546	1989-Oct-26	2012-Oct-26	1	16	A
ZEALAND	1119547	1989-Oct-26	2012-Oct-26	1	16	A
ZEALAND	1119548	1989-Oct-26	2012-Oct-26	1	16	A
ZEALAND	1119549	1989-Oct-26	2016-Oct-26	1	16	A
ZEALAND	1119550	1989-Oct-26	2016-Oct-26	1	16	A
ZEALAND	1119551	1989-Oct-26	2012-Oct-26	1	16	A
ZEALAND	1119552	1989-Oct-26	2012-Oct-26	1	16	A
ZEALAND	1119553	1989-Oct-26	2012-Oct-26	1	16	A
ZEALAND	1119554	1989-Oct-26	2012-Oct-26	1	16	A
ZEALAND	1119555	1989-Oct-26	2012-Oct-26	1	16	A
ZEALAND	1119556	1989-Oct-26	2012-Oct-26	1	16	A
ZEALAND	1119557	1989-Oct-26	2012-Oct-26	1	16	A
ZEALAND	1119558	1989-Oct-26	2012-Oct-26	1	16	A
ZEALAND	1119559	1989-Oct-26	2016-Oct-26	1	16	A
ZEALAND	1119560	1989-Oct-26	2016-Oct-26	1	16	A
ZEALAND	1119561	1989-Oct-26	2012-Oct-26	1	16	A
ZEALAND	1119562	1989-Oct-26	2012-Oct-26	1	16	A
ZEALAND	1119563	1989-Oct-26	2012-Oct-26	1	16	A
ZEALAND	1119564	1989-Oct-26	2012-Oct-26	1	16	A
ZEALAND	1119565	1989-Oct-26	2012-Oct-26	1	16	A
ZEALAND	1119566	1989-Oct-26	2012-Oct-26	1	16	A
ZEALAND	1119567	1989-Oct-26	2012-Oct-26	1	16	A
ZEALAND	1119568	1989-Oct-26	2012-Oct-26	1	16	A
ZEALAND	1144557	1991-Feb-26	2012-Feb-26	1	16	A
ZEALAND	1144558	1991-Feb-26	2012-Feb-26	1	16	A
ZEALAND	1144559	1991-Feb-26	2012-Feb-26	1	16	A
ZEALAND	1144560	1991-Feb-26	2012-Feb-26	1	16	A
ZEALAND	1144561	1991-Feb-26	2012-Feb-26	1	16	A
ZEALAND	1144562	1991-Feb-26	2012-Feb-26	1	16	A
ZEALAND	1144563	1991-Feb-26	2012-Feb-26	1	16	A
ZEALAND	1144564	1991-Feb-26	2012-Feb-26	1	16	A
ZEALAND	1144565	1991-Feb-26	2012-Feb-26	1	16	A
ZEALAND	1144566	1991-Feb-26	2012-Feb-26	1	16	A
ZEALAND	1144567	1991-Feb-26	2012-Feb-26	1	16	A

Township/Area	Claim Number	Claim Recording Date	Claim Due Date	Claim Units	Area (ha)	Status
ZEALAND	1144568	1991-Feb-26	2012-Feb-26	1	16	A
ZEALAND	1144569	1991-Feb-26	2012-Feb-26	1	16	A
ZEALAND	1144570	1991-Feb-26	2012-Feb-26	1	16	A
ZEALAND	1144573	1991-Feb-26	2012-Feb-26	1	16	A
ZEALAND	1144574	1991-Feb-26	2012-Feb-26	1	16	A
ZEALAND	1144575	1991-Feb-26	2012-Feb-26	1	16	A
ZEALAND	1144576	1991-Feb-26	2012-Feb-26	1	16	A
ZEALAND	1144577	1991-Feb-26	2012-Feb-26	1	16	A
ZEALAND	1144578	1991-Feb-26	2012-Feb-26	1	16	A
ZEALAND	1144579	1991-Feb-26	2012-Feb-26	1	16	A
ZEALAND	1144580	1991-Feb-26	2012-Feb-26	1	16	A
ZEALAND	1144581	1991-Feb-26	2012-Feb-26	1	16	A
ZEALAND	1144582	1991-Feb-26	2012-Feb-26	1	16	A
ZEALAND	1144583	1991-Feb-26	2012-Feb-26	1	16	A
ZEALAND	1144584	1991-Feb-26	2012-Feb-26	1	16	A
ZEALAND	1144585	1991-Feb-26	2012-Feb-26	1	16	A
ZEALAND	1144586	1991-Feb-26	2012-Feb-26	1	16	A
ZEALAND	1144587	1991-Feb-26	2012-Feb-26	1	16	A
ZEALAND	1144588	1991-Feb-26	2012-Feb-26	1	16	A
ZEALAND	1145300	1992-Jun-23	2012-Jun-23	4	64	A
ZEALAND	1145301	1992-Jun-23	2012-Jun-23	2	32	A
ZEALAND	3017934	2008-May-21	2012-May-21	4	64	A
ZEALAND	3017936	2008-May-21	2013-May-21	5	80	A
ZEALAND	3017937	2008-May-21	2012-May-21	9	144	A
ZEALAND	3017938	2008-May-26	2012-May-26	2	32	A
ZEALAND	3017939	2008-Jul-04	2012-Jul-04	6	96	A
ZEALAND	3017940	2008-Sep-10	2013-Sep-10	4	64	A
ZEALAND	3017941	2008-Oct-10	2013-Oct-10	4	64	A
ZEALAND	4211252	2007-Sep-06	2012-Sep-06	8	128	A
TOTAL	**137**			**254**	**4064**	

Notes: Source: Ontario Provincial Recording Office (MNDMF), November 9, 2011

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	16.00	M+SR
Zealand[1]	Johnson	15401	N ½ of S ½ Lot 5, Con IV	32.00	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	NE ¼ of 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	Lot 8, Con III	31.1	MRO
Zealand[4]	Schultz	13492	Lot 7, Con III	57.0	M+SR
Zealand	Brisson	23R2434	Part of Broken Lot 9, Con IV	40.8711	SRO
TOTAL:		**17**		**763.9111**	

[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

Table 1-3. Private Land

TOWNSHIP	PARTY	PARCEL	LOT/CONCESSION	AREA (ha)	*RIGHTS
Zealand	Private Land	Dryden Tree Nursery		101	M+SR
TOTAL:		**1**		**101**	

*MRO=Mineral Rights only; SRO = Surface Rights only; M+SR=Mineral and Surface Rights

Appendix 2: Number of samples in assay file "dh-Assay.dat"

HOLE	Number of Samples in Assay File
TL001	74
TL002	79
TL003	30
TL004	118
TL005	75
TL006	11
TL007	23
TL008	4
TL009	66
TL010	257
TL011	211
TL012	250
TL013	116
TL014	67
TL015	16
TL016	50
TL017	34
TL018	119
TL019	35
TL020	84
TL021	104
TL022	36
TL023	235
TL024	33
TL025	112
TL026	128
TL027	133
TL028	99
TL029	105
TL030	58
TL031	84
TL032	89
TL033	86
TL034	104
TL035	113
TL036	132
TL037	32
TL038	12
TL039	1
TL039A	112
TL040	108
TL041	215
TL042	54
TL043	138
TL043W1	30
TL043W2	36
TL043W3	53
TL044	69
TL044W1	32
TL044W2	40
TL044W3	51
TL045	41
TL046	35
TL047	32
TL048	48
TL049	53
TL050	48
TL051	35

HOLE	Number of Samples in Assay File
TL052	51
TL053	49
TL054	36
TL055	74
TL056	52
TL057	25
TL058	67
TL059	57
TL060	41
TL061	44
TL062	50
TL063	46
TL064	44
TL065	82
TL066	151
TL067	56
TL068	46
TL069	52
TL070	57
TL071	80
TL072	48
TL073	70
TL074	61
TL075	45
TL076	83
TL077	45
TL078	51
TL079	51
TL080	69
TL081	58
TL082	67
TL083	57
TL0835-	53
TL084	74
TL085	39
TL086	68
TL087	66
TL088	68
TL089	52
TL090	78
TL091	75
TL092	84
TL093	115
TL094	67
TL095	35
TL096	83
TL096W	42
TL097	39
TL098	27
TL099	74
TL100	169
TL101	87
TL102	147
TL103	79
TL104	72
TL105	69
TL106	52
TL107	60

HOLE	Number of Samples in Assay File
TL108	163
TL109	128
TL110	53
TL111	27
TL112	32
TL113	38
TL114	29
TL115	34
TL116	37
TL117	163
TL118	22
TL119	49
TL120	51
TL121	62
TL122	67
TL123	54
TL124	44
TL125	83
TL125W1	39
TL125W2	40
TL126	164
TL126W1	24
TL126W2	38
TL126W3	49
TL127	229
TL127W1	45
TL127W2	52
TL127W3	58
TL128	174
TL128W1	30
TL128W2	41
TL128W3	48
TL129	108
TL129W1	35
TL129W2	42
TL129W3	50
TL130	117
TL131	134
TL132	104
TL133	92
TL133W1	37
TL133W2	45
TL133W3	60
TL134	133
TL135	135
TL136	156
TL136W1	49
TL136W2	58
TL137	156
TL137W1	51
TL137W2	62
TL138	207
TL139	209
TL140	198
TL141	181
TL142	102
TL143	120
TL144	86

HOLE	Number of Samples in Assay File
TL145	72
TL146	57
TL147	72
TL148	39
TL149	112
TL150	72
TL151	154
TL152	139
TL153	99
TL154	89
TL155	164
TL156	80
TL157	125
TL158	135
TL159	97
TL160	160
TL161	102
TL162	101
TL163	152
TL164	107
TL165	159
TL166	99
TL167	209
TL168	83
TL169	152
TL170	87
TL170W1	25
TL171	80
TL172	123
TL173	125
TL174	76
TL175	87
TL176	56
TL177	83
TL178	55
TL179	50
TL180	71
TL181	53
TL182	64
TL183	108
TL184	103
TL185	86
TL186	86
TL187	92
TL188	77
TL189	51
TL190	46
TL191	47
TL192	45
TL193	48
TL194	51
TL195	48
TL196	88
TL197	123
TL198	151
TL199	79
TL200	166
TL201	128

HOLE	Number of Samples in Assay File
TL202	102
TL203	124
TL204	111
TL205	228
TL206	332
TL207	270
TL208	409
TL209	263
TL210	29
TL211	118
TL212	99
TL213	130
TL214	97
TL215	86
TL216	20
TL217	78
TL218	24
TL219	27
TL220	22
TL221	21
TL222	27
TL223	22
TL224	52
TL225	41
TL226	37
TL227	56
TL228	43
TL229	67
TL230	54
TL231	36
TL232	44
TL233	71
TL234	25
TL235	51
TL236	28
TL237	69
TL238	24
TL239	25
TL240	42
TL241	24
TL242	20
TL243	73
TL244	32
TL245	87
TL246	21
TL247	18
TL248	77
TL249	80
TL250	70
TL251	72
TL252	63
TL253	107
TL254	54
TL255	52
TL256	47
TL257	163
TL258	35
TL259	27

HOLE	Number of Samples in Assay File
TL260	45
TL261	51
TL262	140
TL263	26
TL264	144
TL265	121
TL266	103
TL267	98
TL268	74
TL269	115
TL270	39
TL271	118
TL272	75
TL273	61
TL274	39
TL275	28
TL276	49
TL277	70
TL0801	261
TL0802	185
TL0803	298
TL0804	153
TL0805	250
TL0806	203
TL0807	238
TL0808	166
TL0809	260
TL0810	168
TL0811	326
TL0812	214
TL0813	204
TL0814	170
TL0815	148
TL0816	161
TL0817	133
TL0818	75
TL0819	217
TL0820	323
TL0821	335
TL0822	267
TL0823	519
TL0824	284
TL0825	85
TL0826	218
TL0827	234
TL0828	234
TL0829	577
TL0830	132
TL0831	119
TL0832	277
TL0833	336
TL0834	379
TL0835	801
TL0836	343
TL0837	296
TL0838	90
TL0839	216
TL0840	218

HOLE	Number of Samples in Assay File
TL0841	159
TL0842	174
TL0843	98
TL0844	117
TL0845	154
TL0846	142
TL0847	108
TL0848	106
TL0849	105
TL0850	64
TL0851	202
TL0852	63
TL0853	33
TL0854	89
TL0855	81
TL0956	138
TL0957	218
TL0958	132
TL0959	102
TL0960	98
TL0961	103
TL0962	81
TL0963	91
TL0964	110
TL0965	106
TL0966	102
TL0967	122
TL0968	80
TL0969	68
TL0970	58
TL0971	75
TL0972	50
TL0973	52
TL0974	67
TL0975	118
TL0976	154
TL0977	120
TL0978	111
TL0979	96
TL0980	83
TL0981	82
TL0982	93
TL0983	67
TL0984	63
TL0985	105
TL0986	100
TL1087	251
TL1088	418
TL1089	345
TL1090	168
TL1091	184
TL1092	166
TL1093	140
TL1094	84
TL1095	55
TL1096	75
TL1097	81
TL1098	87

HOLE	Number of Samples in Assay File
TL1099	52
TL10100	45
TL10101	23
TL10102	113
TL10103	19
TL10104	70
TL10105	99
TL10106	52
TL10107	63
TL10108	140
TL10109	35
TL10110	64
TL10111	57
TL10112	71
TL10113	61
TL10114	124
TL10115	127
TL10116	173
TL10117	145
TL10118	70
TL10119	153
TL11120	86
TL11121	82
TL11122	82
TL11123	101
TL11124B	119
TL11125	83
TL11126	94
TL11127	75
TL11128	88
TL11129	106
TL11130	122
TL11131	108
TL11132	143
TL11133	87
TL11134	115
TL11135	107
TL11136	102
TL11137	99
TL11138	96
TL11139	65
TL11140	112
TL11141	64
TL11142	129
TL11143	98
TL11144	106
TL11145	99
TL11146	83
TL11147	109
TL11148	117
TL11149	109
TL11150	135
TL11151	121
TL11152	155
TL11153	116
TL11154	139
TL11155	130
TL11156	28

HOLE	Number of Samples in Assay File
TL11157	103
TL11158	51
TL11159	35
TL11160	61
TL11161	52
TL11162	84
TL11163	37
TL11164	48
TL11165	110
TL11166	88
TL11167	112
TL11168	82
TL11169	142
TL11170	139
TL11171	72
TL11172	135
TL11173	113
TL11174	105
TL11175	171
TL11176	123
TL11177	137
TL11178	143
TL11179	114
TL11180	119
TL11181	205
TL11182	130
TL11183	115
TL11184	87
TL11185A	136
TL11186A	123
TL11187	143
TL11188	195
TL11189	168
TL11190	158
TL11191	143
TL11192	79
TL11193	120
TL11194A	78
TL11195	90
TL11196	90
TL11197	70
TL11198	52
TL11199	130
TL11200	100
TL11201	136
TL11202	115
TL11203	129
TL11204A	142
TL11205	189
TL11206A	133
TL11207	182
TL11208	130
TL11209A	111
TL11210	161
TL11211	112
TL11212	143
TL11213	166
TL11214	218

HOLE	Number of Samples in Assay File	HOLE	Number of Samples in Assay File	HOLE	Number of Samples in Assay File
TL11215	138	UG11	3	UG69	3
TL11216	180	UG12	4	UG70	3
TL11217	235	UG13	2	UG71	1
TL11218	192	UG14	2	UG72	1
TL11219	126	UG15	4	UG73	3
TL11220	213	UG16	4	UG74	3
TL11221	54	UG17	3	UG75	4
TL11222	178	UG18	3	UG76	3
TL11223	144	UG19	3	UG77	4
TL11226	113	UG20	3	UG78	3
TL11228	83	UG21	3	UG79	2
TLE11	103	UG22	3	UG80	3
TLE12	21	UG23	3	UG81	3
TLE13	38	UG24	3	UG82	3
TLE14	41	UG25	2	UG83	3
TLE15	185	UG26	3	UG84	3
TLE16	109	UG27	3	UG85	3
TLE17	132	UG28	3	UG86	1
TLE18	101	UG29	3	UG87	1
TLE19	110	UG30	3	UG88	4
TLE20	129	UG31	4	UG89	4
TLE21	111	UG32	4	UG90	4
TLE22	78	UG33	3	UG91	4
TLE23	126	UG34	3	UG92	3
TLE24	81	UG35	4	UG93	3
TLE25	122	UG36	4	UG94	3
TLE26	129	UG37	4	UG95	3
TLE27	120	UG38	4	UG96	3
TLE28	96	UG39	4	UG97	4
TLE30	99	UG40	4	UG98	3
TLE32	61	UG41	4	UG99	4
TLE33	123	UG42	4	UG100	3
TLOB01	1	UG43	4	UG101	3
TLOB02	1	UG44	3	UG102	3
TLOB03	1	UG45	3	UG103	4
TLOB04	1	UG46	3	UG104	3
TLOB05	1	UG47	4	UG105	3
TLOB06	1	UG48	3	UG106	3
TLOB07	1	UG49	3	UG107	3
TLOB08	1	UG50	3	UG108	3
TLOB09	1	UG51	4	UG109	3
TLOB10	1	UG52	4	UG110	3
TLOB11	1	UG53	3	UG111	3
TLOB12	1	UG54	3	UG112	3
TLOB13	1	UG55	4	UG113	3
TLOB14	1	UG56	4	UG114	4
TLOB15	1	UG57	4	UG115	3
TLOB16	1	UG58	3	UG116	3
UG1	4	UG59	3	UG117	4
UG2	3	UG60	3	UG118	4
UG3	5	UG61	3	UG119	2
UG4	7	UG62	3	UG120	4
UG5	6	UG63	3	UG121	4
UG6	6	UG64	4	UG122	4
UG7	4	UG65	3	UG123	4
UG8	3	UG66	4	UG124	3
UG9	3	UG67	3	UG125	3
UG10	3	UG68	3	UG126	3

HOLE	Number of Samples in Assay File
UG127	3
UG128	3
UG129	4
UG130	4
UG131	5
UG132	3
UG133	3
UG134	3
UG135	4
UG136	3
UG137	3
UG138	3
UG139	3
UG140	3
UG141	3
UG142	3
UG143	3
UG144	3
UG145	3
UG146	4
UG147	4
UG148	3
UG149	4
UG150	4
UG151	4
UG152	3
UG153	3
UG154	3
UG155	3
UG156	4
UG157	4
UG158	3
UG159	3
UG160	3
UG161	3
UG162	3
UG163	3
UG164	3
UG165	3
UG166	3
UG167	3
UG168	3
UG169	4
UG170	3
UG171	3
UG172	3
UG173	3
UG174	3
UG175	4
UG176	3
UG177	3
UG178	3
UG179	3
UG180	3
UG181	4
UG182	1
UG183	3
UG184	2

HOLE	Number of Samples in Assay File
UG185	1
UG186	1
UG187	1
UG188	2
UG189	2
UG190	1
UG191	1
UG192	1
UG193	2
UG194	2
UG195	1
UG196	1
UG197	2
UG198	2
UG199	1
UG200	2
UG201	2
UG202	1
UG203	2
UG204	2
UG205	1
UG206	1
UG207	1
UG208	2
UG209	2
UG210	1
UG211	2
UG212	1
UG213	22
UG214	12
Total	**58,481**

Appendix 3: Resource estimation notes

Wireframing

Using the assay file, the hangingwall and footwall contacts were calculated and saved in the point file "Contacts - Oct 2011 Resource Update.dat". These contacts had hole names and zones associated with them.

The contacts were viewed in long section. Supplemental contacts were added down-dip and along strike, approximately 150 m down-dip from the deepest intercepts. A boundary string joined those contacts, which did not have holes associated with them.

Separate DTM surfaces were made using the hanging wall and footwall contacts. The boundary strings constrained the surfaces. The surfaces were saved as type "HWs and FWs". The "surface to solid" function was used to form a solid wireframe from the two surfaces.

This was carried out for each zone.

The process did not go exactly smoothly the first time. When sections were viewed, there were some places where the hanging wall and footwall surfaces overlapped and crossed. To solve these problems, supplemental contacts were added and the procedure was repeated.

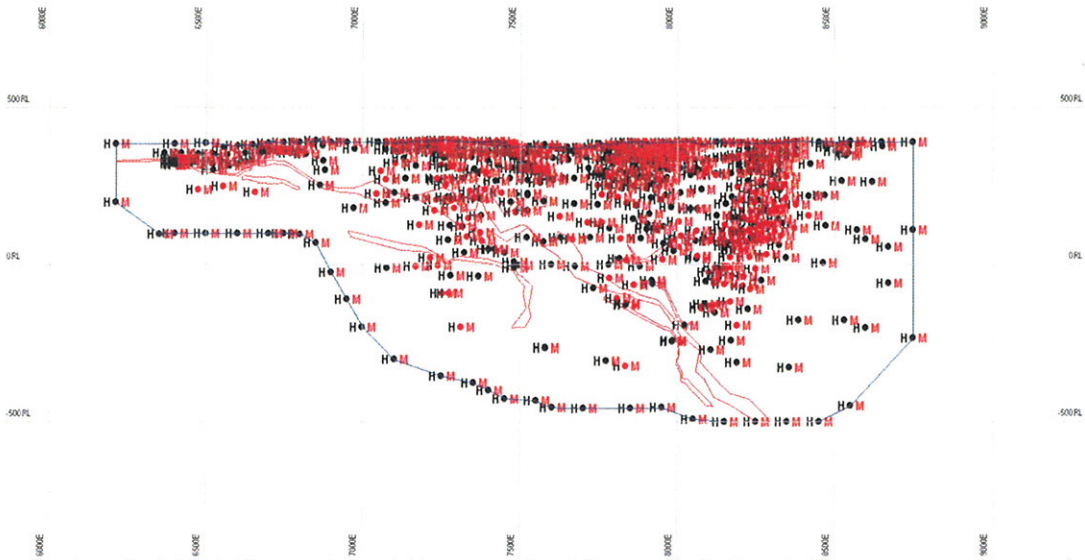


Figure A3-1: Wireframing the Main zone hanging wall. Red points represent contacts from the newer holes.

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 metallics 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 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).

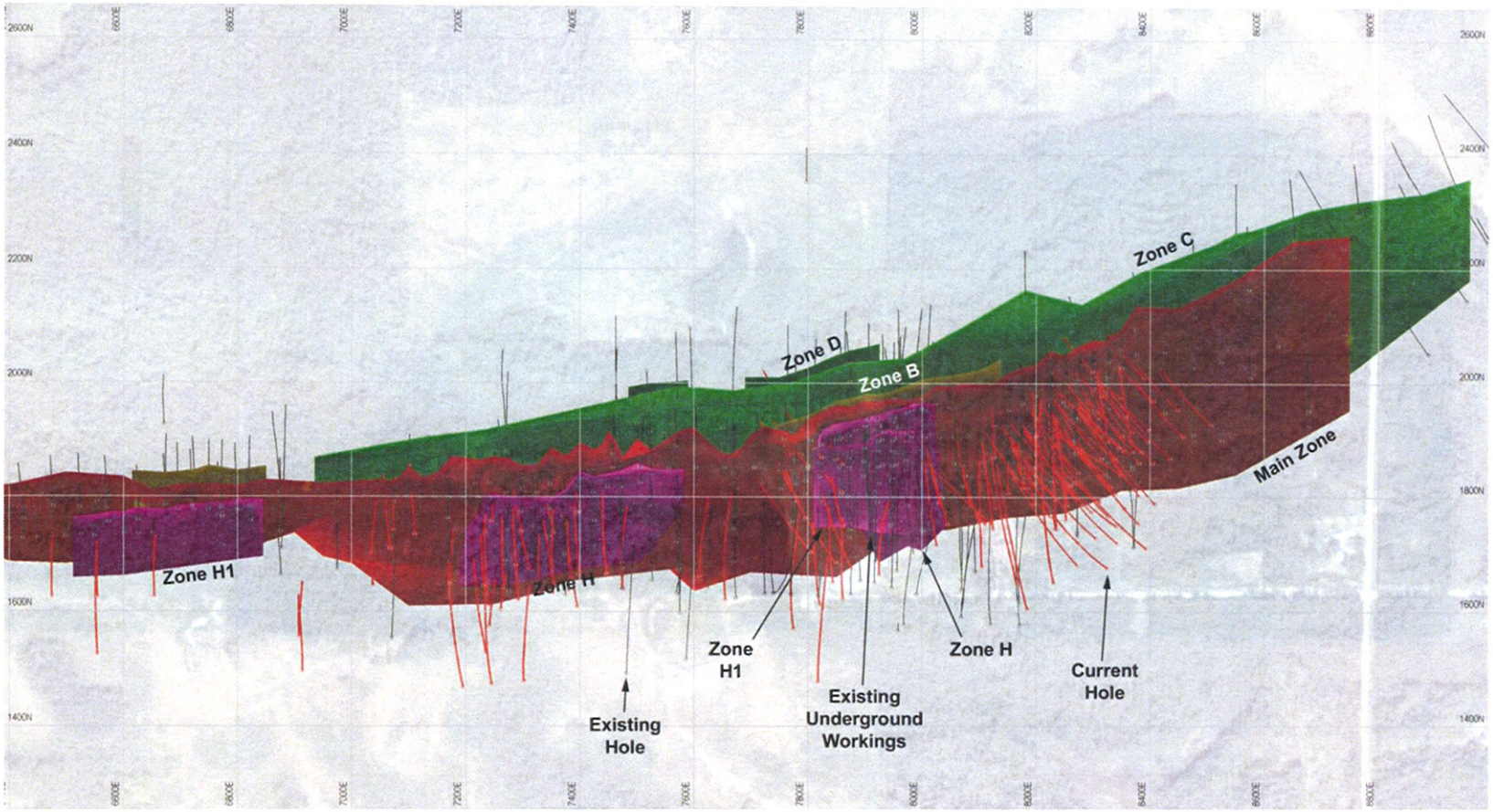


Figure 14-1: Plan view showing older and "current" drilling.

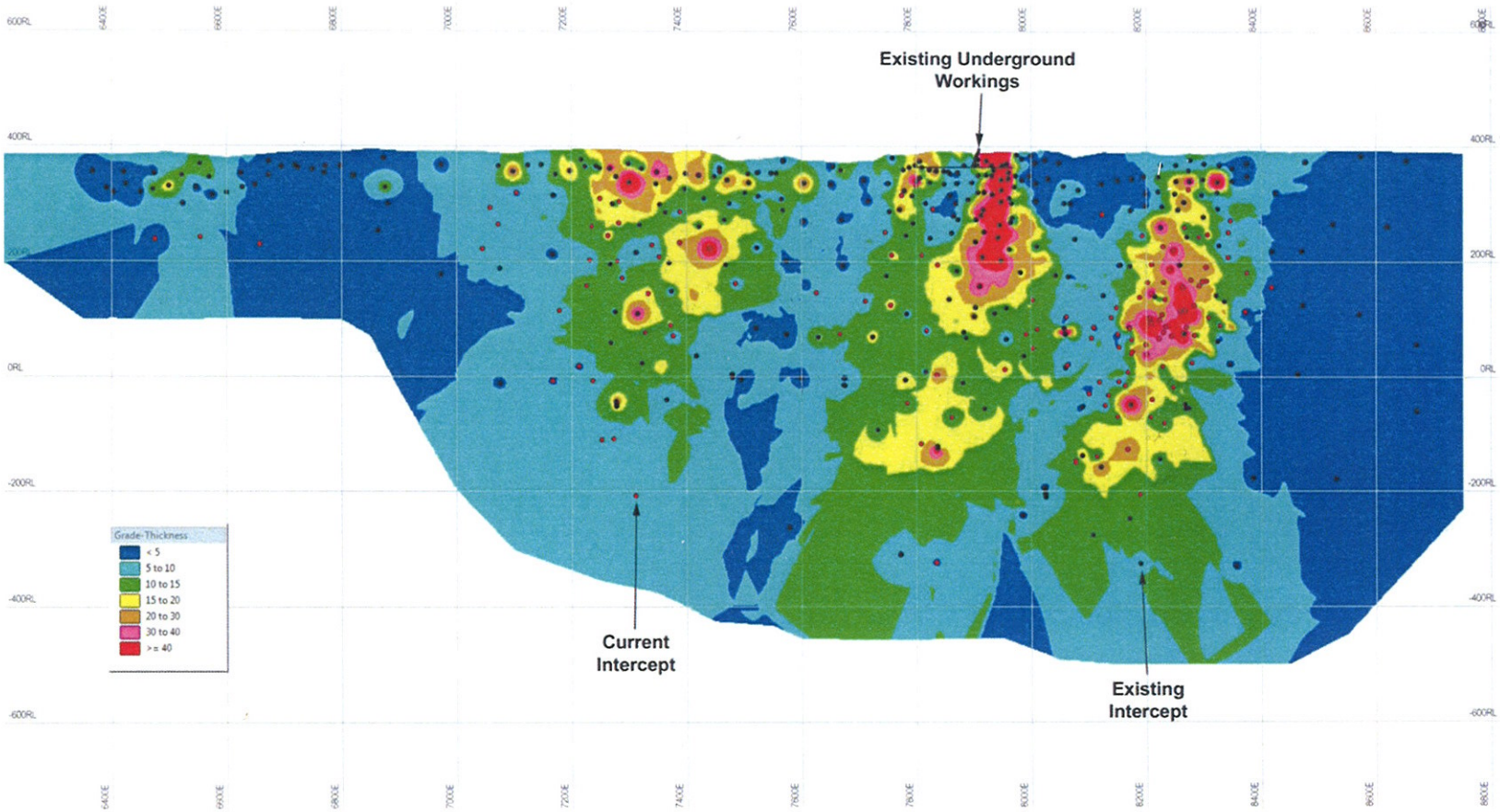


Figure 14-2: Longitudinal section of Main Zone (facing north) showing gram-metres (grade x thickness *or* g/tonne per metre, true thickness), existing intercepts (black) and "current" intercepts (red).

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 MINERALISED ZONE INTERPRETATION

Mineralised zones were outlined to enforce geological control during block modelling. The interpretations that ACA 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, mineralised zones were outlined by following "gold-positive" samples. Cut-off grades are further discussed in Section 14.8.
2. The minimum horizontal zone width was approximately 2 metres.
3. Along strike, zones were extended halfway to the next, under-mineralised 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 14.14.3 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

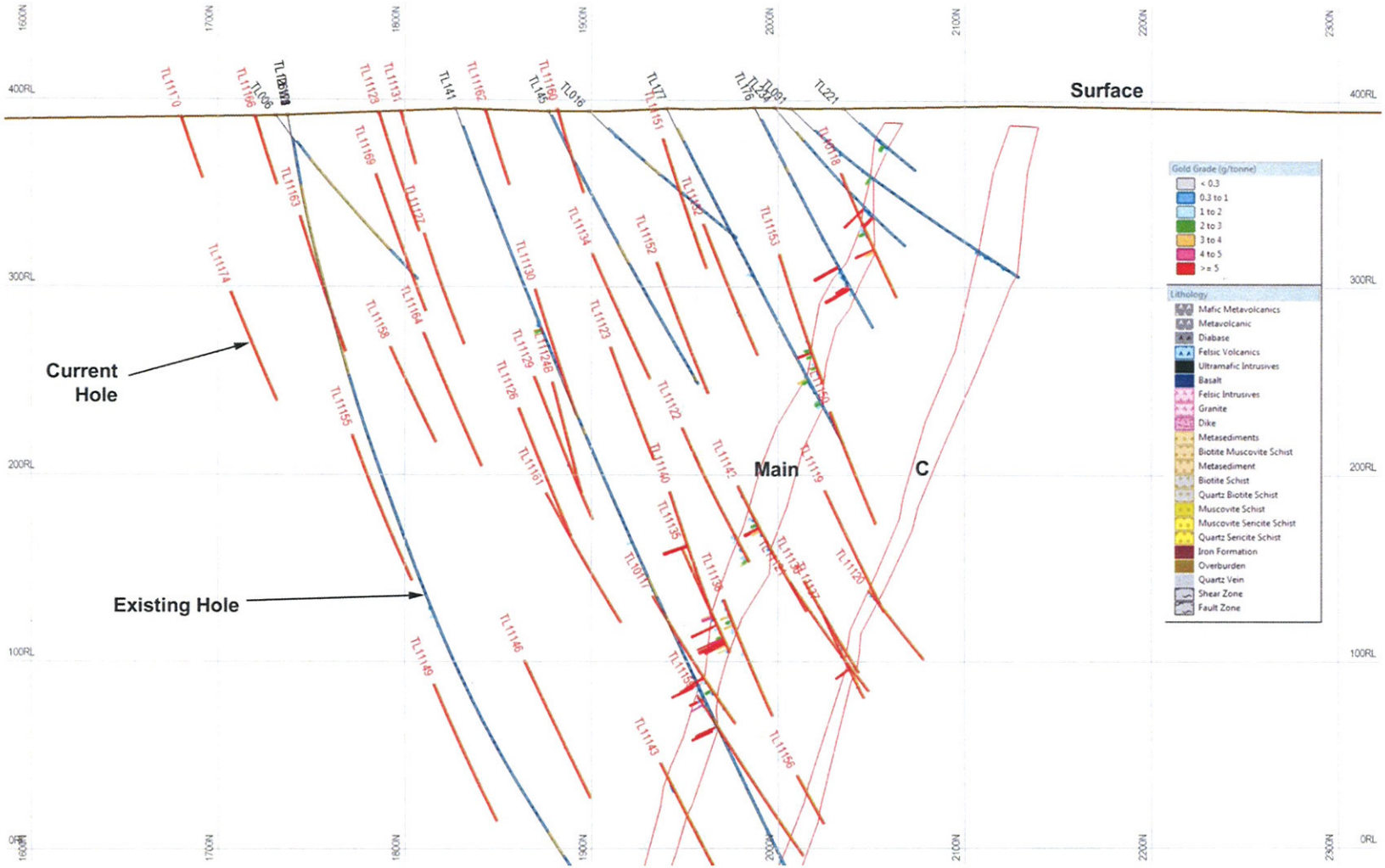


Figure 14-3: Cross-section 8275 m East, facing west.

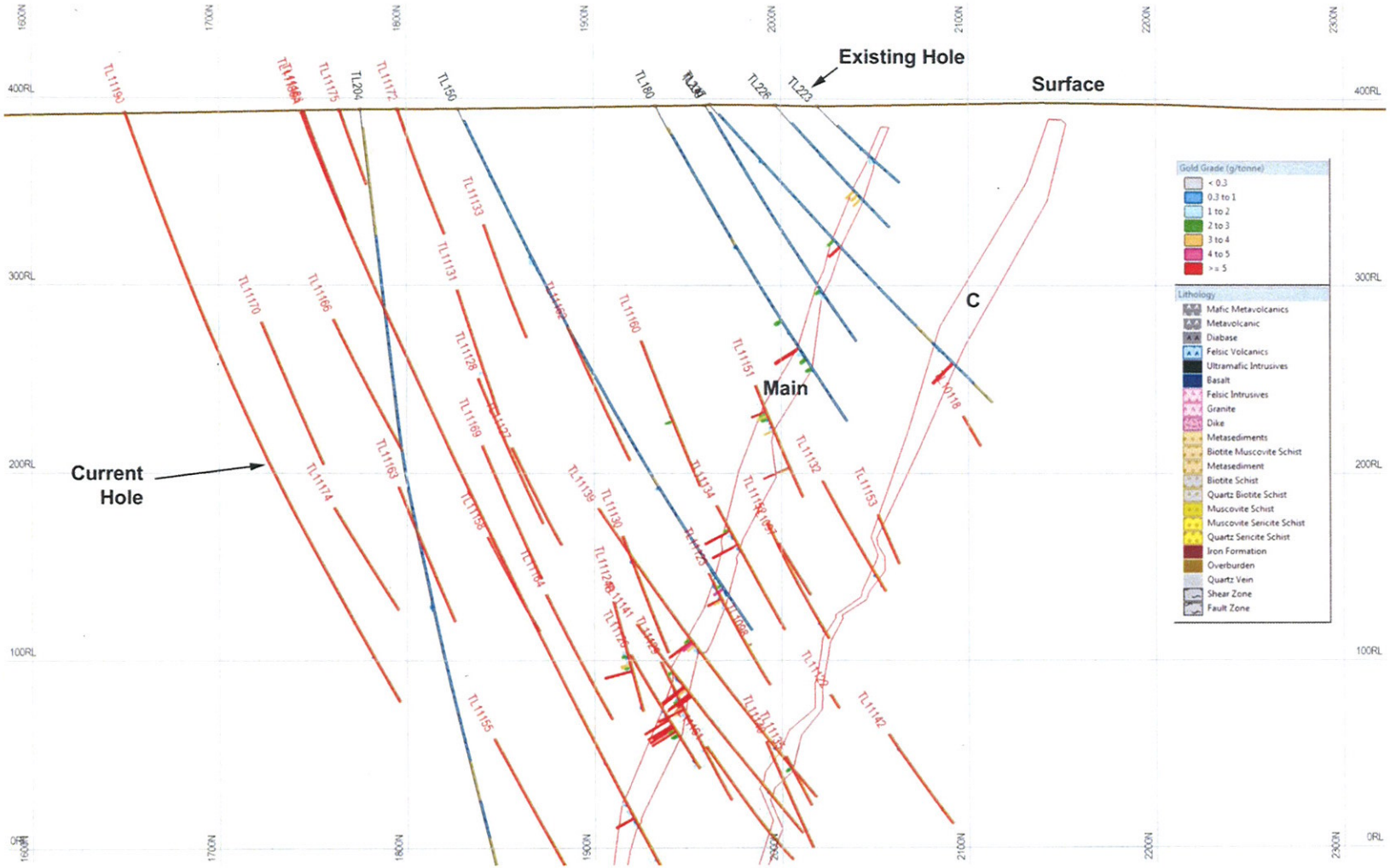


Figure 14-4: Cross-section 8225 m East - facing west.

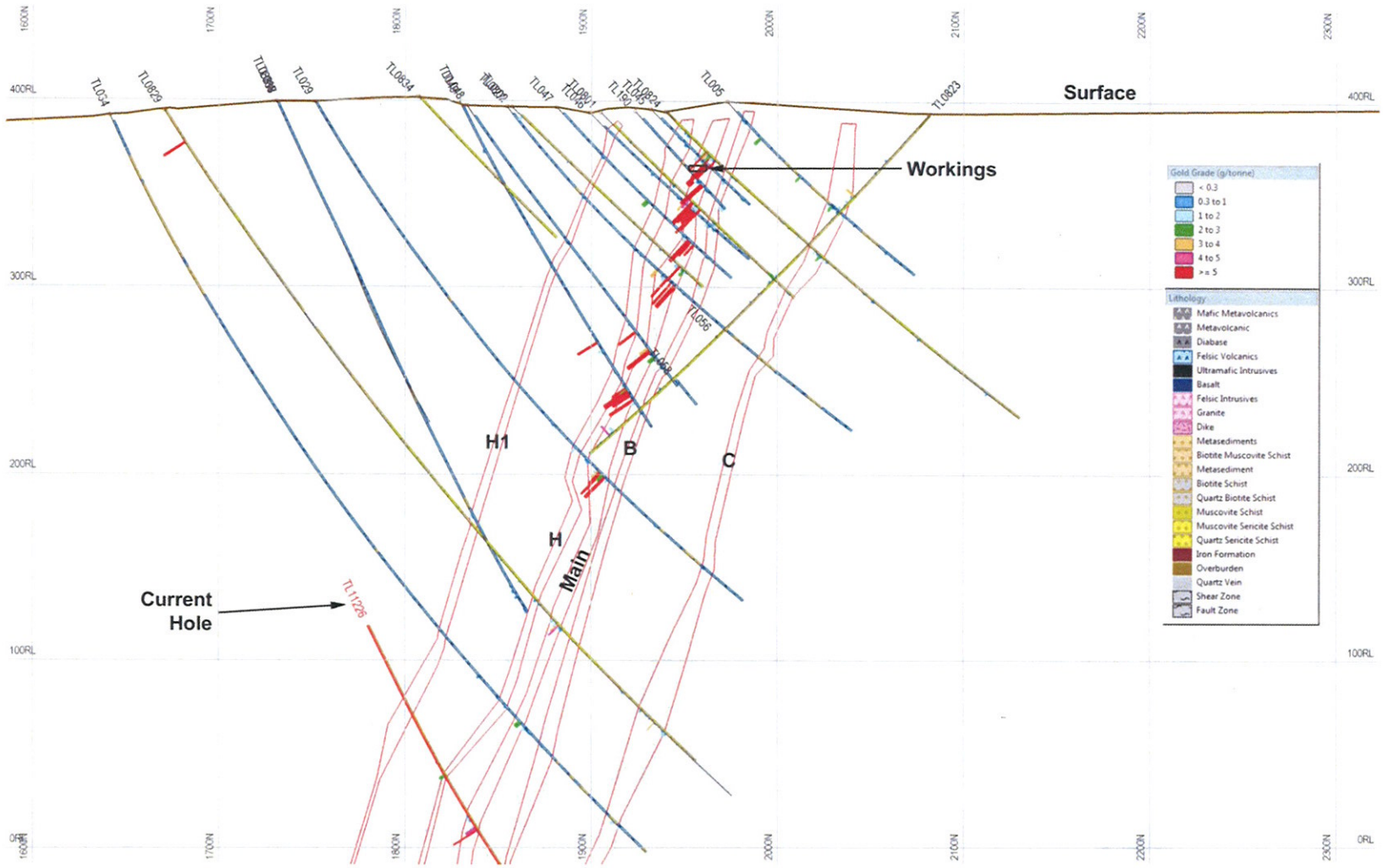


Figure 14-5: Cross-section 7950 m East – facing west.

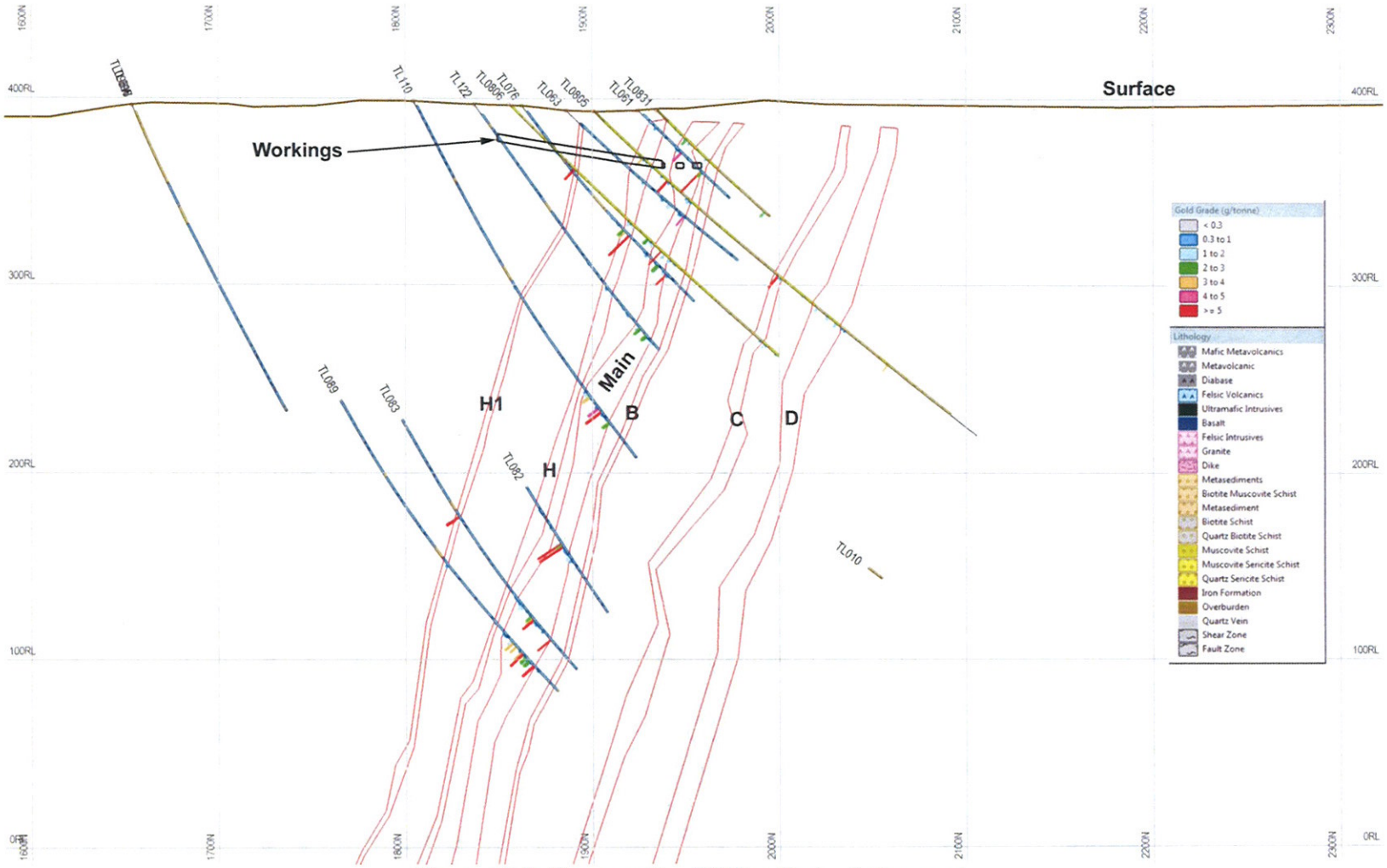


Figure 14-6: Cross-section 7900 m East – facing west.

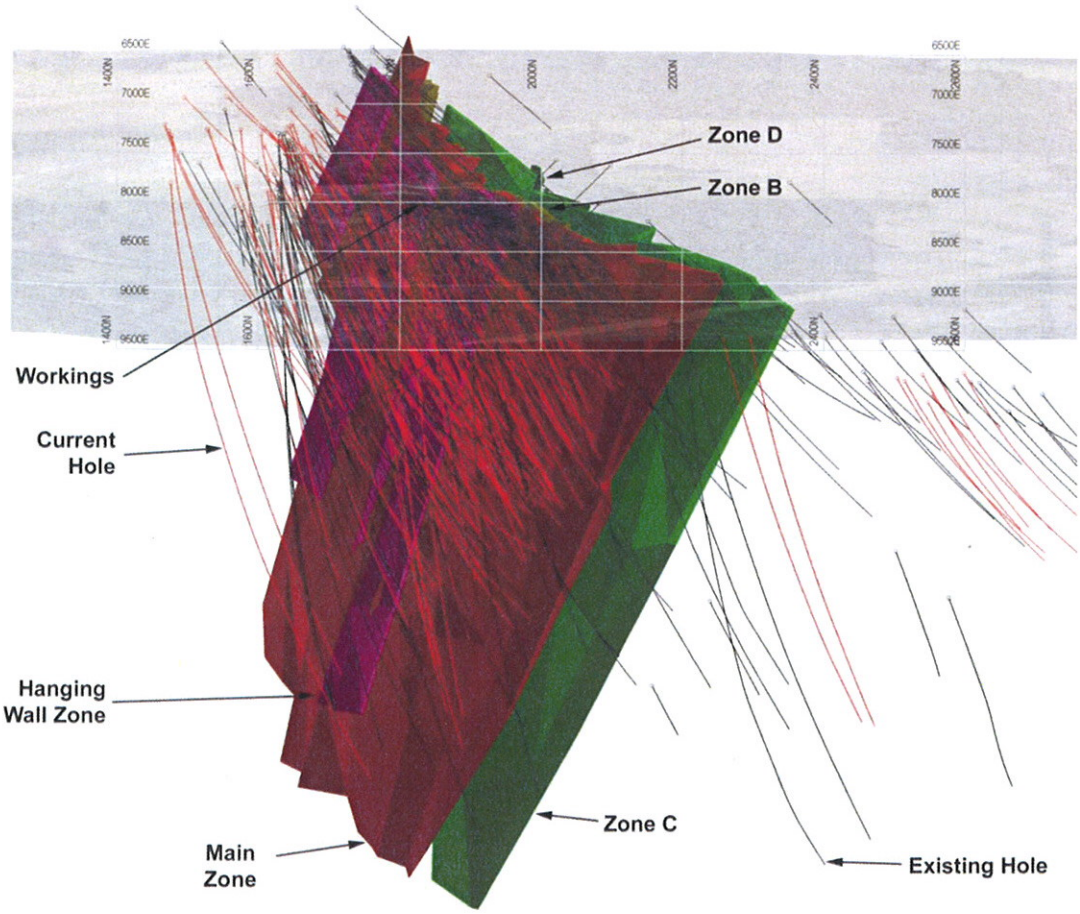


Figure 14-7: 3-D view of outlined zones, facing west.

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.

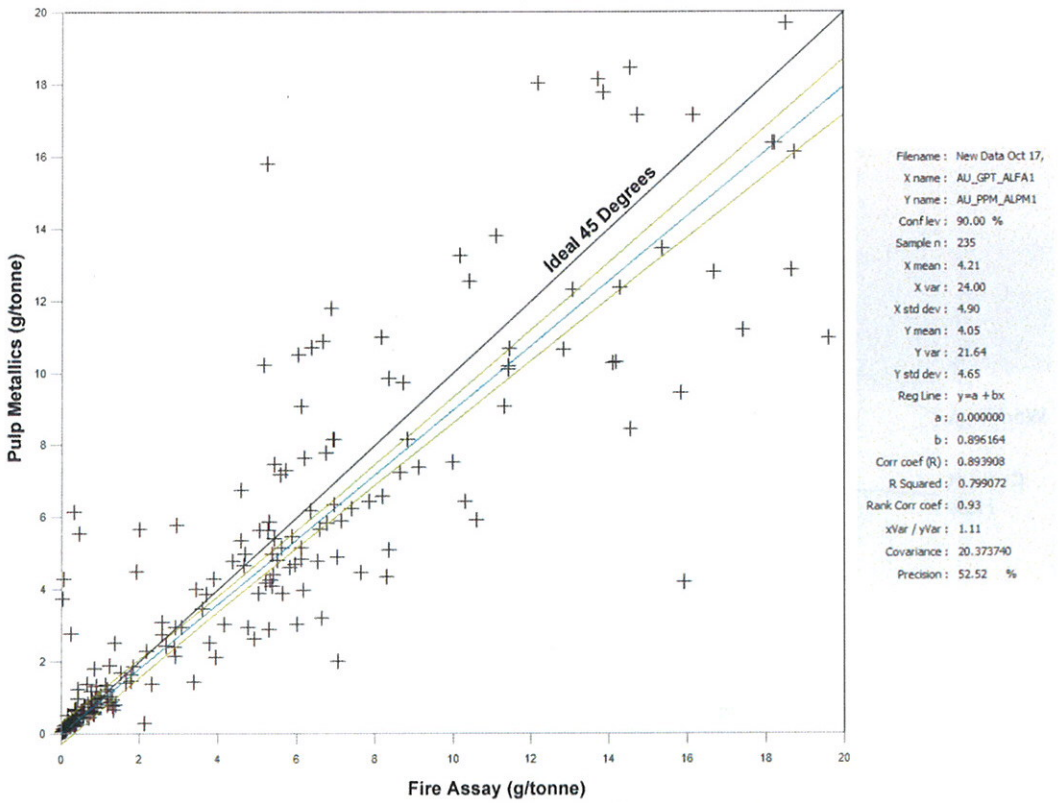
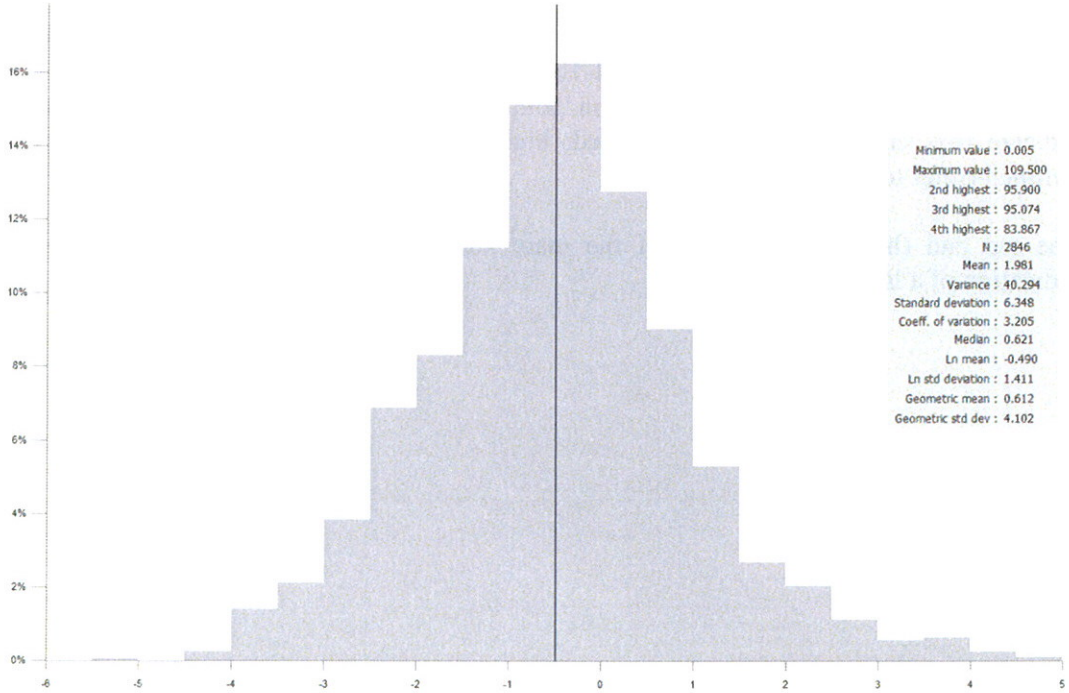


Figure 14-8: Scattergram: pulp metallics versus fire assay.

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.

Histogram - Main Zone, Higher Grade Domain



Histogram - Main Zone, Lower Grade Domain

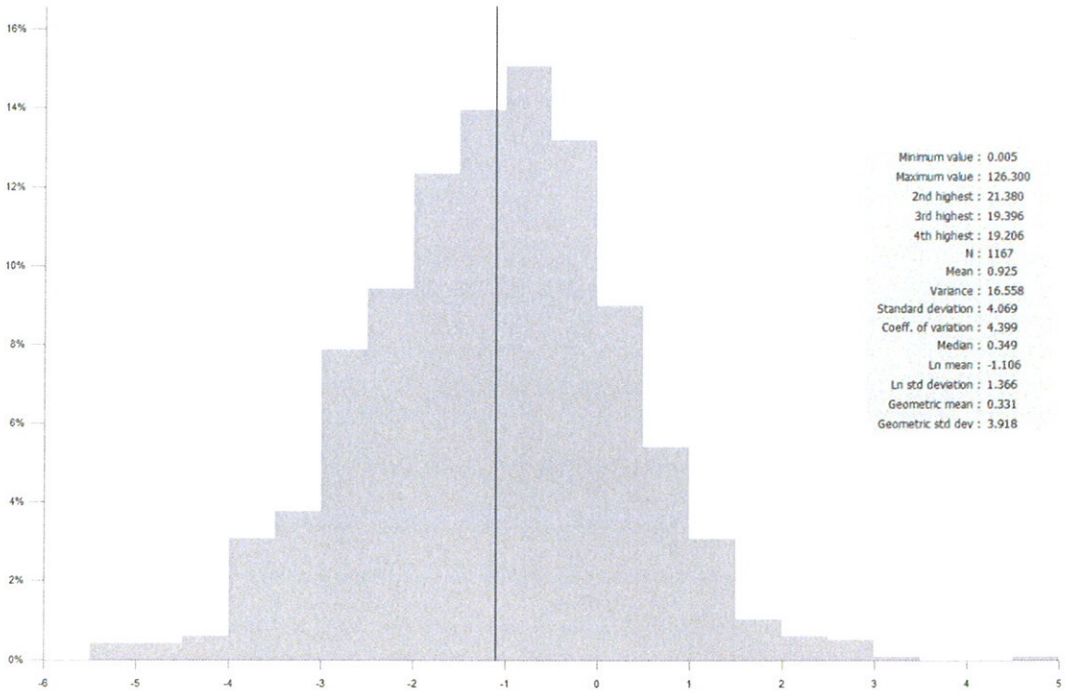


Figure 14-9: Statistics of regularised (1.5 metre), natural log-transformed gold assays [Ln(g/tonne)] within the main zone.

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.

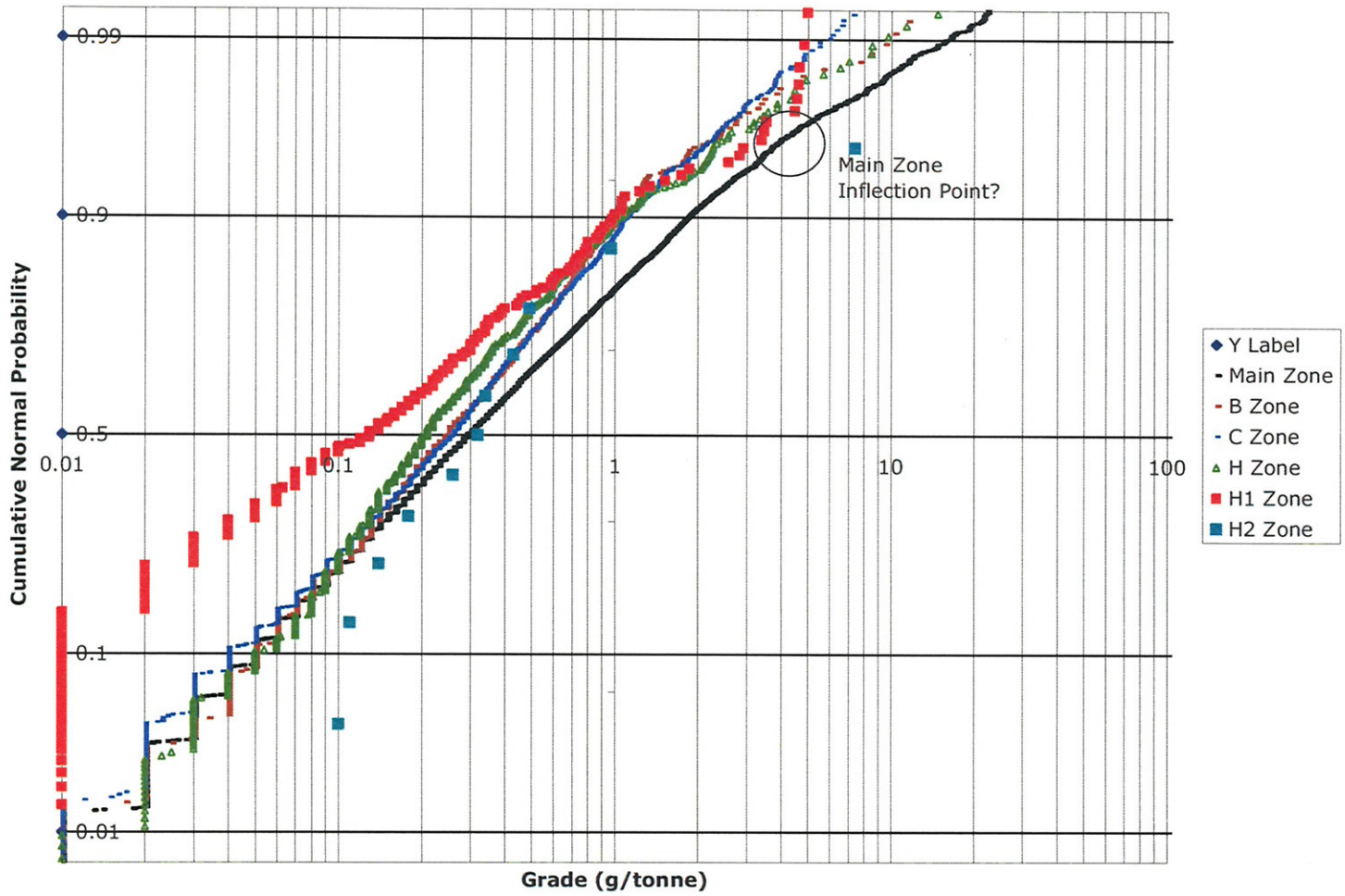


Figure 14-10: Cumulative normal probability for each zone.

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.

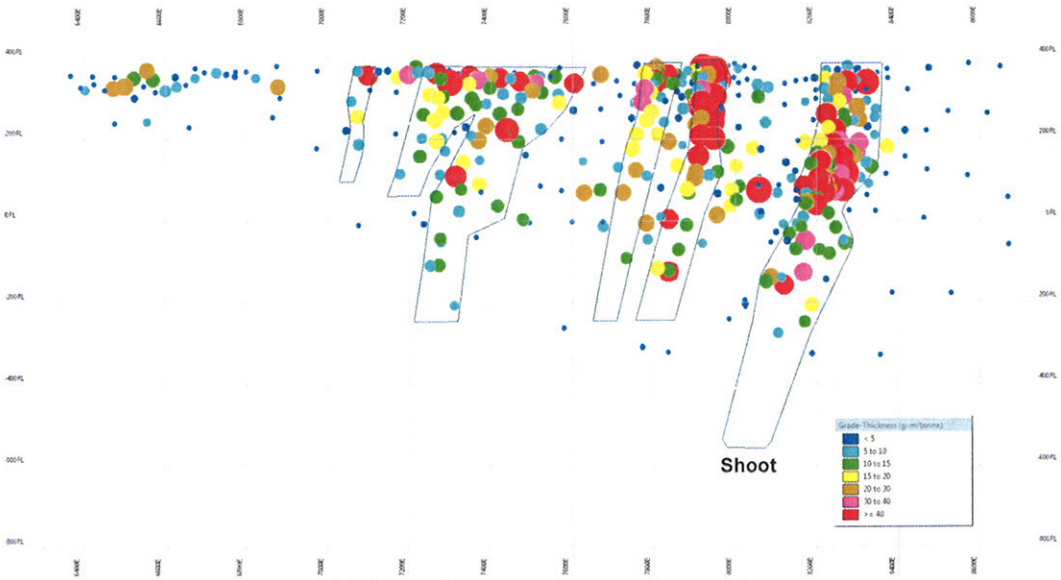


Figure 14-11: Higher-grade domains, Main Zone.

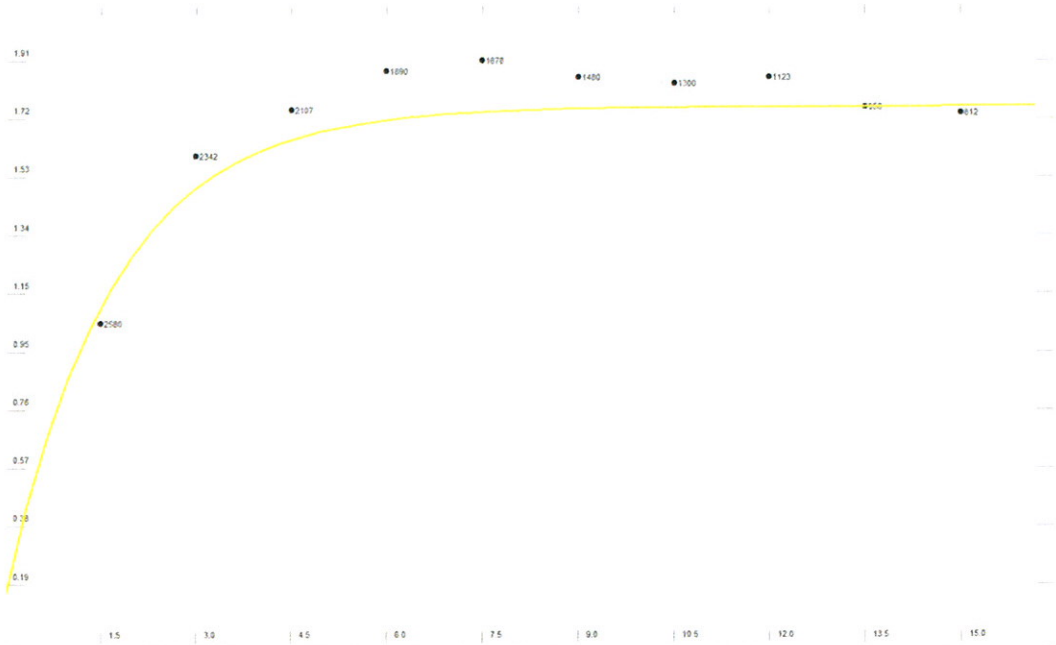


Figure 14-12: Higher-grade domain, downhole (nugget = 0.15, range approx. 5 m, partial sill = 1.61).

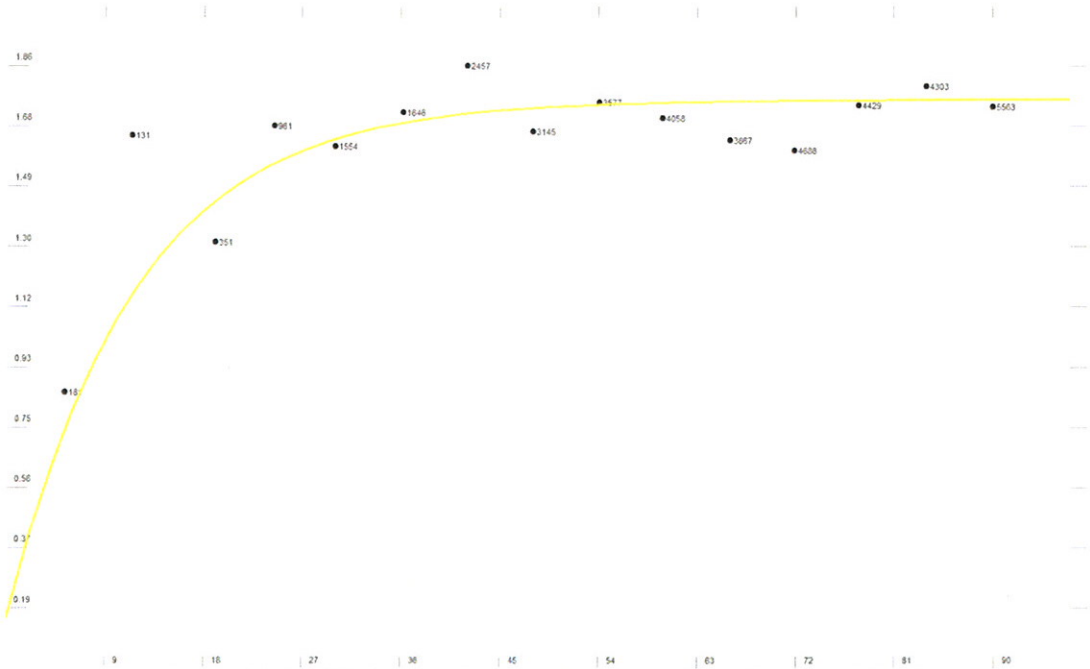


Figure 14-13: Higher-grade domain, along shoot (range = 35 m).

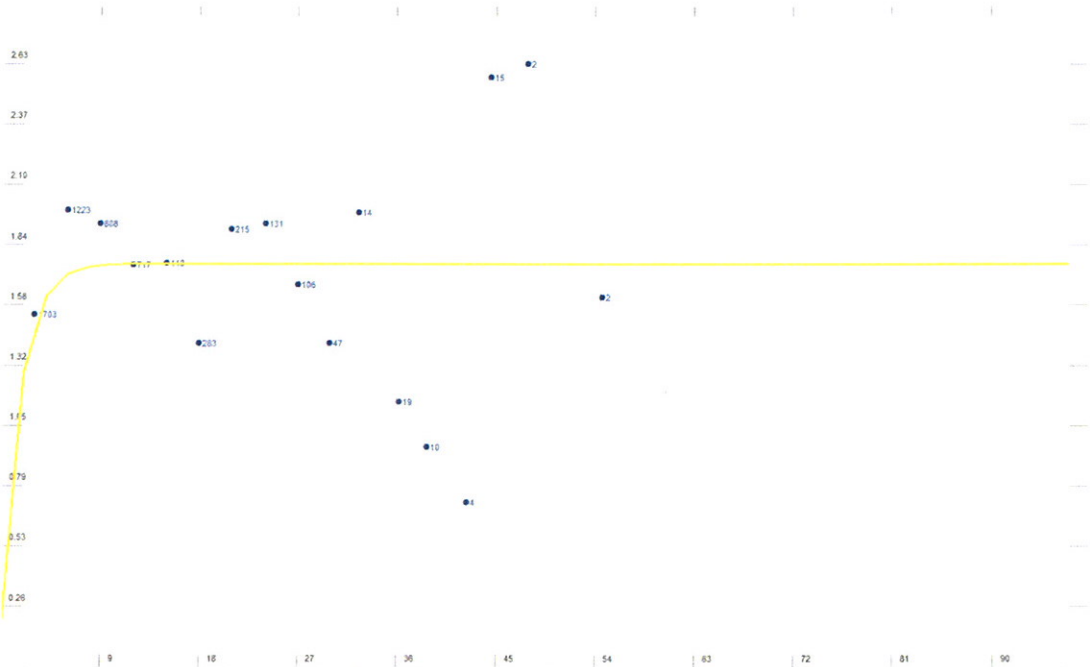


Figure 14-14: Higher-grade domain, across-shoot (very poor quality, range approx. 5 m).

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.

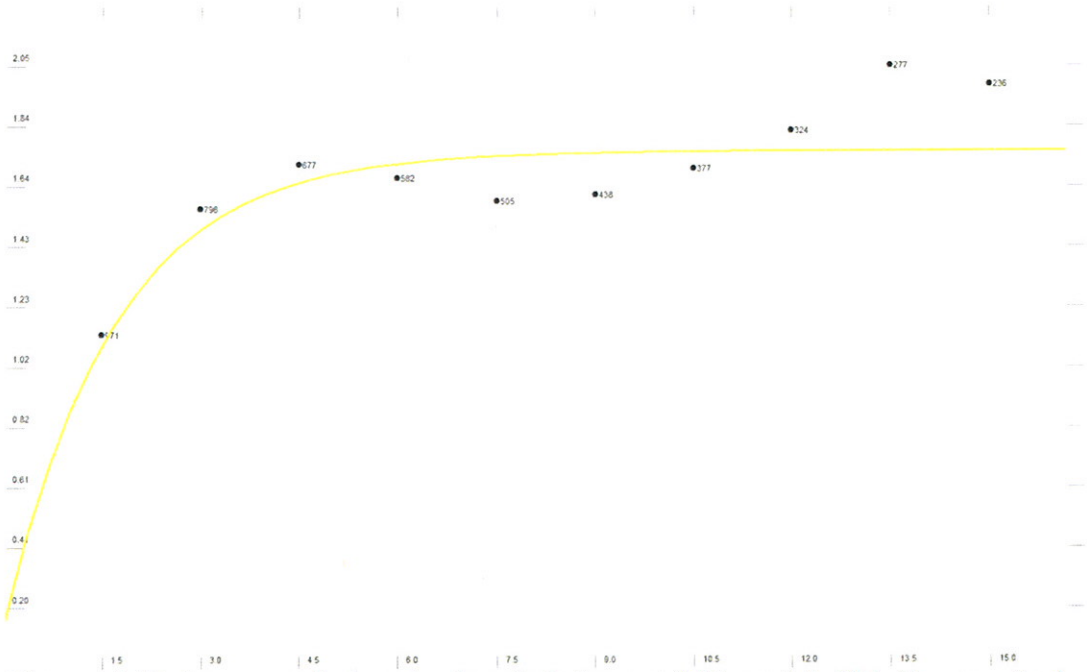


Figure 14-15: Lower-grade domain, downhole (nugget 0.15, partial sill 1.61, range 5 m).

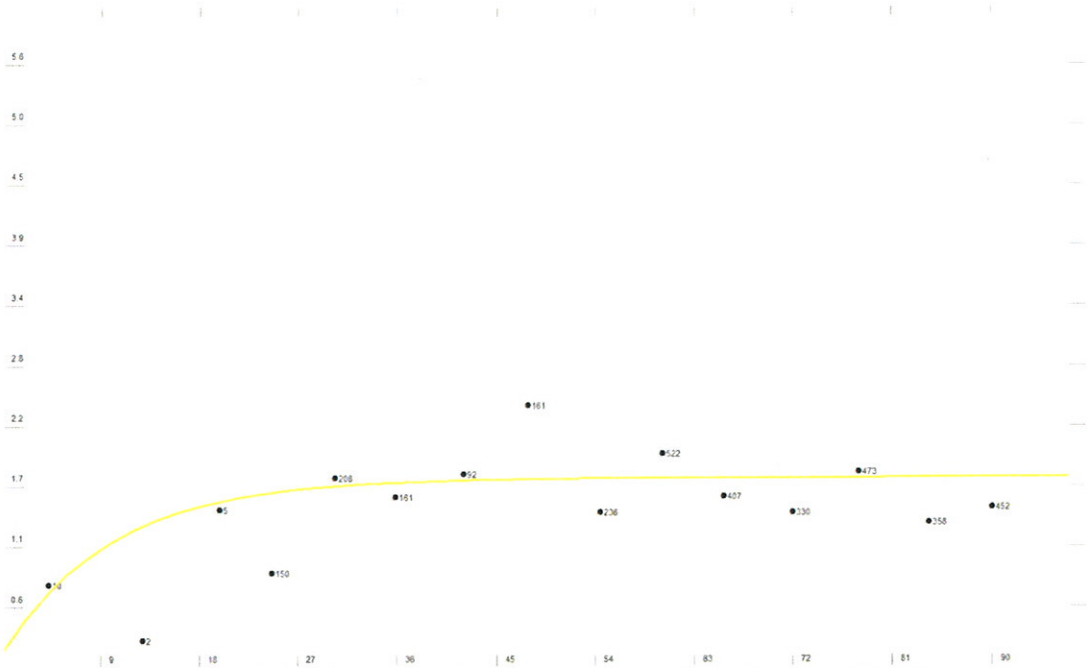


Figure 14-16: Lower-grade domain, down-shoot (range = 30 m).

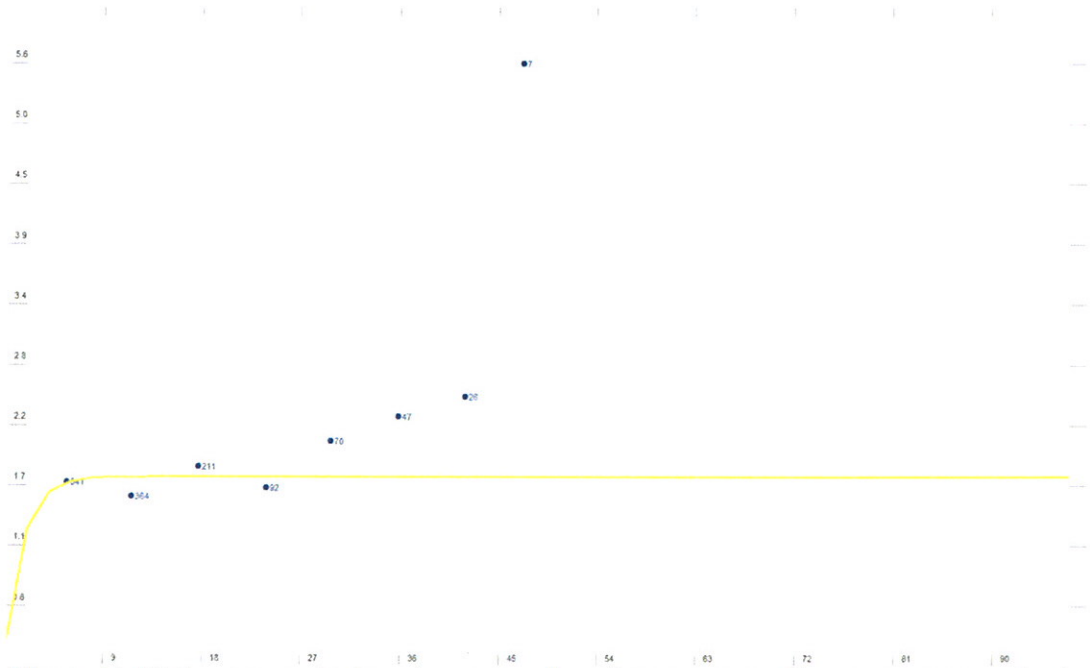


Figure 14-17: Lower-grade domain, across-shoot (very poor quality, range approx. 5 m).

Table 14.3: Variography results for the Main Zone, higher-grade domain.

Direction	Azimuth	Plunge	Data	Model Type	Model Range (m)	Nugget $[Ln(g/tonne)]^2$	Partial Sill $[Ln(g/tonne)]^2$	Fit
Normal to Plane of Mineralisation (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.

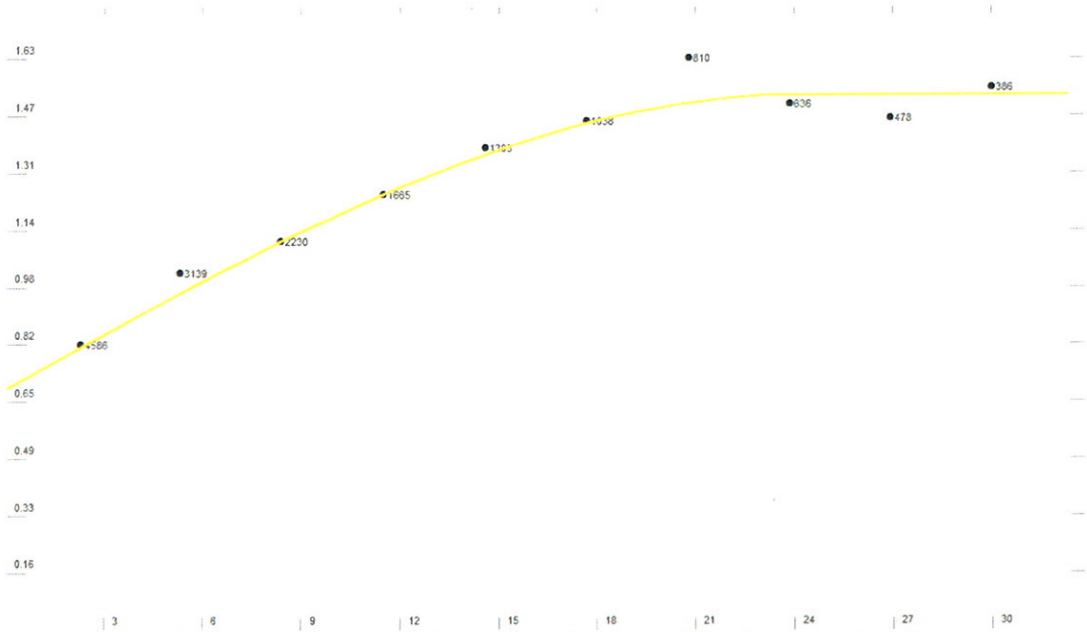


Figure 14-18: Downhole semi-variogram, silver (nugget =0.69).

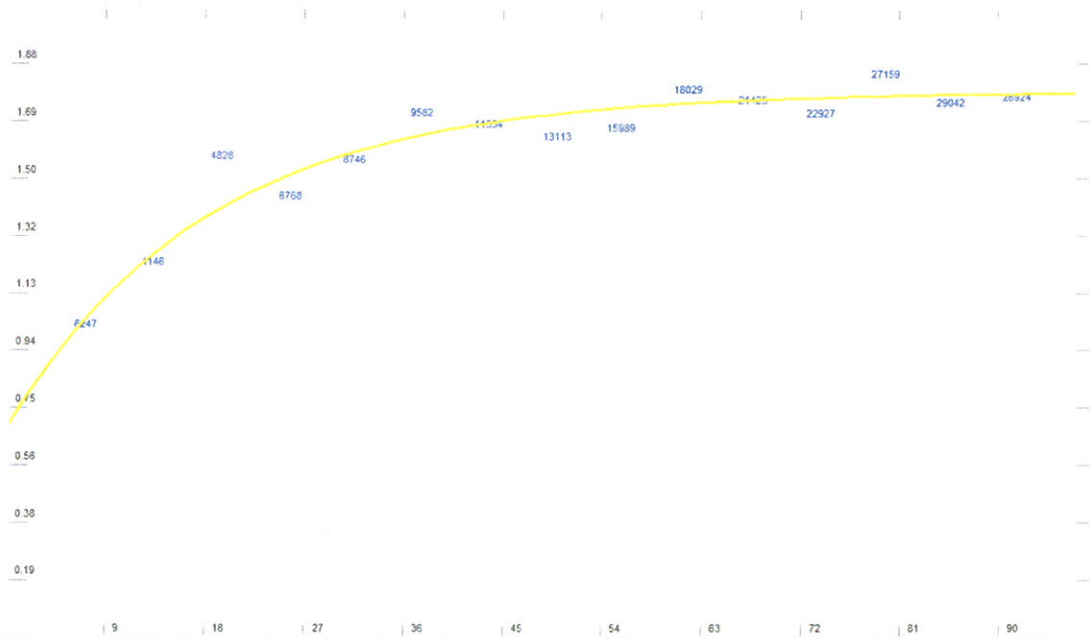


Figure 14-19: Omni-directional semi-variogram, silver (exponential model, range = 55 m, partial sill = 1.10).

14.8 CUT-OFF GRADES

14.8.1 Zone Interpretation

In agreement with the 2010 mineral resource estimate, the chosen cut-off grade for mineralised 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.

14.8.2 Surface Resources

The chosen "block cut-off" [2] 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 a 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).

[2] The grade at which it is possible to mine and process and exposed block (*i.e.*: stripping not included).

14.8.3 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 a revenue of approximately $US 64 per tonne.

14.9 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 $1500 per ounce and $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, one gram of gold is equivalent to 57 grams of silver.

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

$$= 0.018$$

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

14.10 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 mineralised 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**

14.11 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.

14.12 Block Modelling

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 mineralised 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.

14.13 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 mineralised zones were included in the Inferred mineral resource category. Indicated mineral resource blocks were identified using the procedure that was described in Section 14.15.

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.

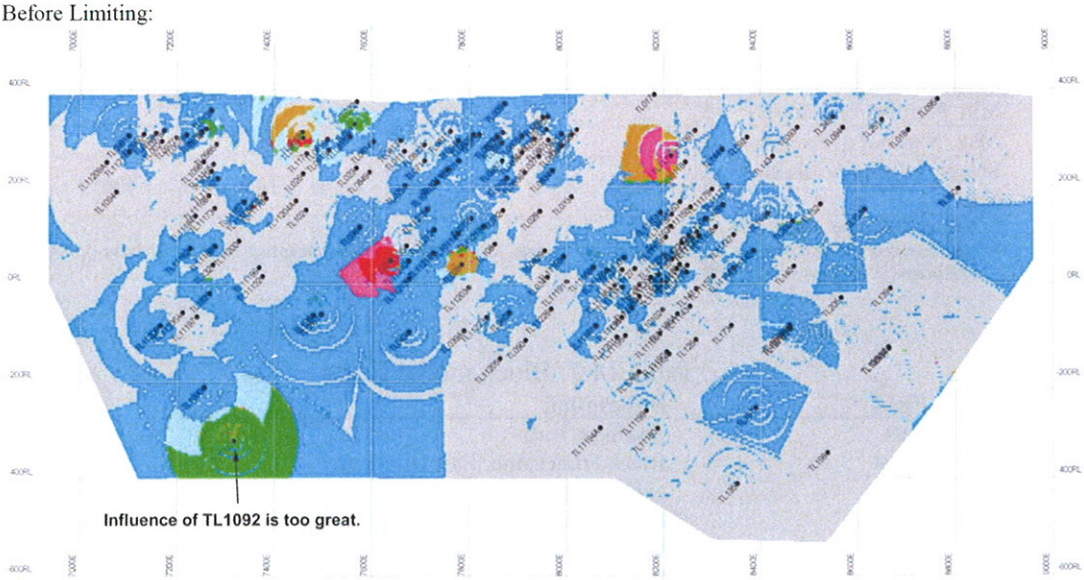
Zone	File
H1	Blocks -Kriged - Zone H1, Rad 400.DAT
H	Blocks -Kriged - Zone H, Rad 400.DAT
Main	Blocks -Kriged - Zone M, Dom LG, Rad 400.DAT (actually contains both higher grade and lower grade domains)
B	Blocks -Kriged - Zone B, Rad 400.DAT
C	Blocks -Kriged - Zone C, Rad 400.DAT
D	Blocks -Kriged - Zone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

14.14 Need to Limit Certain Holes

14.14.1 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:

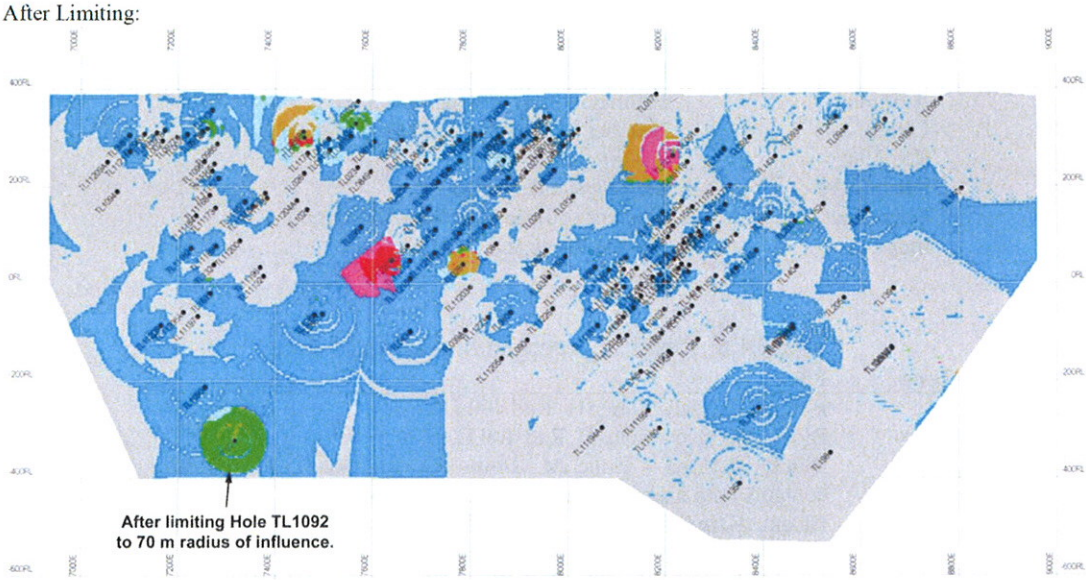


Figure 14-20: Long section of Zone C showing estimated gold grades, before (top) and after (bottom) limiting Hole TL1092`s radius of influence to 70 metres.

14.14.2 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 14-21). 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.

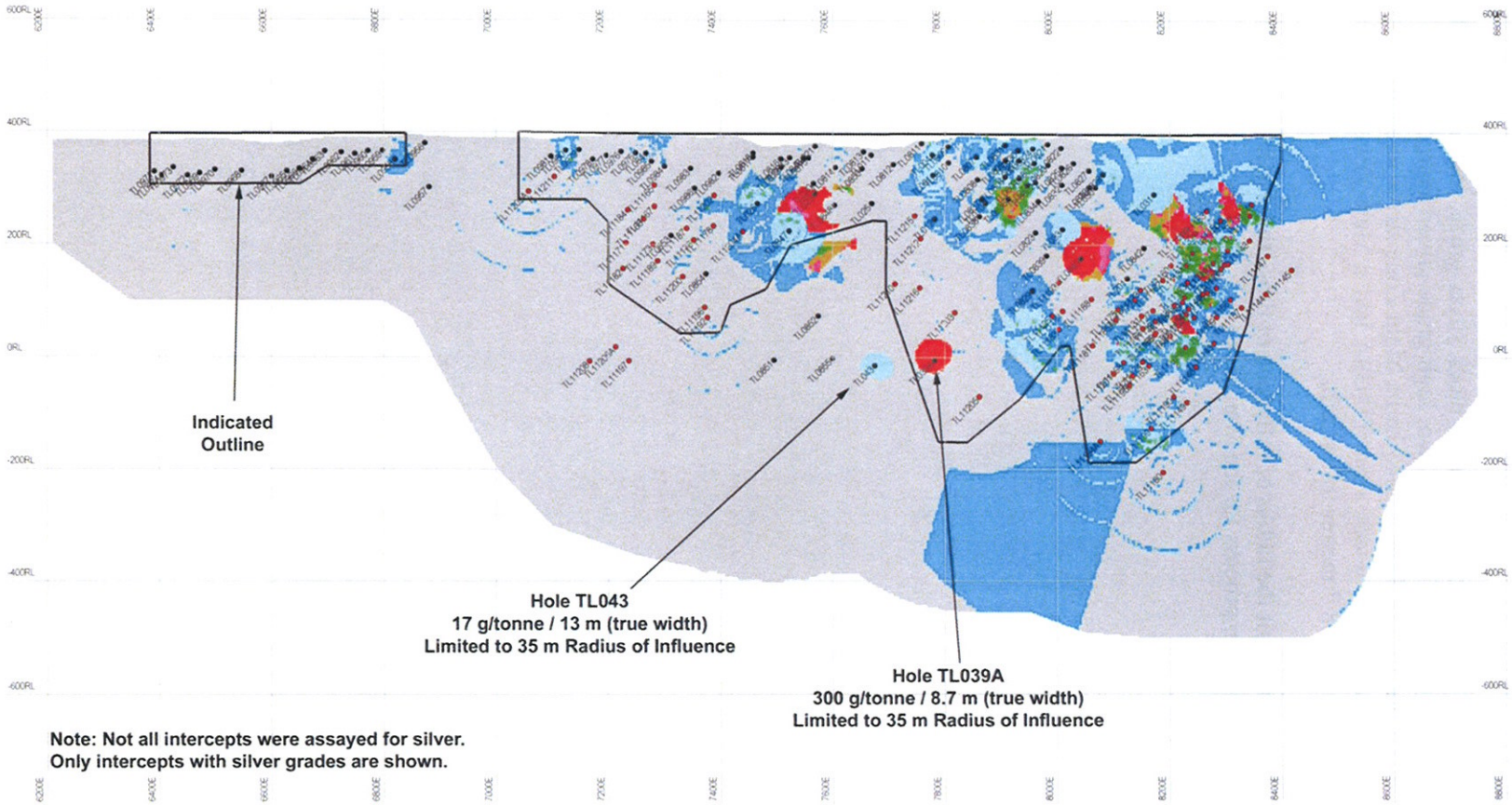
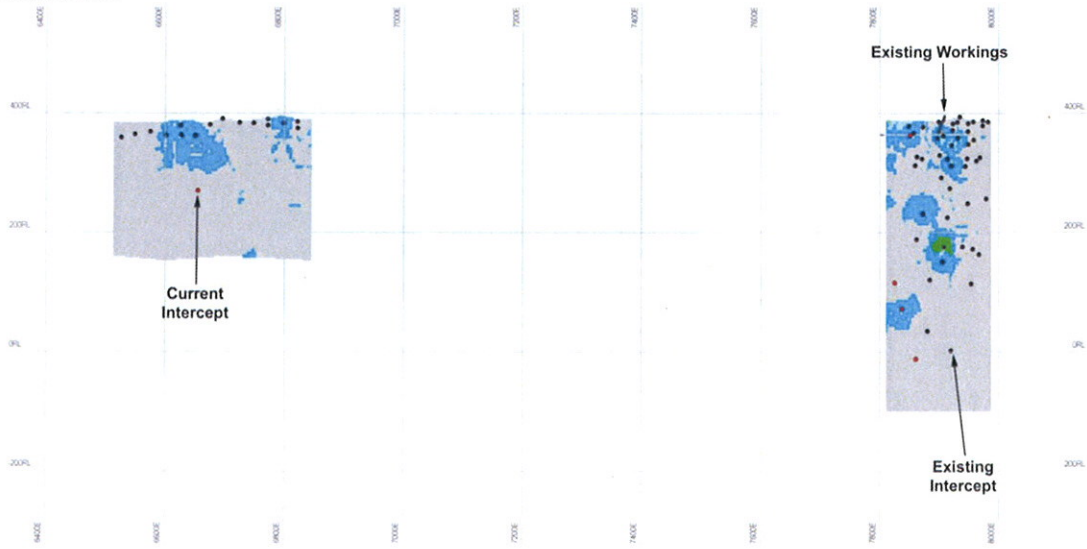


Figure 14-21: Longitudinal section of the Main Zone showing silver assay positions and block silver grade values (facing north, after limiting the influence of Holes TL043 and TL039A).

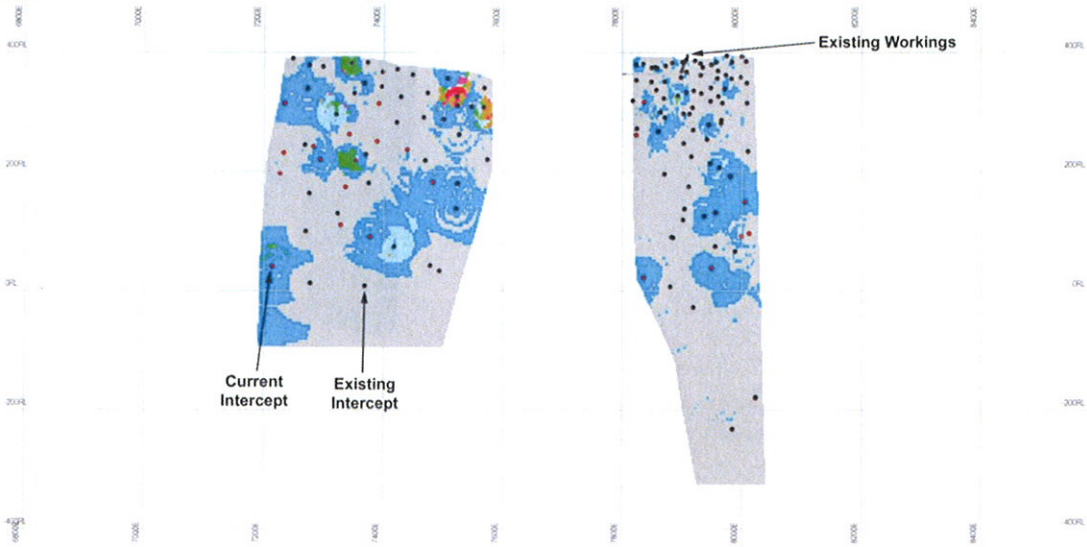
14.14.3 Longitudinal Sections Showing Gold Grades

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

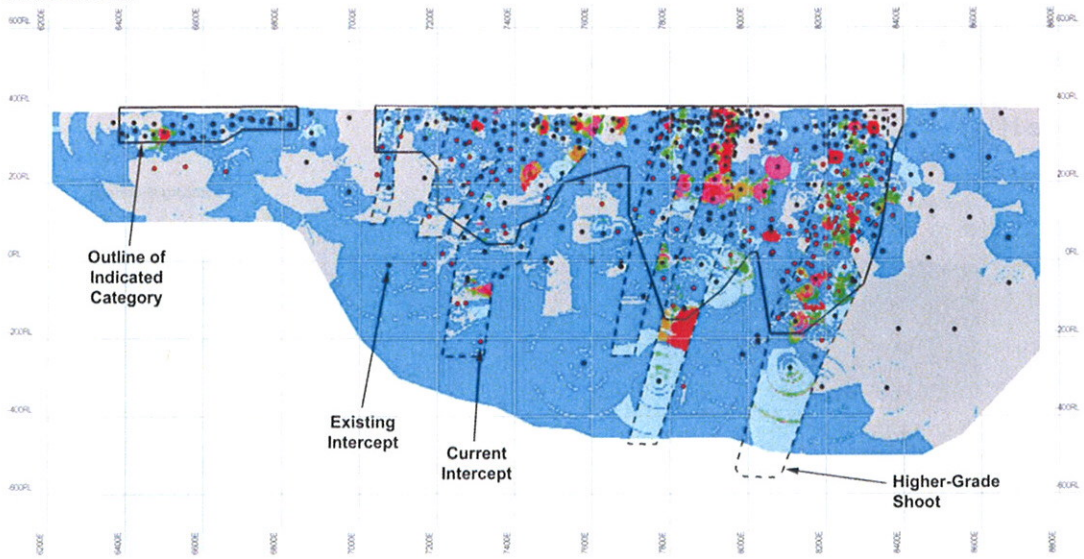
Zone H1:



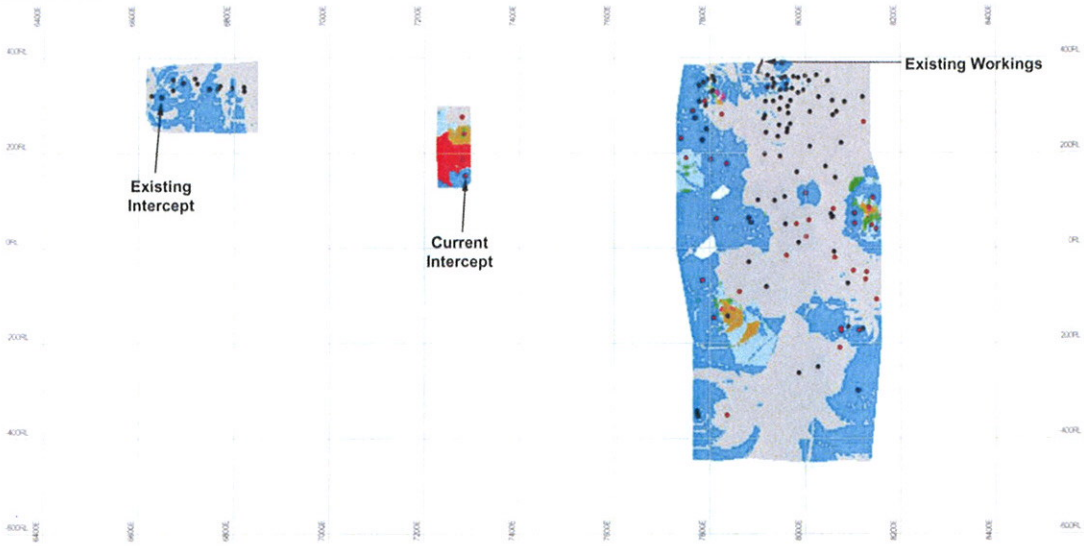
Zone H:



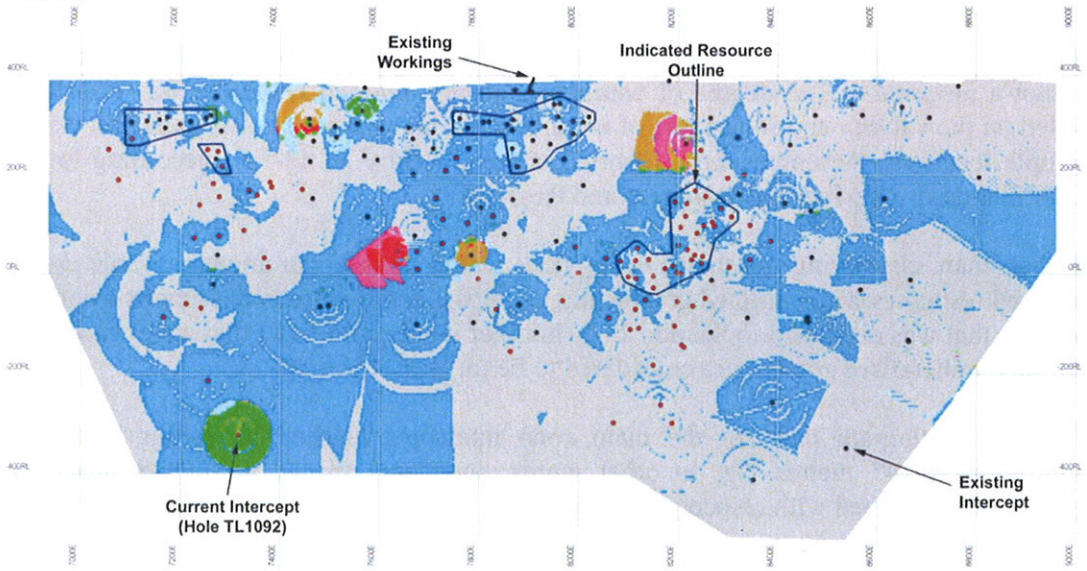
Main Zone:



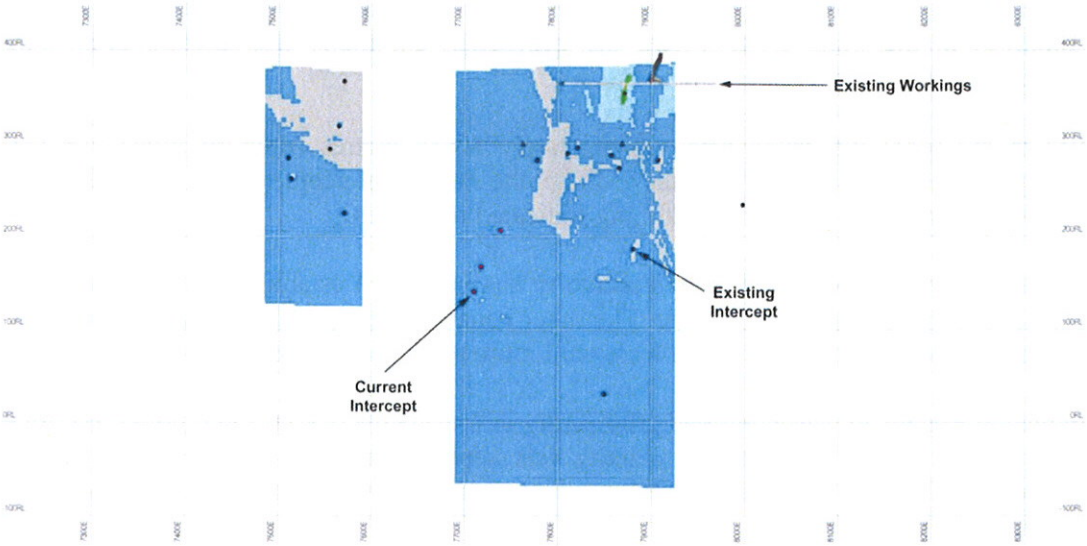
Zone B:



Zone C:


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

Zone D:


Existing Workings
Existing Intercept
Current Intercept

14.15 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 mineralised 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 14-23). 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 14-24 and Figure 14-25, respectively). Outlines were drawn around areas where the circles were generally touching or very close to touching, in two dimensions.

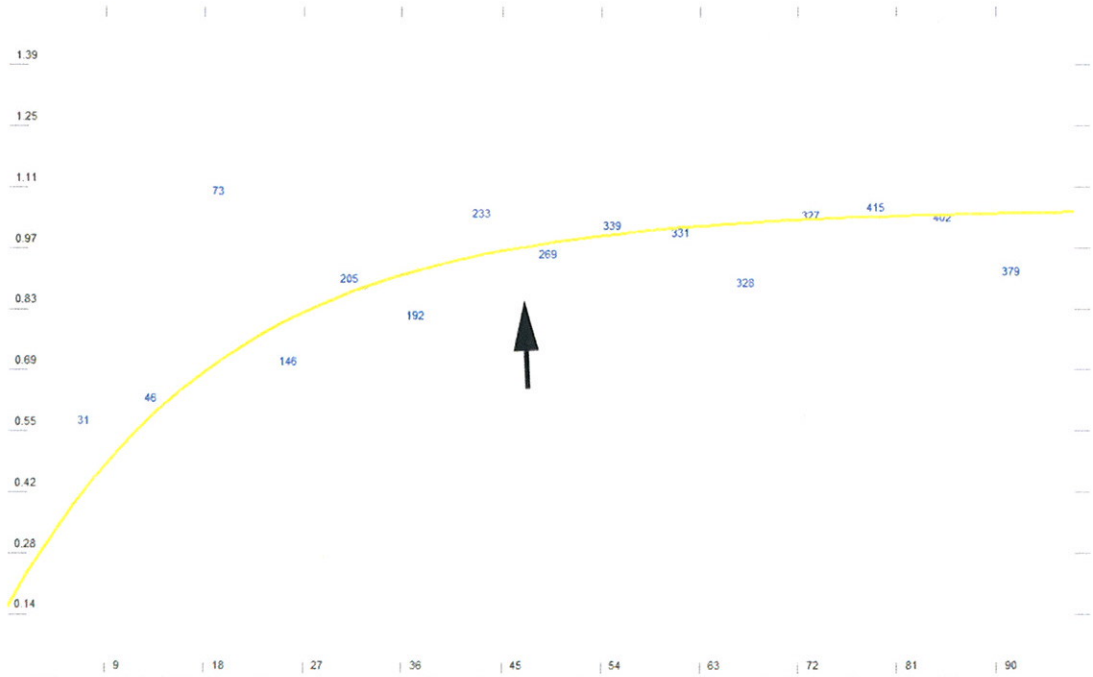


Figure 14-22: Semi-variogram of main zone intercepts (omni-directional). The arrow indicates the approximate range.

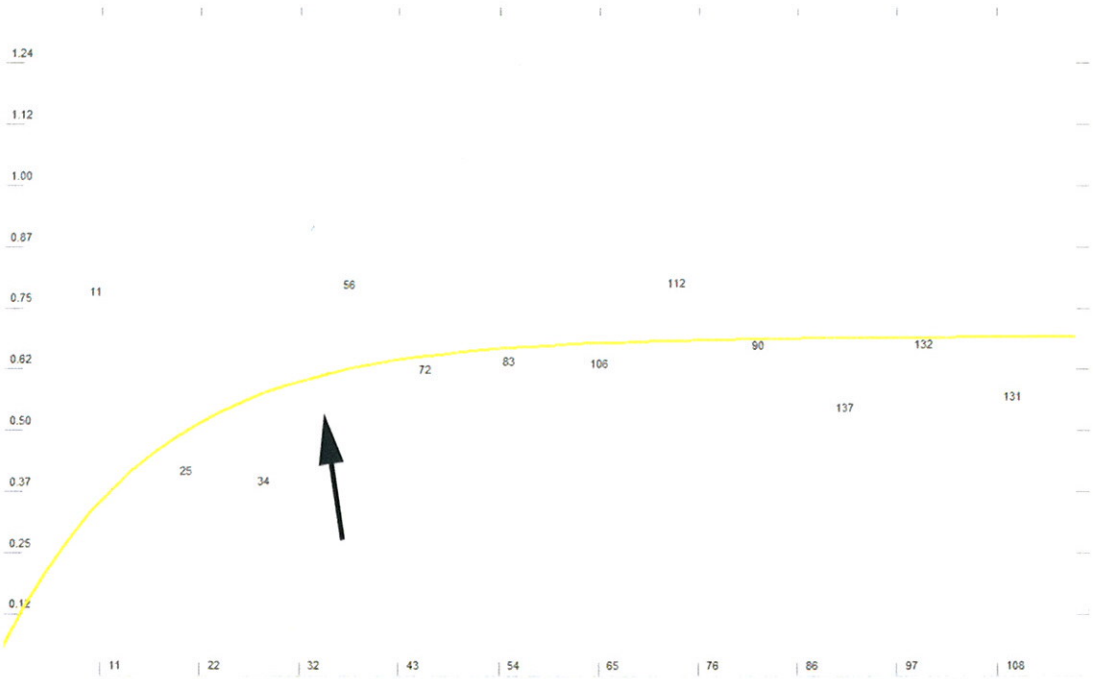


Figure 14-23: Semi-variogram of C-Zone intercepts (omni-directional).

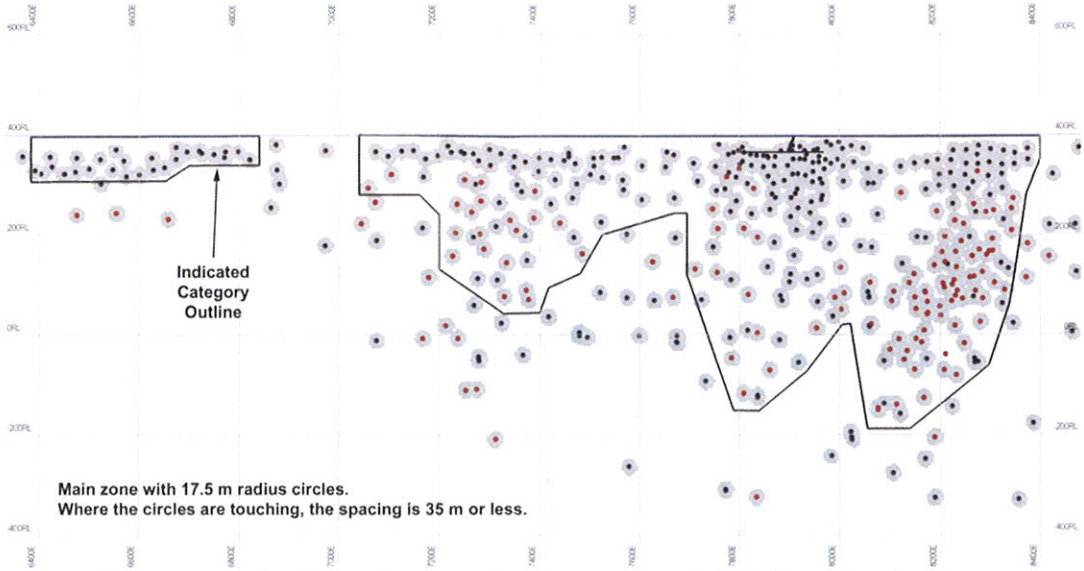


Figure 14-24: Outline of Indicated resources (black line) in the Main Zone.

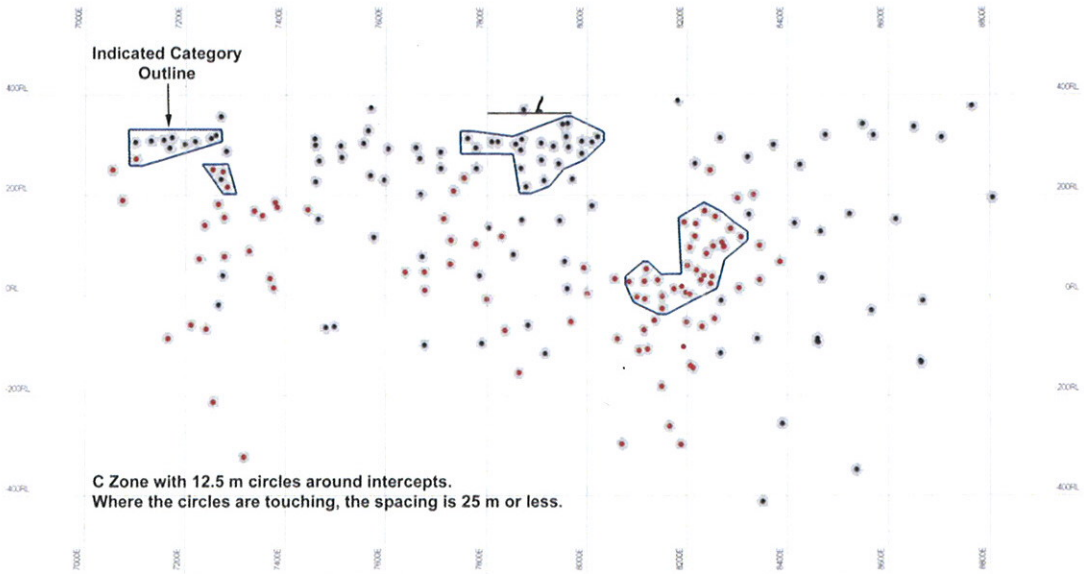


Figure 14-25: Outline of Indicated resources in the C Zone.

14.16 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.

14.16.1 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 mineralised 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 1500 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.

14.17 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 overestimate 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**

14.18 Comparison With Previous Mineral Resource Estimate

A comparison was made with the previous mineral resource estimate that ACA 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

This section is not relevant to this Report. As of the date of this report, no mineral reserves have been estimated for Project.

16 MINING METHODS

This section is not relevant to this Report. As of the date of this report, no mining is conducted at the Project.

17 RECOVERY METHODS

This section is not relevant to this Report. As of the date of this report, no mineral processing is conducted at the Project.

18 PROJECT INFRASTRUCTURE

No infrastructure is currently present at the Project area.

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.

19 MARKET STUDIES AND CONTRACTS

This section is not relevant to this Report. As of the date of this report, no marketing studies have been undertaken for the Project.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

Treasury has commissioned Environmental Base Line Studies using the services of Klohn Crippen Berger ("KCB"), 7-1351C Kelly Lake Rd., Sudbury ON P3E 5P5. Studies were initiated in the Fall of 2010 and have continued to the date of this report. These studies will examine the health of the ecosystem by studying ground and surface water quality, sediment quality, fisheries, terrestrial resources and soil quality. 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.

21 CAPITAL AND OPERATING COSTS

This section is not relevant to this Report. Howe completed a Preliminary Economic Assessment (PEA) on the Goliath Project in July 2010 (Roy et al, 2010)

22 ECONOMIC ANALYSIS

This section is not relevant to this Report. Howe completed a Preliminary Economic Assessment (PEA) on the Goliath Project in July 2010 (Roy et al, 2010)

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, has visited the site and has reviewed sampling procedures and security. Howe believes that the data presented by Treasury are generally an accurate and reasonable representation of the Treasury project mineralisation.

Work by Treasury at the Goliath Project has confirmed the grade of mineralisation outlined by previous owners Teck and Corona, provided further detail on the nature of the mineralised 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 mineralisation, 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 mineralisation.

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 were no Measured mineral resources, or mineral Reserves of any kind identified.

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.

In Howe's opinion, Treasury should continue work to advance the Project, by gathering information and undertaking studies with the view to eventually undertaking a 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.

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. Lateral development, raising and test stoping would be carried out. A bulk sample would be taken. As with the surface sample, this would be split down to 50-100 tonnes then shipped to a pilot plant laboratory facility.

The overall objective of the work would be to determine mining and processing parameters to the preliminary feasibility level of accuracy (plus or minus 15-20%). Should the preliminary feasibility be positive, mineral reserves can be identified.

The grand total budgetary cost for this work is estimated to be $3.2 million.

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**

27 REFERENCES

Accurassay Laboratories (2008).www.Accurassay.com/analyses

Beakhouse, G.P., Blackburn, C.E., Breaks, F.W., Ayer, J., Stone, D. and Stott, G.M. (1995) Western Superior Province, Fieldtrip Guidebook for *Precambrian '95*, Ontario Geological Survey, Open File Report 5924, 102p.

Beakhouse, G.P. (2000) Precambrian geology of the Wabigoon Area; *in* Summary of Field Work and Other Activities 2000, Ontario Geological Survey, Open File Report 6032, page 20-1 to 20-8.

Beakhouse, G.P. (2001) Precambrian geology of the Thunder Lake segment, Wabigoon Area; *in* Summary of Field Work and Other Activities 2001, Ontario Geological Survey, Open File Report 6100, page 15-1 to 15-6.

Beakhouse, G.P. (2002) Field Trip Guide, Wabigoon Region, Gold Mineralisation Occurring North and South of the Wabigoon Fault, 6p.

Beakhouse, G.P. and Pigeon, L. (2003) Precambrian geology of the Thunder Lake area; Ontario Geological Survey, Preliminary Map P.3529, scale 1:20 000.

Beakhouse, G.P. and Idziszek, M. (2006) Precambrian geology of the Butler Lake – Dinorwic Lake area; Ontario Geological Survey, Preliminary Map P.3582, scale 1:20 000.

Berger, B.R. (1990) Precambrian Geology, Laval and Hartman townships; Ontario Geological Survey, Report 272, 74p (with Map 2534, scale 1:20,000).

Burnham, C.W., 1979,Magma and hydrothermal. fluids, in Barnes, H.L., ed., Geochemistry of Hydrothermal Ore Deposits, 2nd edition: Wiley Interscience, New York, p. 71-136.

CAMH (2005) Canadian & American Mines Handbook 2004-2005, Corona Gold Corp., Business Information Group, p.127.

CAMH (2007) Canadian & American Mines Handbook 2006-2007, Corona Gold Corp., Business Information Group, p.128.

CCIC (2010a) Earth Probe DCIP Survey for Treasury Metals Inc., May 2010 Summary Powerpoint Presentation.

CCIC (2010b) Trench Map Summary Report, Goliath Property, Ontario, Canada for Treasury Metals Inc., Date of Report: August 23, 2010.

Conestoga Rovers & Associates (2010) Mine Permitting Scoping Study, Treasury Metals – Goliath Project, Zealand Township, Ontario, prepared for Treasury Metals Inc.

Corona (1997) Corona Gold Corporation (1997) Annual Report 1996.

Corona (1998) Corona Gold Corporation (1998) Annual Report 1997.

Corona (1999) Corona Gold Corporation (1999) Annual Report 1998.

Corona (2001) Corona Gold Corporation (2001) Annual Information Form, dated May 11, 2001.

Dubé, B., Gosselin, P., Hannington, M., and Galley, A. (2006) Gold-rich Volcanogenic Massive Sulphide Deposits, In Goodfellow, W.D. ed, Mineral Deposits of Canada: A Synthesis of Major Deposit-Types, District Metallogeny, the Evolution of Geological Provinces, and Exploration Methods, Special Publication 5, p.75-94.

Folinsbee, J, and Johnston, H, "Pre-feasibility metallurgical testing, Goliath Gold Project," report prepared for Treasury Metals Incorporated, June 1, 2011.

Fugro Airborne Surveys (2011) Logistics And Processing Report - Airborne Magnetic And Dighem Survey - Goliath And Goldcliff Properties Dryden, Ontario, Project 11052 for Treasury Metals Incorporated, Date Of Report: November 08, 2011

G&T Metallurgical Services Ltd. (2011) Pre-Feasibility Metallurgical Testing, Goliath Gold Project for Treasury Metals Incorporated, Report KM2906 dated June 1, 2011

Gordon, R. L. (2007) An Integrated 3D Approach to Deep Search Exploration. Exploration '07 Technical Session: Advances in 3D visualization and Data Integration. September 2007.

Gray, J. and Donkersloot, P. (1999) Thunder Lake Gold Deposit Mineral Resource; Teck Exploration Ltd., internal report, 7p.