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Technical Report on the Tuligtic Project, Puebla State, Mexico

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Contents

Tables

Figures

1 Summary

This Technical Report (the “Report”) is written for the Tuligtic Project (the “Property” or the “Tuligtic Property”), which is held 100 percent (%) by Compania Minera Gorrión S.A. de C.V. (Minera Gorrión), a wholly owned subsidiary of Almaden Minerals Ltd. (together referred to as “Almaden”). The Tuligtic Project comprises two mineral claims totalling 14,229.55 hectares (ha) located within Puebla State, 80 kilometres (km) north of Puebla City, and 130 km east of Mexico City. This report is written to comply with standards set out in National Instrument (NI) 43-101 for the Canadian Securities Administration (CSA), and is a technical summary of available geologic, geophysical, geochemical and diamond drill hole information.

During 2012, Almaden retained APEX Geoscience Ltd. (“APEX”), Giroux Consultants Ltd. (Giroux), and BC Mining Research Ltd. (“BC Mining Research”) to complete an independent technical report on behalf of Almaden specific to the Ixtaca Zone within the Tuligtic Property. The lead author, Mr. Kristopher J. Raffle, P.Geo., Principal of APEX, an independent qualified person as defined by NI 43-101, conducted a property visit on September 23, 2012; and on a previous occasion between October 17 and 20, 2011. The second author, Mr. Gary H. Giroux, P.Eng., MASc., an independent qualified person and Principal of Giroux is responsible for the Mineral Resource Estimate presented in Section 14 of the Technical Report. Mr. Andrew Bamber, B.Sc. (Mech.), Ph.D. (Mining), P.Eng., an independent qualified person and Principal of BC Mining Research is responsible for Section 13: Mineral Processing and Metallurgical Testing. Mr. Raffle is responsible for all other sections of the Technical Report.

Almaden acquired the Cero Grande claim of the Tuligtic Project in 2001 following the identification of surficial clay deposits that were interpreted to represent high-level epithermal alteration. Subsequent geologic mapping, rock, stream silt sampling and induced polarization (IP) geophysical surveys identified porphyry copper and epithermal gold targets within an approximately 5 x 5 km area of intensely altered rock. In July 2010 Almaden initiated a diamond drilling program to test epithermal alteration within the Tuligtic Property, resulting in the discovery of the Ixtaca Zone. The first hole, TU-10-001 intersected 302.42 metres of 1.01 g/t Au and 48 g/t Ag and multiple high grade intervals including 1.67 metres of 60.7 g/t Au and 2122 g/t Ag.

Within the Tuligtic Project, argillaceous limestone of the Late Jurassic to Early Cretaceous Upper Tamaulipas formation is underlain by transitional calcareous clastic rocks including siltstone, grainstone, mudstone, and shale. During the Laramide orogeny the carbonate package was intensely deformed into a series of thrust-related east verging anticlines. Calcareous shale units appear to occupy the cores of the anticlines while the thick bedded limestone/mudstone units occupy the cores of major synclines at the Ixtaca Zone. Limestone basement units are crosscut by intensely altered intermediate composition dykes. The deformed Mesozoic sedimentary sequence is discordantly overlain by epithermal altered Cenozoic bedded crystal tuff of the upper Coyoltepec subunit.

Between 2001 and 2012, Almaden's exploration at the Tuligtic Property included rock and soil geochemical sampling, ground magnetics, IP and resistivity, Controlled Source Audio-frequency Magnetotelluric (CSAMT), and Controlled Source Induced Polarization (CSIP) geophysical surveys.

Of the 436 rock grab samples collected, a total of 45 samples returned assays of greater than 100 parts-per-billion (ppb) gold (Au), and up to 6.14 grams-per-tonne (g/t) Au. A total of 49 rock samples returned assays of greater than 10 g/t silver (Ag) and up to 291 g/t Ag. Basement carbonate units, altered intrusive, and locally calc-silicate skarn mineralization occur as erosional windows beneath unmineralized tuff of the upper Coyoltepec subunit. Surface mineralization at the Ixtaca Zone occurs as limestone boulders containing quartz vein fragments and high level epithermal alteration within overlying volcanic rocks. Epithermal alteration and mineralization is observed overprinting earlier skarn and porphyry style alteration and mineralization. Numerous small skarn-related showings exist on the project. At the Caleva soil anomaly, a 200 x 100 m skarn zone hosts sphalerite, galena and chalcopyrite quartz vein stockwork mineralization along the contact zone between limestone and altered and mineralized intrusive rocks to the east.

The collection of 4,760 soil samples by Almaden between 2005 and 2011 resulted in the identification of five anomalous areas: the Ixtaca, Ixtaca East, Caleva, Azul, and Sol zones. Anomalous thresholds (95^{th} percentile) for gold and silver were calculated to be 20.63 ppb Au and 0.71 ppm Ag, respectively. A total of 238 samples containing anomalous Au were found, including 120 samples with coincident Ag anomalies. The Ixtaca Zone produces the largest Au and Ag response within the Tuligtic Property. Based metals do not correlate significantly with the Ixtaca Zone, and Hg and Sb anomalies occur peripherally within altered volcanic rocks. Base metals correlate well with Au-Ag at the Caleva, Azul, and Sol zones to such an extent they are best termed Cu-Zn (Au-Ag) anomalies. Based on the distribution of soil geochemical anomalies and the mapped geology it is apparent that the overlying post mineral volcanics significantly suppress sedimentary and intrusive basement rock geochemical anomalies. Soil responses are consistent with these zones being prospective for both epithermal and earlier skarn mineralization.

IP and CSAMT resistivity surveys largely reflect surface geology, which is controlled by local topography. Resistivity anomalies occur where surface exposures are dominated by limestone and intrusive lithologies. The anomalies are controlled in part by topographic lows that down-cut through overlying tuff rocks and expose resistive basement lithologies. Conductive anomalies occur along local topographic high ridges and plateaus where accumulations of conductive tuff rocks remain. At the Ixtaca Zone, a northwest trending resistivity and weak chargeability anomaly is centered on the North and Main Ixtaca zones. The anomaly is coincident with the east-verging limestone-cored syncline that hosts the high-grade North and Main Ixtaca zones of mineralization.

From July, 2010 to the November 13, 2012 maiden mineral resource estimate cut-off, Almaden has drilled 225 holes totalling 81,971 m on the Main Ixtaca, Ixtaca North and

Northeast Extension zones. Diamond drilling at 25 to 50 m section spacing has defined the Main Ixtaca and Ixtaca North zones over a strike length of approximately 650 m. High-grade mineralization has been intersected to depths of 200 to 300 m vertically from surface and occurs within a broader zone of mineralization extending laterally (NNW-SSE) over 600 m and to a vertical depth of 600 m below surface. The epithermal vein system at the Main Ixtaca and Ixtaca North zones is associated with two subparallel ENE (060 degrees) trending, subvertical to steeply north dipping dyke zones.

At the Main Ixtaca Zone, a series of 2 m to over 20 m true width dykes occur within an approximately 100 m wide zone. The Ixtaca North dyke zone is narrower and comprises a steeply north-dipping zone of two or three discrete dykes ranging from 5 to 20 m in width. Epithermal vein mineralization occurs both within the dykes and sedimentary host rocks, with the highest grades often occurring within or marginal to the dykes. Vein density decreases outward to the north and south from the dyke zones resulting in the formation of two high-grade zones that lack sharp geologic boundaries. On surface, the Main Ixtaca and Ixtaca North zones are separated by a steep sided ENE trending valley.

The bulk of Main Ixtaca and Ixtaca North zone mineralization is bound within an ENE-verging asymmetric synform. The synfom is cored by a structurally thickened sequence of argillaceous limestone that grades laterally and at depth through transition units, into calcareous shale at depth. The Limestone sequence thins to the west along the rising limb of an ENE-verging antiform. The Main Ixtaca and Ixtaca North vein systems and the dykes transect the antiform sub-perpendicular to the strike of the fold axis. Vein density decreases within shale units coring the antiform and mineralization is confined near the axis of the antiform within a west dipping tabular zone of low-grade mineralization having a true thickness ranging from 150 to 200 m. Mineralized basement rocks are unconformably overlain by crystal tuff, which is also mineralized. High-grade zones of mineralization are present within the tuff vertically above the Main Ixtaca and Ixtaca North vein systems. The high-grade zones transition laterally into low grade mineralization, which together form a broad tabular zone of mineralization at the base of the tuff unit.

The Northeast Extension Zone has a strike length of approximately 350 m as defined by drilling along a series of five ENE (070 degrees) oriented sections spaced at intervals of 50 to 100 m, and near-surface oblique NNW-SSE oriented drill holes. The Northeast Extension Zone dips moderately-steeply to the WSW. High grade mineralization having a true-width ranging from less than 30 and up to 60 m has been intersected beneath approximately 30 m of tuff to a vertical depth of 550 m, or approximately 600 m down-dip. Northeast Extension Zone mineralization is interpreted to occur within the hinge zone of a shale cored antiform. Near surface along the axis of the antiform a narrow zone structurally thinned, brecciated, and mineralized limestone is unconformably overlain by mineralized tuff rocks. At a vertical depth of approximately 80 m below surface, high-grade shale-hosted mineralization dips moderately-steeply WSW subparallel to the interpreted axial plane of the antiform. The footwall of the high-grade zone is marked by a distinct 20 to 30 m true-thickness felsic porphyry dyke (Chemalaco Dyke), which is also mineralized. The Chamelaco Dyke has been interested in multiple drill holes ranging from 250 to 550 m vertically below surface, and its lower contact currently marks the base of Northeast Extension Zone mineralization.

Giroux Consultants Ltd. prepared the Maiden mineral resource estimate for the Ixtaca Deposit based on the results of diamond drilling completed by Almaden. Preliminary metallurgy has shown roughly equivalent metal recoveries for Au and Ag, therefore the mineral resource estimate is presented at a series of Au-equivalent (AuEq) cut-offs based on a three years trailing average price of $1,500 per-ounce Au, and $29 per-ounce Ag, and assuming one could mine to the limits of the mineralized solids and no edge dilution is included. Ixtaca Deposit mineralization has been classified as an inferred and indicated mineral resource according to the definitions from NI 43-101 and from CIM (2005). A cut-off of 0.50 g/t Au has been highlighted as a possible cut-off for open pit mining (Table 17-1 and 17-2). At this time, however, no economic studies have been completed and the economic cut-off is unknown.

Table 1-1. Indicated Resource with AuEq Cut-off for Mineralized Portion of Blocks

AuEq Cut-off (g/t)	Tonnes > Cut-off (tonnes)	Grade>Cut-off			Contained Metal x1000		
		Au (g/t)	Ag (g/t)	AuEq (g/t)	Au (ozs)	Ag (ozs)	AuEQ (ozs)
0.10	191,390,000	0.24	13.54	0.50	1,465	83,320	3,077
0.20	133,100,000	0.31	17.81	0.66	1,335	76,210	2,807
0.25	113,720,000	0.35	19.80	0.73	1,269	72,390	2,669
0.30	97,840,000	0.38	21.80	0.80	1,202	68,580	2,526
0.40	73,610,000	0.45	25.87	0.95	1,074	61,230	2,258
0.50	**56,990,000**	**0.52**	**29.91**	**1.10**	**960**	**54,800**	**2,019**
0.60	44,920,000	0.59	34.05	1.25	856	49,180	1,807
0.70	36,130,000	0.66	38.15	1.40	767	44,320	1,624
0.80	29,690,000	0.73	42.10	1.54	692	40,190	1,469
1.00	20,920,000	0.85	49.82	1.81	570	33,510	1,218
2.00	5,740,000	1.31	88.14	3.01	241	16,270	556

Table 1-2. Inferred Resource with AuEq Cut-off for Mineralized Portion of Blocks

AuEq Cut-off (g/t)	Tonnes > Cut-off (tonnes)	Grade>Cut-off			Contained Metal x1000		
		Au (g/t)	Ag (g/t)	AuEq (g/t)	Au (ozs)	Ag (ozs)	AuEQ (ozs)
0.10	121,520,000	0.28	14.32	0.56	1,098	55,950	2,180
0.20	86,290,000	0.36	18.81	0.73	1,010	52,190	2,017
0.25	75,110,000	0.40	20.86	0.80	964	50,370	1,937
0.30	65,880,000	0.43	22.93	0.88	917	48,570	1,855
0.40	51,800,000	0.50	27.12	1.02	826	45,170	1,700
0.50	**41,530,000**	**0.56**	**31.41**	**1.16**	**741**	**41,940**	**1,552**
0.60	33,450,000	0.62	35.95	1.31	662	38,660	1,410
0.70	27,370,000	0.68	40.46	1.46	595	35,600	1,283

0.80	23,200,000	0.73	44.37	1.59	544	33,100	1,183
1.00	17,830,000	0.82	50.60	1.80	469	29,010	1,030
2.00	5,080,000	1.14	83.18	2.75	186	13,590	449
3.00	1,420,000	1.49	113.47	3.68	68	5,180	168

Diamond drilling by Almaden has resulted in the identification of an indicated mineral resource of 56.99 million-tonnes, comprising 2.02 million-ounces AuEq at an average grade of 1.10 g/t AuEq; and an inferred mineral resource of 41.53 million-tonnes, comprising 1.55 million-ounces AuEq at an average grade of 1.16 g/t AuEq, each using a cut-off grade of 0.5 g/t AuEq. Roughly 90% of the deposit is hosted by the carbonate units, the remaining 10% in volcanic rocks.

Metallurgical testwork was completed on each of the Ixtaca Zone geologic domains: limestone, limestone/dyke high grade (HG), shale (Northeast Extension Zone) and volcanic tuff material. Modelling shows that a combination of grinding to a p_{80} of 100-150μm plus gravity recovery on the cyclone underflow, with recovery of gold and silver by means of bulk flotation, followed by intensive leaching of the combined gravity and flotation concentrates is a viable process route for the Ixtaca resource. A summary of metallurgical parameters for the main zones tested for this process route is presented in Table 17-3. While an acceptable economic baseline has been established, further opportunities exist for optimising the gold and silver recoveries from the resource, and a programme of metallurgical optimization, including further flotation and cyanidation work is planned.

Table 1-3. Overall Projected Gravity + Flotation + Intensive Leach Recoveries

Zone	Overall Recovery	
	Au (Wt%)	Ag (Wt%)
Dyke	96.8	85.3
Limestone	88.7	78.3
Limestone HG	94.9	87.0
Shale	95.9	81.8
Tuff (Volcanic)	54.1	61.9

Based on the results of diamond drilling and the Maiden mineral resource estimate, additional drilling is warranted to expand the Ixtaca Deposit mineral resource. Further diamond drilling is should test the possibility of additional limestone-hosted dyke zones to the north and south of the Main Ixtaca and Ixtaca North zones. Additional diamond drilling to the north and south along the hinge of axis of shale-cored antiforms at the Northeast Extension Zone and west of the Main Ixtaca and Ixtaca North zones is also warranted.

Subsequent to the November 13, 2012 drilling cuttoff for the resource, Almaden announced the discovery of a new volcanic-hosted high grade area along the trend of the Main Ixtaca Zone with holes TU-12-222, 224, 225 and 227, all drilled form the same setup. These holes were drilled on section 11+000E, outside the resource shell, and located 50 m northeast of the closest drill holes that were part of the resource. For the first time in the Ixtaca drill program visible gold was identified in one of these holes, TU-

12-224. Intersections in this new zone included 134.20 m of 4.1 g/t AuEq (3.76 g/t Au and 18.1 g/t Ag). This new zone is indicative of the potential for teh resource to grow in this area as well as elsewhere where mineralization has yet to be constrained.

Diamond drilling should include, but not be limited to, diamond drilling of an additional 40,000 metres to expand the Ixtaca Deposit mineral resource. The estimated cost to complete additional diamond drilling is $4,400,000 (Phase 1). Concurrent with ongoing exploration of the Ixtaca Deposit, baseline environmental, hydro-geological and open pit optimization engineering studies should be initiated towards completion of a preliminary economic assessment (PEA). The estimated cost to complete engineering studies is $500,000 (Phase 2).

2 Introduction

This Technical Report (the "Report") is written for the Tuligtic Project (the "Property" or the "Tuligtic Property"), which is held 100 percent (%) by Compania Minera Gorrión S.A. de C.V. (Minera Gorrión), a wholly owned subsidiary of Almaden Minerals Ltd. (together referred to as "Almaden"). The Tuligtic Project comprises two mineral claims totalling 14,229.55 hectares (ha) within Puebla State, Mexico (Figure 4-1).

During 2012, Almaden retained APEX Geoscience Ltd. ("APEX"), Giroux Consultants Ltd. (Giroux), and BC Mining Research Ltd. ("BC Mining Research") to complete an independent technical report on behalf of Almaden specific to the Ixtaca Zone within the Tuligtic Property. The lead author, Mr. Kristopher J. Raffle, P.Geo., Principal of APEX, an independent qualified person as defined by NI 43-101, conducted a property visit on September 23, 2012; and on a previous occasion between October 17 and 20, 2011. The second author, Mr. Gary H. Giroux, P.Eng., MASc., an independent qualified person and principal of Giroux is responsible for the Mineral Resource Estimate presented in Section 14 of the Technical Report. Mr. Andrew Bamber, B.Sc. (Mech.), Ph.D. (Mining), P.Eng., an independent qualified person and principal of BC Mining Research is responsible for Section 13: Mineral Processing and Metallurgical Testing. Mr. Raffle is responsible for all other sections of the Technical Report.

This report is written to comply with standards set out in National Instrument (NI) 43-101 for the Canadian Securities Administration (CSA), and is a technical summary of available geologic, geophysical, geochemical and diamond drill hole information. The authors, in writing this report use sources of information as listed in the references section. Government reports were prepared by qualified persons holding post-secondary geology, or related university degree(s), and are therefore deemed to be accurate. These reports, which were used as background information, are referenced in this Report in the "Geological Setting and Mineralization" section below. All currency amounts referred to in this Report are in Canadian dollars or Mexican pesos where indicated. All units in this Report are metric and Universal Transverse Mercator (UTM). Coordinates in this report and accompanying illustrations are referenced to North American Datum (NAD) 1983, Zone 14.

3 Reliance on Other Experts

With respect to legal title to the Cerro Grande and Cerro Grande 2 mineral claims, which comprise the Tuligtic Property, the authors have relied on the opinion of Lic. Mauricio Heiras Garibay. In a report provided to the authors on August 20, 2012, Mr. Heiras warrants that Minera Gorrión maintains 100% ownership of the two mineral claims comprising the Tuligtic Property via a December 13, 2011 Assignment of Rights Agreement completed with Minera Gavilán, S.A. de C.V., also a wholly owned subsidiary of Almaden. The claims are shown as being in good standing and held 100% by Minera Gavilán, S.A. de C.V on the Mexico Integrated System of Mining Administration (SIAM) website (http://www.economia-dgm.gob.mx/cartografia/).

4 Property Description and Location

The Tuligtic Project consists of two mineral claims totaling 14,229.55 ha (Table 4-1, and Figure 4-2). Almaden acquired the claims during 2001 as part of a regional exploration program. Minera Gorrión maintains 100% ownership of the two mineral claims comprising the Tuligtic Property via a December 13, 2011 Assignment of Rights Agreement completed with Minera Gavilán S.A. de C.V. also a wholly owned subsidiary of Almaden. The Property is not subject to any royalties, back-in rights, payments or other agreements and encumbrances. Almaden holds three (3) additional mineral claims having a total area of approximately 58,700 ha that surround the Tuligtic Project (Figure 4-2).

Table 4-1. Tuligtic Project Mineral Claims

Claim Name	Claim Number	Valid Until Date	Area (hectares)
Cerro Grande	219469	March 5, 2059	11,201.55
Cerro Grande 2	233434	February 23, 2059	3,028
		Total	**14,229.55**

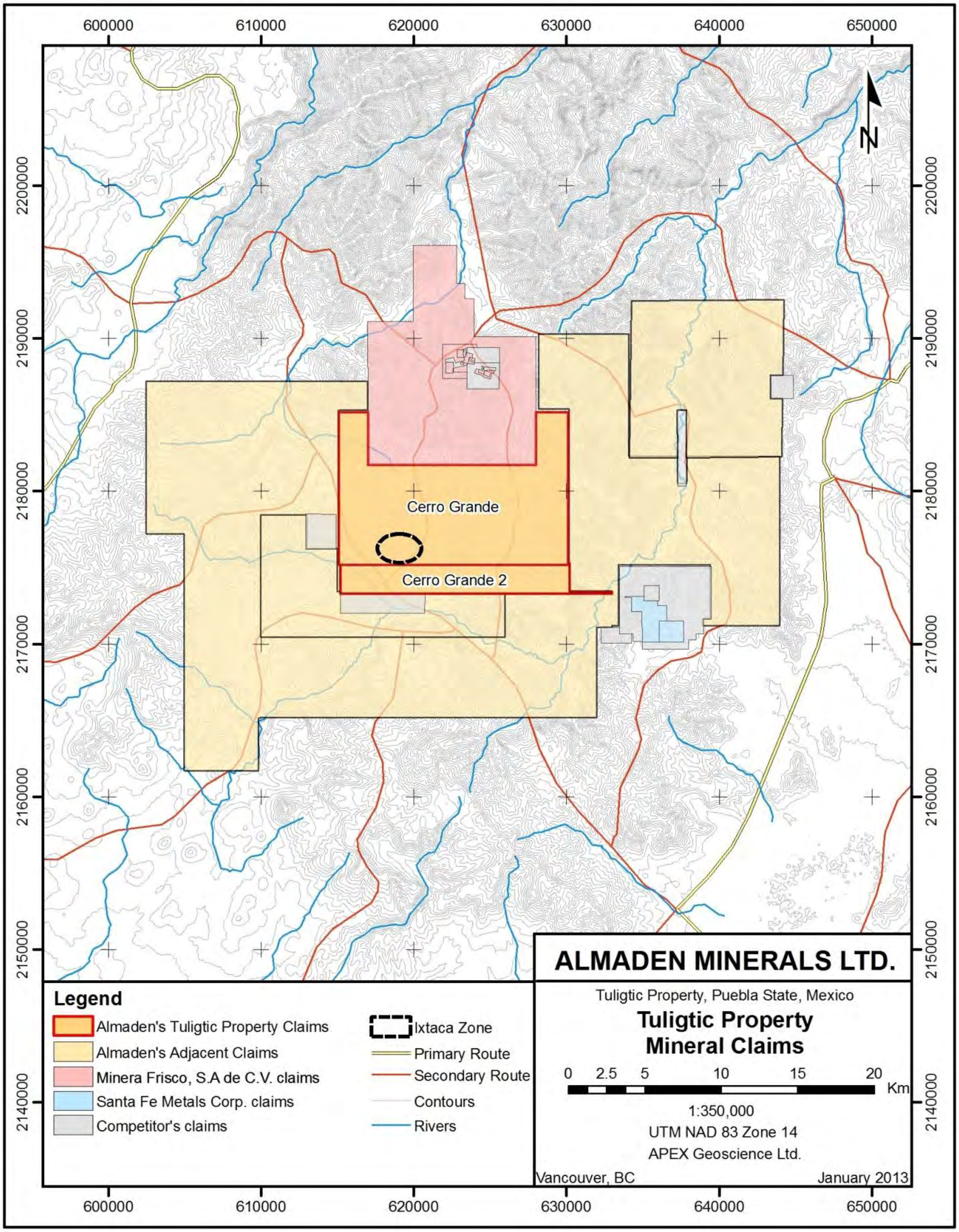
The Property is located at: 19 degrees 40 minutes north latitude and 97 degrees 51 minutes west longitude; or UTM NAD83 Zone 14 coordinates: 618,800 m east and 2,176,100 m north. The Tuligtic Project is road accessible and is located within Puebla State, 80 kilometres (km) north of Puebla City, and 130 km east of Mexico City.

Following an amendment to the Mining Law of Mexico (the “Mining Law”) on April 28, 2005, there is no longer a distinction between the exploration mining concessions and exploitation mining concessions. The Mining Law permits the owner of a mining concession to conduct exploration for the purpose of identifying mineral deposits and quantifying and evaluating economically usable reserves, to prepare and to develop exploitation works in areas containing mineral deposits, and to extract mineral products from such deposits. Mining concessions have a duration of 50 years from the date of their recording in the Registry and may be extended for an equal term if the holder requests an extension within five years prior to the expiration date.

To maintain a claim in good standing holders are required to provide evidence of the exploration and/or exploitation work carried out on the claim under the terms and conditions stipulated in the Mining Law, and to pay mining duties established under the Mexican Federal Law of Rights, Article 263. Exploration work can be evidenced with investments made on the lot covered by the mining claim, and the exploitation work can be evidenced the same way, or by obtaining economically utilizable minerals. The

Figure 4-1. General location

Figure 4-2. Tuligtic Project Mineral Claims



Regulation of the Mining Law indicates the minimum exploration expenditures or the value of the mineral products to be obtained (Table 4-2).

Table 4-2. Exploitation Claim Minimum Expenditure/Production Value Requirements

Area (hectares)	Fixed quota in Pesos (CAD$)	Additional annual quota per hectare in Pesos (CAD$ per hectare)			
		1st year	2nd to 4th year	5th to 6th year	7th year and after
<30	262.24 (20.98)	10.48 (0.84)	41.95 (3.36)	62.93 (5.03)	63.93 (5.11)
30 - 100	524.49 (41.96)	20.97 (1.68)	83.91 (6.71)	125.88 (10.07)	125.88 (10.07)
100 - 500	1,048.99 (83.92)	41.95 (3.36)	125.88 (10.07)	251.75 (20.14)	251.75 (20.14)
500 - 1000	3,146.98 (251.76)	38.81 (3.10)	119.91 (9.59)	251.75 (20.14)	503.51 (40.28)
1000 - 5000	6,293.97 (503.52)	35.66 (2.85)	115.39 (9.23)	251.75 (20.14)	1,007.03 (80.56)
5000 - 50000	22,028.92 (1,762.31)	32.52 (2.60)	111.19 (8.90)	251.75 (20.14)	2,014.07 (161.13)
> 50000	209,799.28 (16,783.94)	29.37 (2.35)	104.9 (8.39)	251.75 (20.14)	2,014.07 (161.13)

*Using a conversion of 1 MEX peso = 0.08 CAD$

The Tuligtic Property is currently subject to annual exploration/exploitation expenditure requirements of approximately CAD$130,000.00 per year.

Subject to the Mexico Mining Laws, any company conducting exploration, exploitation and refining of minerals and substances requires previous authorization from the Secretary of Environment and Natural Resources (SEMARNAT). Because mining exploration activities are regulated under Official Mexican Norms (specifically NOM-120) submission of an Environmental Impact Statement (“Manifestacion de Impacto Ambiental” or “MIA”) is not required provided exploration activities to not exceed disturbance thresholds established by NOM-120. Exploration activities require submission to SEMARNAT of a significantly less involved “Preventive Report” (Informe Preventivo) which outlines the methods by which the owner will maintain compliance with applicable regulations. If the exploration activities detailed within the Preventive Report exceed the disturbance thresholds established by NOM-120, SEMARNAT will inform the owner that an MIA is required within a period of no more than 30 days.

The present scale of exploration activities within the Tuligtic Project are subject to NOM-120 regulation. In future, if significantly increased levels of exploration activities are anticipated submission of an Environmental Impact Statement may be required.

Almaden has negotiated surface land use agreements with landowners within the area affected by diamond drilling activities.

At present, the author is not aware of any environmental liabilities to which the Property may be subject, or any other significant risk factors that may affect access, title, or Almaden's right or ability to perform work on the Property.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The Ixtaca Zone, the epithermal gold-silver target within the Tuligtic Property, is located 8 km northwest of the town of San Francisco Ixtacamaxtitlán, the county seat of the municipality of Ixtacamaxtitlán, Puebla State.

The project is accessible by driving 40 km east along Highway 119 from Apizaco, an industrial centre located approximately 50 km north of Puebla City, and then north approximately 20 km along a gravel road to the town of Santa Maria. The trip from Apizaco to site can be driven in approximately 1.5 hours. There is also access to the Property using gravel roads from the northeast via Tezhuitan and Cuyoaco, from the south via Libres and from the northwest via Chignahuapan. The Xicohtencatl Idustrial complex lies 30 km southwest of the Tuligtic Project, and houses agricultural chemical, biomedical and industrial manufacturing facilities. Puebla, the fourth largest city in Mexico has a population in excess of 4 million people, and includes one of the largest Volkswagen automotive plants outside Germany.

The Topography on the Tuligtic Project is generally moderate to steep hills with incised stream drainages. Elevation ranges from 2,300 metres (m) above sea level in the south to 2,800 m in the north. Vegetation is dominantly cactus and pines and the area is also somewhat cultivated with vegetables, beans, corn and pastures. The region has a temperate climate with average temperatures ranging from 19°C in June to 10°C in December. The area experiences about 600 mm of precipitation annually with the majority falling during the rainy season, between June and September.

Exploration can be conducted year round within the Property; however, road building and drilling operations may be impacted by weather to some degree during the rainy season.

Electricity is available on the Property as the national electricity grid services nearby towns such as Santa Maria and Zacatepec. Water for exploration is available from year-round spring fed streams, wells, and the Apulco River 4 km south of the Ixtaca Zone.

Almaden has negotiated surface land use agreements with landowners within the area affected by diamond drilling activities. Additional or revised landowner agreements may be required in the event advanced operations are anticipated (for example potential tailings storage areas, potential waste disposal areas, and potential processing plant sites). The Mining Law provides claim owners the right to obtain the expropriation, temporary occupancy or creation of land easements necessary to carry out exploration and mining operations.

6 History

Throughout the Property there is evidence that surficial clay deposits were once mined. This clay alteration attracted Almaden to the area and was interpreted to represent high-level epithermal alteration. To the best of the authors knowledge no modern exploration was conducted on the project prior to Almaden's acquisition of claims during 2003.

On May 9, 2002 Almaden entered into a joint venture agreement with BHP Billiton World Exploration Inc. (BHP) to undertake exploration in eastern Mexico. Initial helicopter-borne reconnaissance programs were completed in May 2003 and March 2004 on select targets within the joint venture area of interest. The work resulted in the acquisition of five (5) separate properties; in addition to the previously acquired Cerro Grande claim of the present day Tuligtic Property. Following a review of the initial exploration data, effective January 20, 2005, BHP relinquished its interest in the six properties to Almaden (Almaden, 2005). The joint venture was terminated in 2006 (Almaden, 2006).

During January 2003, Almaden completed a program of geologic mapping, rock, stream silt sampling and induced polarization (IP) geophysical surveys at the Tuligtic Property (then known as the "Santa Maria Prospect"). The exploration identified both a porphyry copper and an epithermal gold target within an approximately 5 x 5 km area of intensely altered rock. At the porphyry copper target, stockwork quartz-pyrite veins associated with minor copper mineralization overprint earlier potassic alteration within a multi-phase intrusive body. A single north-south oriented IP survey line indentified a greater than 2 km long elevated chargeability response coincident with the exposed altered and mineralized intrusive system. Volcanic rocks exposed 1 km to the south of the mineralized intrusive display replacement silicification and sinter indicative of the upper parts of an epithermal system (the "Ixtaca Zone"). Quartz-calcite veins returning anomalous values in gold and silver and textural evidence of boiling were identified within limestone roughly 100 m below the sinter. The sinter and overlying volcanic rocks are anomalous in mercury, arsenic, and antimony (Almaden, 2004).

Additional IP surveys and soil sampling were conducted in January and February 2005, further defining the porphyry copper target as an area of high chargeability and elevated copper, molybdenum, silver and gold in soil. A total of eight (8) east-west oriented lines, 3 km in length, spaced at intervals of 200 m were completed over mineralized intrusive rocks intermittently exposed within gullies cutting through the overlying unmineralized ash deposits (Almaden, 2006).

The Tuligtic Property was optioned to Pinnacle Mines Ltd. in 2006 and the option agreement was terminated in 2007 without completing significant exploration (Almaden, 2007). During 2008, Almaden completed a program of alteration mapping and stream sediment sampling (Almaden, 2008).

The Property was subsequently optioned to Antofagasta Minerals S.A. (Antofagasta) on March 23, 2009. During 2009 and 2010 Antofagasta, under Almaden operation, carried out IP geophysical surveys and a diamond drill program targeting the copper porphyry prospect. Three additional IP survey lines were completed, and in conjunction with the previous nine (9) IP lines, a 2 x 2.5 km chargeability high anomaly, open to the west and south, was defined (Almaden, 2011). The 2009 drilling consisted of 2,973 m within seven (7) holes that largely intersected skarn type mineralization. Highlights of the drill program include 38 metres of 0.13% Cu (copper) from 164 to 202 m and 0.11% Cu from 416 to 462 m within hole DDH-01; 20 m of 0.17% Cu from 94 to 114 m and 26 m of 0.14% Cu from 316 to 342 m in hole DDH-02; 58 m of 0.17% Cu from 366 to 424 m in hole DDH-03 (including 14 m of 0.27% Cu from 410 to 424 m); 2 m of 0.63% Cu from 18 to 20 m in hole DDH-04; and 20 m of 0.11% Cu from 276 to 296 m and 8 m of 0.13% Cu in hole DDH-05. Molybdenum values were anomalous ranging up to 801 parts-per-million (ppm) (0.08%). Elevated gold values were also encountered including 2 m of 1.34 grams-per-tonne (g/t) from 178 to 180 m in DDH-01. On February 16, 2010, Almaden announced that Antofagasta has terminated its option to earn an interest in the Property (Almaden, 2009).

In July 2010 Almaden initiated a preliminary diamond drilling program to test epithermal alteration within the Tuligtic Property, resulting in the discovery of the Ixtaca Zone. The target was based on exploration data gathered by Almaden since 2001 including high gold and silver in soil and a chargeability and resistivity high anomaly (derived from an IP geophysical survey conducted by Almaden) topographically beneath Cerro Caolin, a prominent clay and silica altered hill. This alteration, barren in gold and silver, had been interpreted by Almaden to represent the top of an epithermal system which required drill testing to depth. The first hole, TU-10-001 intersected 302.42 metres of 1.01g/t gold and 48g/t silver and multiple high grade intervals including 1.67 metres of 60.7g/t gold and 2122g/t silver.

7 Geological Setting and Mineralization

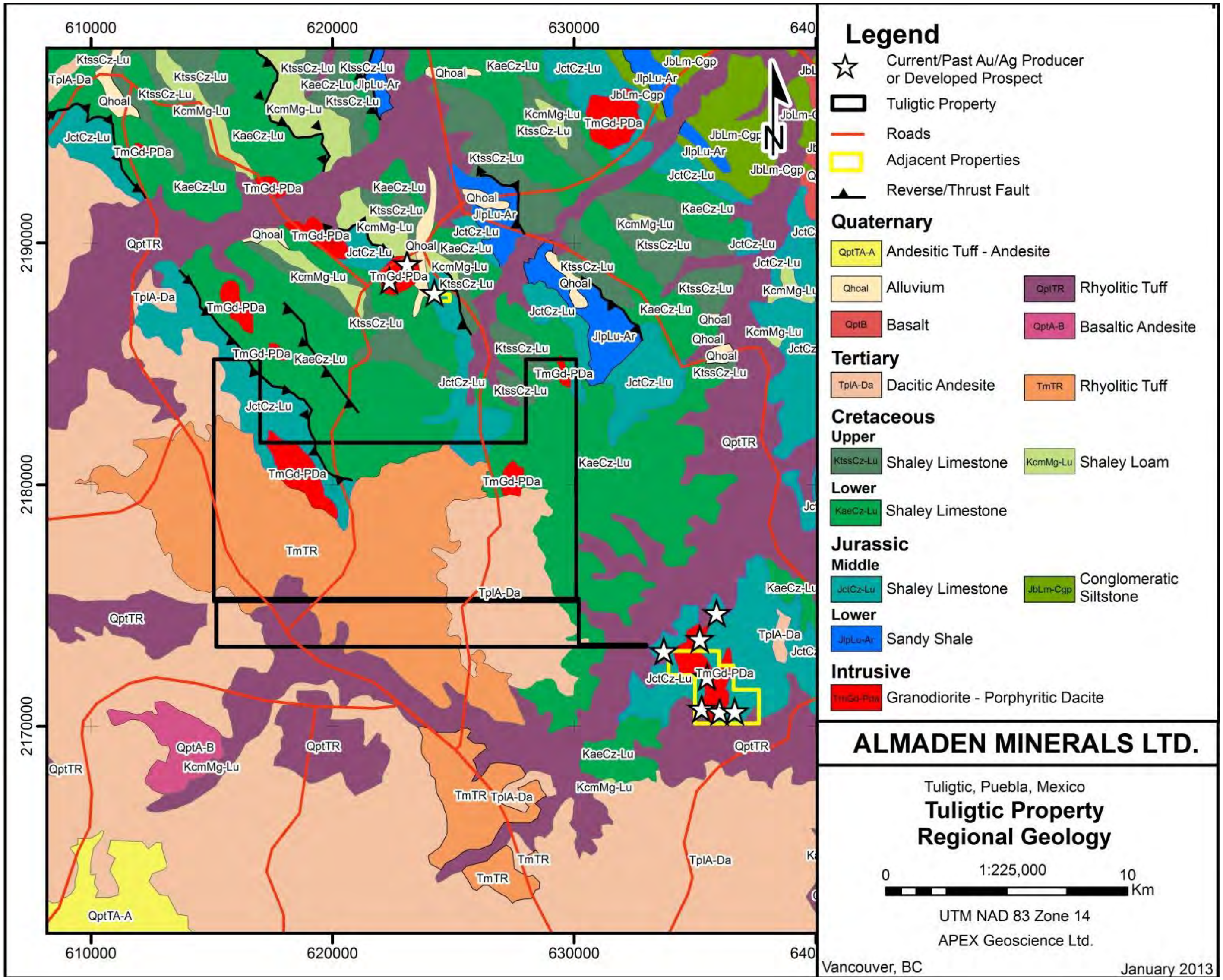
7.1 Regional Geology

The Tuligtic project is situated within the Trans Mexican Volcanic Belt (TMVB), a Tertiary to recent intrusive volcanic arc extending approximately east-west across Mexico from coast to coast and ranging in width from 10 to 300 km (Figure 7-1). The TMVB is the most recent episode of a long lasting magmatic activity which, since the Jurassic, produced a series of partially overlapping arcs as a result of the eastward subduction of the Farallon plate beneath western Mexico (Ferrari, 2011). The basement rocks of the eastern half of the TMVB are Precambrian terranes, including biotite orthogneiss and granulite affected by granitic intrusions, grouped into the Oaxaquia microcontinent (Ferrari et al., 2011; Fuentes-Peralta and Calderon, 2008). These are overlain by the Paleozoic Mixteco terrane, consisting of a metamorphic sequence known as the Acatlan complex and a fan delta sedimentary sequence known as the Matzitzi formation. Another sedimentary complex is found on top of the Mixteco terrane, represented by various paleogeographic elements such as the Mesozoic basins of Tlaxiaco, Zongolica, Zapotitlan, and Tampico-Misantla (Fuentes-Peralta and Calderon, 2008). The subducting plates associated with the TMVB are relatively young, with the Rivera plate dated at 10 Ma (million years) and the Cocos plate at 11 to 17Ma.

The timing and nature of volcanism in the TMVB has been described by Garcia-Palomo et al. (2002). The oldest volcanic rocks in the central-eastern part of the TMVB were erupted ~13.5 Ma ago, followed by a nearly 10 Ma hiatus. Volcanic activity in the area resumed around 3.0-1.5 Ma. The composition of volcanic rocks ranges from basalt to rhyolite and exhibits calc-alkaline affinity. Extensive silicic volcanism in this area has been related to partial melting of the lower crust, hydrated by infiltration of slab-derived fluids during flat subduction (Ferrari et al., 2011). The Sierra Madre Occidental (SMO) style of volcanism is silicic and explosive as opposed to intermediate and effusive volcanism characteristic of the TMVB. Volcanic centres in the region were controlled by NE-SW trending normal faults, associated with horst-and-graben structures, resulting from a stress field with a least principal stress (σ_3) oriented to the NW.

The regional trend of the arc rocks is WNW, though more northerly trending transform faults, forming at a high angle to the TMVB, provide a structural control on the volcanic units (Coller, 2011). Compressional strike-slip and extensional faults also developed as a result of compressional and extensional periods during subduction. The NE-SW San Antonio fault system, which was still active during Late Pliocene, before the reactivation of the Taxco-Queretaro fault system, is characterized by extensional left-lateral oblique-slip kinematics (Coller, 2011). Bellotti et al. (2006) showed that NNW trending regional faults were right lateral in the Miocene, whereas the NNE to N-S trending faults observed at Ixtaca by Coller (2011) are related to the regional horst-and-graben

Figure 7-1. Regional Geology



development and likely to be purely extensional with possibly a component of right lateral movement, or transtensional.

7.2 Property Geology

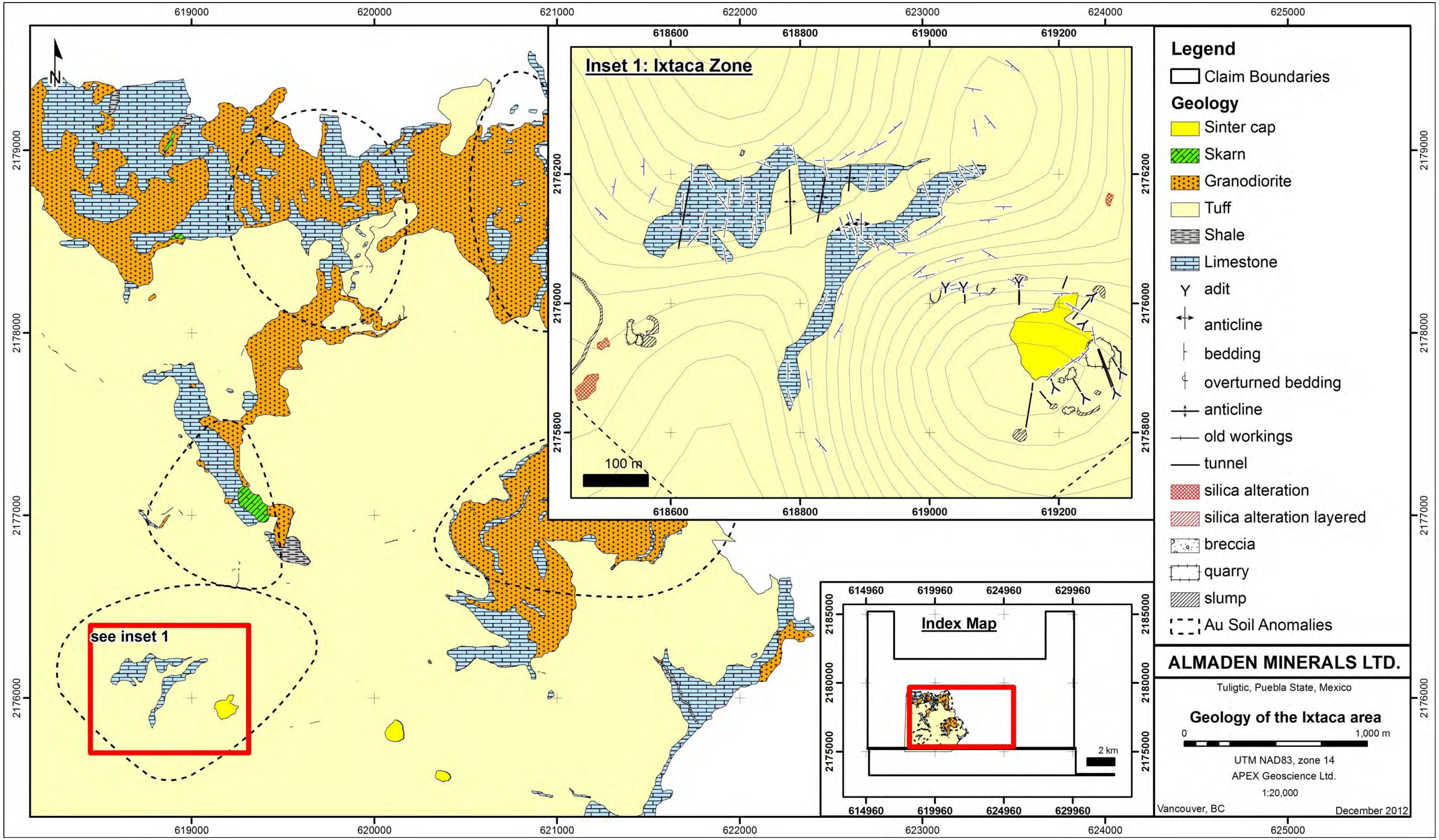
The stratigraphy of the Tuligtic area can be divided into two main sequences: a Mesozoic sedimentary rock sequence related to the Zongolica basin and a sequence of late Tertiary igneous extrusive rocks belonging to the TMVB (Fuentes-Peralta & Calderon, 2008; Tritlla et al., 2004). The sedimentary sequence is locally intruded by plutonic rocks genetically related to the TMVB (Figure 7-2). The sedimentary complex at Tuligtic corresponds to the Upper Tamaulipas formation (Reyes-Cortes 1997). This formation, Late Jurassic to Early Cretaceous in age, has regionally been described (Reyes-Cortes, 1997) as a sequence of grey-to-white limestone, slightly argillaceous, containing bands and nodules of black flint. The drilling conducted by Almaden has allowed for more detailed characterisation of the Upper Tamaulipas Formation carbonate units in the Tuligtic area. The sequence on the Project consists of clastic calcareous rocks. An argillaceous limestone (termed mudstone) grades into what have been named transition units and shale. The transition units are calcareous siltstones and grainstones. These rocks are not significant in the succession but mark the transition from mudstone to underlying calcareous shale. Typical of the transition units are coarser grain sizes. The lower calcareous “shale” units exhibit pronounced laminated bedding and are typically dark grey to black in colour, although there are green coloured beds as well. The shale units appear to have been subjected to widespread calc-silicate alteration.

Both the shale and transition units have very limited surface exposure and may be recessive. The entire carbonate package of rocks were intensely deformed by the Laramide orogeny, showing complex thrusting and chevron folding in the hinge zones of a series of thrust-related east verging anticlines in the Ixtaca area (Tritlla et al., 2004; Coller, 2011). The calcareous shale units appear to occupy the cores of the anticlines while the thick bedded limestone/mudstone units occupy the cores of major synclines identified in the Ixtaca zone.

The Tamaulipas limestones were intruded in the mid-Miocene by a series of magmatic rocks. The compositions are very variable, consisting of hornblende-biotite-bearing tonalites, quartz-plagioclase-hornblende diorites, and, locally, aphanitic diabase dykes (Carrasco-Nunez et al., 1997). In the central part of the Tuligtic project porphyry mineralisation is hosted by and associated with a hornblende-biotite-quartz phyric granodiorite body. The contact between the granodiorite and the limestone is marked by the development of a prograde skarn.

In the Ixtaca epithermal area of the project, the limestone basement units are crosscut by intermediate dykes that are often intensely altered. In the vicinity of the Ixtaca zone these dykes are well mineralised especially at their contacts with limestone country rock. Petrography has shown that epithermal alteration in the dykes, marked by illite, adularia, quartz and pyrite has overprinted earlier calc-silicate endoskarn mineralogies (Leitch, 2011). Two main orientations have been identified for dykes in the Ixtaca area;

Figure 7-2. Property Geology



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060 degrees (parallel to the Main Ixtaca and Ixtaca North zones) and 330 degrees (parallel to the Northeast Extension Zone).

An erosional unconformity surface was formed subsequent to the intrusion of the porphyry mineralisation-associated granodiorites. This paleo topographical surface locally approximates the current topography. Although not well exposed the unconformity is marked by depression localised accumulations of basal conglomerate comprised of intrusive and sedimentary boulders.

This deformed Mesozoic sedimentary sequence is discordantly overlain by late Cenozoic extrusive rocks whose genetic and tectonic interrelations are yet to be fully explained. Two main volcaniclastic units have been recognized in the area of Tuligtic: the Coyoltepec Pyroclastic deposit and the Xaltipan Ignimbrite (Carrasco-Nunez et al., 1997). Both units were covered by a thin (up to 1 m) quaternary ‘tegument’ (Morales-Ramirez 2002) of which only a few patches are left in the area of the property, but it is still widespread in the surrounding areas. This tegument is unconsolidated and composed of a very recent ash fall tuff rich in heavy minerals (mainly magnetite, apatite, and pyroxene).

The extensively altered pre-mineral Coyoltepec pyroclastic deposit has been divided by Carrasco-Nunez et al. (1997) into two subunits: the lower Coyoltepec subunit, which is not exposed in the area of the project, consists of a stratified sequence of surge deposits and massive, moderately indurated pyroclastic flow deposits with minor amounts of pumice and altered lithic clasts.

The upper Coyoltepec subunit, which is the main unit outcropping in the Tuligtic area, consists of a basal breccia or conglomerate overlain by bedded crystal tuff. The basal breccia is comprised of a lithic rhyolite tuff matrix composed of massive, indurated, coarse-gravel sized, lithic-rich pyroclastic flow deposits with pumice, andesitic fragments, free quartz, K-feldspar, plagioclase crystals, and minor amounts of limestone and shale clasts (Tritlla et al., 2004). The Coyoltepec volcanics are altered and mineralised. Gold silver mineralisation is marked by widespread disseminated pyrite and quartz-calcite veinlets.

The post-mineral Xaltipan ignimbrite is not seen in hte Ixtaca area and mainly found in topographic lows south of the Tuligtic property. It consists of a very recent (0.45 ± 0.09 Ma, Carrasco-Nunez et al., 1997), pinkish to brownish-grey rhyolitic ignimbrite unit with different grades of welding, containing abundant pumice fragments, andesite lithic fragments, and small clasts of black obsidian (Tritlla et al., 2004).

7.3 Mineralization

Two styles of alteration and mineralization have been identified in the area: (1) copper-molybdenum porphyry style alteration and mineralization hosted by diorite and quartz-diorite intrusions; (2) silver-gold low-sulphidation epithermal quartz-bladed calcite veins hosted by carbonate rocks and spatially associated with overlying volcanic hosted texturally destructive clay alteration and replacement silicification.

Outcropping porphyry-style alteration and mineralization is observed in the bottoms of several drainages where the altered intrusive complex is exposed in erosional windows beneath post mineral unconsolidated ash deposits. Multiple late and post mineral intrusive phases have been identified crossing an early intensely altered and quartz-veined medium-grained feldspar phyric diorite named the Principal Porphyry. Other intrusive types include late and post mineral mafic dykes and an inter-mineral feldspar-quartz phyric diorite. Late mineral mafic dykes are fine grained and altered to chlorite with accessory pyrite. Calc-silicate (garnet-clinopyroxene) altered limestone occurs in proximity to the intrusive contacts and is crosscut by late quartz-pyrite veins. Early biotite alteration of the principal porphyry consists of biotite-orthoclase flooding of the groundmass. Quartz veins associated with early alteration have irregular boundaries and are interpreted to be representative of A-style porphyry veins. These are followed by molybdenite veins which are associated with the same wall rock alteration. Chalcopyrite appears late in the early alteration sequence. Late alteration is characterized by intense zones of muscovite-illite-pyrite overprinting earlier quartz-K-feldspar-pyrite ± chalcopyrite veining and replacing earlier hydrothermal orthoclase and biotite. Stockwork quartz-pyrite crosscuts the A-style veins and is associated with muscovite-illite alteration of biotite. The quartz-sericite alteration can be texturally destructive resulting in white friable quartz veined and pyrite rich rock. Pyrite is observed replacing chalcopyrite and in some instances chalcopyrite remains only as inclusions within late stage pyrite grains.

Epithermal mineralisation on the Tuligtic property is considered to have no genetic relationship to the porphyry alteration and mineralisation described above. The epithermal system is well preserved and there is evidence of a paleosurface as steam heated kaolinite and replacement silica alteration occur at higher elevations where the upper part of the Coyoltepec pyroclastic deposit is preserved.

The veining of Ixtaca epithermal system displays characteristics representative of intermediate and low sulphidation deposits. These include typical ore and gangue mineralogy (electrum, sphalerite, galena, adularia, carbonates), mineralisation dominantly in open space veins (colloform banding, cavity filling). Assaying has indicated high contents of gold and silver. The high gold contents are rare in Mexico, where epithermal systems are dominantly silver-rich. Mineralized hydrothermal breccias showing multiphase development are commonly encountered within the main veins. Hydrothermal silicic/carbonate breccia zones occur within the limestone and dip steeply. These breccias are dominantly controlled by the main faults.

The Upper Tamaulipas formation, the dykes that crosscut it and the upper Coyoltepec volcanic subunit are the main host rocks to the epithermal vein system at Ixtaca. In the Main and Ixtaca North zones veining strikes dominantly ENE-WNW (060 degrees) parallel to a major dyke trend and at a very high angle to the N to NNW bedding and fold structures within the limestones. The veins of the Northeast Extension Zone are hosted by the shaley carbonate units and strike to the NNW, dipping to the SSW. In the footwall to Northeast Extension Zone a parallel dyke has been identified which is altered and mineralised. The Northeast Extension Zone and the dyke are interpreted to strike parallel to bedding and to core an antiform comprised of shale.

There appear to be two major sets of veins which are related to the large structural setting. The main set of veins strike ENE (060 degrees) and dip steeply to the north and south and are hosted by limestone and dykes that crosscut the limestone. The second set of veins strike NNW (330 degrees) and dip shallowly to the west and is likely related to pre-existing bedding and structures within the limestone and shaley units. The Northeast Extension Zone of veining strikes NNW (330 degrees) and is hosted by west dipping shale interpreted to core an overturned antiform.

Studies of mineral assemblages in hand specimen, transmitted and reflected light microscopy and SEM analyses were carried out in order to construct a paragenetic sequence of mineral formation. This work completed by Herrington (2011) and Staffurth (2012) revealed that veining occurred in three main stages. The first stage is barren calcite veining. This is followed by buff brown and pink colloform carbonate and silicate veins containing abundant silver minerals and lower gold. The third stage of veining contains both gold and silver mineralization. The dominant gold-bearing mineral is electrum, with varying Au:Ag ratios. The majority of grains contain 40-60 wt (weight) % gold but a few have down to 20 wt% (Staffurth, 2012). Gold content occasionally varies within electrum grains, and some larger grains seem to be composed of aggregates of several smaller grains of differing composition (Staffurth, 2012). Electrum often appears to have been deposited with late galena-clausthalite both of which are found as inclusions or in fractures in pyrite. It is also closely associated with silver minerals as well as sphalerite and alabandite. Gold is also present in uytenbogaardtite (Ag_3AuS_2). This mineral is associated with electrum, chalcopyrite, galena, alabandite, silver minerals and quartz in stage three mineralisation (Herrington, 2011; Staffurth, 2012). Apart from electrum, the dominant silver bearing minerals are polybasite (-pearceite) and argentian tetrahedrite plus minor acanthite-naumannite, pyrargyrite and stephanite. They are associated with sulphides (Figure 8-1) or are isolated in gangue minerals (Staffurth, 2012).

The vein-related mineralisation at Ixtaca does not have hard geologic boundaries. The mineralised zones are essentially vein zones the outer boundaries of which are grade boundaries associated with decreased vein density.

7.3.1 Steam Heated Alteration, Replacement Silification and Other Surficial Geothermal Manifestations

One of the most striking features of the Ixtaca epithermal system is the kaolinite alteration, replacement silicification, and sinter carapace that remains uneroded in the vicinity of the Ixtaca Zone. This alteration has been identified over a roughly 5 x 5 km area and is interpreted to represent the upper levels of a preserved epithermal system. All three alteration types have formed in the tuffaceous units. When the source alkali-choride epithermal fluids boil, along with water vapour, CO_2 and H_2S also separate. These gases rise and above the water table H_2S condenses in the vadose zone forming H_2SO_4. Near surface the H_2SO_4 alters volcanic rocks to kaolinite and alunite and can dissolve volcanic glass (Hedenquist and Henley 1985b). This process is interpreted to be responsible for the kaolinite alteration, known as steam-heated alteration in the economic geology literature (eg White and Hedenquist, 1990). The resulting silica laden fluid can transport and re precipitate silica at the water table in permeable host rocks. This mechanism can result in large tabular alteration features often referred to as a silica caps. Since gold is not transported by the gases or sulphuric acid, the silica cap is usually devoid of gold and silver, which is the case at Ixtaca (White and Hedenquist, 1990).

Sinter is diagnostic of modern epithermal systems where silica-rich fluids emanate as hot springs at the earth's surface. Sinters are the highest level manifestation of an epithermal system and consequently the first feature to be removed by erosion. Most epithermal gold-silver deposits that have been recognized show some degree of erosion and ancient sinters are typically poorly preserved in the geological record. The presence of preserved steam heated and replacement silica alteration and sinter at Ixtaca is thus a clear indication that the deposit was not significantly affected by erosion. At Ixtaca, the sinter facies and replacement silicification, where preserved, are located within the altered volcanic units.

8 Deposit Types

The principal deposit-type of interest on the Tuligtic Property is low- to intermediate-sulphidation epithermal gold-silver mineralisation. This style of mineralisation has been recognised at the Ixtaca Zone but property scale high level epithermal alteration suggests that mineralisation of this type can exist elsewhere on the project. These deposits are described more fully below. The Tertiary bodies intruding the Tamaulipas Limestones and the tertiary volcanics, makes the property also prospective for Porphyry copper-gold-molybdenum (Cu-Au-Mo) and peripheral Pb-Zn Skarn deposits.

8.1 Epithermal Gold-Silver Deposits

Gold and silver deposits that form at shallow crustal depths (<1,500 m) are interpreted to be controlled principally by the tectonic setting and composition of the mineralizing hydrothermal fluids. Three classes of epithermal deposits (high-sulphidation, intermediate-sulphidation and low-sulphidation) are recognized by the oxidation state of sulphur in the mineralogy, the form and style of mineralization, the geometry and mineralogy of alteration zoning, and the ore composition (Hedenquist et al., 2000; Hedenquist and White, 2005). Overlapping characteristics and gradations between epithermal classes may occur within a district or even within a single deposit. The appropriate classification of a newly discovered epithermal prospect can have important implications to exploration.

High-sulphidation and intermediate-sulphidation systems are most commonly hosted by subduction-related andesite dacite volcanic arc rocks, which are dominantly calc-alkaline in composition. Low-sulphidation systems are more restricted, generally to rift-related bimodal (basalt, rhyolite) or alkalic volcanic sequences. The gangue mineralogy, metal contents and fluid inclusion studies indicate that near neutral pH hydrothermal fluids with low to moderate salinities form low- and intermediate-sulphidation class deposits whereas high-sulphidation deposits are related to more acidic fluids with variable low to high salinities. Low- and intermediate-sulphidation deposits are typically more vein-style while high-sulphidation deposits commonly consist primarily of replacement and disseminated styles of mineralization with subordinate veining. The characteristics of silver-gold mineralization in the Ixtaca Zone include banded, colloform and brecciated carbonate-quartz veining including locally abundant Mn-carbonate and rhodochrosite indicate that this is primarily an intermediate-sulphidation epithermal district.

The mineralization discovered to date at Ixtaca exhibits features of both the low- and intermediate sulphidation epithermal classes (see Table 8-1). Several of the larger examples of this deposit type occur in Mexico and include the prolific historic epithermal districts of Pachuca, Guanajuato and Fresnillo.

Table 8-1. Classification of Epithermal Deposits

	Low-Sulphidation	Intermediate-Sulphidation	High-Sulphidation
Metal Budget	Au- Ag, often sulphide poor	Ag - Au +/- Pb - Zn; typically sulphide rich	Cu - Au - Ag; locally sulphide-rich
Host Lithology	bimodal basalt-rhyolite sequences	andesite-dacite; intrusion centred district	andesite-dacite; intrusion centred district
Tectonic Setting	rift (extensional)	arc (subduction)	arc
Form and Style of Alteration/Mineralization	vein arrays; open space veins dominant; disseminated and replacement ore minor stockwork ore common; overlying sinter common; bonanza zones common	vein arrays; open space veins dominant; disseminated and replacement ore minor; stockwork ore common; productive veins may be km-long, up to 800 m in vertical extent	verins subordinate, locally dominant; disseminated and replacement ore common; stockwork ore minor.
Alteration Zoning	ore with quartz-illite-adularia (argillic); barren silicification and propylitic (quartz-chlorite-calcite +/- epidote) zones; vein selvedges are commonly narrow	ore with sericite-illite (argillic-sericitic); deep base metal-rich (Pb-Zn +/- Cu) zone common; may be spatially associated with HS and Cu porphyry deposits	ore in silicic core (vuggy quartz) flanked by quartz-alunite-kaolinite (advanced argillic); overlying barren lithocap common; Cu-rich zones (enargite) common
Vein Textures	chalcedony and opal common; laminated colloform-crustiform; breccia; bladed calcite (evidence for boiling)	chalcedony and opal uncommon; laminated colloform-crustiform and massive common; breccias; local carbonate-rich, quartz-poor veins; rhodochrosite common,	chalcedony and opal uncommon; laminated colloform-crustiform veins uncommon; breccia veins; rhodochrosite uncommon

		especially with elevated base metals	
Hydrothermal Fluids	low salinity, near neutral pH, high gas content (CO_2, H_2S); mainly meteoric	moderate salinities; near neutral pH	low to high salinities; acidic; strong magmatic component?
Examples	McLaughlin, CA; Sleeper and Midas, NV; El Penon, Chile; Hishikari, Japan	Arcata Peru; Fresnillo Mexico; Comstock NV; Rosia Montana Romania	Pierina Peru; Summitville CO

*Altered after Taylor, 2007

The low- and intermediate-sulphidation epithermal gold-silver deposits are generally characterised by open space fill and quartz-carbonate veining, stockworks and breccias associated with gold and silver often in the form of electrum, argentite and pyrite with lesser and variable amounts of sphalerite, chalcopyrite, galena, rare tetrahedrite and sulphosalt minerals, which form in high-level (epizonal) to near-surface environments.

The epithermal veins form when carbonate minerals and quartz precipitate from a cooling and boiling alkali-chloride fluid. Alkali-chloride geothermal fluid are formed from magmatic gases and convecting groundwater and are near neutral in composition. These fluids convect in the upper crust perhaps over a 10 kilometer deep vertical interval and can transport gold, silver and other metals. At roughly 2 km depth these fluids begin to boil, releasing CO_2 and H_2S (carbon-dioxide and hydrogen-sulphide). Both these now separated gases form separate fluids, each forming alteration zones with distinct mineralogy (Hedenquist et al., 2000).

Above the water table H_2S condenses in the vadose zone to form a low pH H_2SO_4 (hydrogen-sulphate) dominant acid sulphate fluid (Hedenquist and White, 1990). These fluids can result in widespread tabular steam-heated alteration zones dominated by fine grained and friable kaolinite and alunite. Steam-heated waters collect at the water table and create aquifer-controlled stratofrom blankets of dense silicification known as silica caps (Shoenet al., 1974; Hedenquist et al.,, 2000). Since gold is not transported by the gases or sulphuric acid, the silica cap and overlying kaolinite alteration is usually devoid of gold and silver (Hedenquist et al. 2000).

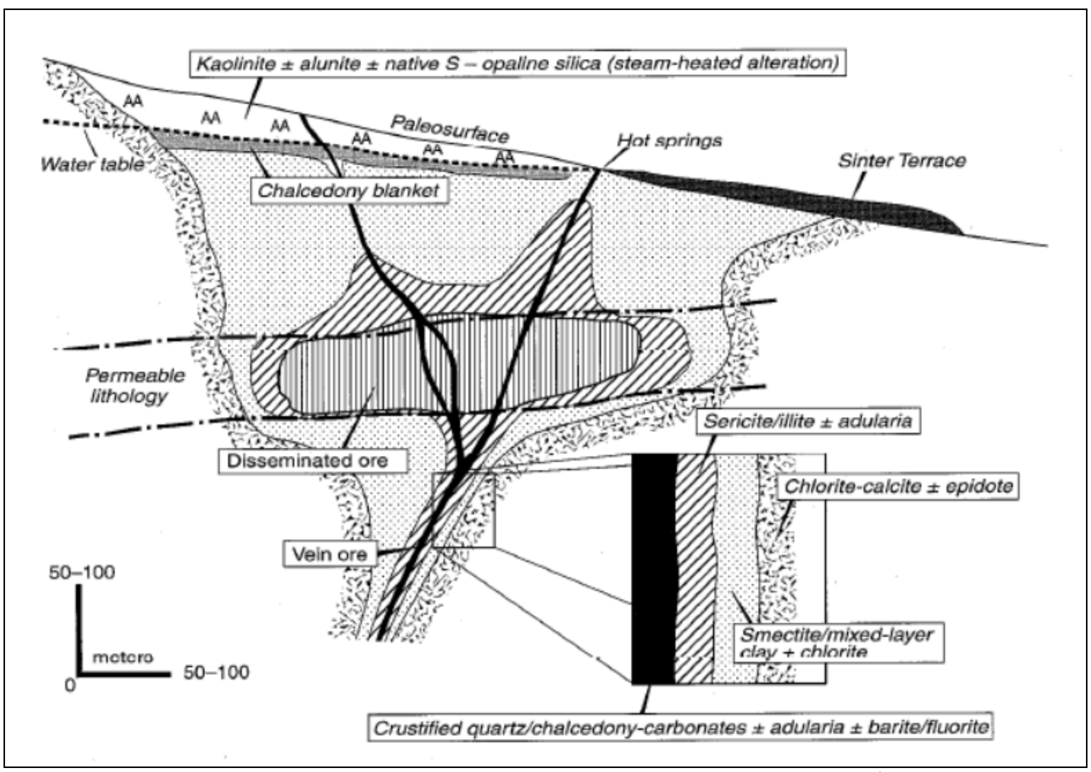
Bicarbonate fluids are the result of the condensation of CO_2 in meteoric water. These fluids are also barren of gold and silver and generally form carbonate dominated alteration on the margins of the geothermal cell.

As the source alkali chloride fluids boil and cool quartz and carbonate deposit in the fractures along which the fluids are ascending to form banded carbonate-quartz veins. Gold and silver present within the fluid also precipitate in response to the boiling of the fluid. Potassium-feldspar adularia is also a common mineral that deposits in the veins in response to boiling. As carbonate and quartz precipitates individual fractures can be sealed and the boiling fluid must then find another weak feature to continue rising. Gases which accumulate beneath the sealed fracture causes the pressure to increase until the seal is broken. This results in a substantial change in pressure which propagates catastrophic boiling in turn causing gold, bladed calcite and amorphous silica to precipitate rapidly. Once the fluids return to equilibrium the quartz crystals again precipitate under passive conditions and seal the vein again until the process recurs. This episodic sealing and fracturing results in the banded textures common in these vein systems.

Ore zones are typically localized in structures, but may occur in permeable lithologies. Upward-flaring ore zones centred on structurally controlled hydrothermal conduits are typical. Large (bigger than 1 m wide and hundreds of metres in strike length) to small veins and stockworks are common with lesser disseminations and replacements. Vein systems can be laterally extensive but ore shoots have relatively restricted vertical extent. High-grade ores are commonly found in dilational zones in faults at flexures, splays and in stock work.

These deposits form in both subaerial, predominantly felsic, volcanic fields in extensional and strike-slip structural regimes and island arc or continental andesitic stratovolcanoes above active subduction zones. Near-surface hydrothermal systems, ranging from hotspring at surface to deeper, structurally and permeability focused fluid flow zones are the sites of mineralization. The ore fluids are relatively dilute and cool solutions that are mixtures of magmatic and meteoric fluids. Mineral deposition takes place as the solutions undergo cooling and degassing by fluid mixing, boiling and decompression.

Figure 8-1. Schematic Cross-section of an Epithermal Au-Ag Deposit



*Hedenquist, 2000

8.2 Porphyry Copper-Gold-Molybdenum and Lead-Zinc Skarn Deposits

In Porphyry Cu-Au-Mo deposit types, stockworks of quartz veinlets, quartz veins, closely spaced fractures, and breccias containing pyrite and chalcopyrite with lesser molybdenite, bornite and magnetite occur in large zones of economically bulk-mineable mineralization in or adjoining porphyritic intrusions and related breccia bodies. Disseminated sulphide minerals are present, generally in subordinate amounts. The mineralization is spatially, temporally and genetically associated with hydrothermal alteration of the host rock intrusions and wall rocks.

These deposit types are commonly found in orogenic belts at convergent plate boundaries, commonly linked to subduction-related magmatism. Also in association with emplacement of high-level stocks during extensional tectonism related to strike-slip faulting and back-arc spreading following continent margin accretion (Panteleyev, 1995).

Many Au skarns are related to plutons formed during oceanic plate subduction, and there is a worldwide spatial, temporal and genetic association between porphyry Cu provinces and calcic Au skarns. The Au skarns are divided into two types. Pyroxene-rich Au skarns tend to be hosted by siltstone-dominant packages and form in hydrothermal systems that are sulphur-rich and relatively reduced. Garnet-rich Au skarns tend to be hosted by carbonate-dominant packages and develop in more oxidising and/or more sulphur-poor hydrothermal systems. The gold is commonly present as micron-sized inclusions in sulphides, or at sulphide grain boundaries. To the naked eye, ore is generally indistinguishable from waste rock. Due to the poor correlation between Au and Cu in some Au skarns, the economic potential of a prospect can be overlooked if Cu-sulphide-rich outcrops are preferentially sampled and other sulphide-bearing or sulphide-lean assemblages are ignored (Ray, 1998).

9 Exploration

Almaden Minerals Ltd. completed an exploration program at the Tuligtic Property that included both rock and soil geochemical sampling campaigns and a number of ground geophysical surveys. Ground magnetics, Induced Polarization (IP) and resistivity, Controlled Source Audio-frequency Magnetotelluric (CSAMT), and Controlled Source Induced Polarization (CSIP) surveys were competed over parts of the Property. Summary results for these exploration programs are presented below.

9.1 Rock Geochemistry

Between 2004 and 2011 a total of 436 rock samples were collected on the Property. Rock sampling, guided by concurrent soil geochemical surveys, is concentrated around the Ixtaca Zone and extends a distance of 4 km to the NNE over the copper porphyry target. Gold and silver mineralization occurs within the Ixtaca Zone, and is associated with anomalous arsenic, mercury (± antimony). To the northeast zinc, copper and locally anomalous gold, silver and lead (± arsenic) values occur in association with calc-silicate skarn and altered intrusive rocks. Epithermal alteration and mineralization is observed overprinting earlier skarn and porphyry style alteration and mineralization.

The Upper Tamaulipas formation, the dykes that crosscut it and lower parts of the upper Coyoltepec volcanic subunit host the epithermal vein system at Ixtaca. Much of the Ixtaca Zone and area surrounding it are overlain by unmineralized cover rocks of the upper Coyoltepec subunit, which has hindered rock geochemical sampling efforts. Outcroppings of the underlying Upper Tamaulipas Formation carbonate units, altered intrusive, and locally calc-silicate skarn mineralization occur as erosional windows within incised drainages.

Rock grab samples collected by Almaden were from both from representative and apparently mineralized lithologies in outcrop, talus and transported boulders within creeks throughout the Tuligtic Property. Rock samples ranging from 0.5 to 2.5 kilograms (kg) in weight and were placed in uniquely labelled poly samples bags and their locations were recorded using handheld GPS accurate to plus or minus 5 m accuracy.

Geochemical results for the Tuligtic rock samples are calculated into breakdowns of the 70th, 90th, 95th and 97.5th percentiles for gold (Figure 9-1), silver (Figure 9-2), zinc and (Figure 9-3), and copper (Figure 9-4) and are summarized in Table 9-1.

Table 9-1. Tuligtic Project Rock Geochemical Sampling Summary Statistics

	Au (ppb)	Ag (ppm)	Cu (ppm)	Mo (ppm)	Zn (ppm)
70th Percentile	14.50	1.50	60	7	272
90th Percentile	103.00	13.20	652	22	3260
95th Percentile	284.25	31.65	1671	46	5940
97.5th Percentile	428.00	58.36	4333	72	10000
Mean	71.33	6.52	360	12	1241
Max.	6140	291	12400	598	65600
Detection Limit	5	0.2	1	1	2
Range	6135	290.8	12399	597.00	65598
Correlation Coefficient (with Au): value from 1.00 to -1.00	1.00	0.35	0.09	0.21	0.02

Of the 436 rock grab samples collected, a total of 45 samples returned assays of greater than 100 parts-per-billion (ppb) gold (Au), and up to 6.14 grams-per-tonne (g/t) Au. A total of 49 rock samples returned assays of greater than 10 g/t silver (Ag) and up to 291 g/t Ag. A total of 27 samples returned values between 0.52% Zn and up to 6.6% Zn; and 33 returned between 0.1% Cu and 1.2% Cu.

9.1.1 Ixtaca Zone

Within the Ixtaca Zone, mineralization is not widely exposed at surface. Mineralization occurs primarily as float boulders of limestone breccia containing quartz vein fragments and high level epithermal alteration within overlying volcanic rocks. Epithermal mineralization of the type intersected by diamond drilling is observed in a single small (about 2 x 5 m) outcrop within the southwest trending creek bisecting the Ixtaca Main and North Zones. Here narrow (0.1 to 3 centimetre) quartz-carbonate veins with epithermal textures cutting limestone returned assays of 1 g/t Au and 100 g/t Ag.

9.1.2 Caleva Zone

Rock geochemical anomalies extend north of the Chemalco Zone within the Caleva soil anomaly. Here a 200 x 100 m zone of skarn zone occurs along the contact zone between limestone and altered and mineralized intrusive rocks to the east. A small 4 m long adit locally known as “Mina Eleva” is driven east into sphalerite, galena and chalcopyrite quartz vein stockwork mineralization within the skarn zone. Three rock grab samples collected from a small ore dump and adits driven into both sides of the creek returned values ranging from 2.8% to 6.6 % zinc (Zn), 0.27% to 0.66% Cu, 28 to 78 ppm Ag, and 32 to 145 ppb Au.

9.1.3 Azul and Sol Zones

The Azul and Sol Zones occur 1,500 to 2,500 m to the northwest of Caleva Zone. At both zones roof pendants of silicified limestone rocks intruded by quartz-monzonite porphyry host quartz-chlorite sphalerite-pyrite (± chalcopyrite) mineralization.

Within the Azul Zone, a total of 20 rock grab samples widely distributed over a 1 x 1 km area returned values ranging from 0.12 % to 2.0% Zn. Zinc has a high correlation with silver mineralization within the Azul Zone, and a total of nine (9) samples returned between 11.4 and 100 g/t Ag. Gold values are generally low to anomalous; however a single silicified limestone float sample returned assays of 2 g/t Au, 37.8 g/t Ag; with values of 2.2% lead (Pb), 0.22% Zn, 2.4% manganese (Mn), 0.19% arsenic (As) and 383 ppm antimony (Sb).

A distance of 800 m to the west of the Azul Zone, skarn mineralization exposed within a creek gully was mapped intermittently over a 1 km north-south trending zone. Within this area, at “Mina Pancho”, a short adit was driven into semi-massive pyrite and sphalerite calc-silicate skarn. Rock grab sampling over a 500 m distance within the creek, from the adit and a small ore dump resulted in eight (8) samples returning 0.15% to 1.8% Zn, of which six (6) samples returned 01.% to 0.8% Cu.

At the Sol Zone, a total of 5 rock grab samples returned values range from 0.51% to 1.0% Zn. As with the Azul Zone gold and silver values were generally low, however where anomalous they have a high correlation with zinc, arsenic and antimony. Two main zones of mineralization have been discovered. At the south end of the Sol zone, two (2) rock grab samples of silicified, iron-manganese oxide and clay altered limestone returned 0.92% and 1.0% Zn, 184 ppb and 284 ppb Au, 8.9 and 8.6 g/t Ag, anomalous arsenic and antimony. At the north end of the Sol Zone, at the “Mina Pablo” area a number of shallow artisanal shafts have been excavated on bedding conformable limestone skarn hosted semi-massive to massive pyrite and sphalerite vein mineralization. A total of 7 rock grab samples collected from Mina Pablo returned values ranging from 0.11% to 0.94% Zn.

Figure 9-1. Rock Geochemistry (Au)

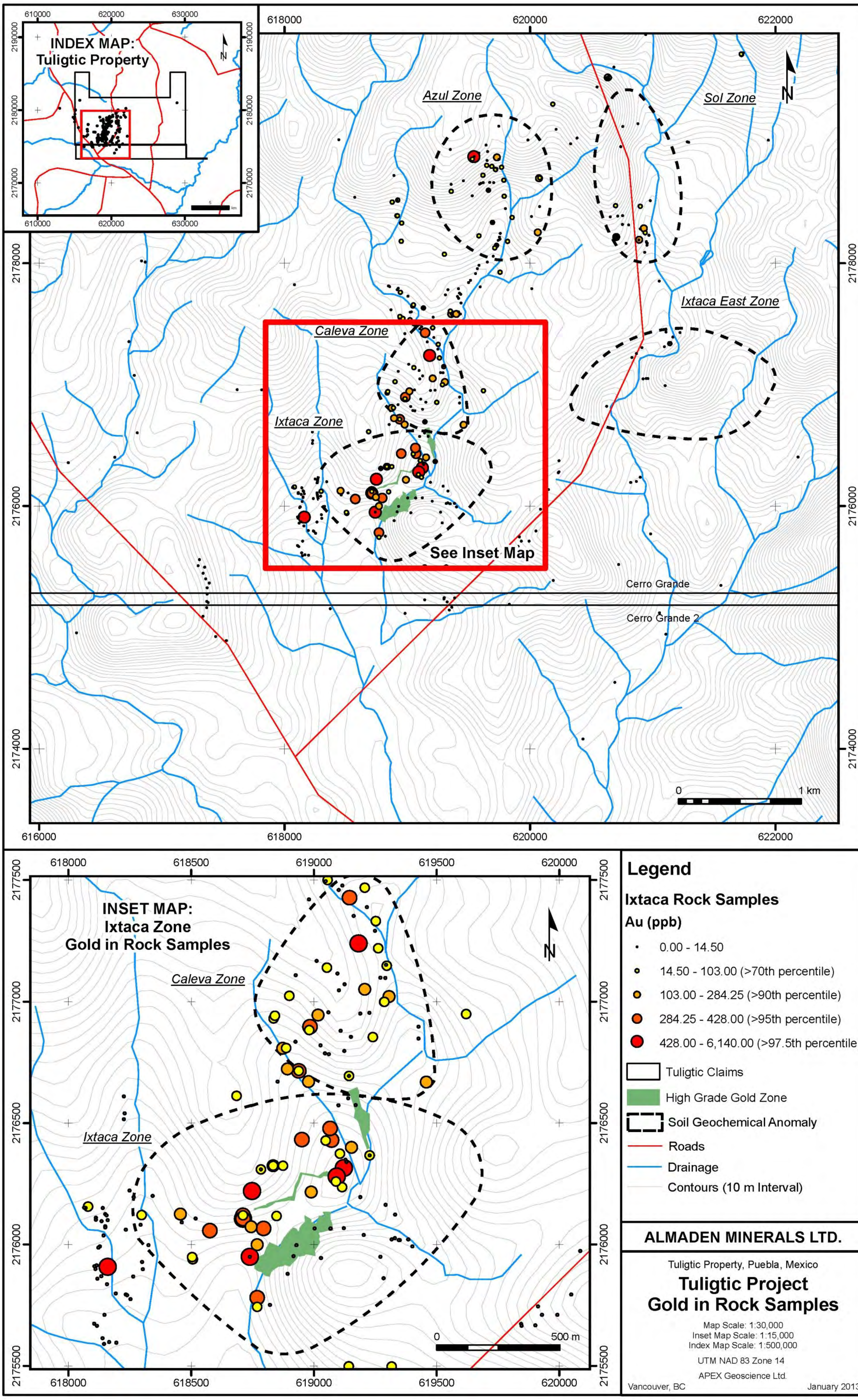


Figure 9-2. Rock Geochemistry (Ag)

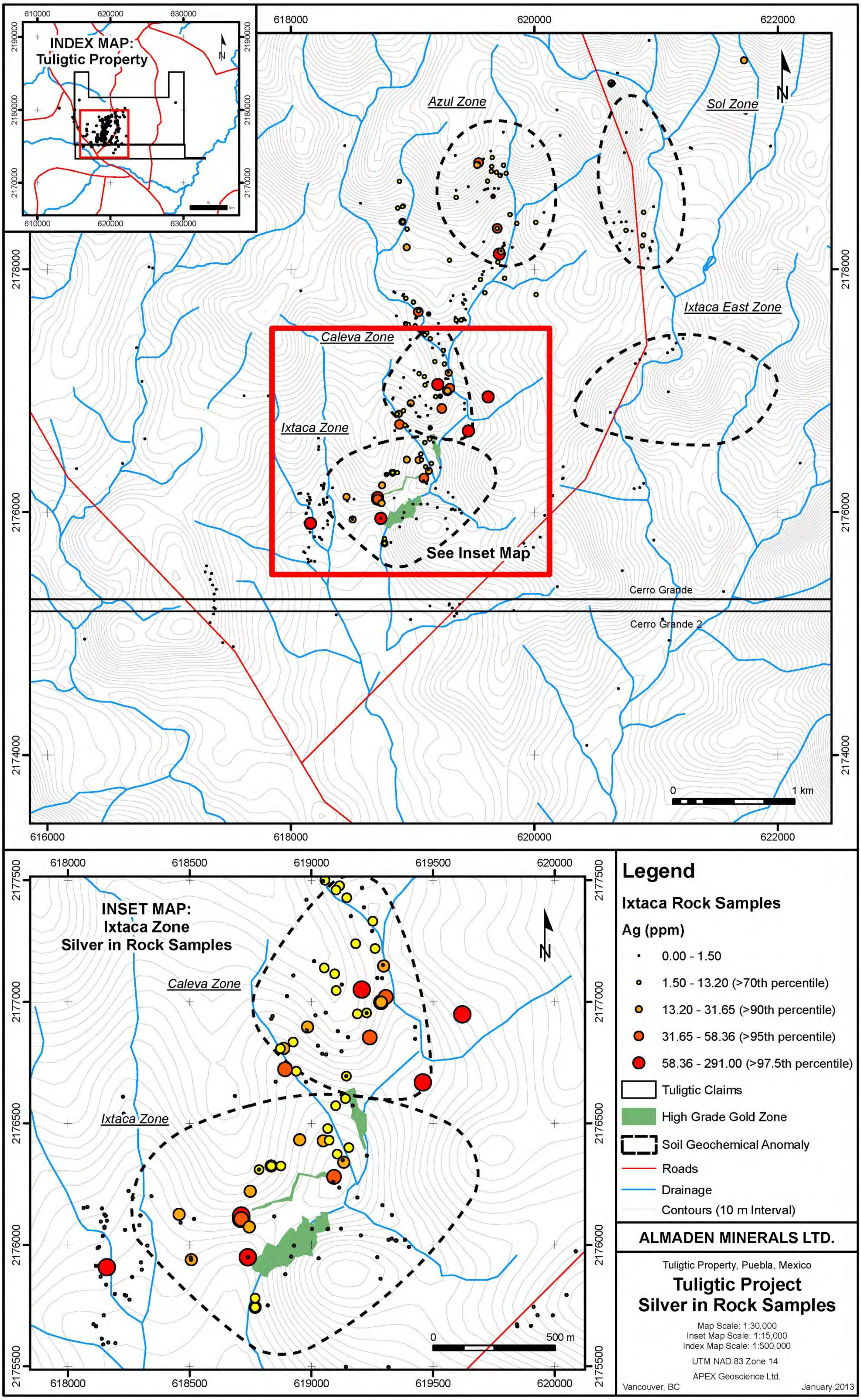


Figure 9-3. Rock Geochemistry (Cu)

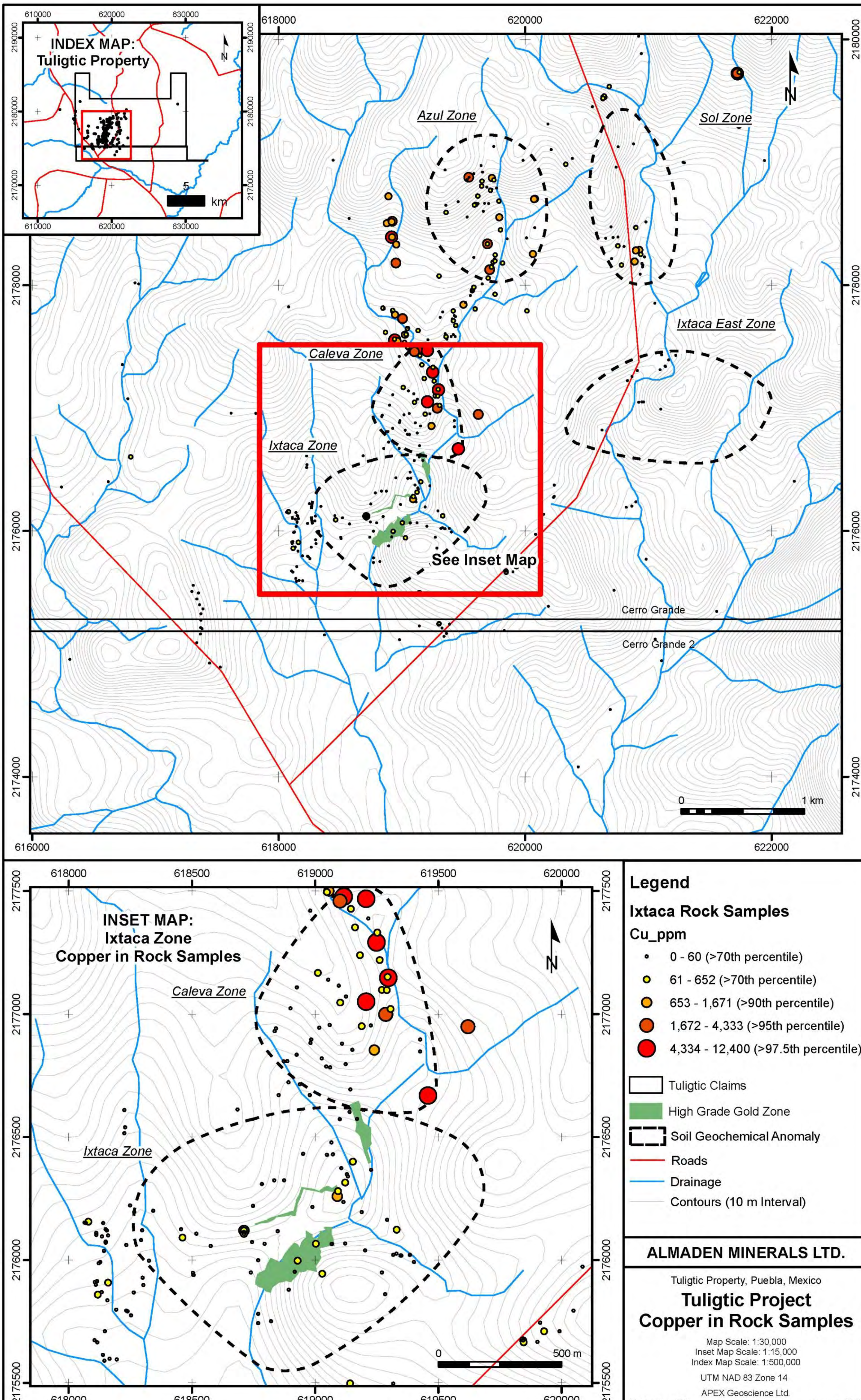
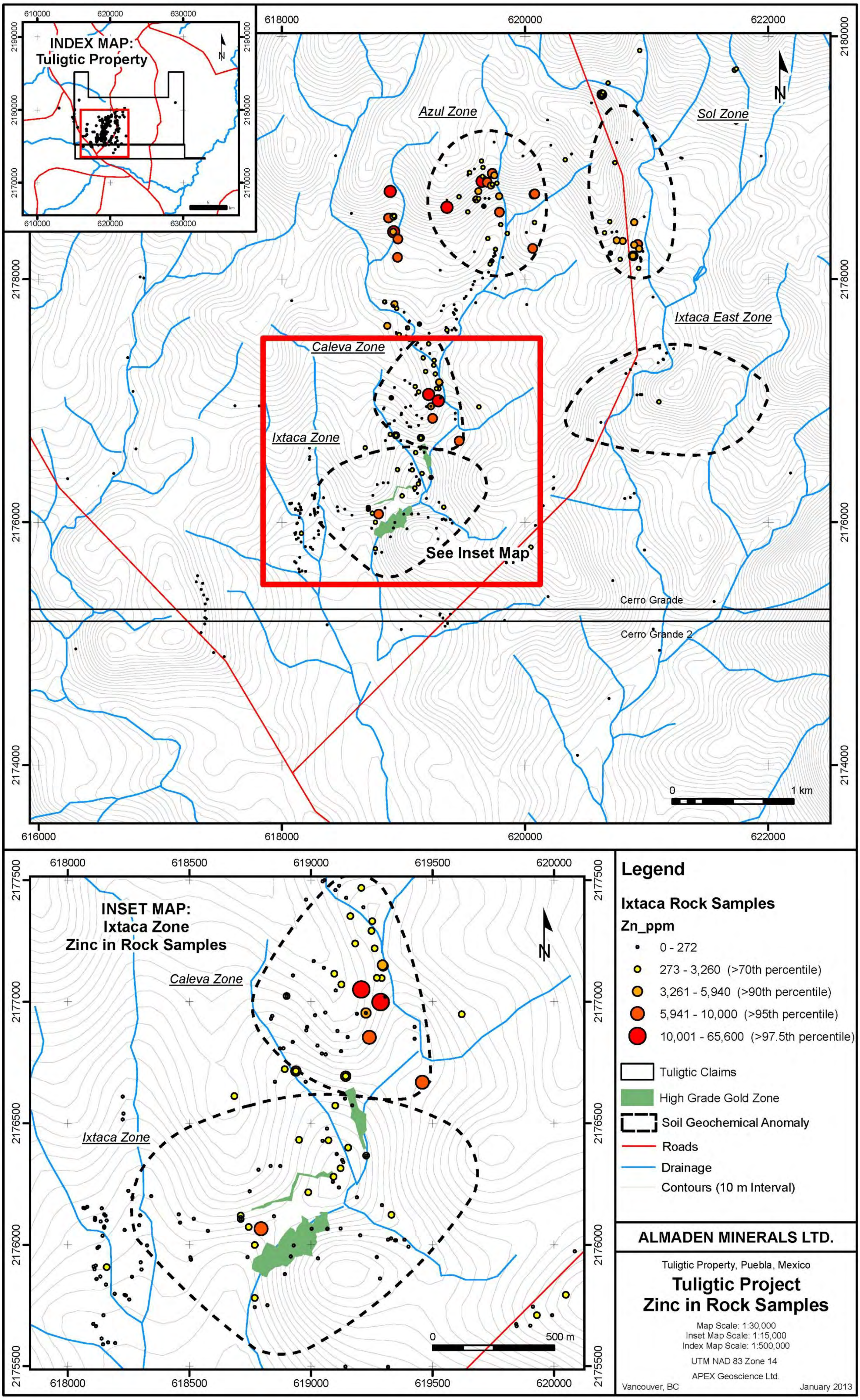


Figure 9-4. Rock Geochemistry (Zn)



9.2 Soil Geochemistry

Between 2005 and 2011, Almaden completed soil geochemical surveys over a 6 x 6 km area centred on the Ixtaca Zone within the western half of the Tuligtic Property. A total of 4,760 soil geochemical samples were collected and analyzed between 2005 and 2011. Five anomalous areas were identified, corresponding to the Ixtaca, Ixtaca East, Caleva, Azul, and Sol zones (Figures 9-2 and 9-2).

Samples were collected at 50 m intervals along a series of 200 m spaced east-west oriented lines. Infill lines spaced at 100 m were completed over gold and silver anomalies at the Caleva and Ixtaca East zones, and an unnamed anomaly 2.5 km west of the Ixtaca Zone. Subsequently, detailed 50 m x 50 m grid sampling of the Ixtaca Zone and select grid infill of the Azul and Sol zones was completed. Soil samples were collected by hand from a small hole dug with a non-metallic pick or hoe. The sample depth was typically 10 cm, or at least deep enough to be below the interpreted surficial organic layer. Sample bags were labelled with a unique sample number, and the sample location recorded with handheld GPS to plus or minus 5 m accuracy.

Geochemical results for the Tuligtic soil samples are calculated into breakdowns of the 90th, 95th, and 97.5th percentiles, and shown as thematic maps for gold (Figure 8) and silver (Figure 9). Anomalous thresholds (95th percentile) for gold and silver were calculated to be 20.63 ppb Au and 0.71 ppm Ag, respectively. A total of 238 samples containing anomalous Au were found, including 120 samples with coincident Ag anomalies. Summary statistics for the soil sampling campaign are shown in Table 9-2.

Table 9-2. Tuligtic Project Soil Geochemical Sampling Summary Statistics

	Au (ppb)	Ag (ppm)	As (ppm)	Cu (ppm)	Hg (ppm)	Mn (ppm)	Pb (ppm)	Sb (ppm)	Zn (ppm)
70th Percentile	2.70	0.11	10.0	19.5	0.13	571	13.9	2.54	78
90th Percentile	9.40	0.33	33.7	34.2	0.58	811	23.1	8.78	250
95th Percentile	20.63	0.71	61.7	54.2	1.30	1181	41.2	15.11	566
97.5th Percentile	45.02	1.60	99.3	83.1	2.24	1811	69.7	23.80	1180
Mean	6.46	0.24	15.9	23.4	0.31	551	17.4	4.16	190
Max.	720	30.7	1070	1450	53.80	11750	1405	564	31800
Detection Limit	0.01	0.01	0.1	0.2	0.01	5	0.2	0.05	2
Range	719.99	30.69	1069.9	1449.8	53.79	11745	1404.8	563.95	31798
Correlation Coefficient (with Au): value from 1.00 to -1.00	1.00	0.57	0.40	0.12	0.13	0.43	0.18	0.21	0.11

The Ixtaca Zone produces the largest Au and Ag response within the Tuligtic Property. Importantly, based metals do not correlate significantly with the Ixtaca Zone, and Hg and Sb anomalies occur peripherally within steam heated and replacement silicification altered volcanic rocks. Hg and Sb anomalies at the Ixtaca Zone occur with a broader As anomaly. Base metals correlate well with Au-Ag at the Caleva, Azul, and Sol zones to such an extent they are best termed Cu-Zn (Au-Ag) anomalies. Based on the distribution of soil geochemical anomalies (Figure 9-5 and 9-6) and the mapped geology (Figure 7-2) it is apparent that the overlying post mineral volcanics significantly suppress sedimentary and intrusive basement rock geochemical anomalies. Soil responses are consistent with these zones being prospective for both epithermal and earlier skarn mineralization.

9.2.1 Ixtaca Zone

Epithermal gold and silver mineralization at the Ixtaca Zone was originally identified by geologic mapping and surface rock sampling, and subsequently delineated by diamond drilling. At the Ixtaca Zone a northeast-southwest oriented soil geochemical anomaly occupies an area approximately 1000 x 400 m showing elevated Au and Ag values, within a broader 1,500 x 1,500 m zone returning anomalous a As, Sb and Hg values (Figures 9-5 and 9-6). Out of 1,165 soil samples collected at the Ixtaca Zone, a total of 97 anomalous samples returned assays greater than 20.63 ppb Au (>95th percentile), including 52 samples with coincident silver anomalies greater than 0.71 ppm Ag (>95th percentile). The Ixtaca Zone presents the largest and most concentrated Au and Ag response within the Tuligtic Property.

9.2.2 Caleva, Azul, Sol and Ixtaca East Zones

Four additional areas with elevated levels of Au and Ag were identified by the 2011 soil sampling program: the Caleva, Azul, Sol and Ixtaca East zones (Figures 9-5 and 9-6). These zones are not as large or as concentrated as the Ixtaca Zone soil geochemical anomaly, but have returned significant Au and Ag responses. The Caleva Zone, located directly north of the Ixtaca Zone is defined by a north-northwest trending area of elevated Au and Ag values up to 310 ppb Au and 14.75 ppm Ag, including five anomalous samples with greater than 45 ppb Au and 2 ppm Ag. The Azul and Sol Zones are located 2.5 km to the northeast of the Ixtaca Zone, respectively. Together, they form a broad zone of elevated Ag values (Figure 9-6) including clusters of anomalous Au and Ag responses (Figures 9-5 and 9-6). The Azul and Sol Zones contain 30 samples with greater than 45.02 ppb Au (>97.5th percentile), including 20 samples with coincident Ag anomalies (>97.5th percentile). The Ixtaca East Zone, located 2 km along strike from the Ixtaca Zone, corresponds to a broad, northeast trending area of weakly elevated Au and Ag values in soil, including two samples with greater than 45.02 ppb Au (>97.5th percentile).

9.2.3 Pathfinder and Base Metal Anomalies

Comparison of correlation coefficients from the Tuligtic soil geochemical data reveals that the elements Ag, As, and Mn show a good positive correlation with Au values. Hg and Sb show a moderate positive correlation with Au (Table 9-2). These elements, along with the base metals (Cu-Zn-Pb) exhibit sufficient variability in comparison to detection limit (range >> detection limit) to permit anomaly discrimination. Comparison of the anomalous Au zones with Ag, As, and Mn yields a strong, discernible spatial correlation in all five zones. Hg and Sb have a good spatial correlation with anomalous Au in the Caleva, Azul, Sol and Ixtaca East Zones. At the Ixtaca Zone, Hg and Sb

Figure 9-5. Soil Geochemistry (Au)

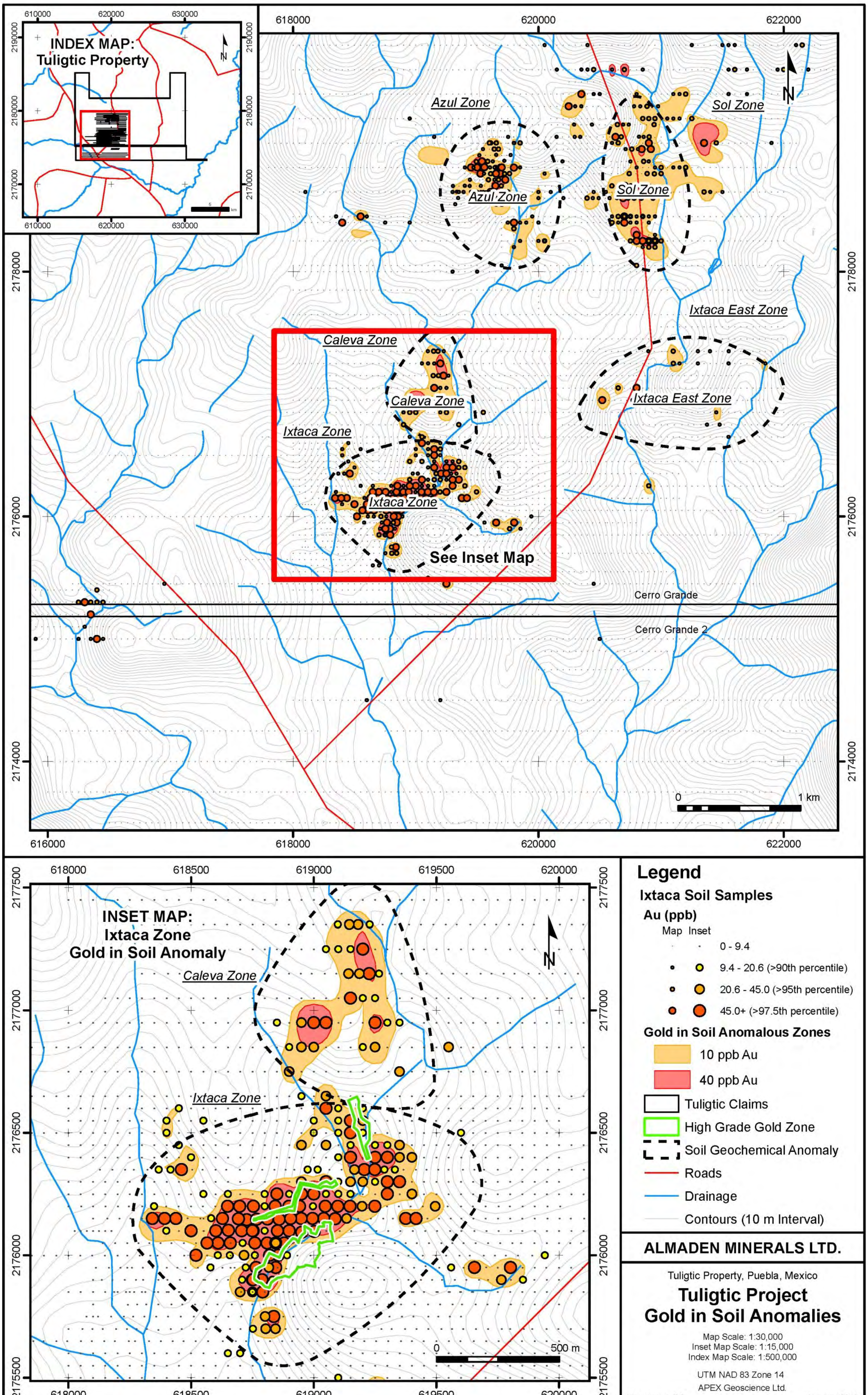
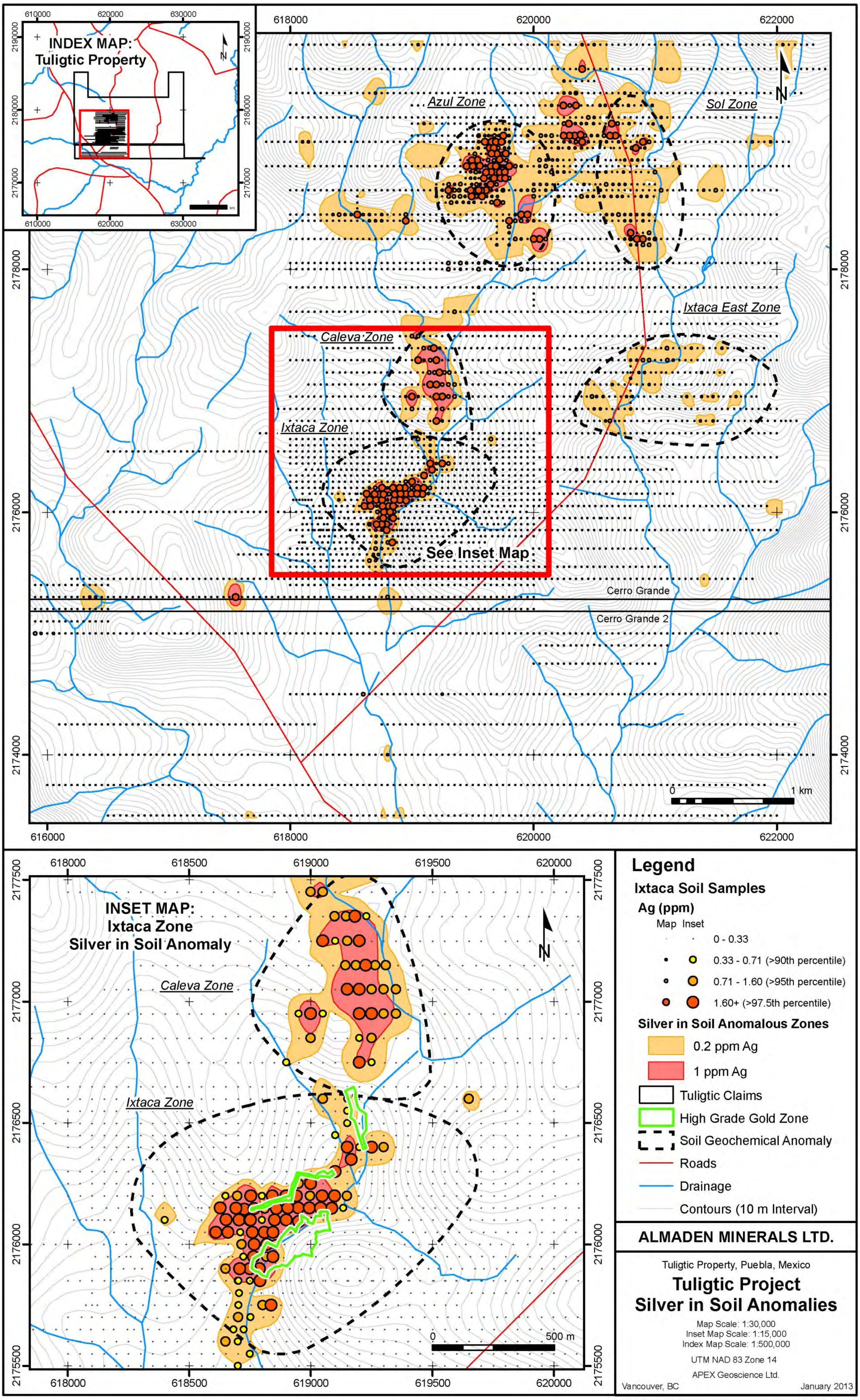


Figure 9-6. Soil Geochemistry (Ag)



anomalies appear to occur in association with within steam heated and replacement silicification in altered volcanic rocks. Hg and Sb anomalies at the Ixtaca Zone occur with a broader As anomaly. Base metals correlate well with gold at the Caleva, Azul, and Sol zones to such an extent they are best termed Cu-Zn (Au-Ag) anomalies. Importantly, based metals do not correlate significantly with the Ixtaca Zone.

9.3 Ground Geophysics

9.3.1 Magnetic

During September 2010, 84 line-km of ground magnetic geophysical surveying was completed over an area of 4 km by 4.5 km covering the copper porphyry target area north of the Ixtaca Zone (Figure 9-7).

Survey data was collected using one Gem Systems Inc. GSM mobile magnetometer and a GSM base magnetometer providing diurnal correction. The survey was conducted over a series of 27 east-west oriented lines spaced at 200 m intervals. One additional survey line (6050N) located 1 km to the south of the main survey area crosses the Ixtaca Zone. The survey lines range between 2.53 km to 4.4 km in length, with magnetic reading collected at 12.5 m intervals along each line.

Within the centre of the survey area, a broad poorly defined, approximately 100 nano-Tesla (nT) magnetic high anomaly is present. The anomaly corresponds in part with mapped altered quartz-monzonite porphyry rocks. Numerous, 30 to 50 nT short strike length NNW trending linear magnetic high anomalies parallel the regional structural grain, and the strike of bedding within Upper Tamaulipas formation calcareous rocks. The most significant linear magnetic anomaly has a magnitude of up to 100 nT and and extends from the Caleva Zone NNW through the Mina Pancho area. Here limestone and local calc-silicate skarn mineralization are preserved along the NNW trending contact zone with intrusive rocks to the east, again suggesting structural and/or lithologic control of magnetic anomalies.

9.3.2 Induced Polarization (IP) / Resistivity

During September 2010, Prospec MB Inc, on behalf of Almaden, completed and induced polarization (IP) / resistivity geophysical survey on areas covering the Ixtaca Zone and Cavela Zone, and parts of Azul zone and Ixtaca East Zone (Figures 9-7 and 9-8). A total of 108 line-km was collected over the 22 east-west trending lines and 12 perpendicular north-south trending lines. The lines were spaced at 100 m intervals and ranged in length from 2.2 to 4.5 km.

An Elrec IP-6 receiver and a 5000 watt GDD TxII transmitter were used for the IP survey employing a pole-dipole array. Readings were taken with an “a” spacing of 100 m at “n” separations of 1 to 8. The on line current electrode was located to the west and south of the potential electrodes. Elrec IP-6 receiver was used with a 2000 millisecond (ms) window. The delay was set to 600 ms, and the chargeability window used for integration was set from 0.1 to 1.0 seconds, or 100 ms to 1000 ms.

Figure 9-7. Ground Magnetic Survey (Plan)

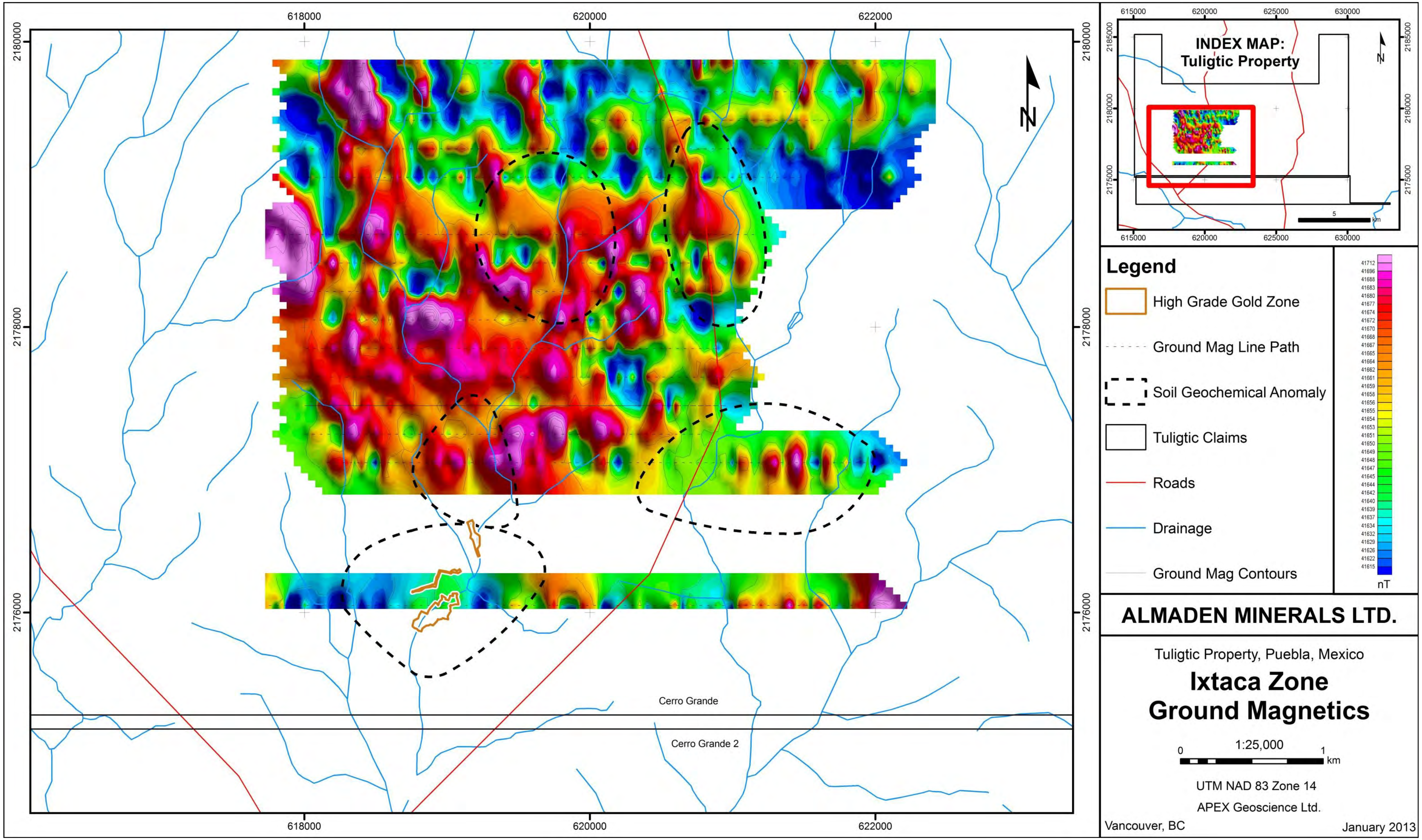


Figure 9-8. Inverted Chargeability (Plan)

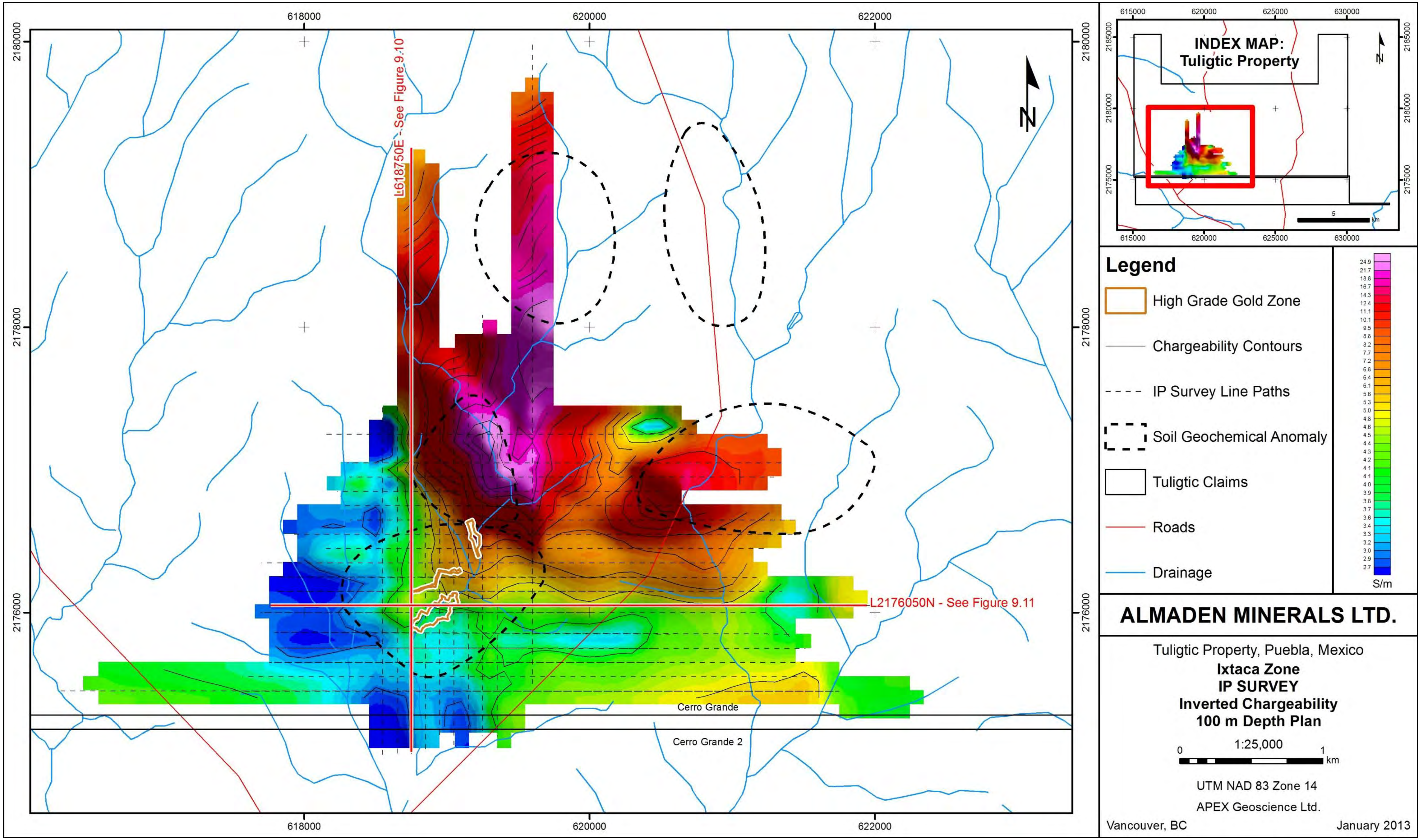


Figure 9-9. Inverted Resistivity (Plan)

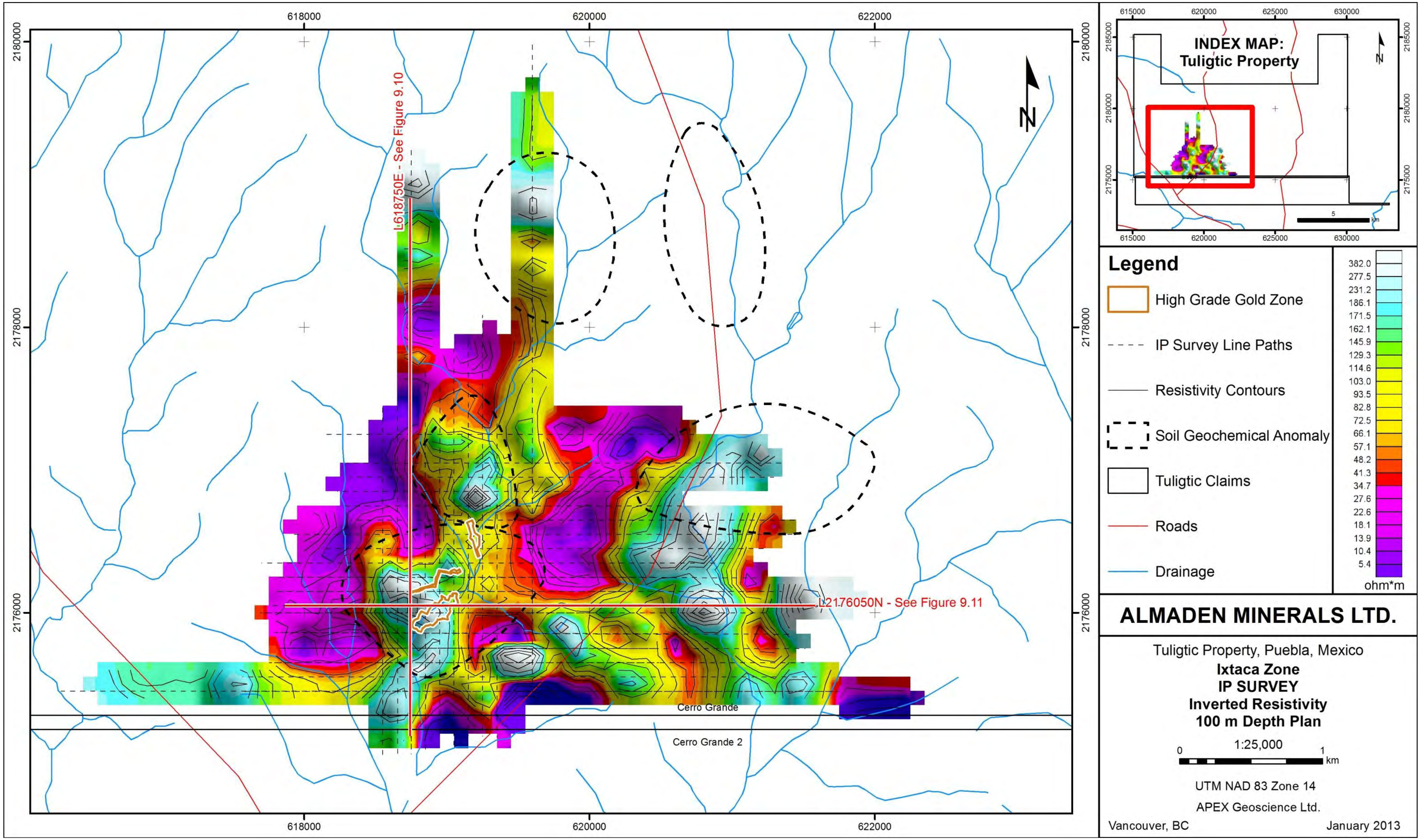


Figure 9-10. Inverted Section (Line 618,750E)

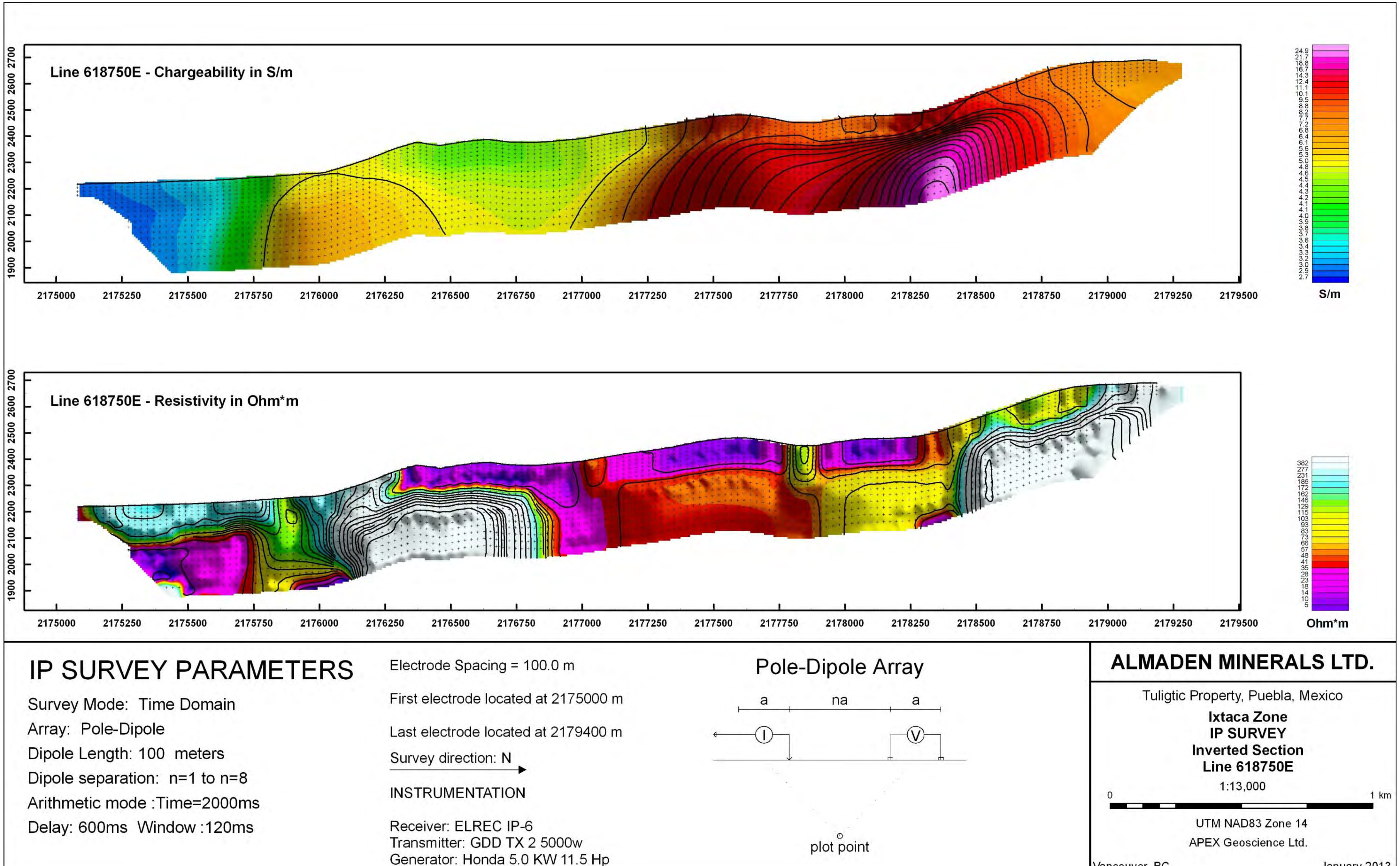
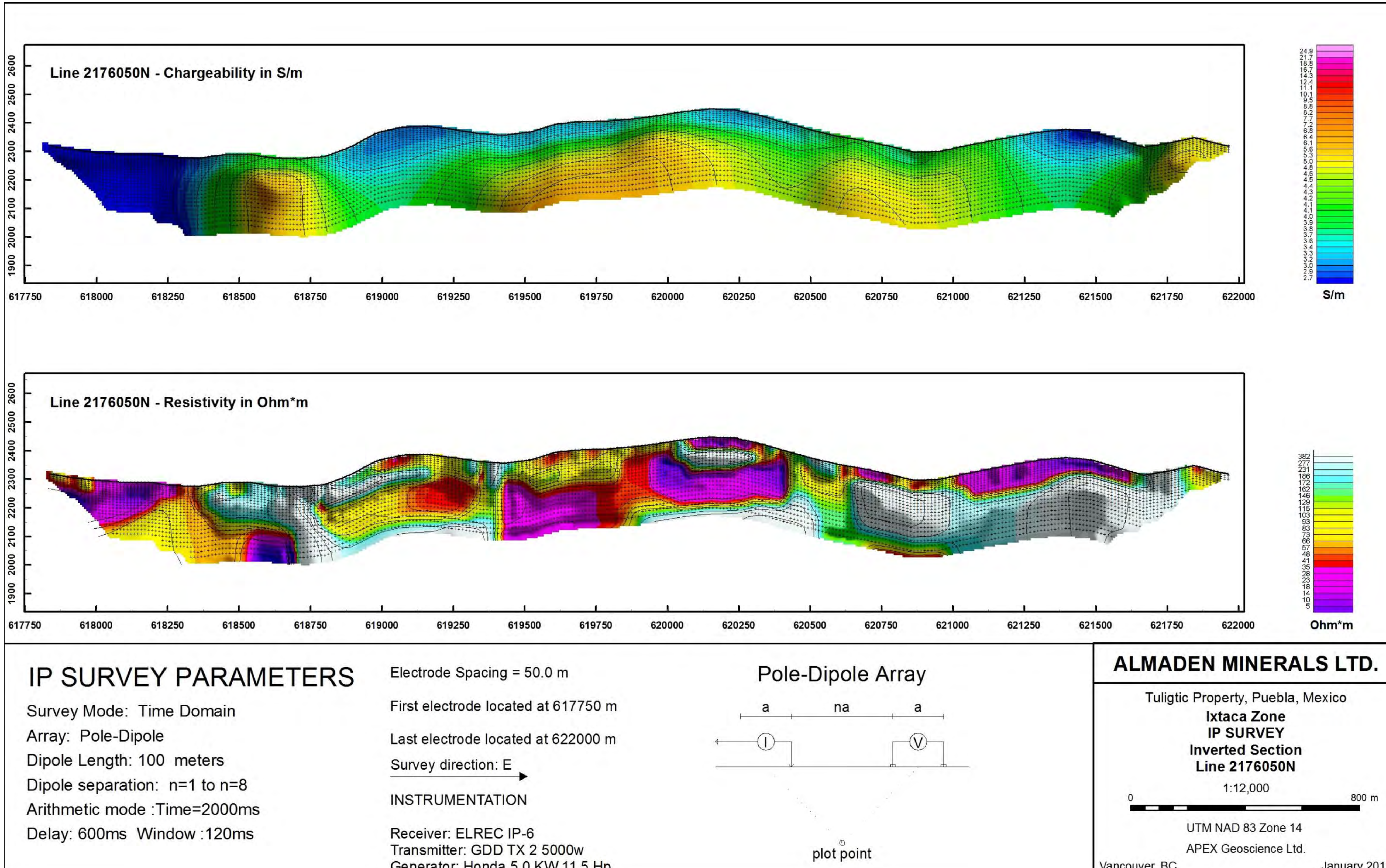


Figure 9-11. Inverted Section (Line 2,176,050N)



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Average chargeability values fall between 5 to 8 millivolts-per-volt (mV/V), and chargeability anomalies range from 15 to 30 mV/V. At the Caleva Zone a 1000 x 200 m north-northwest trending 20 to 30 mV/V chargeability anomaly is coincident with a mapped zinc-copper-silver mineralized calc-silicate skarn body along the western margin of the intrusive (Figures 9-8 and 9-10). To the north of this anomaly, a single north-south oriented survey line defines a 1 km long 20 to 30 mV/V chargeability. anomaly, within a broader 2.5 km long 10 to 20 mV/V zone of chargeability. While poorly constrained by the existing survey coverage, the anomaly appears to be coincident with a north-northeast trending Cu-Zn soil geochemical anomaly passing through the Caleva and Azul Zones.

Partial survey coverage of the Ixtaca East Zone multi-element soil geochemical anomaly defines a 700 x 500 m elliptical 7 to 15 mV/V chargeability anomaly along its western margin.

Resistivity data appears to largely reflect surface geology, which is controlled by local topography (Figure 9-9). Resistivity anomalies occur at the Ixtaca Zone (300 ohm-metres) where surface exposures are dominated by limestone lithologies, and intrusive rocks exposed at the Ixtaca East Zone (400 to 700 ohm-m). Resistivity anomalies appear to be controlled in part by topographic lows that down-cut through overlying tuff rocks and expose more resistive basement lithologies. Resistivity low (conductive) anomalies are common along local topographic high ridges and plateaus where significant thicknesses of more conductive tuff rocks remain.

At the Ixtaca Zone, a northwest trending resistivity and weak chargeability anomaly is centered on the North and Main Ixtaca zones (Figure 9-10 and 9-12). The anomaly is coincident with the east-verging limestone-cored syncline that hosts the high-grade North and Main Ixtaca zones of mineralization (Figure 10-2). A flanking north-south oriented moderately conductive anomaly to the east (60 to100 ohm-m) may reflect an interpreted calcareous-shale cored anticline, host to mineralization at the Northeast Extension Zone.

9.3.1 CSAMT/CSIP

In spring 2011 Zonge International Inc. was contracted by Almaden to conduct a Controlled Source Audio-frequency Magnetotelluric (CSAMT) and Controlled Source Induce Polarization (CSIP) geophysical investigation on the Tuligtic property. The survey started on May 13, 2011 and was completed during two separate periods, ending October 22, 2011.

The survey comprised 14 lines totalling 48.5 line-km, including six lines oriented N-S (N16E azimuth, CSAMT and CSIP), and 8 perpendicular E-W oriented lines (N104E azimuth, CSAMT only). The line spacing ranged from 170 to 550 m metres. CSAMT and CSIP data were collected via an array of six 25 m dipoles, with almost 2000 stations acquired. Data were collected with a 6 channel Zonge GDP-32^{II} multipurpose receiver. The electric field signals were measured using non-polarizing ceramic porous-pot electrodes connected to the receiver with insulated 14-gauge wire. A square-wave signal was provided by 10 kilowatt (Kw) Zonge GGT-10, and 30 kW GGT-30 transmitters, with transmitter power provided by motor-generator sets. Magnetic field data was collected using a single ANT/G magnetic field sensor.

CSAMT and CSIP electric field data and magnetic field (CSAMT only) were collected at each station. Data from the electric and magnetic field measurements were then used to calculate resistivity and impedance phase values at each of 12 discrete frequencies (from 4 to 8,192 Hz in binary increments), as well as odd harmonics (3rd, 5th, 7th, and 9th) of the transmitted frequencies. For CSIP, electric field data was collected at 0.125, 0.25 and 0.375 Hz (a mixture of fundamental and harmonic frequencies) to approximate the IP response. 1D and 2D modeling of resistivity pseudo-section data was completed using Zonge SCSINV (2.20I) and SCS2D (3.20y) smooth-model inversion programs.

Zonge completed 1-D and 2-D smooth-model section and plan-view projections for CSAMT at depths of 100, 200, 300, 400 and 500 m, calculated from the modelled topographic surface, and a plan-view of CSIP data. The 100 m depth slice for 1-D (N-S and E-W lines combined), 2-D (N-S), and 2-D (E-W) smooth-model resistivity are presented in Figures 9-12, 9-13, and 9-14 below; in addition to 2-D smooth-model resistivity sections for one N-S (Line 1), and one E-W (Line 17) (Figure 9-15), and CSIP plan-view (Figure 9-16).

The 100 m depth plan 1-D and 2-D smooth-model resistivity data are in broad agreement and appear to identify similar broad scale resistive feature. However, 1-D models are sensitive to topography. Valleys tend to create high-angle resistive anomalies and peaks tend to create high angle conductive features. 1-D model data is strongly affected by steep topography present throughout the Ixtaca Zone grid. A narrow, steep-sided, northeast trending gully separating the Ixtaca Main and North zones, and the extension of this gully to the northwest, is coincident with a strong resistive anomaly (Figure 9-12). Similar broad N-S trending conductive anomalies to the east and west of the Caleva Zone are coincident with significant topographic high ridgelines.

The 100 m depth plan 2-D smooth-model resistivity data for both N-S and E-W lines suggest that broad scale features of the 1-D model reflect the underlying geology. The 2-D (N-S Line) data defines a NW trending resistivity anomaly west of the Ixtaca Main Zone, and an E-W trending resistivity anomaly through the Ixtaca Zone (Figure 9-13). The NW trending anomaly passes through drill sections 10+200E to 10+400E, and may reflect limestone rocks on the west limb of an east-verging antiform (Figures 9-15 and 10-2). A similar NW trending conductive anomaly immediately to the east may represent calcareous shale rocks within the core of the antiform. The significance of the E-W trending anomaly is not known given the context of the current geologic model.

There are differences between the 2-D smooth-model plots for N-S and E-W oriented lines. These differences are thought to relate to the line orientation and location of the transmitter with respect to structural geology of the Ixtaca Zone. It is therefore expected that certain structures may be more or less apparent depending on their geometry with respect to the line orientation. Because of these differences it is best to view the 2-D models of the N-S and E-W lines separately, and any interpretation of geology must consider both. The 2-D (E-W Line) data shows a strong resistivity anomaly associated with the core of the Ixtaca Main Zone, and surface outcropping limestone (Figures 9-14 and 9-15). To the northeast, a resistivity anomaly may reflect complex structural geology patterns and the relatively resistive limestone and the Chemalaco Dyke lithologies (Figure 10-2).

Significantly, that the 100 m depth 2-D smooth model resistivity (25 m dipole spacing) and conventional resistivity (100 m dipole spacing) survey data correlate reasonably well. Although dipole separation, line spacing and orientation differ, this repeatability suggests effectiveness in mapping sub-surface resistivity at the Ixtaca Zone. Given the dominant NW structural, NE (Ixtaca Main and North zones) and NW (Northeast Extension Zone) mineralization trends it is unlikely a single line orientation will effectively map both geologic and mineralization trends. This fact is compounded by logistical considerations imposed by the rugged terrain surrounding the Ixtaca Zone.

CSIP anomalies may be associated with mineralization or conductive geology associated with high-angle resistivity contacts. CSIP data can only detect an anomalous IP-like response. In most cases indentifying the IP source, as well as the precise location of this IP source, requires more information. This can be provided by a conventional IP survey. CSIP data does not appear to have identified significant anomalies. A broad CSIP anomaly at the south end of the survey grid is underlain by low angle valley topography suggesting the anomaly is due to conductive overburden (Figure 9-16). Part of the Ixtaca Zone is coincident with a CSIP anomaly; however its orientation, parallel to survey direction, suggests it may be a result of line artifacts. A CSIP anomaly at the centre of the Caleva Zone is offset from a conventional IP response associated with skarn mineralization along the intrusive contact (Figure 9-8). Given the CSIP anomaly here occurs on a topographic high, it may be due to the presence of conductive tuff rocks.

Figure 9-12. CSAMT 1D Smooth-Model Resistivity 100 m Depth Plan

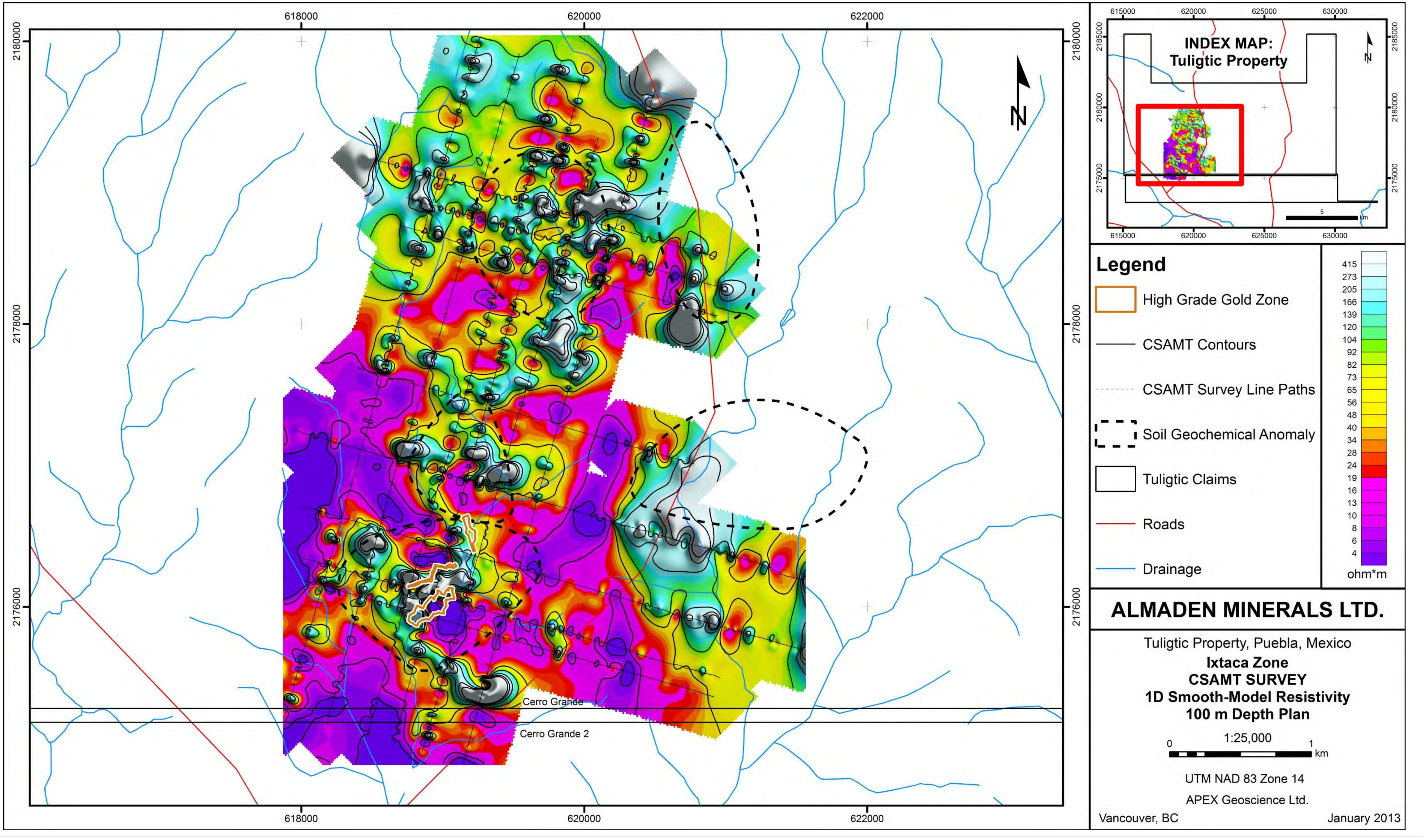

Figure 9-13. CSAMT 2D Smooth-Model Resistivity 100 m Depth Plan (N-S Lines)

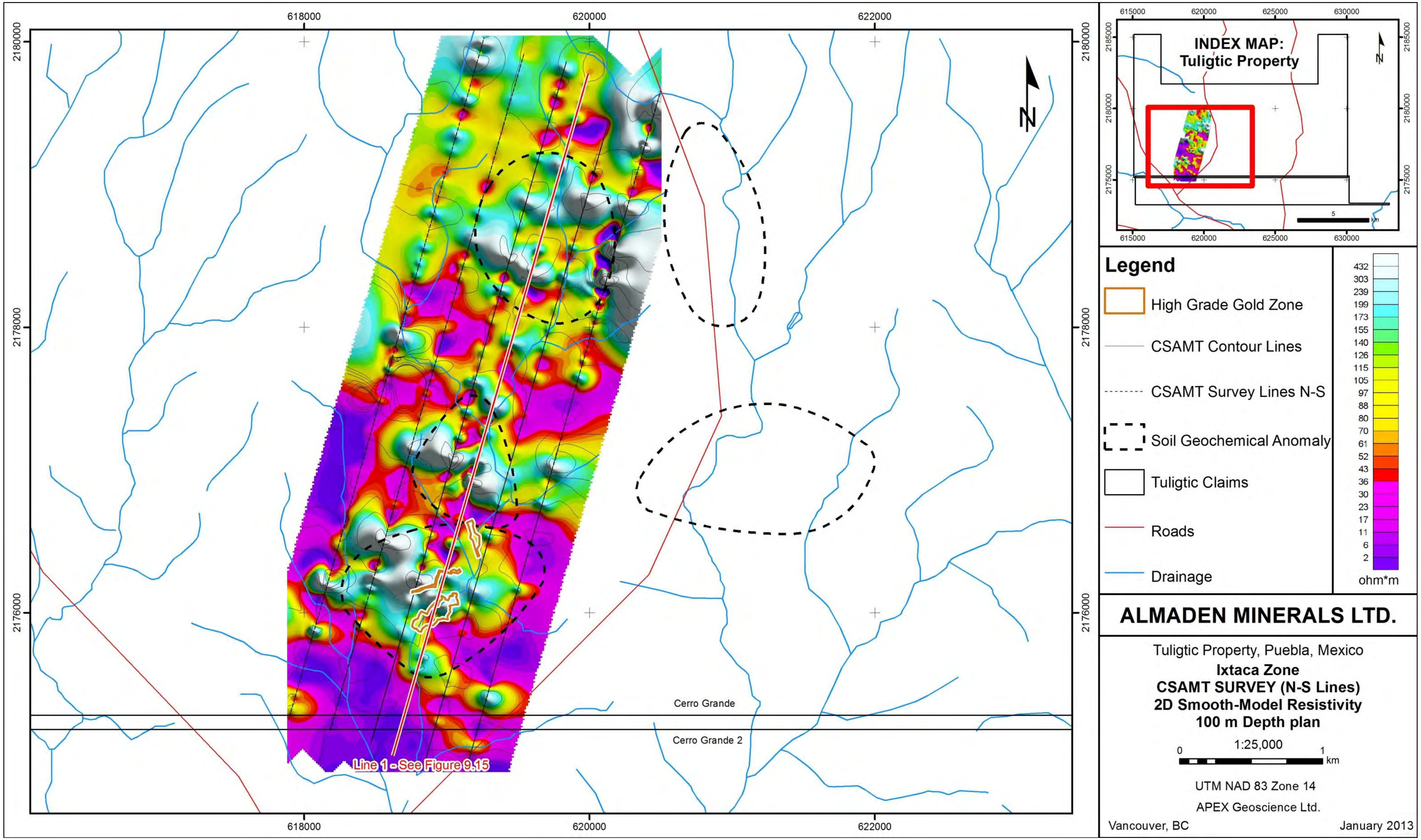

Figure 9-14. CSAMT 2D Sooth-Model Resistivity 100 m Depth Plan (E-W Lines)

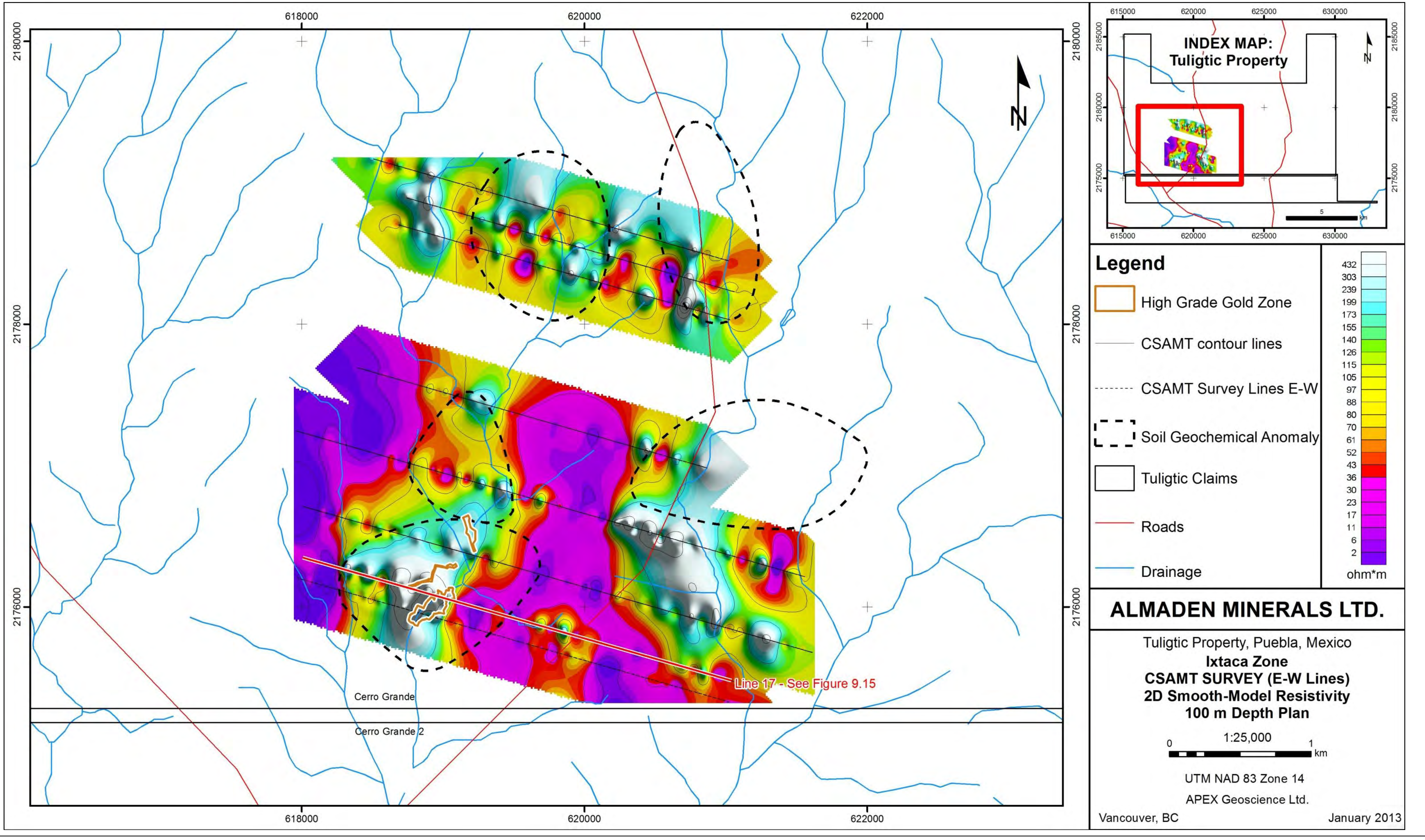

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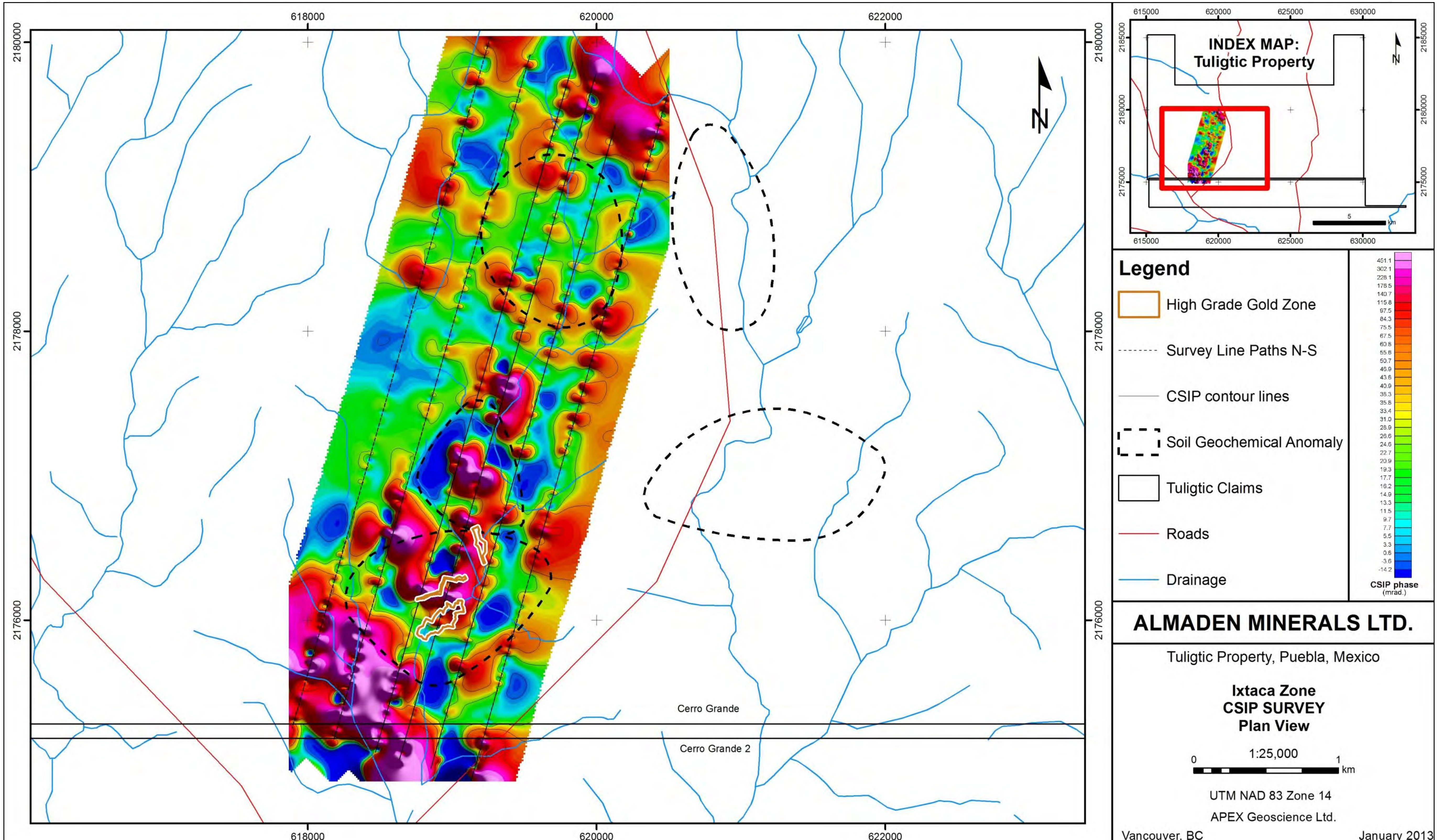
Figure 9-15. CSAMT 2D Smooth-Model Resistivity Cross Section Line 1 (N-S) and Line 17 (E-W)



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10 Drilling

In July 2010 Almaden initiated a preliminary diamond drilling program to test epithermal alteration within the Tuligtic Property, resulting in the discovery of the Main Ixtaca Zone. The first hole, TU-10-001 intersected 302.42 m of 1.01g/t Au and 48g/t Ag and multiple high grade intervals including 1.67 m of 60.7g/t Au and 2,122g/t Ag. Almaden drilled 14 holes totalling 6,465 m during 2010, defined the Main Ixtaca Zone over a 400 m strike length, and initiated drilling along 50 m NNW oriented sections. During 2011, Almaden drilled an additional 85 holes totalling 30,644 metres, which resulted in the discovery of the Ixtaca North Zone and testing of the Main Ixtaca Zone over a 600 m strike length on 50 m sections. Almaden discovered the Northeast Extension (Chemalaco) Zone in early 2012 and continued drilling of the Ixtaca North and Main Ixtaca zones. Almaden drilled 126 holes totalling 44,862 m on the Property from the beginning of 2012 until the November 13, 2012 maiden mineral resource estimate cut-off, for a total of 81,971 m in 225 drill holes. Of the 225 holes, approximately 110 holes have been completed on the Main Ixtaca zone, 72 holes on the Ixtaca North Zone and 43 holes on the Northeast Extension (Figure 10-1).

The diamond drill holes range from a minimum length of 130 m to a maximum of 701 m, and average 364 m. All drilling completed at the Ixtaca Zone has been diamond core of NQ2 size (5.08 cm diameter). Drilling was performed using four diamond drills owned and operated by Almaden via its wholly owned operating subsidiary Minera Gavilán, S.A. de C.V. The 2010 through 2012 diamond drill programs were completed under the supervision of Almaden personnel. Drill hole collars were spotted using a handheld GPS and compass, and subsequently were surveyed using a differentially corrected GPS. Each of the holes is marked with a small cement cairn inscribed with the drill hole number and drilling direction.

Drill holes were surveyed down hole using Reflex EZ-Shot or EX-Trac instruments following completion of each hole. Down hole survey measurements were spaced at 100 m intervals during 2010 drilling and were decreased to 50 m intervals in 2011. During 2012, select drill holes within all three mineralized zones were surveyed at 15 m intervals. A total of 2,206 drill hole orientation measurements (including 225 collar surveys) were collected for an average down hole spacing of 35 m. A total of 20 drill holes (6,657 m), apart from the collar survey, were not surveyed downhole; and a total of 4 drill holes (1,410 m) were surveyed at the collar and end of hole only. Drill holes having no down hole survey were assumed to have the orientation of the collar. Drill hole data was plotted in the field and was inspected. Down hole data returning unrealistic hole orientations were considered suspect and the data was not used. Down hole survey summary statistics are provided in Table 10-1, below.

At the rig, drill core was placed in plastic core boxes labeled with the drill hole number, box number, and an arrow to mark the start of the tray and the down hole direction. Wooden core blocks were placed at the end of each core run (usually 3 m, or less in broken ground). Throughout the day and at the end of each shift drill core is transported to Almaden's Santa Maria core logging, sampling and warehouse facility.

Table 10-1. Tuligtic Project Down Hole Survey Statistics

	Number of Drill Holes	Metres
Number of Surveys (including collar)	2,206	81,971
Average Survey Spacing (not including casing)	225	35.0
Drill Holes (No Down Hole Survey)	20 (8%)	6,657
Drill Holes (End Of Hole Survey Only)	4 (2%)	1,410
Drill Holes (15 m Survey Spacing)	69 (30%)	24,390
Drill Holes (50m Survey Spacing)	116 (53%)	43,195
Drill Holes (100 m Survey Spacing)	16 (8%)	6,319

Geotechnical logging comprised measurements of total core recovery per-run, RQD (the total length of pieces of core greater than twice the core width divided by the length of the interval, times 100), core photography (before and after cutting), hardness testing and measurements of bulk density using the weight in air-weight in water method. Core recovery for the 225 Ixtaca Zone drill holes averaged 92%, with RQD averaging 76%.

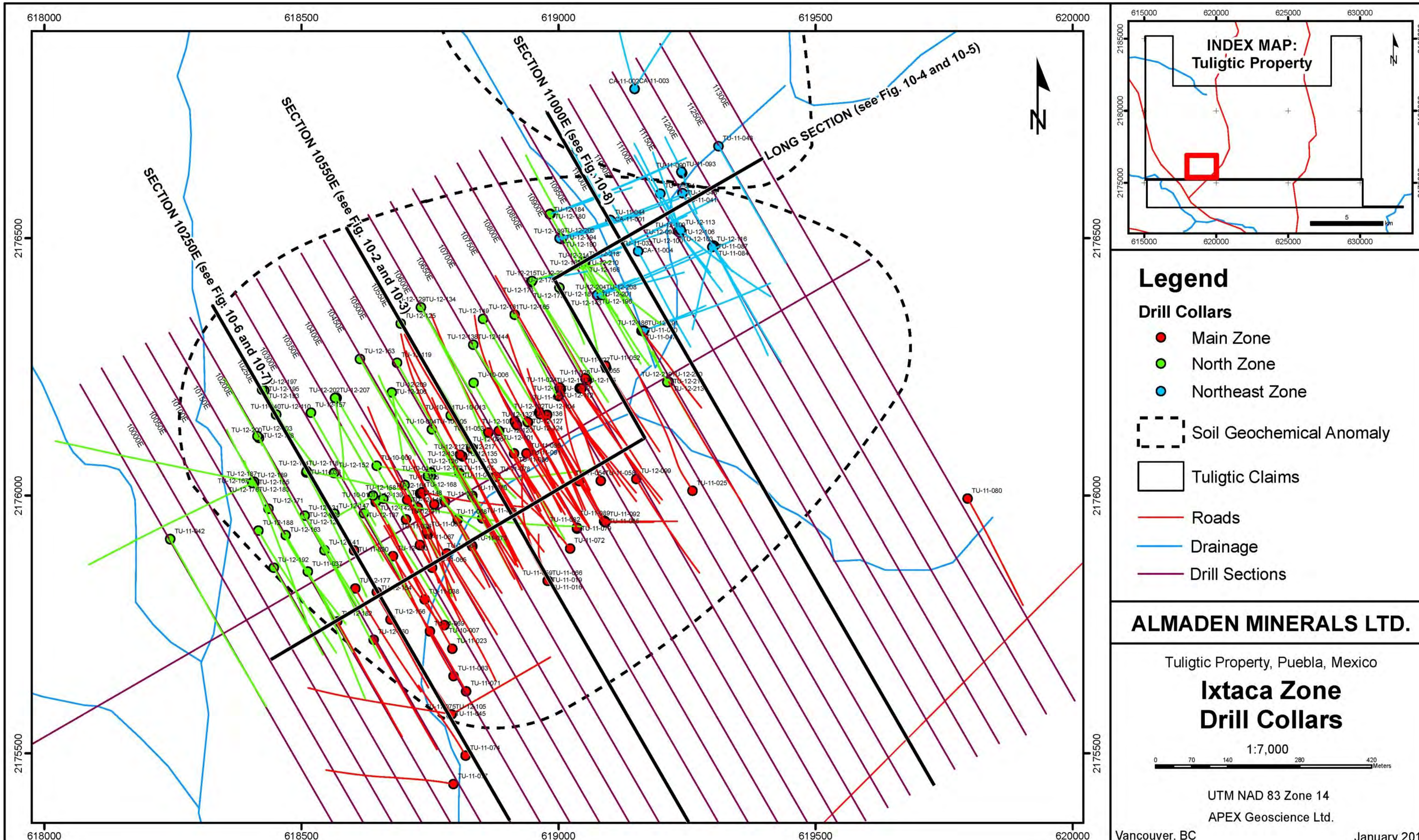
Drill core was logged based on lithology, and the presence epithermal alteration and mineralization. All core is sampled. Almaden employed a maximum sample length of 2 m in unmineralized lithologies, and a maximum sample length of 1 m in mineralized lithologies (50 cm minimum sample length). Geological changes in the core such as major alteration or mineralization intensity (including large discrete veins), or lithology were used as sample breaks.

The Upper Tamaulipas formation, the dykes that crosscut it and the upper Coyoltepec volcanic subunit are the main host rocks to the epithermal vein system at Ixtaca. In the Main and Ixtaca North zones veining strikes dominantly ENE-WNW (060 degrees) parallel to a major dyke trend and at a very high angle to the N to NNW bedding and fold structures within the limestones. The veins of the Northeast Extension Zone are hosted by the shaley carbonate units and strike to the NNW, dipping to the SSW. In the footwall to Northeast Extension Zone a parallel dyke has been identified which is altered and mineralised. The Northeast Extension Zone and the dyke are interpreted to strike parallel to bedding and to core an antiform comprised of shale.

10.1 Main Ixtaca and Ixtaca North Zones

The Main Ixtaca and Ixtaca North zones have a strike length of approximately 650 m and have been drilled at 50 m section spacing. The vast majority of holes were drilled at an azimuth of 150 or 330 degrees and at dips between 45 and 60 degrees from horizontal. Limited 25 m section infill drilling has also been completed in the central area of the Main Ixtaca Zone. Diamond drilling has intersected high-grade mineralization within the Main Ixtaca and Ixtaca North zones to depths of 200 to 300 m vertically from surface. High-grade zones occur within a broader zone of mineralization extending

Figure 10-1. Drill Hole Locations



laterally (NNW-SSE) over 600 m and to a vertical depth of 600 m below surface (Table 10-2 and Figure 10-2).

The epithermal vein system at the Main Ixtaca and Ixtaca North zones is roughly associated with two parallel ENE (060 degrees) trending, subvertical to steeply north dipping dyke zones. The dykes predate mineralisation and trend and at a high angle to the N to NNW bedding and fold structures within calcareous sediments of the Upper Tamaulipas formation.

At the Main Ixtaca Zone, a series of dykes ranging from less that 2 m to over 20 m true width occur within an approximately 100 m wide zone (Figure 10-3). Wider dykes often correlate within individual drill sections, where they are inferred to pinch or splay. The boarder dyke zone itself is correlatable between sections; although individual dykes are typically not continuous between sections. The dyke zone hosting the Ixtaca North Zone is narrower, comprising a steeply north-dipping zone of two or three discrete dykes ranging from 5 to 20 m in width. Epithermal vein mineralization occurs both within the dykes and sedimentary host rocks, with the highest grades often occurring within or proximal to the dykes. Vein density decreases outward to the north and south from the dyke zones resulting in the formation of two high-grade zones that lack sharp geologic boundaries. The dykes are often intensely altered and are interpreted to control the distribution of epithermal vein system at Ixtaca to the extent that they provided a conduit for ascending hydrothermal fluids, and an important rheological contrast resulting in vein formation within and along the margins individual dykes, and laterally within the adjacent limestone. On surface, the Main Ixtaca and Ixtaca North zones are separated by a steep sided ENE trending valley.

The lateral (WSW-ENE) extent of the epithermal vein system is controlled by N to NNW bedding and fold structures in basement rocks of the Upper Tamaulipas formation. Drilling indicates Main Ixtaca and Ixtaca North zone mineralization is bound within an ENE-verging asymmetric synform (Figure 10-4). The synfom is cored by a structurally thickened sequence of argillaceous limestone that grades laterally and at depth through calcareous siltstone and grainstone transition units, into dark grey to laminated calcareous shale at depth. Based on increased vein density, including the presence of broad alteration zones and networks of intersecting epithermal veins, the relatively brittle limestone is a preferential host to Main Ixtaca and Ixtaca North zone mineralization.

Table 10-2. Section 10+550E Significant Drill intercepts (Main Ixtaca and Ixtaca North Zones)

Hole ID	From (m)	To (m)	Interval (m)	Au (g/t)	Ag (g/t)	AuEq* (g/t)
TU-10-011	122.70	124.20	1.50	0.65	229.8	**5.1**
TU-10-011	185.09	185.64	0.55	1.13	405.7	**9.0**
TU-10-011	204.98	408.63	203.65	1.01	44.3	**1.9**
including	207.82	208.40	0.58	1.27	274.5	**6.6**
including	223.05	224.50	1.45	3.02	284.7	**8.5**

including	241.03	242.94	1.91	6.72	551.5	17.4
including	255.42	338.50	83.08	1.83	77.7	3.3
including	258.68	260.45	1.77	48.98	1391.7	75.9
including	279.23	280.63	1.40	7.82	560.3	18.7
including	292.93	296.34	3.41	2.91	133.9	5.5
including	303.09	306.90	3.81	2.79	113.1	5.0
including	333.85	336.36	2.51	6.30	237.1	10.9
TU-10-013	64.90	89.00	24.10	1.43	99.0	3.3
TU-10-013	193.65	201.33	7.68	0.21	19.2	0.6
TU-10-013	212.80	213.42	0.62	2.72	269.0	7.9
TU-10-013	289.50	289.92	0.42	6.67	304.0	12.5
TU-10-013	420.01	420.42	0.41	5.54	35.7	6.2
TU-10-013	426.62	427.70	1.08	1.69	37.2	2.4
TU-11-016	208.00	409.35	201.35	0.99	86.2	2.7
including	208.00	237.19	29.19	0.67	105.7	2.7
including	235.30	237.19	1.89	3.68	776.1	18.7
including	256.48	286.60	30.12	1.62	187.9	5.3
including	269.28	273.68	4.40	4.33	577.3	15.5
including	270.68	272.68	2.00	6.78	1038.5	26.9
including	281.79	282.84	1.05	18.15	2250.0	61.7
including	317.20	351.48	34.28	1.73	95.2	3.6
including	326.32	329.34	3.02	6.13	601.9	17.8
including	338.91	349.10	10.19	2.85	72.4	4.2
including	365.90	409.35	43.45	1.62	118.9	3.9
including	374.22	378.75	4.53	4.19	280.3	9.6
including	374.22	376.83	2.61	5.74	336.9	12.3
including	386.70	387.70	1.00	6.88	524.0	17.0
including	395.63	409.35	13.72	1.74	138.7	4.4
including	395.63	402.99	7.36	2.46	208.2	6.5
TU-11-016	439.00	443.00	4.00	1.11	13.0	1.4
TU-11-019	203.40	328.90	125.50	0.48	39.9	1.3
including	234.45	235.15	0.70	2.38	642.2	14.8
including	285.59	328.90	43.31	0.91	74.4	2.3
including	285.59	294.14	8.55	3.04	184.7	6.6
including	287.24	292.03	4.79	4.64	273.1	9.9
including	305.92	308.36	2.44	1.59	161.2	4.7
TU-11-019	369.20	372.12	2.92	3.45	418.9	11.5
TU-11-056	58.95	66.95	8.00	1.84	46.8	2.7
TU-11-056	72.54	106.60	34.06	1.63	56.2	2.7
including	73.25	78.50	5.25	5.26	77.7	6.8
including	85.65	86.65	1.00	5.95	412.5	13.9

including	92.65	100.00	7.35	1.88	100.8	**3.8**
TU-11-056	226.15	251.00	24.85	0.75	163.4	**3.9**
including	234.14	235.40	1.26	2.45	853.5	**19.0**
including	248.45	249.40	0.95	13.86	2576.8	**63.7**
TU-11-059	145.00	189.80	44.80	0.29	9.9	**0.5**
including	154.00	179.80	25.80	0.39	12.2	**0.6**
TU-11-059	277.08	277.88	0.80	0.99	112.2	**3.2**
TU-11-059	345.50	346.25	0.75	1.25	130.5	**3.8**
TU-11-059	356.50	367.95	11.45	0.33	8.1	**0.5**
TU-11-059	488.92	490.55	1.63	1.50	30.9	**2.1**
TU-11-059	522.40	534.22	11.82	0.22	3.8	**0.3**
TU-11-059	611.95	614.33	2.38	0.38	26.8	**0.9**
TU-11-059	617.97	624.82	6.85	0.44	33.8	**1.1**
including	618.17	620.33	2.16	0.76	60.9	**1.9**
TU-11-066	145.00	189.80	44.80	0.51	8.8	**0.7**
including	176.12	182.08	5.96	1.26	10.4	**1.5**
TU-11-078	3.59	76.50	72.91	0.65	28.5	**1.2**
including	24.50	54.50	30.00	1.24	50.5	**2.2**
including	37.00	48.00	11.00	2.04	100.8	**4.0**
TU-11-078	100.00	119.60	19.60	0.42	40.6	**1.2**
TU-11-078	150.00	173.00	23.00	0.72	44.0	**1.6**
including	155.00	167.42	12.42	1.11	70.5	**2.5**
including	163.70	167.42	3.72	1.84	154.8	**4.8**
TU-11-078	208.70	250.00	41.30	0.51	49.0	**1.5**
TU-11-083	55.45	60.80	5.35	0.33	38.4	**1.1**
TU-11-083	120.45	283.90	163.45	1.27	61.6	**2.5**
including	146.10	200.24	54.14	2.32	105.7	**4.4**
including	146.10	154.10	8.00	9.82	492.8	**19.3**
including	179.70	182.97	3.27	6.42	83.0	**8.0**
including	244.65	255.50	10.85	1.95	98.3	**3.9**
including	267.50	272.30	4.80	3.18	93.9	**5.0**
TU-11-088	109.00	113.00	4.00	0.37	50.8	**1.4**
TU-11-088	120.00	127.80	7.80	0.33	58.9	**1.5**
TU-11-088	150.40	170.50	20.10	0.32	26.7	**0.8**
including	167.62	170.50	2.88	1.41	103.8	**3.4**
TU-11-088	181.50	256.57	75.07	0.87	59.5	**2.0**
including	216.70	255.96	39.26	1.36	91.2	**3.1**
including	238.55	246.28	7.73	3.93	249.5	**8.8**
TU-12-125	332.00	351.50	19.50	1.20	64.1	**2.4**

*Gold Equivalent based on a three-year trailing average price of $1,500/ounce gold and $29/ounce silver

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The Limestone sequence thins to the west in response to a rising ENE-verging antiform. The Main Ixtaca and Ixtaca North veins systems and the dykes transect the antiform sub-perpendicular to the strike of the fold axis. Vein density decreases within the shale units that core the antiform and mineralization is confined near the axis of the antiform within a west dipping tabular zone of low-grade mineralization having a true thickness ranging from 150 to 200 m (Table 10-3, Figure 10-6 and 10-7).

Table 10-3. Section 10+250E Significant Drill intercepts (Main Ixtaca and Ixtaca North Zones)

Hole ID	From (m)	To (m)	Interval (m)	Au (g/t)	Ag (g/t)	AuEq* (g/t)
TU-11-030	60.00	212.00	152.00	0.91	13.6	1.2
including	60.00	68.00	8.00	9.38	3.4	9.4
including	64.00	65.00	1.00	66.80	18.4	67.2
including	79.00	212.00	133.00	0.47	15.3	0.8
including	136.00	189.44	53.44	0.82	22.9	1.3
including	166.00	180.08	14.08	1.14	48.2	2.1
including	170.00	173.00	3.00	3.31	116.9	5.6
TU-11-033	26.75	350.00	323.25	0.44	14.5	0.7
including	26.75	105.50	78.75	0.53	9.8	0.7
including	120.65	144.30	23.65	0.46	7.5	0.6
including	169.23	203.50	34.27	0.39	12.1	0.6
including	228.60	314.00	85.40	0.57	17.6	0.9
TU-11-033	402.00	404.85	2.85	1.34	7.4	1.5
TU-11-040	42.00	197.00	155.00	0.60	3.9	0.7
including	42.00	135.20	93.20	0.29	4.2	0.4
including	77.04	197.00	119.96	0.71	4.7	0.8
including	77.04	108.80	31.76	0.43	7.1	0.6
including	151.36	186.45	35.09	1.75	4.5	1.8
including	159.50	184.80	25.30	2.26	5.5	2.4
including	171.56	173.13	1.57	18.20	22.2	18.6
including	182.55	184.80	2.25	3.87	23.8	4.3
TU-11-045	65.00	146.30	81.30	0.78	4.6	0.9
including	65.00	129.00	64.00	0.94	4.7	1.0
including	69.70	118.00	48.30	1.03	4.3	1.1
including	108.85	117.00	8.15	1.99	4.2	2.1
TU-11-074	86.95	211.00	124.05	0.31	8.2	0.5
including	144.50	211.00	66.50	0.36	13.7	0.6
including	157.00	189.80	32.80	0.49	19.8	0.9
including	157.00	173.00	16.00	0.61	24.2	1.1
including	188.10	189.80	1.70	1.77	102.5	3.8
TU-12-110	40.50	93.00	52.50	0.79	3.5	0.9

including	60.50	93.00	32.50	1.12	5.1	**1.2**
TU-12-110	140.50	145.60	5.10	6.87	10.9	**7.1**
TU-12-114	23.70	118.85	95.15	0.59	4.0	**0.7**
including	111.00	117.65	6.65	2.01	10.3	**2.2**
TU-12-114	138.99	157.60	18.61	2.59	3.8	**2.7**
including	138.99	139.50	0.51	85.80	42.2	**86.6**
TU-12-114	203.00	221.10	18.10	0.75	12.2	**1.0**
TU-12-147	21.34	262.00	240.66	1.09	16.6	**1.4**
including	26.00	87.50	61.50	1.95	7.4	**2.1**
including	45.50	70.50	25.00	4.22	15.4	**4.5**
including	47.15	52.25	5.10	14.96	56.1	**16.0**
including	49.90	51.21	1.31	49.79	207.2	**53.8**
including	126.00	153.00	27.00	0.32	22.2	**0.7**
including	140.50	151.00	10.50	0.52	30.4	**1.1**
including	155.50	165.00	9.50	0.56	78.0	**2.1**
including	181.00	214.50	33.50	2.96	35.1	**3.6**
including	211.25	212.00	0.75	87.60	207.0	**91.6**
including	223.00	262.00	39.00	0.47	15.2	**0.8**
TU-12-147	276.35	279.00	2.65	0.54	65.9	**1.8**
TU-12-154	94.93	154.23	59.30	0.78	15.2	**1.1**
including	128.50	135.25	6.75	1.52	19.2	**1.9**
including	138.75	149.00	10.25	1.16	47.4	**2.1**
TU-12-156	109.12	130.85	21.73	1.56	11.7	**1.8**
TU-12-193	67.00	96.00	29.00	0.53	3.4	**0.6**
TU-12-193	107.50	112.50	5.00	0.22	3.1	**0.3**
TU-12-195	51.25	58.30	7.05	0.22	2.1	**0.3**
TU-12-195	181.00	186.50	5.50	0.23	2.0	**0.3**
TU-12-197	173.00	177.00	4.00	0.52	0.7	**0.5**

*Gold Equivalent based on a three-year trailing average price of $1,500/ounce gold and $29/ounce silver

Mineralized limestone, shale and the cross-cutting dykes are unconformably overlain by bedded crystal tuff, which is also mineralized. Mineralization within of tuff rocks overlying the Ixtaca Zone occurs as broad zones of alteration and disseminated sulphides having relatively few veins. High-grade zones of mineralization are locally present within the tuff vertically above the Main Ixtaca and Ixtaca North vein systems and dykes. The high-grade zones transition laterally into low grade mineralization, which together form a broad tabular zone of mineralization at the base of the tuff unit.

10.2 Northeast Extension Zone

The Northeast Extension Zone of the Ixtaca deposit has an approximate strike length of 350 m and has been drilled via a series of five ENE (070 degrees) oriented sections spaced at intervals of 50 to 100 m, and near-surface oblique NNW-SSE oriented drill holes (Figure 10-1). The Northeast Extension Zone dips moderately-steeply WSW. High grade mineralization having a true-width ranging from less than 30 and up to 60 m has been intersected beneath approximately 30 m of tuff to a vertical depth of 550 m, or approximately 600 m down-dip (Table 10-4, Figures 10-4 and 10-5).

The Northeast Extension Zone vein lies east of the Main Ixtaca Zone and occurs within the hinge zone of a shale cored antiform (Figure 10-4). Near surface, along the axis of the antiform, a zone structurally thinned, brecciated, and mineralized limestone is unconformably overlain by mineralized tuff rocks (Figure 10-8). At a vertical depth of 80 m below surface, high-grade shale-hosted mineralization dips moderately-steeply WSW sub-parallel to the interpreted axial plane of the antiform. The footwall of the high-grade zone is marked by a distinct 20 to 30 m true-thickness felsic porphyry dyke (Chemalaco Dyke), which is also mineralized. The Chamelaco Dyke has been interested in multiple drill holes ranging from 250 to 550 m vertically below surface, and its lower contact currently marks the base of marks the base of Northeast Extension Zone mineralization.

Table 10-4. Section 50+000N Significant Drill intercepts (Northeast Extension Zone)

Hole ID	From (m)	To (m)	Interval (m)	Au (g/t)	Ag (g/t)	AuEq* (g/t)
TU-11-090	42.69	130.00	87.31	0.12	26.6	0.6
including	61.53	73.50	11.97	0.13	62.4	1.3
including	115.75	130.00	14.25	0.10	52.3	1.1
including	118.85	123.75	4.90	0.16	75.3	1.6
TU-11-094	45.00	54.00	9.00	0.21	13.0	0.5
TU-11-094	89.10	91.60	2.50	0.33	86.8	2.0
TU-11-094	104.78	233.20	128.42	0.43	49.2	1.4
including	119.30	159.70	40.40	0.41	79.0	1.9
including	143.30	159.70	16.40	0.66	95.9	2.5
including	154.90	159.70	4.80	1.02	122.2	3.4
including	171.60	193.90	22.30	0.86	87.2	2.5
TU-12-155	51.82	106.07	54.25	0.24	7.2	0.4
TU-12-155	194.00	212.00	18.00	0.19	21.5	0.6
including	199.80	204.50	4.70	0.32	34.5	1.0
TU-12-155	227.99	272.00	44.01	1.04	95.8	2.9
including	229.80	240.30	10.50	2.65	244.9	7.4
including	230.80	233.30	2.50	8.49	683.6	21.7
including	235.80	239.30	3.50	1.23	178.4	4.7
including	242.30	248.00	5.70	0.55	65.6	1.8
including	253.00	258.50	5.50	1.81	108.1	3.9
TU-12-155	334.80	337.70	2.90	0.88	89.6	2.6
TU-12-159	51.82	76.50	24.68	0.40	16.8	0.7
TU-12-159	240.50	299.60	59.10	0.59	53.4	1.6

including	240.50	250.00	9.50	1.21	101.2	3.2
including	244.00	246.50	2.50	3.10	232.9	7.6
including	270.00	299.60	29.60	0.75	69.0	2.1
including	271.50	287.55	16.05	0.91	90.3	2.7
including	273.00	276.50	3.50	0.81	91.0	2.6
including	280.00	283.05	3.05	1.34	88.6	3.1
including	286.05	287.55	1.50	2.00	233.0	6.5
including	295.60	298.10	2.50	1.70	141.0	4.4
TU-12-159	337.60	341.10	3.50	1.01	7.2	1.1
including	340.10	341.10	1.00	2.76	9.3	2.9
TU-12-162	51.82	71.80	19.98	0.38	1.3	0.4
TU-12-162	84.00	94.00	10.00	0.16	6.0	0.3
TU-12-162	250.50	319.00	68.50	1.16	36.6	1.9
including	263.50	314.50	51.00	1.47	41.6	2.3
including	264.50	280.00	15.50	2.42	70.5	3.8
including	264.50	268.00	3.50	5.24	125.1	7.7
including	293.50	301.00	7.50	2.25	47.0	3.2
TU-12-162	333.00	347.60	14.60	0.43	16.1	0.7
TU-12-166	54.25	69.00	14.75	0.49	2.4	0.5
TU-12-166	284.00	433.90	149.90	0.90	12.0	1.1
including	302.00	401.80	99.80	1.24	14.9	1.5
including	302.00	305.50	3.50	1.34	16.6	1.7
including	322.00	381.60	59.60	1.59	18.5	1.9
including	334.70	338.20	3.50	2.77	40.0	3.5
TU-12-215	70.30	111.60	41.30	0.54	3.8	0.6
TU-12-215	153.70	166.50	12.80	0.10	7.5	0.2
TU-12-215	473.50	491.30	17.80	0.69	36.1	1.4
including	476.50	488.30	11.80	0.92	50.3	1.9
TU-12-215	509.45	554.15	44.70	0.26	12.4	0.5
TU-12-221	71.70	113.30	41.60	0.68	3.6	0.7
including	73.20	78.10	4.90	2.62	5.2	2.7
TU-12-221	409.50	507.25	97.75	1.49	10.1	1.7
including	451.50	469.50	18.00	6.36	14.0	6.6
including	451.50	453.50	2.00	7.01	25.7	7.5
including	458.75	469.50	10.75	8.22	13.8	8.5
TU-12-221	520.25	523.75	3.50	0.16	10.6	0.4

*Gold Equivalent based on a three-year trailing average price of $1,500/ounce gold and $29/ounce silver

Figure 10-2. Section 10+550E through the Main Ixtaca Zone and Ixtaca North Zone

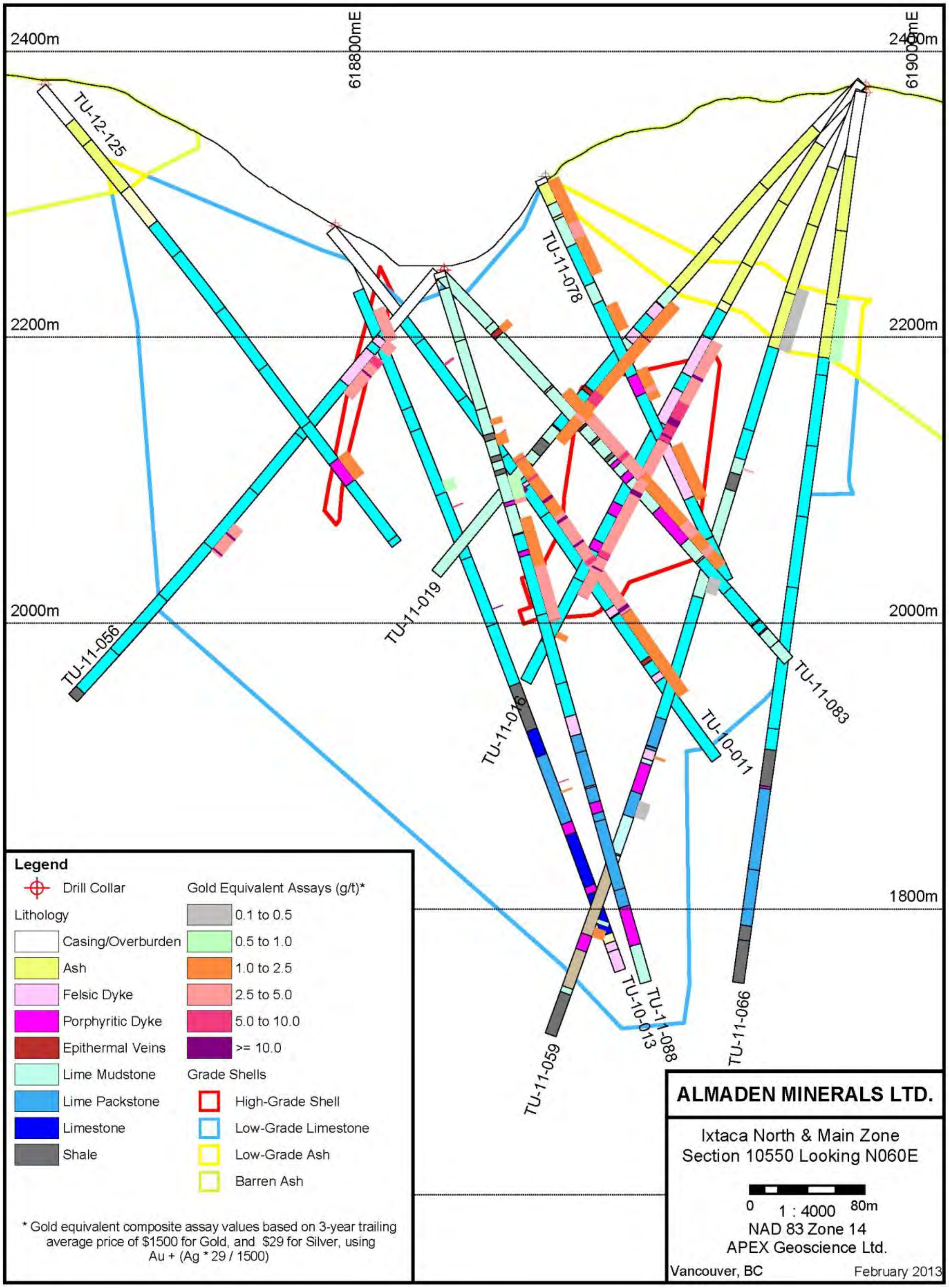

Figure 10-3. Schematic Section 10+550E through the Main Ixtaca Zone and Ixtaca North Zone

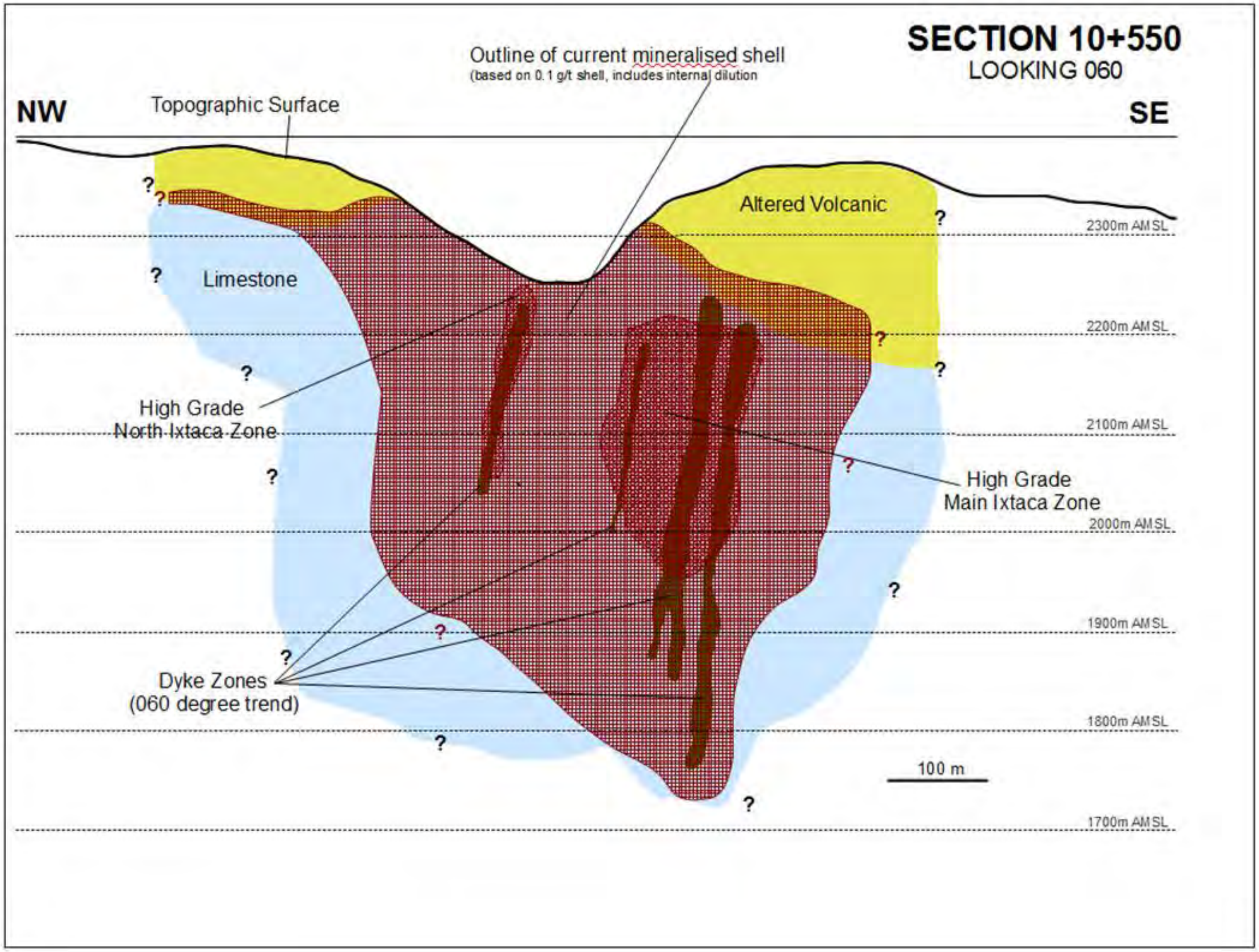

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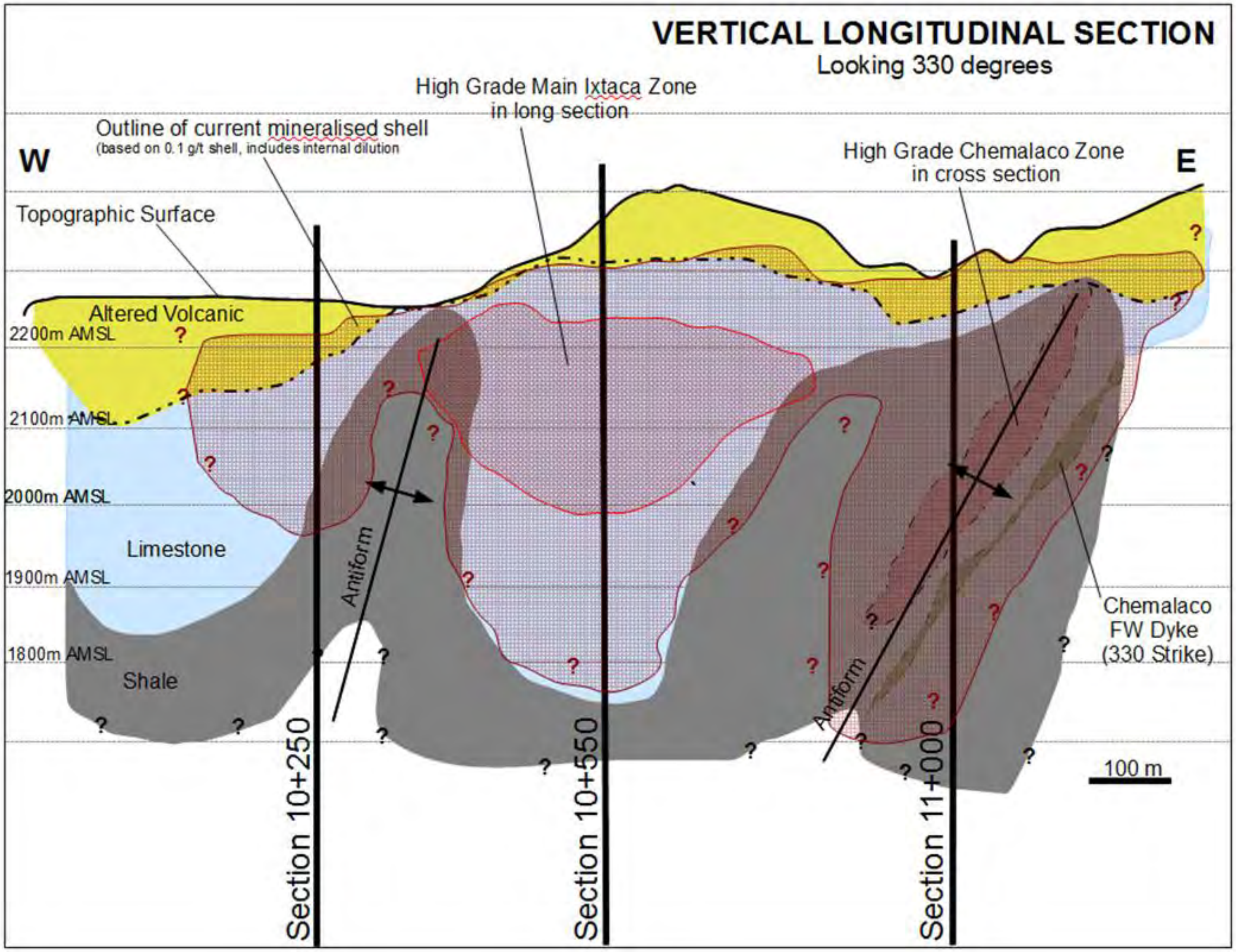
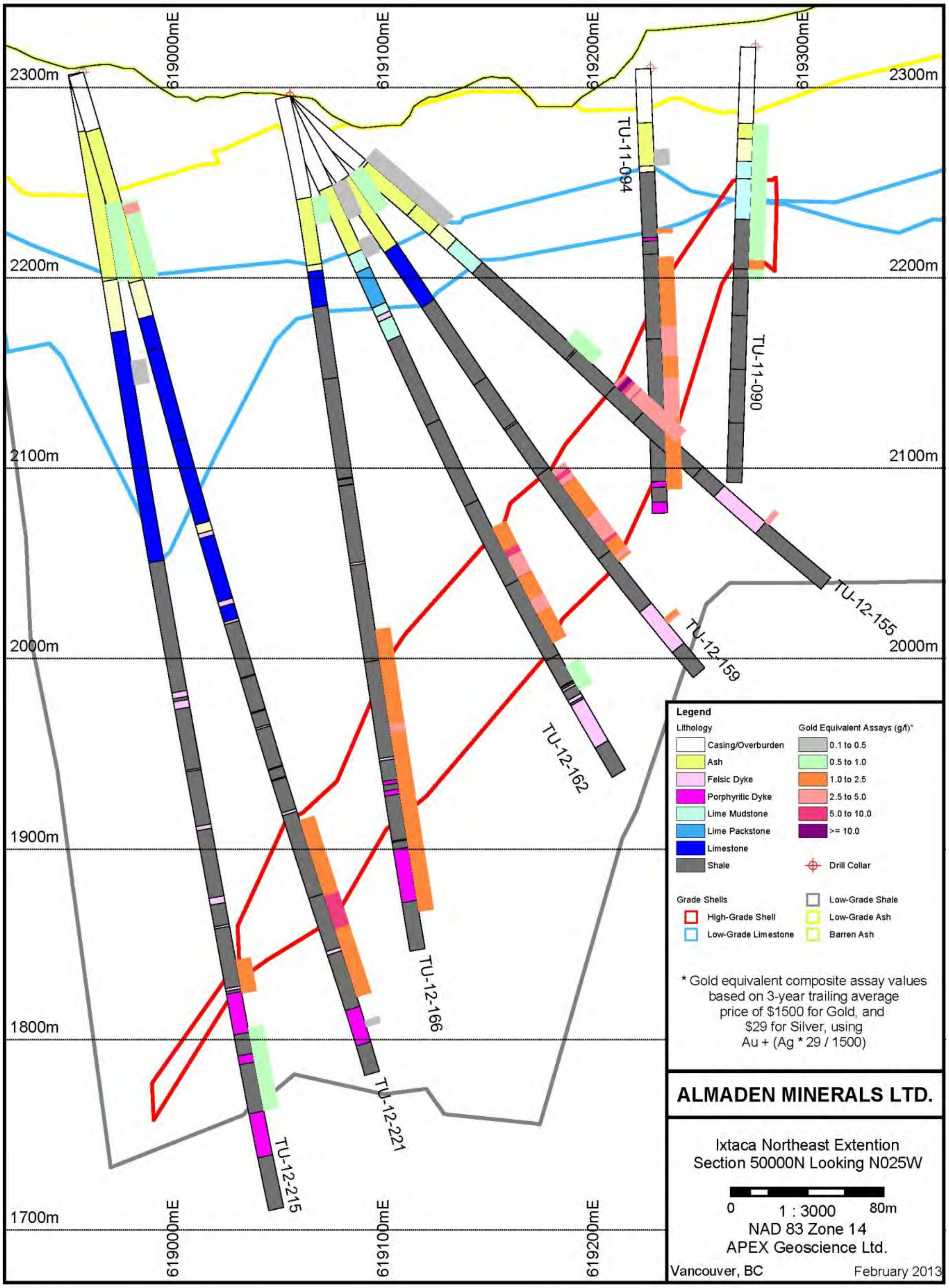
Figure 10-4. Schematic Vertical Longitudinal Section through Main Ixtaca Zone



APEX Geoscience Ltd.

Figure 10-5. Section 50+000N through the Northeast Extension Zone



APEX Geoscience Ltd.

Figure 10-6. Schematic Section 10+250E through the Main Ixtaca Zone and Ixtaca North Zone

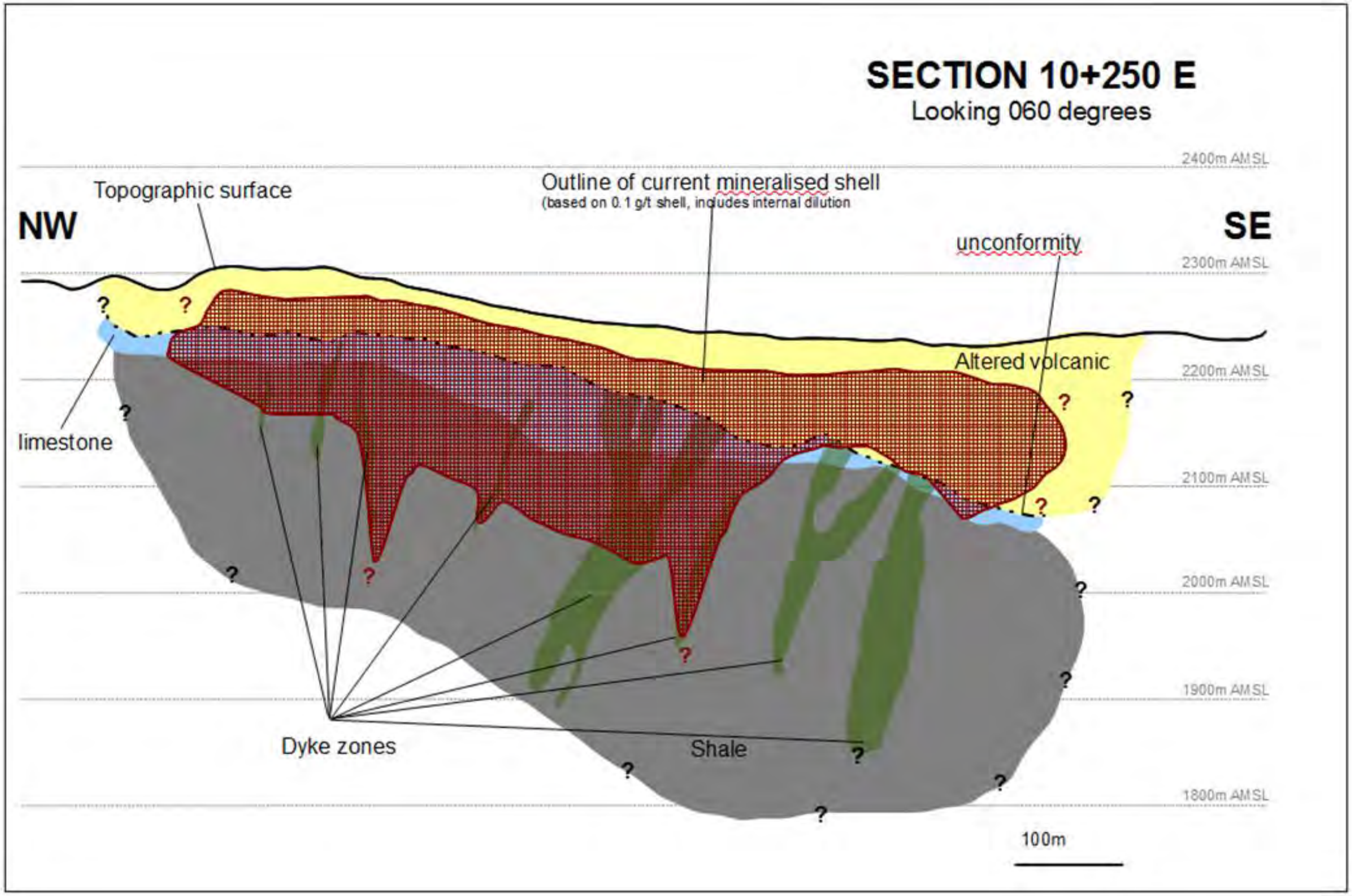


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Figure 10-7. Section 10+250E through the Main Ixtaca Zone and Ixtaca North Zone

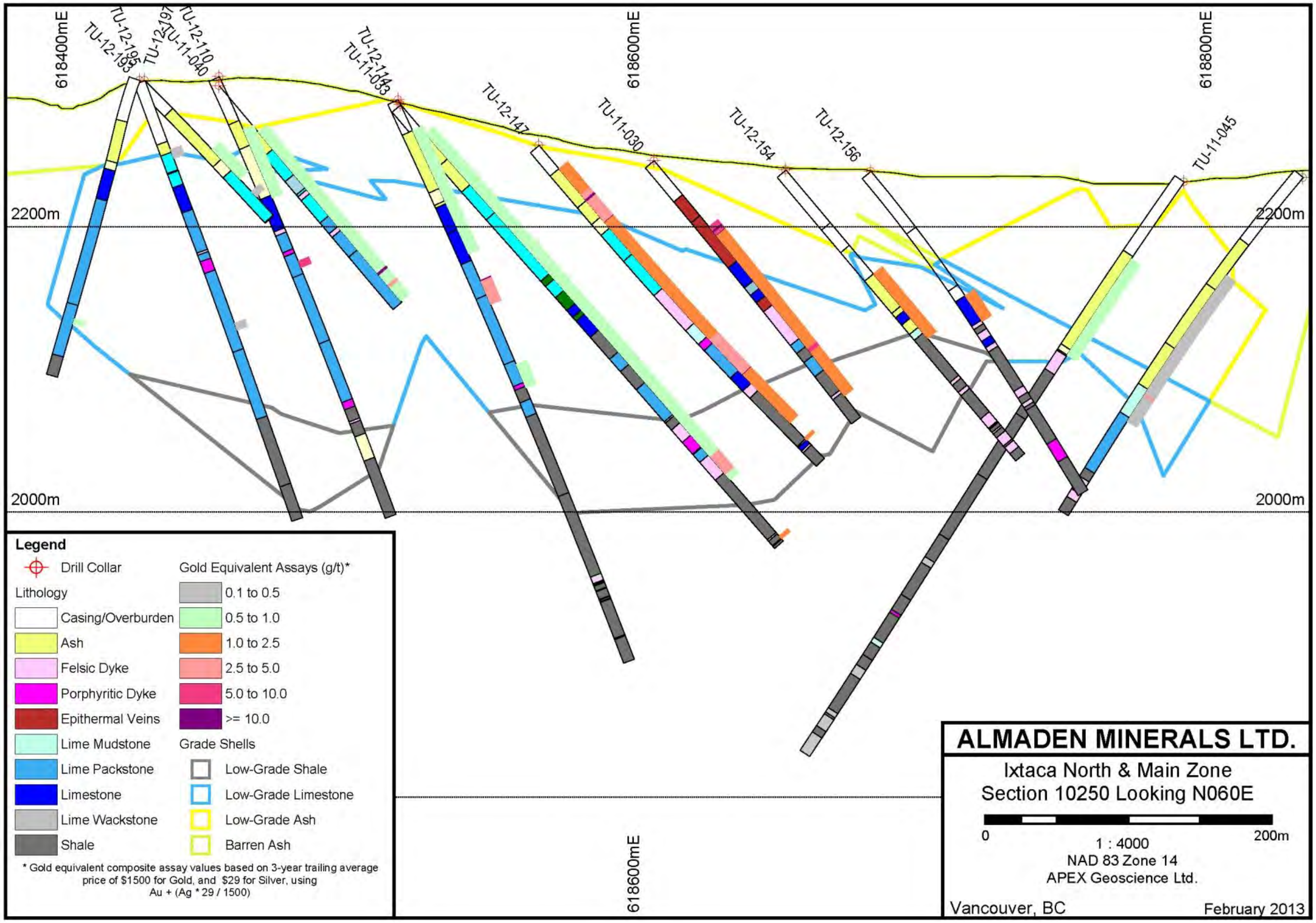
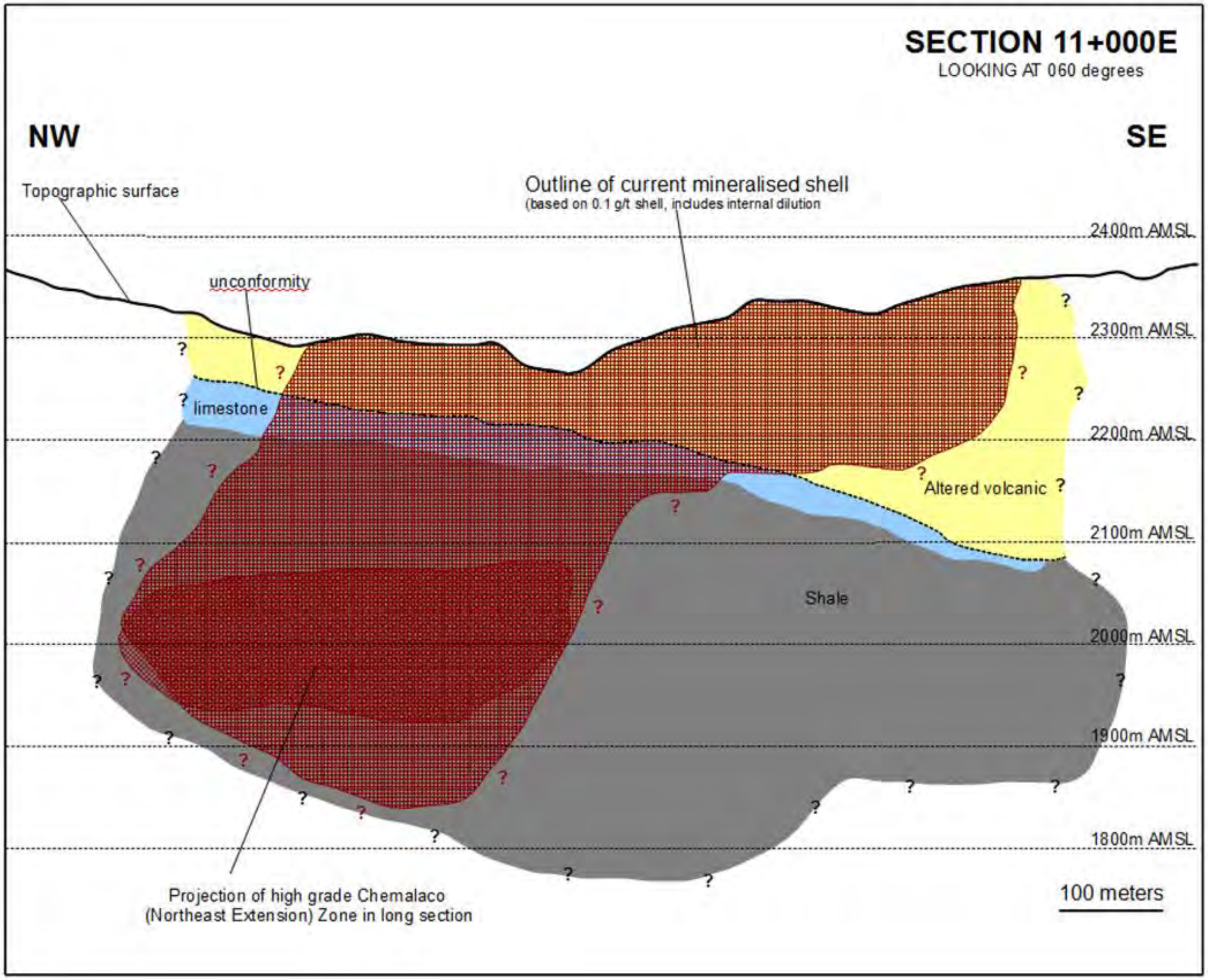


Figure 10-8. Schematic Vertical Longitudinal Section 11+000E through the Northeast Extension Zone



11 Sample Preparation, Analyses and Security

11.1 Sample Preparation and Analyses

11.1.1 Rock Grab and Soil Geochemical Samples

Rock grab and soil geochemical samples were transported by Almaden field personnel to the Santa Maria core facility where they were placed in into sealed plastic twine (rice) sacks, sealed using single plastic cable ties. Custody of samples is handed over to ALS Minerals (ALS) at the Santa Maris core facility. ALS sends its own trucks to the Project to transport samples to its sample preparation facility in Guadalajara or Zacatecas, Mexico. Prepared sample pulps were then forwarded by ALS personnel to the ALS North Vancouver, British Columbia laboratory for analysis.

ALS is an International Standards Organization (ISO) 9001:2008 and ISO 17025-2005 certified geochemical analysis and assaying laboratory. ALS is independent of Almaden and the authors.

ALS reported nothing unusual with respect to the shipments, once received. Almaden and the authors did not have control over the samples at all times during transport, and therefore cannot verify what happened to the samples from shipping up to the time they were received by ALS. However, the author has no reason to believe that the security of the samples was compromised.

At the ALS Zacatecas and Guadalajara sample preparation facilities rock grab samples were dried prior to preparation and then crushed to 10 mesh (70% minimum pass) using a jaw crusher. The samples were then split using a riffle splitter, and sample splits were further crushed to pass 200 mesh (85% minimum pass) using a ring mill pulverizer (ALS PREP-31 procedure). Soil samples were dried and sieved to 80 mesh.

Rock grab samples were subject to gold determination via a 50 gram (g) fire-assay (FA) fusion utilizing atomic absorption spectroscopy (AA) finish with a lower detection limit of 0.005 ppm Au (5 ppb) and upper limit of 10 ppm Au (ALS method Au-AA24). A 50 gram (g) prepared sample is fused with a mixture of lead oxide, sodium carbonate, borax, silica and other reagents as required, inquarted with 6 mg of gold-free silver and then cupelled to yield a precious metal bead. The bead is digested in 0.5 ml dilute nitric acid and 0.5 ml concentrated hydrochloric acid. The digested solution is cooled, diluted to a total volume of 4 ml with de-mineralized water, and analyzed by atomic absorption spectroscopy against matrix-matched standards.

Soil samples were subject to gold determination via is digestion of a 50 g prepared sample in a mixture of 3 parts hydrochloric acid and 1 part nitric acid (aqua regia). Dissolved gold is then determined by ICP-MS.

Silver, base metal and pathfinder elements for rock and soil samples were analyzed by 33-element inductively coupled plasma atomic emission spectroscopy (ICP-AES), with a 4-acid digestion (ALS method ME-ICP61). A 0.25 g prepared sample is digested with perchloric, nitric, hydrofluoric and hydrochloric acids. The residue is topped up with dilute hydrochloric acid and the resulting solution is analyzed by ICP-AES. For rock samples only, following this analysis, the results are reviewed for high concentrations of bismuth, mercury, molybdenum, silver and tungsten and diluted accordingly. Samples meeting this criterion are then analyzed by inductively coupled plasma mass spectrometry (ICP-MS, ALS method ME-MS61). Results are corrected for spectral inter-element interferences. Four acid digestions are able to dissolve most minerals; however, depending on the sample matrix, not all elements are quantitatively extracted.

11.1.2 Almaden Drill Core

All strongly altered or epithermal-mineralized intervals of core were sampled. Almaden employed a maximum sample length of 2 m in unmineralized lithologies, and a maximum sample length of 1 m in mineralized lithologies (50 cm minimum sample length). Sampling always began at last 5 samples above the start of mineralization. Geological changes in the core such as major alteration or mineralization intensity (including large discrete veins), or lithology were used as sample breaks.

Drill core was half-sawn using industry standard gasoline engine-powered diamond core saws, with water fresh water cooled blades and “core cradles” to ensure a straight cut. For each sample, the core logging geologist marks a cut line down the centre of the core designed to produce two halves of equal proportions of mineralization. This is accomplished by marking the cut line down the long axis of ellipses described by the intersection of the veins with the core circumference.

Areas of very soft rock (e.g. fault gouge), are cut with a machete using the side of the core channel to ensure a straight cut. Areas of very broken core (pieces <1 cm) were sampled using spoons. In all cases, the right hand side of the core (looking down the hole) was sampled. After cutting half the core was placed in a new plastic sample bag and half was placed back in the core box. Between each sample, the core saw and sampling areas was washed to ensure no contamination between samples. Field duplicate, blank and analytical standards were added into the sample sequence as they were being cut.

Sample numbers were written on the outside of the sample bags twice and the numbered tag from the ALS sample book was placed inside the bag with the half core. Sample bags were sealed using single plastic cable-ties. Sample numbers were checked against the numbers on the core box and the sample book.

Drill core samples collected by the Almaden were placed into sealed plastic twine (rice) sacks, sealed using single plastic cable ties. ALS takes custody of the samples at the SANTA Maria core facility. and ALS sends its own trucks to the Project to transport samples to its sample preparation facility in Guadalajara or Zacatecas, Mexico. Prepared sample pulps were then forwarded by ALS personnel to the ALS North Vancouver, British Columbia laboratory for analysis.

Drill core samples were subject to gold determination via a 50 gram (g) AA finish FA fusion with a lower detection limit of 0.005 ppm Au (5 ppb) and upper limit of 10 ppm Au

(ALS method Au-AA24). A 50 g prepared sample is fused with a flux mixture, inquarted with 6 mg of gold-free silver and then cupelled to yield a precious metal bead. The bead is digested in 0.5 ml dilute nitric acid and 0.5 ml concentrated hydrochloric acid. The digested solution is cooled, diluted to a total volume of 4 ml with de-mineralized water, and analyzed by atomic absorption spectroscopy against matrix-matched standards.

Over limit gold values (>10 ppm Au) are were subject to gravimetric analysis, whereby a 50 g prepared sample is fused with a mixture of lead oxide, sodium carbonate, borax, silica and other reagents in order to produce a lead button. The lead button containing the precious metals is cupelled to remove the lead. The remaining gold and silver bead is parted in dilute nitric acid, annealed and weighed as gold (ALS method Au-GRA22).

Silver, base metal and pathfinder elements for drill core samples were analyzed by 33-element ICP-AES, with a 4-acid digestion, a lower detection limit of 0.5 ppm Ag and upper detection limit of 100 ppm Ag (ALS method ME-ICP61). A 0.25 g prepared sample is digested with perchloric, nitric, hydrofluoric and hydrochloric acids. The residue is topped up with dilute hydrochloric acid and the resulting solution is analyzed by ICP-AES (ALS method ME-ICP61). Four acid digestions are able to dissolve most minerals; however, depending on the sample matrix, not all elements are quantitatively extracted.

Over limit silver values (>100 ppm Ag) were subject to 4-acid digestion ICP-AES analysis with an upper limit of 1,500 ppm Ag (ALS method ME-OG62). A prepared sample is digested with nitric, perchloric, hydrofluoric, and hydrochloric acids, and then evaporated to incipient dryness. Hydrochloric acid and de-ionized water is added for further digestion, and the sample is heated for an additional allotted time. The sample is cooled and transferred to a 100 ml volumetric flask. The resulting solution is diluted to volume with de-ionized water, homogenized and the solution is analyzed by ICP-AES. Ultra-high grade silver values (>1,500 ppm Ag) were subject to gravimetric analysis with an upper detection limit of 10,000 ppm Ag (Ag-GRA22).

11.1.1 Authors Drill Core

Drill core samples collected by Kristopher J. Raffle, P.Geo. were placed into sealed plastic bags and transported by the author to ALS North Vancouver, British Columbia laboratory for gold FA and ICP-MS analysis. The author did not have control over the samples at all times during transport, and therefore cannot personally verify what happened to the samples from shipping up to the time they were received by ALS. However, the author has no reason to believe that the security of the samples was compromised.

The samples were dried prior to preparation and then crushed to 10 mesh (70% minimum pass) using a jaw crusher. The samples were then split using a riffle splitter, and sample splits were further crushed to pass 200 mesh (85% minimum pass) using a ring mill pulverizer (ALS PREP-31 procedure). Soil samples were dried and sieved to 80 mesh.

Drill core samples collected by the author were subject to gold determination via a 50 gram (g) AA finish FA fusion with a lower detection limit of 0.005 ppm Au (5 ppb) and upper limit of 10 ppm Au (ALS method Au-AA24). A 50 g prepared sample is fused with a flux mixture, inquarted with 6 mg of gold-free silver and then cupelled to yield a precious metal bead. The bead is digested in 0.5 ml dilute nitric acid and 0.5 ml concentrated hydrochloric acid. The digested solution is cooled, diluted to a total volume of 4 ml with de-mineralized water, and analyzed by atomic absorption spectroscopy against matrix-matched standards.

Silver, base metal and pathfinder elements for rock and soil samples were analyzed by 33-element inductively coupled plasma atomic emission spectroscopy (ICP-AES), with a 4-acid digestion. A 0.25 g prepared sample is digested with perchloric, nitric, hydrofluoric and hydrochloric acids. The residue is topped up with dilute hydrochloric acid and the resulting solution is analyzed by ICP-AES. Following this analysis, the results are reviewed for high concentrations of bismuth, mercury, molybdenum, silver and tungsten and diluted accordingly. Samples meeting this criterion are then analyzed by inductively coupled plasma mass spectrometry (ICP-MS, ALS method ME-MS61). Results are corrected for spectral inter-element interferences. Four acid digestions are able to dissolve most minerals; however, depending on the sample matrix, not all elements are quantitatively extracted.

Over limit silver values (>100 ppm Ag) were subject to 4-acid digestion, ICP-AES analysis with an upper limit of 1,500 ppm Ag (ALS method ME-OG62). A prepared sample is digested with nitric, perchloric, hydrofluoric, and hydrochloric acids, and then evaporated to incipient dryness. Hydrochloric acid and de-ionized water is added for further digestion, and the sample is heated for an additional allotted time. The sample is cooled and transferred to a 100 ml volumetric flask. The resulting solution is diluted to volume with de-ionized water, homogenized and the solution is analyzed by ICP-AES.

11.2 Quality Assurance / Quality Control Procedures

For the Tuligitc Rock grab sample and soil geochemical programs Almaden relied on external quality assurance and quality control (QA/QC) measures employed by ALS. QA/QC measures at ALS include routine screen tests to verify crushing efficiency, sample preparation duplicates (every 50 samples), and analytical quality controls (blanks, standards, and duplicates). QC samples are inserted with each analytical run, with the minimum number of QC samples dependant on the rack size specific to the chosen analytical method. Results for quality control samples that fall beyond the established limits are automatically red-flagged for serious failures and yellow-flagged for borderline results. Every batch of samples is subject to a dual approval and review process, both by the individual analyst and the Department Manager, before final approval and certification. The author has no reason to believe that there are any issues or problems with the preparation or analyzing procedures utilized by ALS.

Drill core samples are subject to Almaden's internal QA/QC program that includes the insertion of analytical standard, blank and duplicate samples into the sample stream. A total of 15 QA/QC samples are present in every 100 samples sent to the laboratory.

QA/QC sample results are review following receipt of each analytical batch. QA/QC samples falling outside established limits are flagged and subject to review and possibly re-analysis, along with the 10 preceding and succeeding samples (prior to August 7, 2012, a total of 5 samples preceding and 5 samples succeeding the reviewable QA/QC sample were re-analyzed). Where the re-analyses fall within acceptable QA/QC limits the values are added to the drill core assay database. Summary results of Almaden's internal QA/QC procedures are presented below.

In the author's opinion, Almaden's QA/QC procedures are reasonable for this type of deposit and the current level of exploration. Of the 8,128 QA/QC analytical standard and blank samples submitted for analysis, a total of 52 (0.64%) were subject to initial review based on Almaden's established criteria. Based on the results of the QA/QC sampling summarized below, the analytical data is considered to be accurate; the analytical sampling is considered to be representative of the drill sample, and the analytical data to be free from contamination. The analytical data is suitable for inclusion into a resource estimate.

11.2.1 Analytical Standards

A total of 14 different analytical standards are being used on the project each having an accepted gold and silver concentration as well as known "between laboratory"" standard deviations, or expected variability, associated with each standard. The standards included 6 gold only, 3 silver only, and 6 multi-element gold-silver standards, with accepted values ranging from 0.438 to 29.21 g/t Au, and 13.4 to 205.6 g/t Ag. One analytical standard for every 20 samples (5%) was inserted into the sample stream at the '05', '25', '45', '65' and '85' positions. QA/QC summary charts showing gold and silver values for each analytical standard in addition to the accepted value, the second, and third "between laboratory" standard deviation are shown in Figure 12-3 below.

Between 2010 and 2012 Almaden employed two separate criteria by which standards were assigned "pass" or "reviewable" status.

Up to drill hole TU-12-130 a reviewable standard was defined as any standard occurring within a reported mineralized interval returning greater than three (3) standard deviations (>3SD) above the accepted value for gold or silver. Standards returning >3SD below the accepted value were not flagged as reviewable, similarly >3SD standards occurring outside of reported mineralized intercepts were not flagged as reviewable. Beginning with drill hole TU-12-131, a reviewable standard was defined as any standard occurring anywhere in a drill hole returning >3SD above the accepted value for gold or silver. In addition, two standards analyzed consecutively returning values ranging from >2SD to <3SD above the accepted value for at least one element were classified as reviewable (gold or silver, both must be above the accepted value).

All standard samples returning gold or silver values outside the established criteria were reviewed. A decision to conduct reanalysis of samples surrounding the reviewable standard was based on whether the standard returned a value above or below the accepted value (low, or slightly high >3SD values were allowed after data review) or if it occurred within a reported interval (>3SD values were allowed outside of reported intervals) Prior to August 7, 2012, when a reviewable standard was recognized the 5 preceding and 5 succeeding samples, in addition to the standard were subject to review and possibly re-analysis. After August 7, 2012 when a reviewable standard was recognized the 10 preceding and 10 succeeding samples, in addition to the standard were subject to review and possibly re-analysis. The results of re-analysis were then compared to the original analysis. Provided that no significant systematic increase or decrease in gold and silver values is noted and the re-analyzed standard returned values within the expected limits; the QA/QC concern was considered resolved and the re-analyzed standard value was added to the drill hole database.

A total of 4,066 analytical standards were inserted into the sample stream of 69,175 assays for gold and silver for the 225 drill holes. Of the 4,066 standards a total of 2,357 are subject to review criteria in place up to drill hole TU-12-130. The remaining 1,709 samples are subject to the current review criteria (TU12-131 and later).

Based on an examination of the Ixtaca QA/QC database, a total of seven (7) analytical standards subject to the pre-TU-12-131 criteria are reviewable (0.3%). Upon inspection by Almaden, five (5) of the standards returned “slightly” high values between 3.03 to 3.50 SD above the expected value for gold or silver. All five of the standards occurred within different drill holes, and different laboratory analytical batches. Based on a review of adjacent QA/QC samples no concerns were noted, therefore it was determined re-analysis was not warranted. The remaining two (2) standards returned 3.8 and 3.9 SD above the expect values for gold and silver, respectively; within two separate drill holes. Re-analysis of the remaining standard material returned values of 3.9 and 4.0 SD above the expected values for gold and silver, indicating initial standard analysis were accurate.

Of the 1,709 QA/QC samples subject to post-TU-12-130 criteria a total of 21 (1.2%) were reviewable as a result of two consecutive standards returning >2SD from the accepted value, or a single standard returning >3SD from the accepted value for gold or silver. This included a total of 13 standards returning >3SD from the accepted value, and three (3) pairs of consecutive standards returning >2SD and <3SD from the accepted value.

A total of 10 of the 21 reviewable standards, involved Au-Ag multi-element analytical standard CDN-ME-11. This is considered a high rate of failure that places the accuracy of this standard in question. Of the 13 standards returning >3D from the expected value, a total of five (5) occurred outside reported intervals and were therefore not reviewed further. Of the remaining eight (8) standards a total of six (6) returned “slightly” high values between 3.1 to 3.6 SD above the expected value for gold or silver (including 6 of standard CDN-ME-11). Four (4) of the standards occurred within different drill holes, and different laboratory analytical batches. Two (2) of the standards (including 1 of standard CDN-ME-11) occurred within drill hole TU-12-157 and are within the same analytical batch; however they are not consecutive. Based on a review of adjacent QA/QC samples no concerns were noted, therefore it was determined re-analysis was not warranted. The remaining two (2) standards returned 3.3 and 3.4 SD below the expected value for gold; therefore re-analysis was not warranted.

11.2.2 Blanks

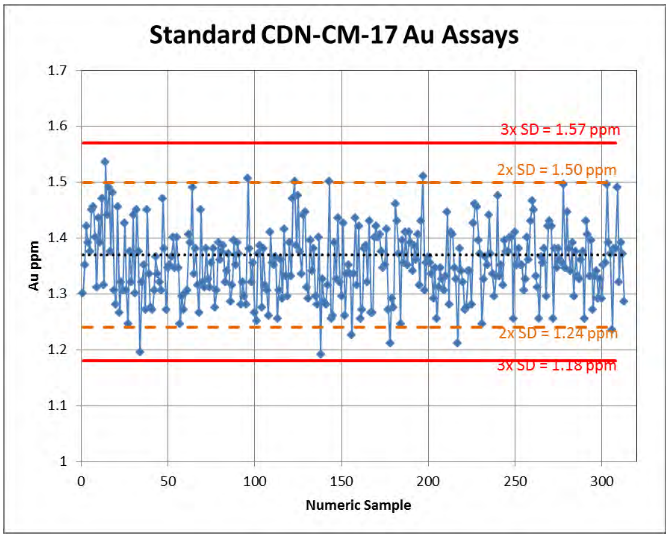
Local limestone gravel was used for coarse “blank” samples to monitor potential contamination during the sample preparation procedure. One blank for every 20 samples (5%) was inserted into the sample stream at the ‘10’, ‘30’, ‘50’, ‘70’, and ‘90’ positions. Blank samples returning values of greater than 50 ppb Au and/or 5 ppm Ag were flagged for review.

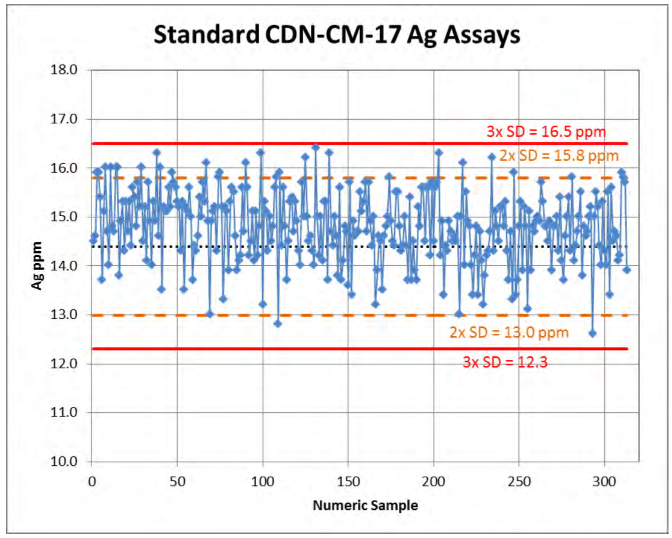
Reviewable blank samples occurring outside a reported mineralized intercept were not subject to re-analysis. In the event that a blank returned values above the accepted limits for gold or silver (prior to August 7, 2012), the blank and 5 samples on either side were re-analyzed. To provide additional confidence, on August 7, 2012, Almaden increased the number of samples re-analyzed to 10 samples. The results of re-analysis were then compared to the original analysis. Provided that no significant systematic increase or decrease in gold and silver values is noted and the re-analyzed blank did not return values above the accepted limits; the QA/QC concern was considered resolved and the re-analyzed blank value was added to the drill hole database.

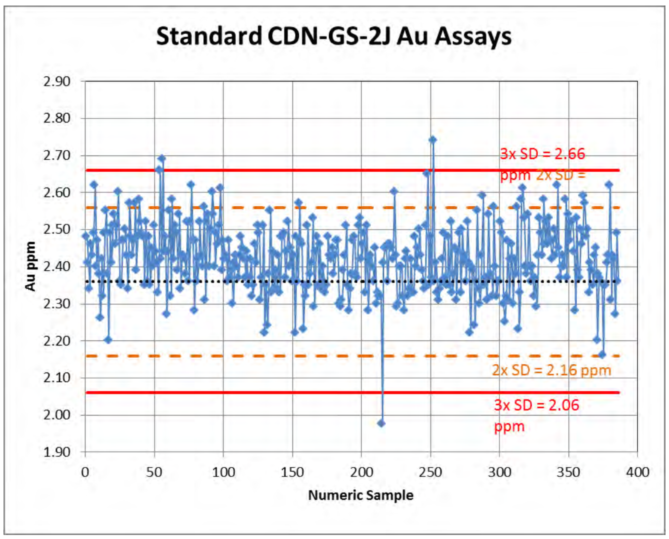
Of the 4,062 blank samples analyzed up to the end of drill hole TU-12-221 (the end or resource estimate cut-off) a total of 19 blanks returned assays of greater than 50 ppb Au, and a total of 16 samples returned greater than 5 ppm Ag (Figure 12-2). 11 samples exceeded the accepted limits for both gold and silver, 8 gold only, and 5 silver only; for a total of 24 reviewable blank samples.

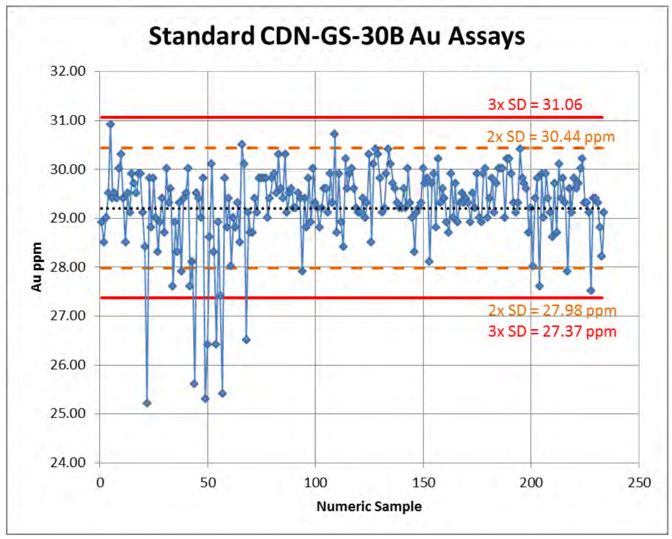
Five (5) of the blank samples, occur within unmineralized zones and therefore were not subject to further review. Eighteen (18) of the remaining 19 reviewable blank samples occur within reported mineralized intercepts and follow very high grade samples that returned values ranging from 0.266 to 55.6 g/t Au, and 38.4 to 2,280 g/t Ag. Blanks returning above accepted values in these cases occur as a result of carryover from very high-grade samples and are considered reasonable given the magnitude of the preceding gold and silver values; therefore no re-analysis was completed. The single remaining blank failure occurred within a wide zone of relatively low grade mineralization and returned 65 ppb Au and insignificant silver.

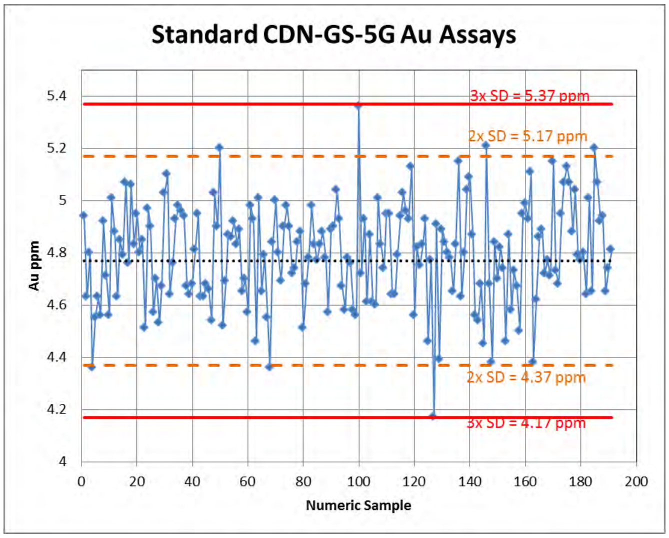
Figure 11-1. QA/QC Analytical Standards

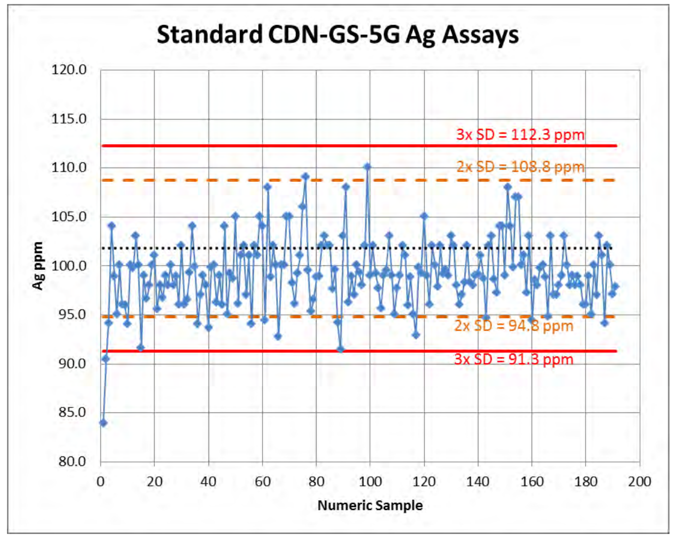


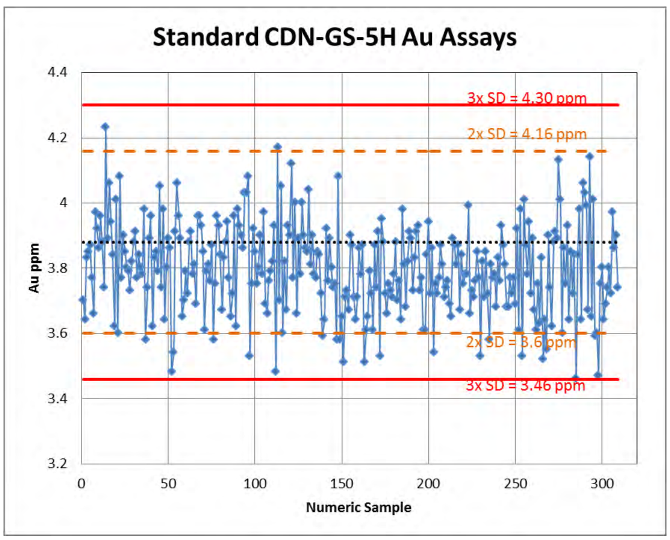


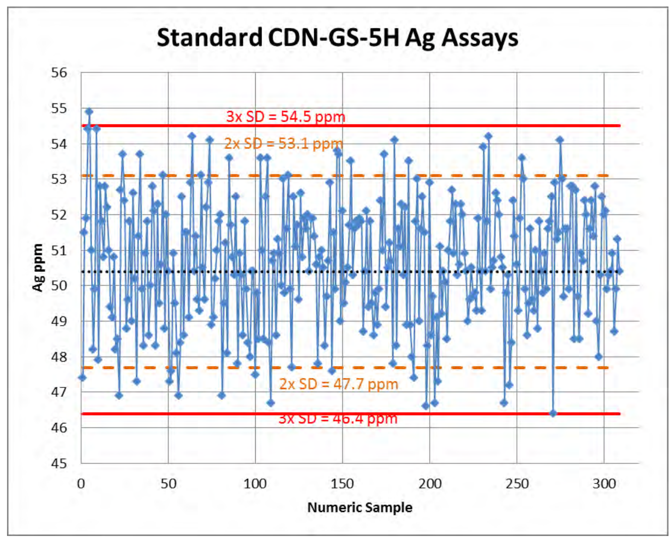


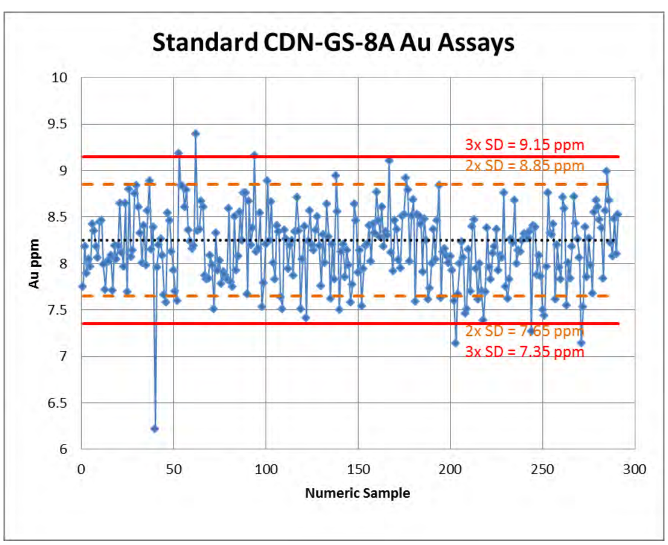


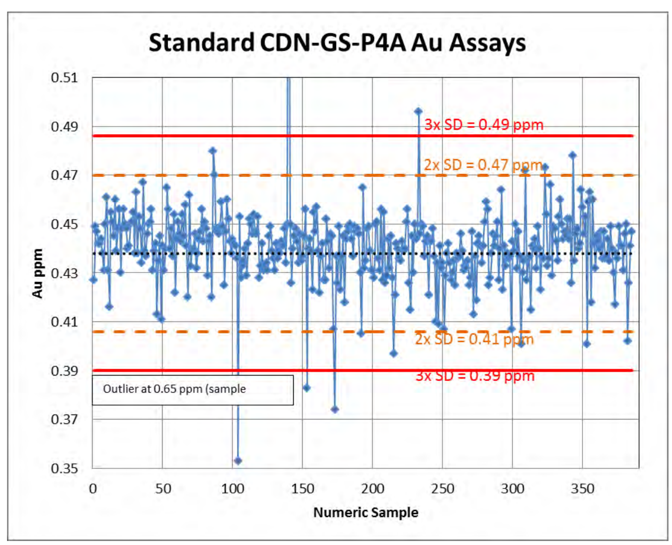

Standard CDN-GS-5G Au Assays
3x SD = 5.37 ppm
2x SD = 5.17 ppm
2x SD = 4.37 ppm
3x SD = 4.17 ppm
Au ppm
Numeric Sample

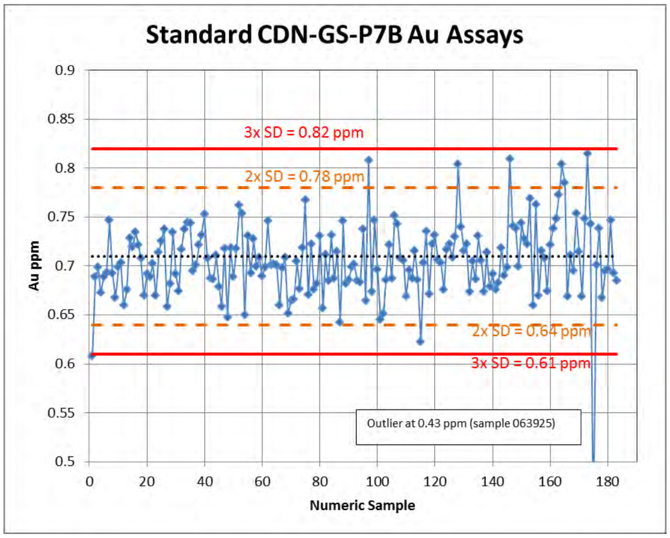

Standard CDN-GS-5G Ag Assays
3x SD = 112.3 ppm
2x SD = 108.8 ppm
2x SD = 94.8 ppm
3x SD = 91.3 ppm
Ag ppm
Numeric Sample

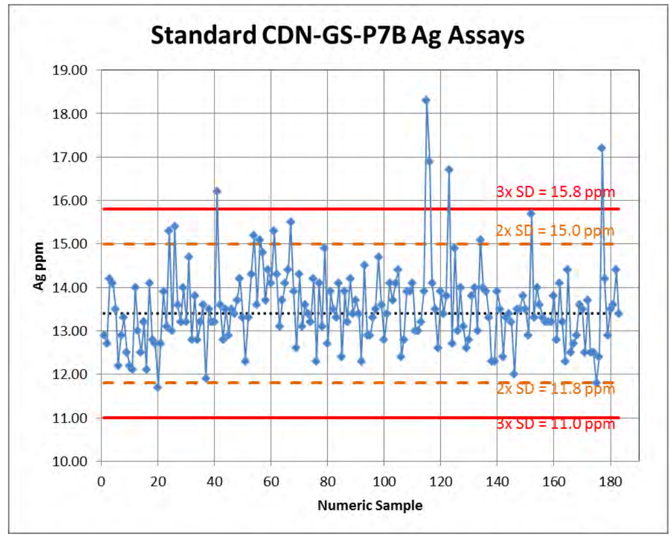

Standard CDN-GS-5H Au Assays
3x SD = 4.30 ppm
2x SD = 4.16 ppm
2x SD = 3.6 ppm
3x SD = 3.46 ppm
Au ppm
Numeric Sample

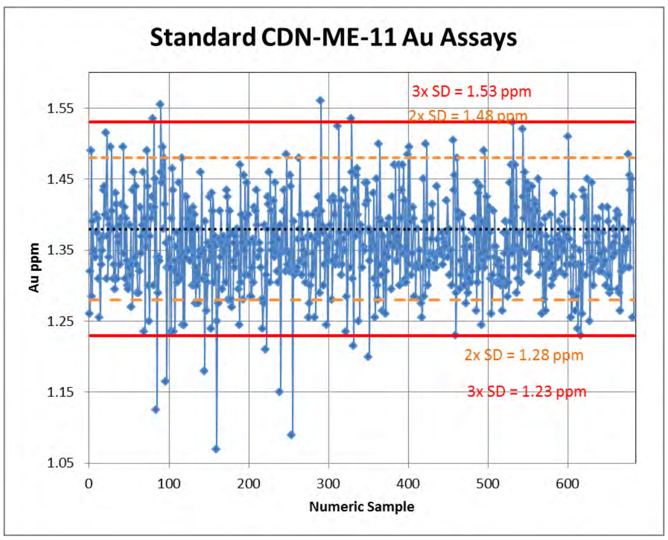

Standard CDN-GS-5H Ag Assays
3x SD = 54.5 ppm
2x SD = 53.1 ppm
2x SD = 47.7 ppm
3x SD = 46.4 ppm
Ag ppm
Numeric Sample

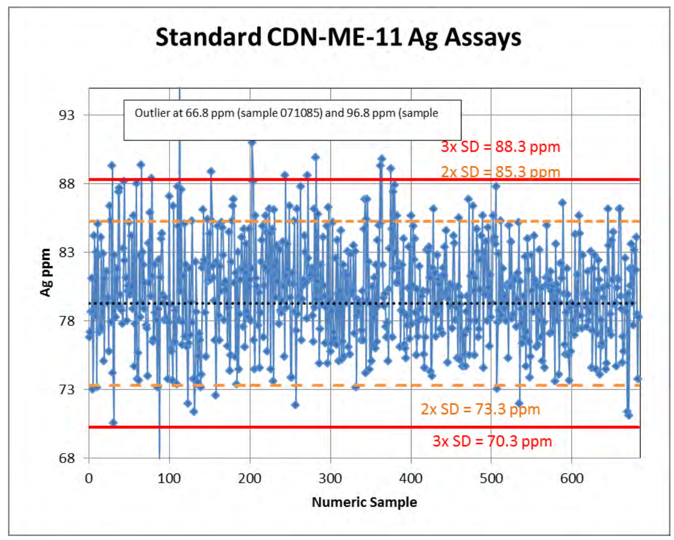

Standard CDN-GS-8A Au Assays
3x SD = 9.15 ppm
2x SD = 8.85 ppm
2x SD = 7.65 ppm
3x SD = 7.35 ppm
Au ppm
Numeric Sample

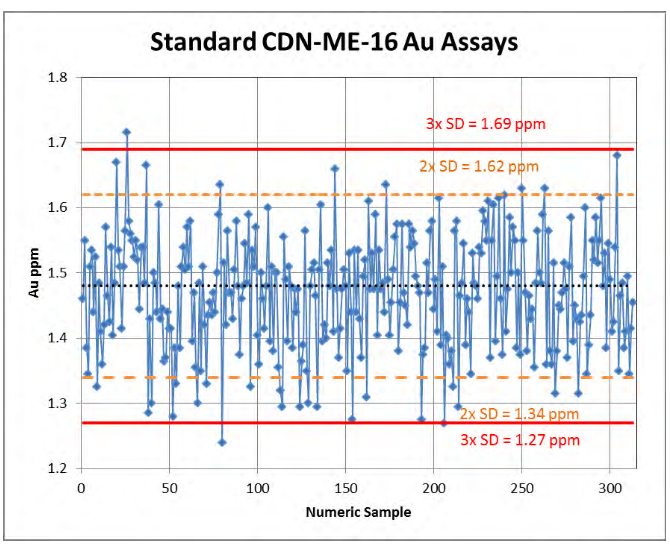

Standard CDN-GS-P4A Au Assays
3x SD = 0.49 ppm
2x SD = 0.47 ppm
2x SD = 0.41 ppm
3x SD = 0.39 ppm
Outlier at 0.65 ppm (sample
Au ppm
Numeric Sample

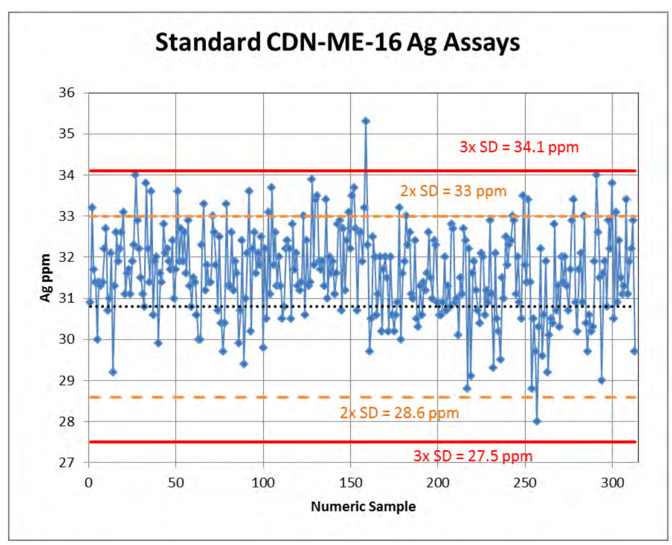

Standard CDN-GS-P7B Au Assays
Au ppm
Numeric Sample
3x SD = 0.82 ppm
2x SD = 0.78 ppm
2x SD = 0.64 ppm
3x SD = 0.61 ppm
Outlier at 0.43 ppm (sample 063925)

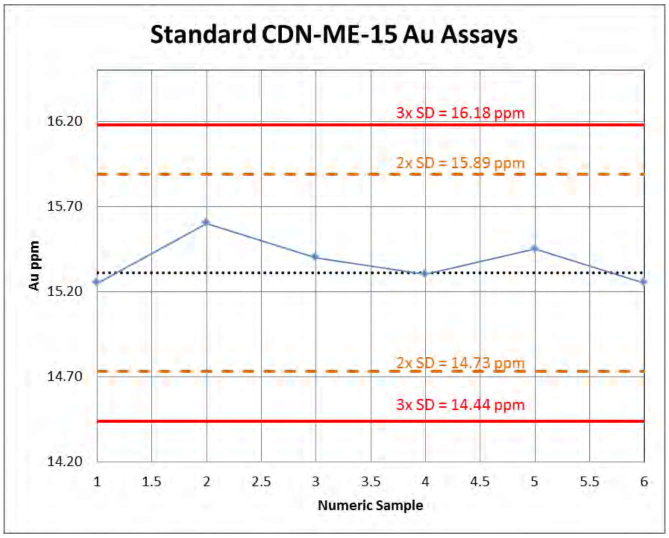

Standard CDN-GS-P7B Ag Assays
Ag ppm
Numeric Sample
3x SD = 15.8 ppm
2x SD = 15.0 ppm
2x SD = 11.8 ppm
3x SD = 11.0 ppm

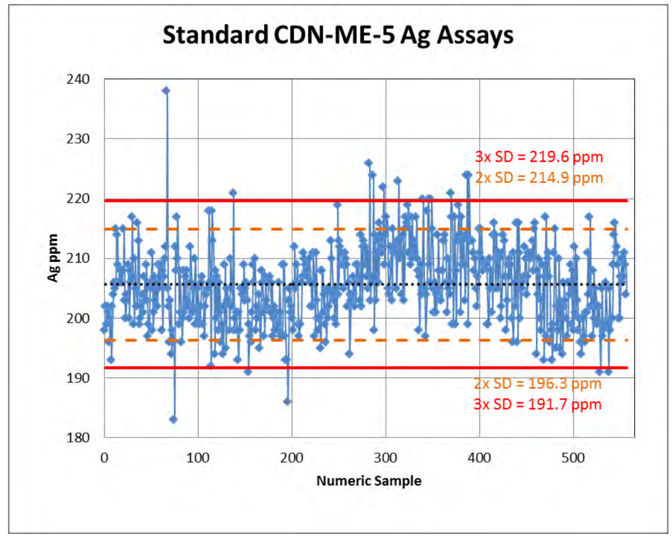

Standard CDN-ME-11 Au Assays
Au ppm
Numeric Sample
3x SD = 1.53 ppm
2x SD = 1.48 ppm
2x SD = 1.28 ppm
3x SD = 1.23 ppm


Standard CDN-ME-11 Ag Assays
Ag ppm
Numeric Sample
Outlier at 66.8 ppm (sample 071085) and 96.8 ppm (sample
3x SD = 88.3 ppm
2x SD = 85.3 ppm
2x SD = 73.3 ppm
3x SD = 70.3 ppm


Standard CDN-ME-16 Au Assays
Au ppm
Numeric Sample
3x SD = 1.69 ppm
2x SD = 1.62 ppm
2x SD = 1.34 ppm
3x SD = 1.27 ppm


Standard CDN-ME-16 Ag Assays
Ag ppm
Numeric Sample
3x SD = 34.1 ppm
2x SD = 33 ppm
2x SD = 28.6 ppm
3x SD = 27.5 ppm


Standard CDN-ME-15 Au Assays
3x SD = 16.18 ppm
2x SD = 15.89 ppm
2x SD = 14.73 ppm
3x SD = 14.44 ppm
Au ppm
Numeric Sample


Standard CDN-ME-5 Ag Assays
3x SD = 219.6 ppm
2x SD = 214.9 ppm
2x SD = 196.3 ppm
3x SD = 191.7 ppm
Ag ppm
Numeric Sample

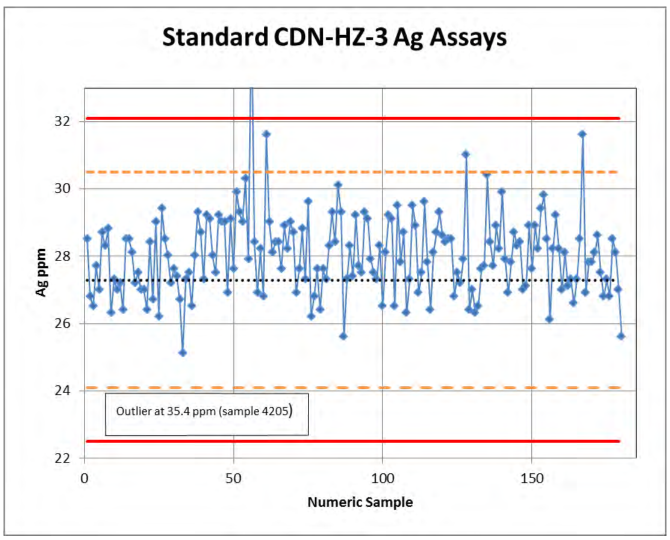
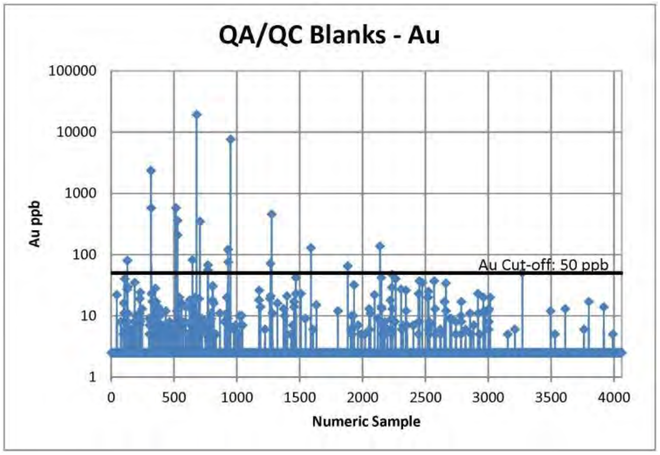
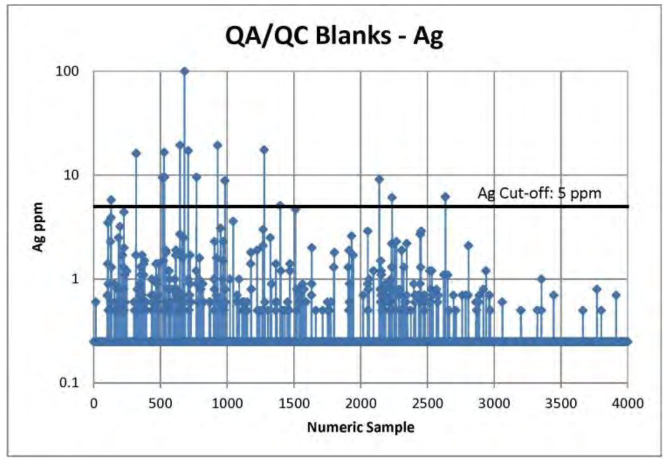

Standard CDN-HZ-3 Ag Assays
Outlier at 35.4 ppm (sample 4205)
Ag ppm
Numeric Sample

Figure 11-2. QA/QC Blanks





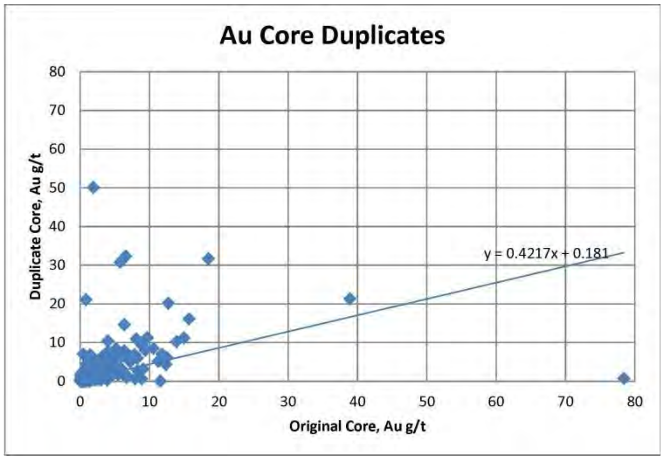
11.2.3 Duplicates

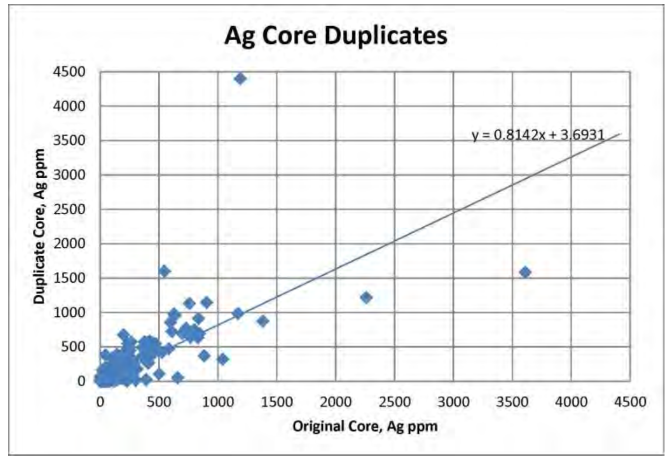
Quartered-core duplicate samples were collected to assess the overall repeatability of individual analytical values. One core duplicate for every 20 samples (5%) was inserted into the sample stream at the '15', '35', '55', '75', and '95' positions. A total of 4,057 quarter-core duplicates were inserted into the sample stream up to the end of drill hole TU-12-221 (the end or resource estimate cut-off).

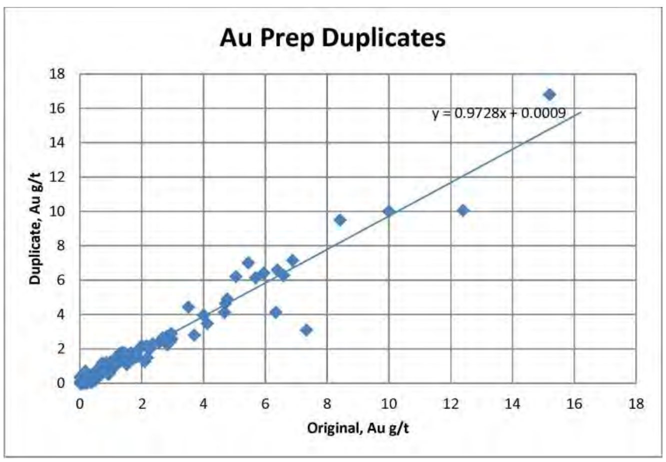
As part of their internal QA/QC program ALS completed routine re-analysis of prep (coarse reject) and pulp duplicate to monitor precision. ALS analyzed a total of 985 prep duplicates (including 5 repeat analyses) for gold, and 1,036 for silver (including 53 repeat analyses). A total of 2,152 pulp duplicates (including 1 repeat analysis) were analyzed for gold and 2,372 (including 9 repeat analyses) for silver.

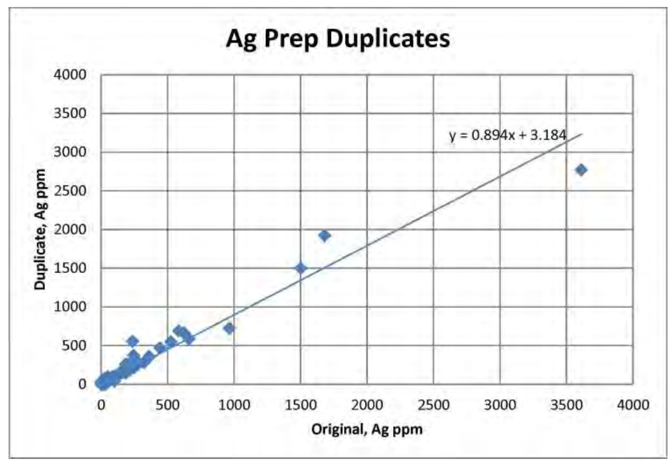
Charts showing original versus duplicate quarter-core, prep and pulp duplicate values for gold and silver show a significant and progressive increase in sample repeatability (Figure 11-3). Increased repeatability is expected as the level of duplicate sample homogenization increases from low (quarter-core) to moderate (prep) and high (pulp). Importantly the date indicates a high level of repeatability for both prep (coarse reject) and pulp duplicates. This is interpreted to indicate a low "nugget" effect with respect to Ixtaca gold and silver analyses. Excluding primary geologic heterogeneity (quarter-core), the data show a homogenous distribution of gold and silver values within Ixtaca drill core.

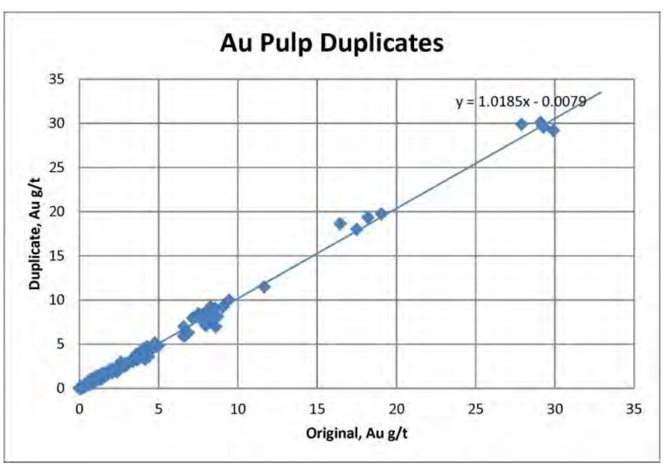
Figure 11-3. QA/QC Duplicates

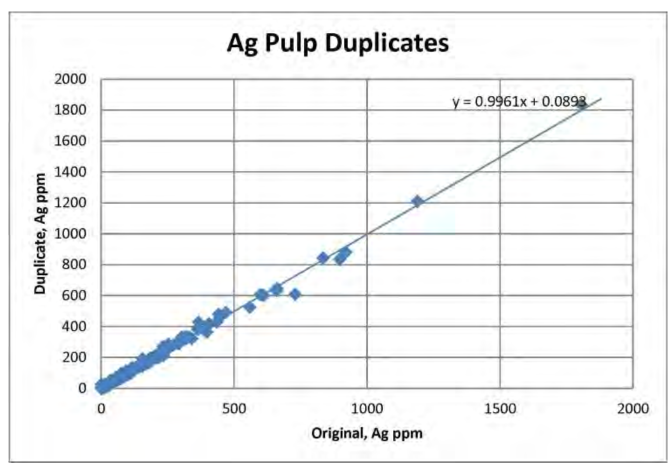












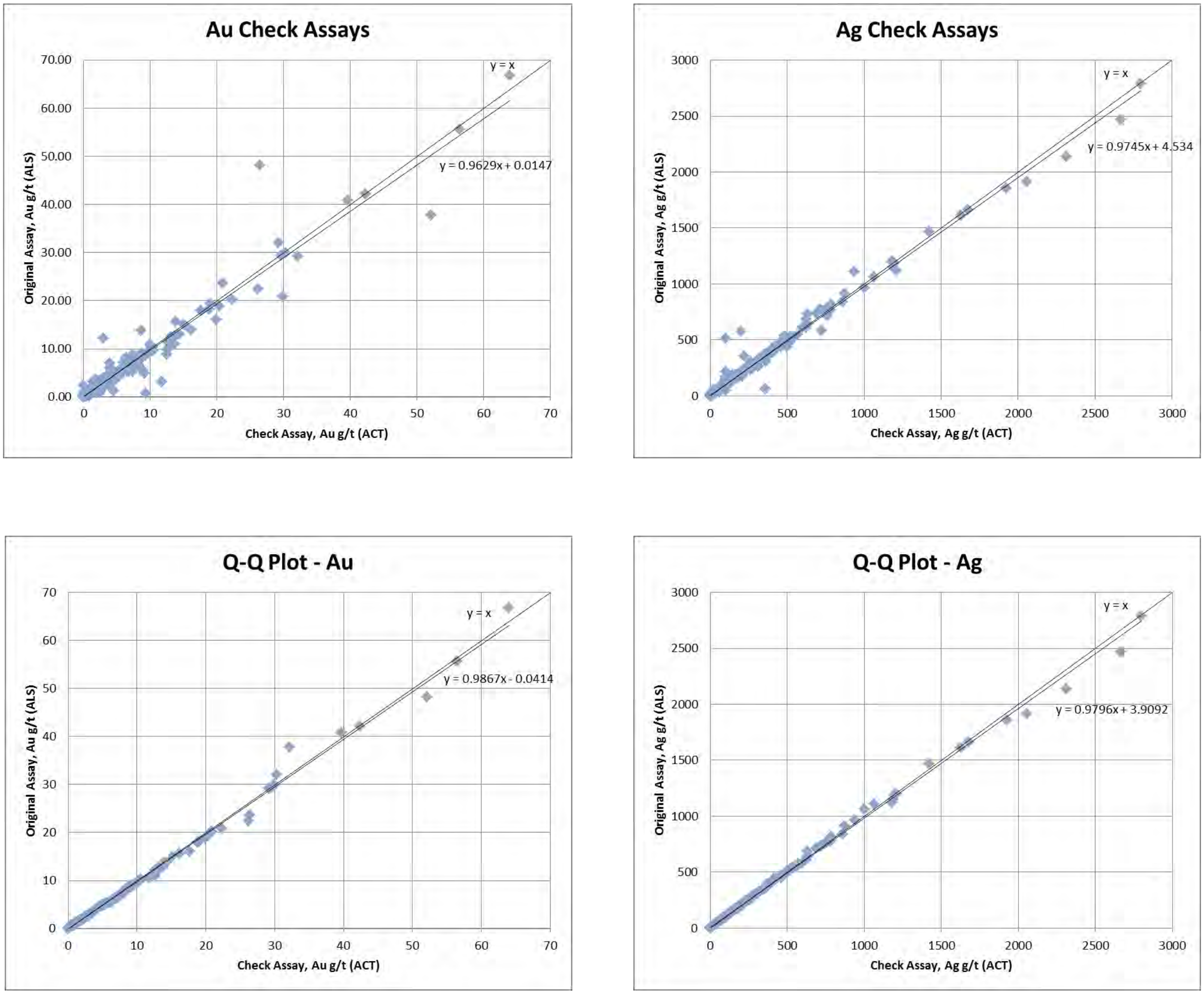
11.2.4 Check Assays

Almaden submitted coarse reject “check assay” samples to Actlabs Mexico S.A. de C.V. (Actlabs), Zacatecas, an ISO 9001:2008 certified sample preparation and chemical analysis laboratory. Actlabs is independent of Almaden and the authors.

A total of 815 samples were submitted to Actlabs as “check assays” to assess “between lab” analytical precision. The samples were selected from 19 drill holes ranging from TU-10-004 and TU-11-051, and included submission of a 39 blank, 41 duplicate and 40 standards samples. All samples were analyzed for gold by 50 g FA fusion, with 4-acid digestion, and AA (3 ppm Au upper limit) or gravimetric finish; and for Ag by 50 g, 4-acid digestion, with ICP-AES (100 ppm Ag upper limit) or gravimetric finish.

Charts showing original ALS versus Actlabs analyses for gold and silver show good “between laboratory” precision (Figure 11-4). A small number of outliers occur and are predicted due to geologic heterogeneity of the coarse reject sample material. Summary Q-Q plots, with ascending gold or silver values for each of the two laboratories plotted against each other, assess potential for “between laboratory” systematic bias across the measured range of gold and silver values (by comparing the population of gold and silver values for each laboratory). The Q-Q plots show good “between laboratory” correlation across the measured range of gold and silver values. This provides confirmation that ALS gold and silver analyses are both accurate and precise.

Figure 11-4. QA/QC Check Assays



11.3 Independent Audit of Almaden Drill Hole Database

Between August 23 and September 26, 2012 APEX personnel, under the direct supervision of Kristopher J. Raffle, P.Geo., conducted an independent audit of Almaden's drill hole database. The audit included systematic checks of database values for drill collar coordinate, downhole survey, and drill core, analytical standard, duplicate, and blank sample assays against the original field survey files and laboratory certificates. In addition, APEX conducted a review of Almaden QA/QC database, summary results of which is presented within section 11.2 above.

11.3.1 Collar Coordinate and Downhole Survey Databases

A total of 11 diamond drill hole collar locations were confirmed by Kristopher J. Raffle, P.Geo. following site visits to the Tuligtic Project on October 18, 2011 and September 23, 2012. The drill locations were compared with the Almaden database used in the mineral resource estimate and are deemed to be accurate. In addition, Almaden provided APEX with copies of all original down hole survey field records. Original field records for a total of 23 drill holes were checked against database values used for the mineral resource estimate. No discrepancies were found.

11.3.2 Drill Core Assay Database

A total of 69,177 drill core samples exist within the drill database up to the completion of drill hole TU-12-22 (225 drill holes in total). The database audit consisted of checking 6,826 database gold and silver values against the original ALS analytical certificates. The audit specifically focused on assays within reported mineralized intercepts. No discrepancies were identified between the original ALS analytical certificates and Almaden's drill hole database values.

12 Data Verification

The author conducted a reconnaissance of the Tuligtic Property from October 17 to October 20, 2011 to verify the reported exploration results. The author completed a traverse of the Ixtaca Zone, observed the progress of ongoing diamond drilling operations and recorded the location of select drill collars consistent with those reported by Almaden. Additionally, Almaden's complete drill core library was made available and the author reviewed mineralized intercepts in drill core from a series of holes across the Ixtaca Zone. The author personally collected quartered drill core samples as 'replicate' samples from select reported mineralized intercepts.

An additional visit to the Tuligtic Property was carried out by the author on September 23, 2012 to observe current operations, review additional mineralized intercepts in drill core, and collect quarter drill core samples from the recently completed drill holes. A comparison of the results of the authors 'replicate' sampling versus original Almaden reported values for gold and silver are presented in Table 12-1.

Table 12-1. Authors Independent Drill Core Sample Assays

Authors Sample	Almaden Sample	Drill Hole	From (m)	To (m)	Interval (m)	Authors Au (ppm)	Authors Ag (ppm)	Almaden Au (ppm)	Almaden Ag (ppm)
11KRP201	51662	TU-11-036	82.97	83.5	0.53	7.85	525	5.59	504
11KRP202	4596	TU-10-006	332.62	333.66	1.04	3.00	164	2.79	191
11KRP203	45073	TU-11-020	190.57	190.87	0.30	5.49	271	5.19	285
11KRP204	56217	TU-11-051	91.7	92.2	0.50	1.98	229	4.04	349
11KRP205	46586	TU-11-034	140.16	140.50	0.34	32.40	691	29.9	712
11KRP206	45347	TU-11-021	168.67	169.16	0.49	17.60	1130	15.55	1460
12KRP601	086459	TU-12-138	299.50	300.00	0.50	1.745	307	1.545	229
12KRP602	094696	TU-12-164	188.00	188.50	0.50	0.819	126	1.745	134
12KRP603	N298311	TU-12-123	228.60	229.10	0.50	3.45	86.6	4.39	92.5
12KRP604	N296249	TU-12-124	174.80	175.30	0.50	1.165	100	2.01	155
12KRP605	098391	TU-12-166	356.40	357.00	0.60	3.94	13.2	3.64	14.5
12KRP606	071443	TU-12-103	273.50	274.00	0.50	5.20	118	4.36	136

Based on the results of the traverses, drill core review, and ‘replicate’ sampling the author has no reason to doubt the reported exploration results. Slight variation in assays is expected due to variable distribution of ore minerals within a core section but the analytical data is considered to be representative of the drill samples and suitable for inclusion in the resource estimate.

13 Mineral Processing and Metallurgical Testing

13.1 Introduction

Preliminary metallurgical work has been undertaken at Almaden’s Ixtaca gold-silver deposit in Mexico in support of the Maiden mineral resource estimate and a potential preliminary economic assessment to be completed in 2013.

Metallurgical testwork on Ixtaca was undertaken between September 2012 and January 2013 at the Blue Coast Research Ltd. (Blue Coast), Parksville, British Columbia. Testwork commenced with the treatment of a range of composite samples, comprising half drill core intersections from each of the main geologic domains: limestone, limestone/dyke high grade (HG), shale (Northeast Extension Zone) and volcanic tuff material. Each composite was made up of five sub composites, each of which was taken from a separate drill hole, representing a different part of the respective geologic domain. Samples were shipped from Ixtaca in late August, 2012 and inspected at the Blue Coast laboratory in early September 2012 prior to processing.

The following work was undertaken on each of four domain samples, Dyke, Limestone, Shale and Tuff:

- Head Assays for each sample
- Bond Ball Work Index
- E- GRG (Gravity Recoverable Gold) test
- Cyanidation of the E-GRG tails
- Rougher flotation tests

A high grade blend of limestone and dyke material ('High Grade') was also tested. Results of the testwork were used to develop a preliminary process strategy and model expected metal recoveries for the purposes of establishing inferred and indicated resources within the deposit. Samples were generally comprised of coarse assay rejects although some samples were received in the form of half and quarter drill core. Metallurgical composites prepared from drill samples received were tested. Average results from characterization work on head samples are shown in Table 13-1.

Summary results of Blue Coast's Ixtaca metallurgical testing are presented below. The complete Ixtaca metallurgical test results are presented in Appendix 4.

Table 13-1. Metallurgical Composite Head Assay

Sample	Pb (%)	Zn (%)	Fe (%)	Au (g/t)	Ag (g/t)	C (%)	S (%)
Dyke	0.02	0.04	3.86	0.71	40.0	1.45	3.64
Limestone	0.01	0.02	0.98	0.58	41.0	7.69	0.77
Limestone/Dyke HG	0.04	0.06	2.28	2.24	127.0	5.03	2.42
Shale	0.23	0.43	3.20	0.98	45.0	3.68	3.38
Tuff (volcanic)	0.01	0.02	2.53	0.86	9.0	1.04	1.95

13.2 Metallurgical Test Results

A programme of Bond Ball mill work index (BWi), gravity gold recovery, cyanidation and rougher flotation testing was undertaken on the samples. Hardness testwork completed suggests that the Tuff domain is the softest at 10.5 kWh/t, followed by limestone at 13.2 kWh/t, dyke at 14.6 kWh/t and Shale the hardest at 18.6kWh/t. E-GRG testing was also undertaken on the 5 zone composites using a Knelson MD-3 gravity concentrator. E-GRG testing was conducted at stage grinds of 850µm, 180µm and 75µm respectively.
A summary of E-GRG results is shown in Table 13-2.

Table 13-2. E-ERG Test Result Summary

Sample	E-GRG Number (%)	Concentrate Grade Au (g/t)
Dyke	48.4	27.02
Limestone	58.7	39.32
Shale	54.9	50.04
Tuff (volcanic)	15.1	11.30

Three of the composites dyke, limestone and shale demonstrated a significant constituent of gravity recoverable gold. The fourth tuff sample indicated lower GRG content, however all samples were considered good candidates for this process route.

Tailings from the E-GRG testing on each sample were subjected to cyanidation to recover residual gold content at a p80 of 76 microns, with selected samples reground to a p80 of between 40-50 microns. A summary of results is presented in Table 13-3.

Table 13-3. Cyanidation Test Result Summary

Sample	Head		Recovery	
	Au (g/t)	Ag (g/t)	Au (Wt%)	Ag (Wt%)
Dyke	0.73	45.6	96.8	85.3
Limestone	0.76	49.25	88.7	78.3
Limestone/Dyke HG	2.01	123.5	94.9	87.0
Shale	0.93	46.4	95.9	81.8
Tuff (volcanic)	0.8	12.95	54.1	61.9

- The Limestone and Dyke domains exhibited the best overall response to cyanidation of the GRG tails. 60-62% of the non GRG gold was extracted into the PLS. Neither regrinding nor increased cyanide concentration had effect on gold extraction.
- The Shale gold extractions were low at 25% regardless of the cyanide concentration employed. Regrinding had no positive effect on gold extraction, however an increase in silver extraction to 56% was noted in regrinding.
- Tuff gold extractions were consistently low at between 37-43%. Regrinding and increased cyanide concentration had no positive effect on gold extraction. Silver extraction was 47% and was increased by ~11% to 59% through regrinding.
- Silver extractions averaged between 81-82% for the Limestone/Dyke composites.

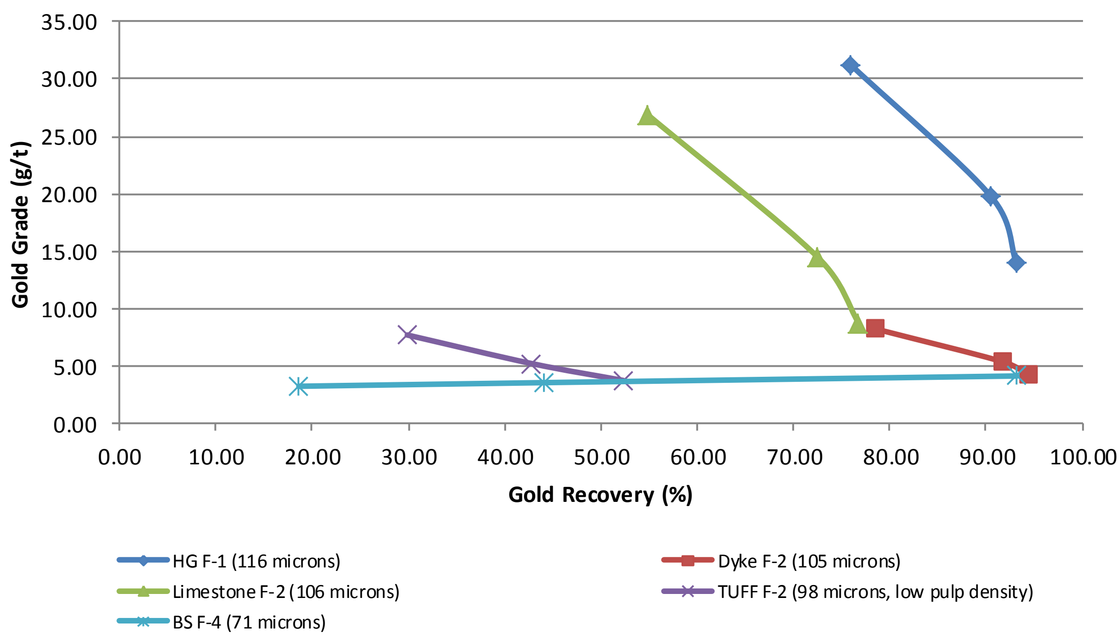
The initial flotation program consisted of bulk flotation tests on the four domain samples in addition to bulk flotation on the High Grade sample with cyanidation of the bulk concentrate. All bulk flotation tests were conducted at natural pH with 300g/t copper sulphate, between 150-200 g/t SIPX, 45g/t 3418A and F-140 frother as needed to produce a stable froth phase. Total rougher flotation residence time was fixed at 11 minutes and flotation was conducted over three rougher stages. The majority of tests were conducted at a nominal p80 of between 100-120 µm, however both coarser and finer grinds were tested on the High Grade composite and Limestone/Dyke domains respectively. A summary of results is shown in Table 13-4.

Table 13-4. Bulk Floatation Test Result Summary

Sample	Head		Recovery	
	Au (g/t)	Ag (g/t)	Au (Wt%)	Ag (Wt%)
Dyke	0.73	45.6	94.4	87.0
Limestone	0.76	49.25	85.7	79.9
Limestone/Dyke HG	2.01	123.5	92.0	88.8
Shale	0.93	46.4	93.2	83.5
Tuff (volcanic)	0.8	12.95	52.3	63.2

In flotation of the Shale, 93% of the gold and 83.5% of the silver was recovered into a bulk concentrate grading 4.1 g/t Au, 196 g/t Ag, 1.8% Zn and 1.1% Pb. The High Grade limestone/dyke sample showed excellent amenability to bulk rougher flotation. High grade gold and silver bulk rougher concentrates were obtained in preliminary tests, although overall gold and silver recovery was limited to 67% and 76% respectively. Test HG F-2 produced a bulk rougher concentrate grading 21 g/t Au and 1220 g/t Ag, with gold and silver recoveries of 92% and 90% respectively. Flotation of the dyke material demonstrated lower grade concentrates, but with excellent recoveries. Dyke flotation test F-1 produced a bulk rougher concentrate grading 4.6g/t Au and 307g/t Ag at gold and silver recoveries of 89% and 88% respectively. The limestone material showed fair flotation response were lower at 77% and 73% with grades of 9g/t Au and 660g/t Ag respectively. Bulk rougher flotation of Tuff F-2 produced a concentrate grading 4g/t Au and 78g/t Ag at gold and silver recoveries of 52% and 63% respectively. Figure 13-1 shows summary bulk flotation results for all four domains plus the High Grade sample.

Figure 13-1. Summary of the Domain Bulk Flotation Results



Clear variability to treatment by flotation exists between the various domains. The High Grade MET sample yielded the highest grade concentrate at >90% gold recovery. Both the Dyke and Shale composites produce gold recoveries >90%, albeit at lower bulk concentrate grades. The TUFF appears to behave differently to all other domains (as was observed in the gravity and cyanidation testwork) and yielded a much lower grade concentrate and lower recovery to said concentrate. Further mineralogical work on the Tuff material, including pre-treatment and depression of clays, is planned to optimize flotation recovery in this domain.

13.3 Evaluation of Process Routes and Projected Zone Recoveries

From the testwork possible process routes for potentially economic material from the Ixtaca deposit include crushing and grinding with cyanidation only, cyanidation with gravity recovery of the gold, flotation only and flotation with gravity recovery of the gold. Preliminary metallurgy indicates good results for gravity gold recovery in all samples excepting the tuff. Cyanidation of the GRG tails also showed economic potential, with the exception of the tuff which exhibited lower recoveries than the other zone samples. Treatment by bulk flotation shows excellent potential for all samples barring the shale. Stage Pb Zn flotation of this material showed improved response however this route is considered incompatible with other process options at this stage. Treatment of the Ixtaca resource material by a combination of grinding, gravity recovery of the gold, and bulk flotation, followed by intensive cyanidation of the combined concentrate was selected as the base case for treatment of the Ixtaca material. Gravity recovery results were factored in to the bulk flotation recovery numbers, and combined with typical recoveries for intensive cyanidation to obtain expected metal recoveries by this route to develop metallurgical recovery parameters for the establishment of a resource. Modelled zones recoveries are presented in Table 13-5.

Table 13-5. Modelled Recovery Parameters for the Ixtaca Deposit

Sample	Head		Flotation only		Gravity	Combined Float + GRG		GRG+Float+ICL	
	Au (g/t)	Ag (g/t)	Au (Wt%)	Ag (Wt%)	Au (Wt%)	Au (Wt%)	Ag (Wt%)	Au (Wt%)	Ag (Wt%)
Dyke	0.73	45.6	94.4	87.0	48.4	98.8	87.0	96.8	85.3
Limestone	0.76	49.3	85.7	79.9	58.7	90.5	79.9	88.7	78.3
Limestone/Dyke HG	2.01	123.5	92.0	88.8	58.7	96.8	88.8	94.9	87.0
Shale	0.93	46.4	93.2	83.5	54.9	97.9	83.5	95.9	81.8
Tuff (volcanic)	0.8	13.0	52.3	63.2	15.1	55.2	63.2	54.1	61.9

13.4 Conclusions

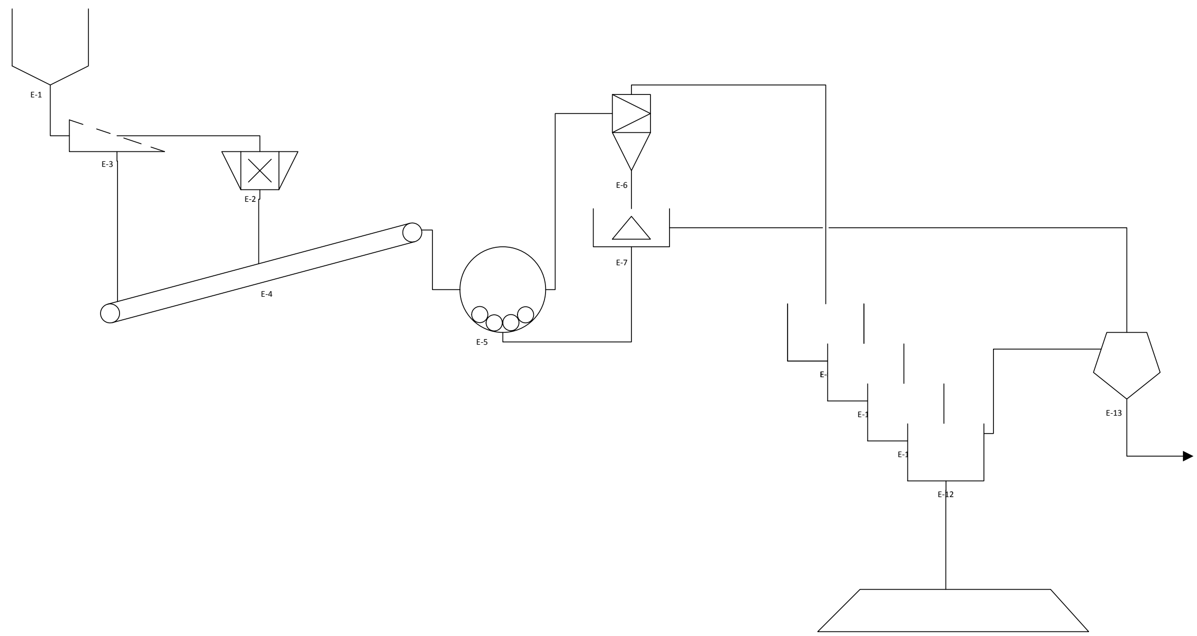
Metallurgical testwork on Ixtaca commenced with the treatment of a range of composite samples, comprising half drill core intersections from each of the main geologic domains: limestone, limestone/dyke high grade (HG), shale (Northeast Extension Zone) and volcanic tuff material. Each composite was made up of five sub composites, each of which was taken from a separate drill hole, representing a different part of the respective geologic domain. Samples were shipped from Ixtaca in late August, 2012 and inspected at the Blue Coast Laboratory in early September 2012 prior to processing. Metallurgical testwork comprising gravity-recoverable gold (GRG) testwork, leaching of the gravity tailings, as well as stage- and bulk flotation tests on each of the 4 zone samples was conducted between October and December 2012. Initial excellent results for GRG testing as well as flotation on the HG samples indicated good potential for these process routes. Combinations of gravity, leaching and flotation indicate excellent potential for gold and silver recovery from the resource. Summary results for the zones tested are shown in Table 13-6.

Table 13-6. Metallurgical Results for Ixtaca Domain Samples

Zone	Gravity Only Recovery		Flotation Only Recovery	
	Au (Wt%)	Ag (Wt%)	Au (Wt%)	Ag (Wt%)
Dyke	48.4	N/A	94.4	87.0
Limestone	58.7	N/A	85.7	79.9
Limestone/Dyke HG	58.7	N/A	92.0	88.8
Shale	54.9	N/A	93.2	83.5
Tuff (volcanic)	15.1	N/A	52.3	63.2

Initial process results indicate that treatment of Ixtaca material by a combination of grinding to a p_{80} of 100-150μm plus gravity recovery on the cyclone underflow, with recovery of gold and silver by means of bulk flotation, followed by intensive leaching of the combined gravity and flotation concentrates is a viable process route for the Ixtaca resource. A block flow sheet for this treatment route is shown in Figure 13-2.

Figure 13-2. Proposed Treatment Route, Ixtaca Project



A summary of metallurgical parameters for the main zones tested for this process route is presented in Table 13-7.

Table 13-7. Overall Projected Gravity + Flotation + Intensive Leach Recoveries

Zone	Overall Recovery	
	Au (Wt%)	Ag (Wt%)
Dyke	96.8	85.3
Limestone	88.7	78.3
Limestone/Dyke HG	94.9	87.0
Shale	95.9	81.8
Tuff (volcanic)	54.1	61.9

Overall Au and Ag recoveries from a combination of flotation, gravity concentration and intensive leaching average 88% for Au and 82% for Ag across all geologic domains. In basement rocks average recoveries were 93% for Au and 82% for Ag (ranging from 88.6 to 96.8% for Au, and 81.8 to 87.0% for Ag); in volcanics 54.1% Au, and 61.9% Ag. Gravity recoveries of Au in basement rocks averaged 55% (ranging from 48% to 59%), and 15% for volcanic rocks. Further metallurgical work, including mineralogical work, process optimization of flotation and leaching responses, and investigation of alternate reagent combinations on existing and fresh domain samples is planned for 2013.

14 Mineral Resource Estimate

At the request of Morgan Poliquin, President of Almaden, Giroux Consultants Ltd. was retained to produce a resource estimate on the Main Ixtaca, Ixtaca North and Northeast Extension zones (the "Ixtaca Deposit"), Tuligtic Project located in Puebla State, Mexico. There have been 225 diamond drill holes completed on the Tuligtic Project by Almaden from 2010 to 2012. The effective date for this Estimate is November 14, 2012.

G.H. Giroux is the qualified person responsible for the resource estimate. Mr. Giroux is a qualified person by virtue of education, experience and membership in a professional association. He is independent of the company applying all of the tests in section 1.5 of National Instrument 43-101. Mr. Giroux has not visited the property.

14.1 Data Analysis

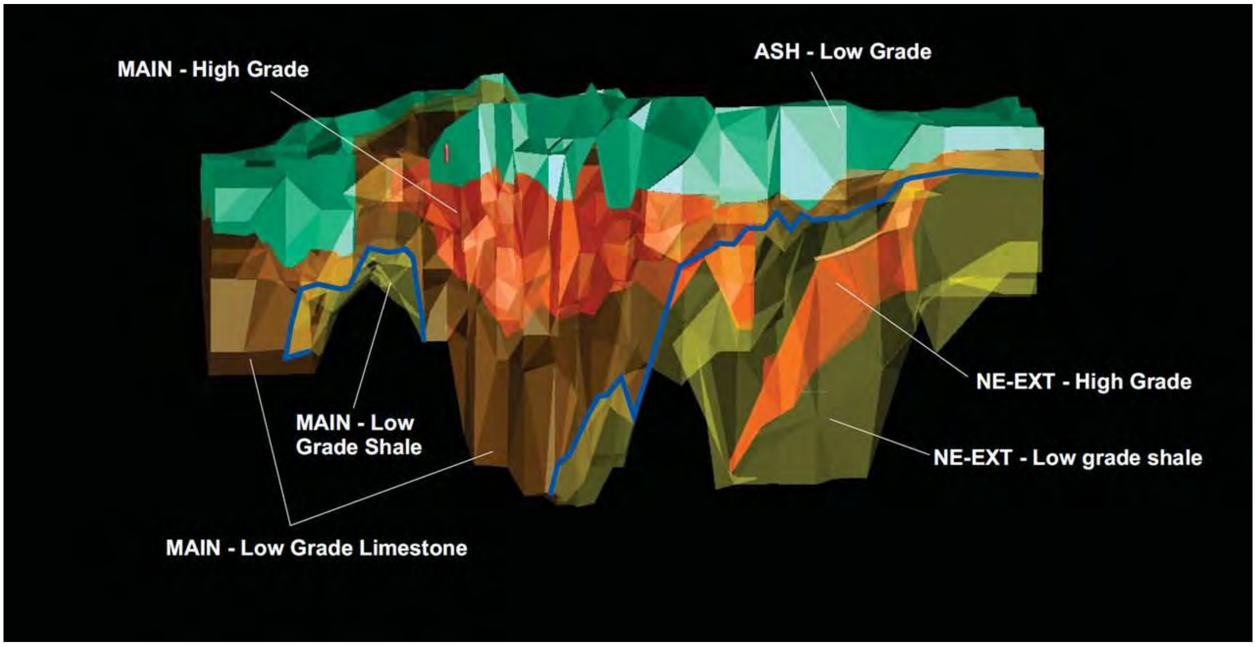
Almaden supplied a total of 225 drill holes with 2,430 down hole surveys and 69,175 assays for gold and silver. Of these drill holes, 222 totalling 80,366 m outline the Ixtaca Main zone and NE Extension which are estimated in this resource. All drill holes are included in Appendix 1 with the holes used in this resource highlighted. A total of 276 gaps were found in the from – to record and in these gaps a value of 0.001 g/t Au and 0.01 g/t Ag was inserted. Two gold and silver assays reported as blank were set to 0.001 g/t and 0.01 g/t respectively. In addition 235 intervals at the start or end of holes were not sampled due to broken rock which was cased or ends of holes that were not considered mineralized. In these 235 cases values of 0.001 g/t Au and 0.01 g/t Ag were inserted.

Almaden also supplied a series of geologic solids for the Ixtaca zone, which outlined the following mineralized domains:

Code	**Description**
BASH	A barren clay altered tuff overlying the mineralized carbonate rocks
LGASH	A mineralized horizon within the clay altered tuff near the contact with the mineralized carbonate rocks
MHG	The Main Ixtaca High Grade Mineralized Zone comprised of varying density of carbonate-quartz epithermal veining
NEHG	A North east trending extension of High Grade carbonate-quartz epithermal veining
LGLS	A lower grade envelope within the Main Zone Limestone unit
LGSH	A lower grade envelope within the Main Zone Shale unit
NELGSH	A lower grade envelope in the North East Extension Shale Unit

From this list, 3 dimensional solids, for each domain, were created in Gemcom software to constrain the estimation. A topographic surface and an overburden surface constrained the top of the solids. Figure 14-1 shows the various zones.

Figure 14-1. Isometric View Looking N Showing the Geologic Solids



Drill holes were then compared to the solids and each assay was tagged with a code. The statistics for gold and silver are tabulated below sorted by mineralized zone. Assays outside the mineralized solids were tagged as waste.

Table 14-1. Assay Statistics for Gold and Silver Sorted by Mineralized Zone

Domain	Variable	Number of Assays	Mean Grade	Standard Deviation	Minimum Value	Maximum Value	Coefficient Of Variation
BASH	Au (g/t)	1,684	0.016	0.054	0.001	1.08	3.33
	Ag (g/t)	1,684	0.33	0.49	0.01	12.90	1.48
LGASH	Au (g/t)	4,966	0.403	1.562	0.001	75.20	3.87
	Ag (g/t)	4,966	7.95	64.80	0.01	4340.00	8.15
MHG	Au (g/t)	8,086	1.309	5.659	0.001	336.00	4.32
	Ag (g/t)	8,086	86.31	220.41	0.01	6390.00	2.55
LGLS	Au (g/t)	30,530	0.253	1.549	0.001	87.60	6.12
	Ag (g/t)	30,530	16.90	94.41	0.01	4270.00	5.59
LGSH	Au (g/t)	2,118	0.133	0.922	0.001	38.00	6.96
	Ag (g/t)	2,118	9.56	61.30	0.01	2370.00	6.41
NEHG	Au (g/t)	2,048	0.953	3.442	0.002	94.00	3.61
	Ag (g/t)	2,048	49.91	113.85	0.25	2720.00	2.28
NELGSH	Au (g/t)	10,502	0.117	0.740	0.001	57.30	6.31
	Ag (g/t)	10,502	10.01	55.16	0.01	2620.00	5.51
WASTE	Au (g/t)	9,759	0.014	0.084	0.001	3.86	6.08
	Ag (g/t)	9,759	1.18	9.91	0.01	646.00	8.38

To determine if each of these geologic domains were unique the lognormal cumulative frequency plots for gold and silver were examined.

Figure 14-2. Lognormal Cumulative Frequency Plot for Au as a Function of Domain

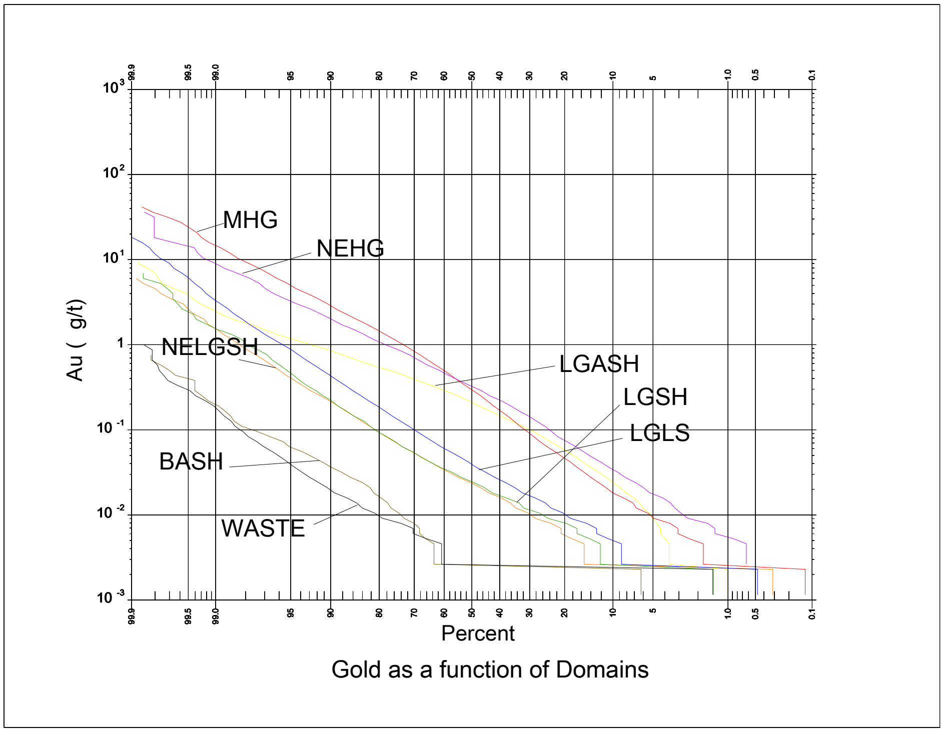
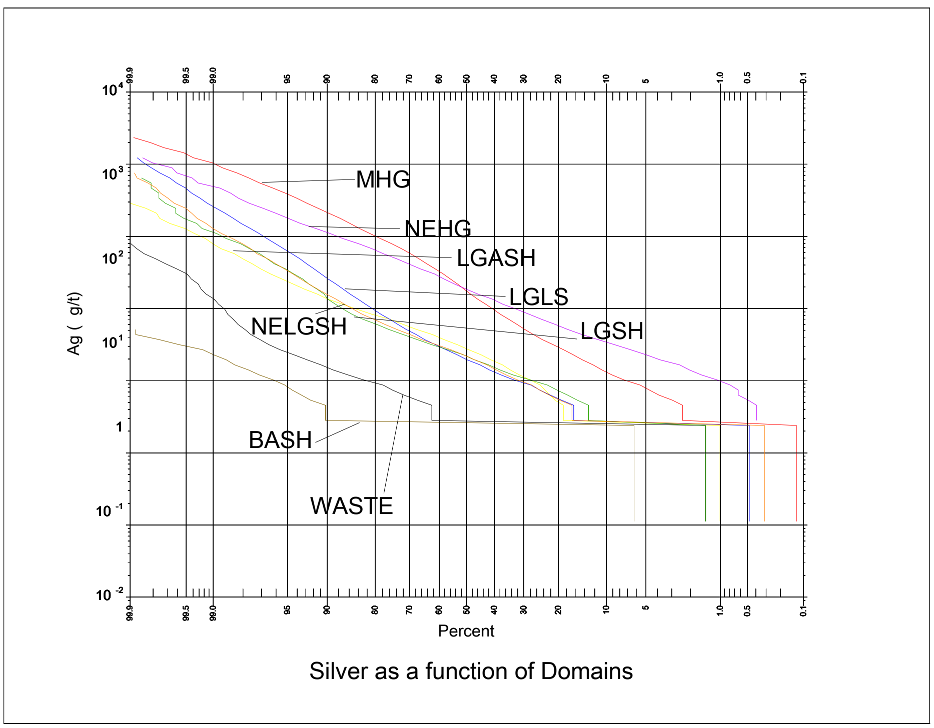


Figure 14-3. Lognormal Cumulative Frequency Plot for Ag as a Function of Domain



For both Au and Ag there is a significant difference between the barren Ash unit and the low grade Ash unit so this subdivision should be maintained. The two high grade units are significantly different from the low grade units so again these subdivisions should be honoured. While the low grade units in the Ash and Limestone are reasonably similar they do occur in different geographic areas so they should be modelled separately. The two shale units are very similar but occur on different ends of the deposit.

The grade distributions for gold and silver, within each mineralized domain, were examined to determine if capping was required and if so at what levels. Both elements showed skewed distributions in all domains and were converted to lognormal cumulative frequency plots. The procedure used is explained in a paper by Dr. A.J. Sinclair titled Applications of probability graphs in mineral exploration (Sinclair, 1976). In short the cumulative distribution of a single normal distribution will plot as a straight line on probability paper while a single lognormal distribution will plot as a straight line on lognormal probability paper. Overlapping populations will plot as curves separated by inflection points. Sinclair proposed a method of separating out these overlapping populations using a technique called partitioning. In 1993 a computer program called P-RES was made available to partition probability plots interactively on a computer (Bentzen and Sinclair, 1993). Screen dumps from this program are shown for each variable within the MHG Domain as Figures 14-4 and 14-5. In each Figure the actual data distribution is shown with black dots. The inflection points that separate the populations are shown as vertical lines and each population is shown by the straight lines of open circles. The interpretation is tested by recombining the data in the proportions selected and the test is shown as triangles compared to the original distribution.

Each variable is examined in the following section with the populations broken out and thresholds selected for capping if required.

For gold in the Ixtaca Main high grade zone (MHG), 6 overlapping lognormal populations were identified. These are shown in Table 14-2 and Figure 14-2. Population 1 with a mean grade of 283.3 g/t and representing 0.03 % of the total assays is clearly made up of erratic outliers and should be capped. An effective cap level would be two standard deviations above the mean of population 2 and as a result 5 gold assays in the MHG zone were capped at 56 g/t Au.

Table 14-2. Gold Populations within the MHG Zone

Population	Mean Au (g/t)	Percentage of Total Data	Number of Assays
1	283.3	0.03 %	2
2	21.60	0.52 %	42
3	8.99	2.71 %	219
4	1.78	24.23 %	1,960
5	0.29	42.74 %	3,456
6	0.03	29.76 %	2,407

Figure 14-4. Lognormal Cumulative Frequency Plot for Au in MHG

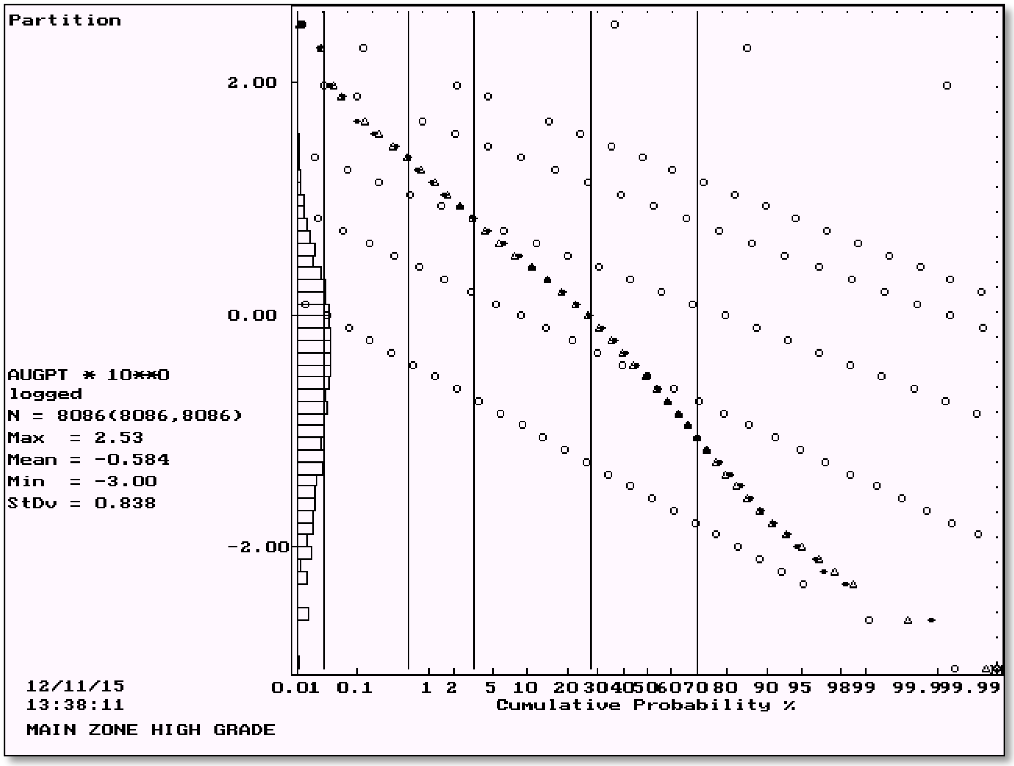
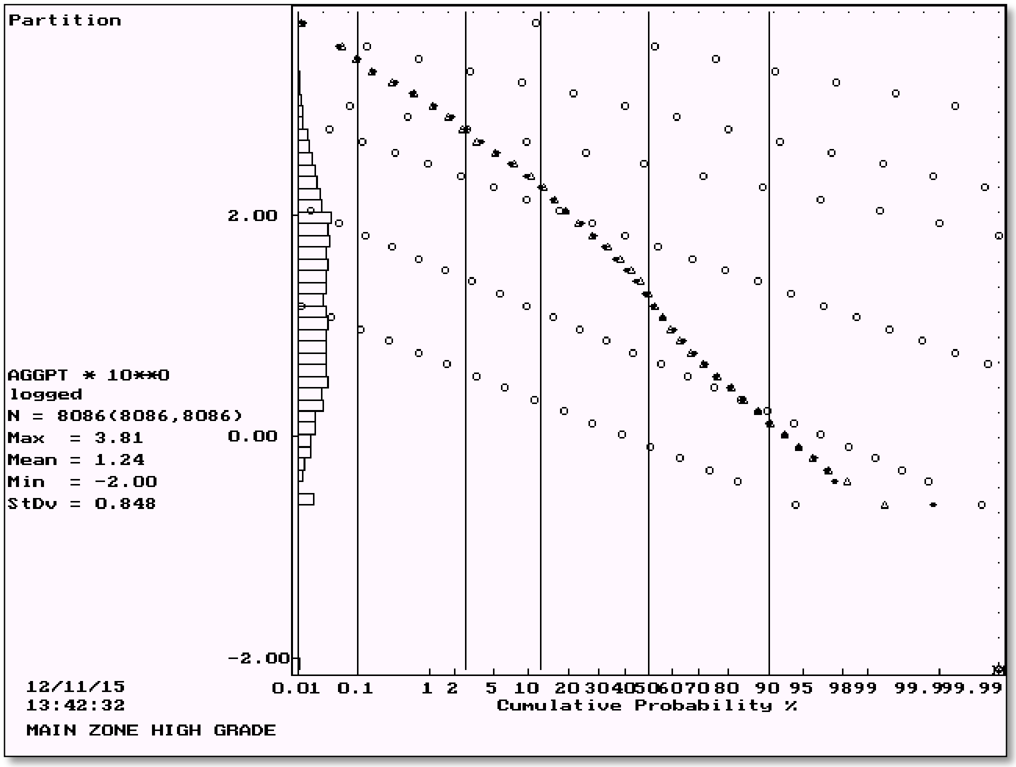


Figure 14-5. Lognormal Cumulative Frequency Plot for Ag in MHG



A similar procedure was used on gold and silver within all zones and the capping strategy is tabulated below.

Table 14-3. Cap Levels for Gold and Silver

Domain	Variable	Cap Level (g/t)	Number of Assays capped
MHG	Au	56.0 g/t	5
	Ag	2100.0 g/t	13
BASH	Au	0.7 g/t	3
	Ag	3.7 g/t	4
LGASH	Au	20.0 g/t	2
	Ag	430.0 g/t	2
LGLS	Au	41.0 g/t	7
	Ag	2411 g/t	8
LGSH	Au	6.0 g/t	3
	Ag	300.0 g/t	5
NEHG	Au	20.0 g/t	4
	Ag	668.0 g/t	9
NELGSH	Au	7.3 g/t	10
	Ag	1312.0 g/t	5
WASTE	Au	0.5 g/t	12
	Ag	60.0 g/t	5

The effects of capping are shown in the following Table 14-4 with minor reductions in mean grade but significant reductions in standard deviations and coefficients of variation.

Table 14-4. Capped Assay Statistics for Gold and Silver Sorted by Domain

Domain	Variable	Number of Assays	Mean Grade	Standard Deviation	Minimum Value	Maximum Value	Coefficient Of Variation
BASH	Au (g/t)	1,684	0.016	0.049	0.001	0.70	3.05
	Ag (g/t)	1,684	0.32	0.37	0.01	3.70	1.15
LGASH	Au (g/t)	4,966	0.383	0.761	0.001	20.00	1.99
	Ag (g/t)	4,966	7.10	19.76	0.01	430.00	2.78
MHG	Au (g/t)	8,086	1.241	3.392	0.001	56.00	2.73
	Ag (g/t)	8,086	84.71	196.46	0.01	2100.00	2.32
LGLS	Au (g/t)	30,530	0.247	1.299	0.001	41.00	5.25
	Ag (g/t)	30,530	16.66	86.15	0.01	2411.00	5.17
LGSH	Au (g/t)	2,118	0.115	0.400	0.001	6.00	2.47
	Ag (g/t)	2,118	7.98	24.42	0.01	300.00	3.06
NEHG	Au (g/t)	2,048	0.868	1.799	0.002	20.00	2.07
	Ag (g/t)	2,048	47.44	83.70	0.25	668.00	1.76
NELGSH	Au (g/t)	10,502	0.110	0.394	0.001	7.30	3.61
	Ag (g/t)	10,502	9.82	48.56	0.01	1312.00	4.95
WASTE	Au (g/t)	9,662	0.011	0.032	0.001	0.50	3.00
	Ag (g/t)	9,662	0.90	3.17	0.01	60.00	3.51

14.2 Composites

Of the 69,688 assays, within the 8 domains, 69,058 or 99.1% were less than or equal to 3 m in length. As a result a 3 m composite length was selected. Down hole composites 3 m in length were formed to honour the domain boundaries. Composite intervals at the domain boundaries that were less than 1.5 m in length were combined with adjoining samples while those greater than or equal to 1.5 m were left alone. As a result the composites formed a uniform support of 3±1.5 m. Material outside the 7 mineralized solids was considered waste.

Table 14-5. 3m Composite Statistics for Gold and Silver Sorted by Mineralized Zone

Domain	Variable	Number of Assays	Mean Grade	Standard Deviation	Minimum Value	Maximum Value	Coefficient Of Variation
BASH	Au (g/t)	2,305	0.007	0.029	0.001	0.574	4.12
	Ag (g/t)	2,305	0.17	0.36	0.01	9.10	2.15
LGASH	Au (g/t)	2,682	0.280	0.465	0.001	8.460	1.66
	Ag (g/t)	2,682	5.18	10.66	0.01	128.61	2.06
MHG	Au (g/t)	1,922	0.892	1.396	0.001	17.98	1.56
	Ag (g/t)	1,922	61.21	85.20	0.01	727.13	1.39
LGLS	Au (g/t)	10,366	0.154	0.448	0.001	10.08	2.91
	Ag (g/t)	10,366	9.70	30.77	0.01	886.89	3.17
LGSH	Au (g/t)	778	0.079	0.187	0.001	2.30	2.36
	Ag (g/t)	778	5.45	11.03	0.01	144.22	2.02

NEHG	Au (g/t)	419	0.771	1.101	0.004	9.10	1.43
	Ag (g/t)	419	42.00	50.01	0.37	368.08	1.19
NELGSH	Au (g/t)	3,588	0.072	0.223	0.001	5.85	3.10
	Ag (g/t)	3,588	6.42	23.37	0.01	940.25	3.64
WASTE	Au (g/t)	4,700	0.007	0.015	0.001	0.45	2.27
	Ag (g/t)	4,700	0.50	1.08	0.01	52.63	2.18

To determine if hard or soft boundaries would be required between some of the geologic domains a series of Contact Plots were produced. These plots examine the contact area between two geologic domains and compare the average grade for the variable being examined as a function of distance away from this contact. Where large differences appear at the contact a Hard Boundary should be used with samples from one side of the contact not allowed to influence blocks on the other side. If, on the other hand, the differences are minimal or gradational then a Soft Boundary could be set up with samples allowed to influence block grades from both sides of a contact.

The results are shown in Appendix 2. The grades for Au and Ag across the contacts are sufficiently different for the BASH-LGASH, LGLS-LGSH and LGLS-LGASH boundaries to make these all Hard Boundaries. In the case of the LGLS-NELGSH contact the grades are sufficiently similar, for both Au and Ag across the contact, to make this a Soft Boundary.

14.3 Variography

Pairwise relative semivariograms were produced for gold and silver within the each of the geologic domains. In all cases except for waste, a geometric anisotropy was observed and nested spherical models were fit to the three principal directions. Due to the high correlation between Au and Ag in each of the domains, gold and silver showed similar directions of anisotropy.

Table 14-6. Pearson Correlation Coefficients for Au – Ag Geologic Domains

	BASH	LGASH	MHG	LGLS	LGSH	NEHG	NELGSH	WASTE
Au:Ag Correlation Coef.	0.7702	0.8542	0.8851	0.8174	0.7563	0.5889	0.7929	0.7347

Within the barren Ash zone both gold and silver were modelled with anisotropic models with longest range along azimuth 45^{o} dip 0^{o}. Within the mineralized low grade ash unit near the contact with the mineralized units the anisotropy for both Au and Ag shifted to longest direction along azimuth 90^{o}. This probably reflects the influence of mineralization from the lower mineralized units seeping up into the ash flows.

Within the Main High Grade zone the longest direction of continuity for both Au and Ag was along azimuth 60^{o} with the second longest range dipping -35^{o} along azimuth 150^{o}. A similar direction of anisotropy was observed within both the low grade limestone unit that surrounds the Main High Grade Zones and within the low grade Shale unit.

For the north east extension mineralization, the longest ranges in both the high grade core and low grade shale that surrounds it, were found along azimuth 347°.

Within waste, both gold and silver showed isotropic nested structures.

The semivariogram parameters are tabulated below and the models for gold are shown in Appendix 3.

Table 14-7. Semivariogram Parameters for Gold and Silver

Domain	Variable	Az/Dip	C_0	C_1	C_2	Short Range (m)	Long Range (m)
MHG	Au	60° / 0°	0.30	0.35	0.28	6.0	96.0
		330° / -55°	0.30	0.35	0.28	12.0	28.0
		150° / -35°	0.30	0.35	0.28	8.0	70.0
	Ag	60° / 0°	0.40	0.20	0.33	12.0	80.0
		330° / -55°	0.40	0.20	0.33	8.0	28.0
		150° / -35°	0.40	0.20	0.33	8.0	60.0
BASH	Au	45° / 0°	0.01	0.20	0.20	40.0	80.0
		315° / 0°	0.01	0.20	0.20	15.0	30.0
		0° / -90°	0.01	0.20	0.20	20.0	40.0
	Ag	45° / 0°	0.05	0.25	0.50	30.0	80.0
		315° / 0°	0.05	0.25	0.50	20.0	60.0
		0° / -90°	0.05	0.25	0.50	30.0	40.0
LGASH	Au	90° / 0°	0.25	0.35	0.25	40.0	100.0
		0° / 0°	0.25	0.35	0.25	36.0	60.0
		0° / -90°	0.25	0.35	0.25	20.0	30.0
	Ag	90° / 0°	0.25	0.25	0.35	40.0	80.0
		0° / 0°	0.25	0.25	0.35	30.0	40.0
		0° / -90°	0.25	0.25	0.35	20.0	30.0
LGLS	Au	60° / 0°	0.36	0.30	0.27	12.0	120.0
		330° / -55°	0.36	0.30	0.27	20.0	78.0
		150° / -35°	0.36	0.30	0.27	18.0	120.0
	Ag	60° / 0°	0.35	0.27	0.30	15.0	100.0
		330° / -55°	0.35	0.27	0.30	12.0	60.0
		150° / -35°	0.35	0.27	0.30	12.0	84.0
LGSH	Au	60° / 0°	0.26	0.25	0.32	30.0	64.0
		330° / -55°	0.26	0.25	0.32	12.0	36.0
		150° / -35°	0.26	0.25	0.32	20.0	60.0
	Ag	60° / 0°	0.20	0.20	0.30	10.0	46.0
		330° / -55°	0.20	0.20	0.30	5.0	20.0
		150° / -35°	0.20	0.20	0.30	12.0	60.0
NELGSH	Au	347° / 0°	0.20	0.25	0.35	32.0	134.0
		257° / -55°	0.20	0.25	0.35	25.0	210.0
		77° / -35°	0.20	0.25	0.35	18.0	70.0
	Ag	347° / 0°	0.10	0.41	0.26	20.0	130.0
		257° / -55°	0.10	0.41	0.26	20.0	200.0
		77° / -35°	0.10	0.41	0.26	18.0	60.0
NEHG	Au	347° / 0°	0.40	0.15	0.25	12.0	40.0
		257° / -55°	0.40	0.15	0.25	20.0	40.0
		77° / -35°	0.40	0.15	0.25	18.0	30.0
	Ag	347° / 0°	0.30	0.25	0.23	10.0	40.0

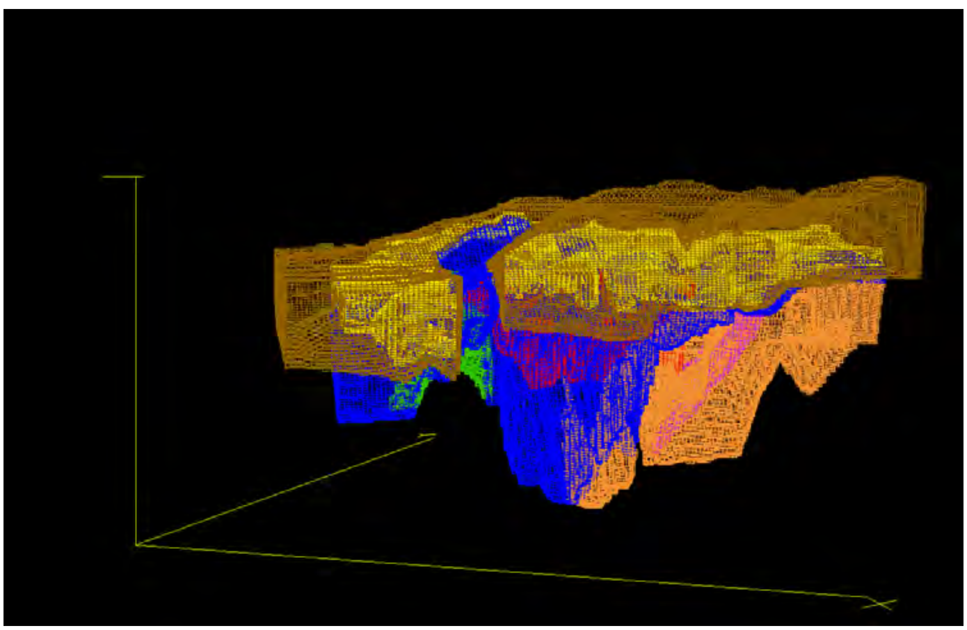
		257° / -55°	0.30	0.25	0.23	20.0	50.0
		77° / -35°	0.30	0.25	0.23	15.0	30.0
WASTE	Au	Omni Directional	0.10	0.10	0.23	10.0	110.0
	Ag	Omni Directional	0.10	0.06	0.25	10.0	100.0

14.4 Block Model

A rotated block model with blocks 10 m NE-SW, 10 m NW-SE and 5 m high was superimposed over the mineralized solids. The model was rotated 30° counter clockwise to line up with drill sections and line up with the mineralized structures. Within each block the percentage below surface topography and the percentage inside each mineralized solid were recorded. These percentages were checked to assure there was no overlap. The block model origin was as follows:

Lower Left Corner		
618578 E	Column size = 10 m	142 columns
2175235 N	Row size = 10 m	116 rows
Top of Model		
2445 Elevation	Level size = 5 m	150 levels
Rotation 30° counter clockwise		

Figure 14-6. Isometric View Looking NW Showing Blocks



*BASH in Brown, LGASH in Yellow, MHG in Red, LGLS in Blue, LGSH in Green, NEHG in Purple and NELGSH in Orange

14.5 Bulk Density

A total of 425 specific gravity determinations were collected on a routine basis across the Ixtaca mineralized zone on cross sections 250 E (western border of Ixtaca), 550 E (central part of zone) and 1150 E (eastern section of zone).

• Section 250E: Drill Holes TU-11-030, TU-11-033, TU-11-040, TU-11-045, TU-11-074 and TU-11-075.
• Section 550E: Drill Holes TU-10-011, TU-10-013, TU-11-016, TU-11-019, TU-11-059, TU-11-066 and TU-11-078.
• Section 1150E: Drill Holes TU-11-041, TU-11-046, CA-11-002 and CA-11-003.

The measurements were made on drill core samples using the weight in air-weight in water method. The relative number of analysis is shown below:

Table 14-8. Specific Gravity Determinations Sorted by Cross Section

Cross Section	Number of Samples	Minimum SG	Maximum SG	Average SG
550 E	223	1.33	3.28	2.57
250 E	88	1.42	2.69	2.41
1150 E	114	1.43	3.21	2.60
Total	**425**	**1.33**	**3.28**	**2.55**

The data can also be sorted by lithology.

Table 14-9. Specific Gravity Determinations Sorted by Lithology

Lithology Code	Lithology	Number of Samples	Average SG
Ash	Ash unit	33	1.67
Bx/Lm	Breccia / Limestone	3	2.45
Df	Felsic Dyke	71	2.46
Dm	Mafic Dyke	7	2.70
Dp	Porphyritic Dyke	25	2.59
Lch	Limestone/chert	58	2.65
Lg	Lime < 10% mud	10	2.67
Lm	Lime Mudstone	72	2.67
Lp	Lime Packstone	37	2.59
Ls	Limestone undifferentiated	2	2.65
Lw	Lime wackestone	2	2.58
Min	Mineralized qtz. veining	7	2.96
Pp	Principal Porphyry	2	2.58
ShB	Shale	56	2.61
ShG	Green Shale	3	2.44
Skn	Skarn	20	2.89
Slt	Siltstone	17	2.71

Table 14-9 summarizes specific gravity values for all lithologies studied in all three sections. Values in the table have been averaged for each lithology. Values from these lithologies were then averaged within the various geologic domains to produce the following specific gravities for converting volumes to tonnes.

- Barren (BASH) and Low Grade Ash (LGASH) Domains had an average specific gravity of 1.67
- Low grade Limestone (LGLS) Domain had an average specific gravity of 2.66
- Main High Grade Zone (MHG) Domain had an average specific gravity of 2.63 (This unit contains about 20% Felsic Dyke)
- Main High Grade Zone (MHGN) North limb had an average specific gravity of 2.60 (This north limb contains about 40% Felsic Dyke and 40% Mafic Dyke)
- Low grade Shale (LGSH) and NE low grade Shale (NELGSH) Domains had an average specific gravity of 2.61
- North East extension High Grade (NEHG) Domain had an average specific gravity of 2.65

14.6 Grade Interpolation

Grades for gold and silver were interpolated into the blocks by Ordinary Kriging. Each domain was treated separately with hard boundaries used, except for the LGLS and NELGSH domains where contact plots showed a soft boundary was appropriate. For example, blocks with some percentage of MHG present were kriged for Au and Ag using only composites from within the MHG domain while blocks with some percentage of LGLS could see composites within both the LGLS and NELGSH domains. Blocks containing more than one domain were estimated for each domain and a weighted average was then produced.

Each kriging run was completed in a series of passes with the search ellipse orientation and dimension a function of the semivariogram for the domain and variable being estimated. The first pass used search dimensions equal to ¼ the semivariogram range in the three principal directions. A minimum of 4 composites were required to estimate a block with a maximum of 3 from any given drill hole. In this manner all blocks were estimated with a minimum of 2 drill holes. For blocks not estimated in pass 1 a second pass using ½ the semivariogram range was completed. A third pass using the full range and a fourth pass using twice the range followed. Finally because there were many blocks containing multiple domains a fifth pass was often required to ensure all domains were estimated. Since silver had shorter ranges in all domains except BASH the fourth pass for silver used the gold 4^{th} pass distances to ensure all blocks were estimated for both variables. For the barren Ash domain (BASH) the fourth pass for Au used the search ellipse distances for Ag, again to ensure all blocks had both variables estimated. In all passes the maximum number of composites used was 12 and if more were found in any search the closest 12 were used.

Once all domains were completed, estimated blocks containing some percentage outside the mineralized domains were estimated in a similar manner using composites from outside the mineralized domains (waste).

Finally for all blocks along the contacts, containing multiple domains, a weighted average grade for gold and silver was produced. The search parameters for gold within each domain and the number of blocks estimated in each pass are tabulated below.

Table 14-10. Kriging Parameters for Gold in Each Domain

Domain	Pass	Number Estimated	Az /Dip	Dist. (m)	Az /Dip	Dist. (m)	Az /Dip	Dist. (m)
MHG	1	1,229	60 / 0	24.0	330 / -55	7.0	150 / -35	17.5
	2	8,422	60 / 0	48.0	330 / -55	14.0	150 / -35	35.0
	3	8,754	60 / 0	96.0	330 / -55	28.0	150 / -35	70.0
	4	1,195	60 / 0	192.0	330 / -55	56.0	150 / -35	140.0
NEHG	1	15	347 / 0	10.0	257 / -55	10.0	77 / -35	7.5
	2	268	347 / 0	20.0	257 / -55	20.0	77 / -35	15.0
	3	2,809	347 / 0	40.0	257 / -55	40.0	77 / -35	30.0
	4	5,497	347 / 0	80.0	257 / -55	80.0	77 / -35	60.0
LGLS	1	32,841	60 / 0	30.0	330 / -55	19.5	150 / -35	30.0
	2	103,532	60 / 0	60.0	330 / -55	39.0	150 / -35	60.0
	3	47,748	60 / 0	120.0	330 / -55	78.0	150 / -35	120.0
	4	8,802	60 / 0	240.0	330 / -55	156.0	150 / -35	240.0
NELGSH	1	23,433	347 / 0	33.5	257 / -55	52.5	77 / -35	17.5
	2	64,711	347 / 0	67.0	257 / -55	105.0	77 / -35	35.0
	3	34,545	347 / 0	134.0	257 / -55	210.0	77 / -35	70.0
	4	3,690	347 / 0	268.0	257 / -55	420.0	77 / -35	140.0
LGASH	1	2,123	90 /0	25.0	0 / 0	15.0	0 / -90	7.5
	2	13,930	90 /0	50.0	0 / 0	30.0	0 / -90	15.0
	3	34,084	90 /0	100.0	0 / 0	60.0	0 / -90	30.0
	4	18,723	90 /0	200.0	0 / 0	120.0	0 / -90	60.0
BASH	1	952	45 / 0	20.0	315 / 0	7.5	0 / -90	10.0
	2	4,797	45 / 0	40.0	315 / 0	15.0	0 / -90	20.0
	3	21,763	45 / 0	80.0	315 / 0	30.0	0 / -90	40.0
	4	81,093	45 / 0	160.0	315 / 0	120.0	0 / -90	80.0
LGSH	1	189	60 / 0	16.0	330 / -55	9.0	150 / -35	15.0
	2	2,069	60 / 0	32.0	330 / -55	18.0	150 / -35	30.0
	3	7,566	60 / 0	64.0	330 / -55	36.0	150 / -35	60.0
	4	6,487	60 / 0	128.0	330 / -55	62.0	150 / -35	120.0
WASTE	1	2,287	Omni Directional			27.5		
	2	10,600	Omni Directional			55.0		
	3	27,218	Omni Directional			110.0		
	4	21,797	Omni Directional			220.0		

14.7 Classification

Based on the study herein reported, delineated mineralization of Ixtaca is classified as a resource according to the following definitions from National Instrument 43-101 and from CIM (2005):

"In this Instrument, the terms "mineral resource", "inferred mineral resource", "indicated mineral resource" and "measured mineral resource" have the meanings ascribed to those terms by the Canadian Institute of Mining, Metallurgy and Petroleum, as the CIM Definition Standards on Mineral Resources and Mineral Reserves adopted by CIM Council, as those definitions may be amended."

The terms Measured, Indicated and Inferred are defined by CIM (2005) as follows:

"A Mineral Resource is a concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge."

"The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of technical, economic, legal, environmental, socio-economic and governmental factors. The phrase 'reasonable prospects for economic extraction' implies a judgement by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. A Mineral Resource is an inventory of mineralization that under realistically assumed and justifiable technical and economic conditions might become economically extractable. These assumptions must be presented explicitly in both public and technical reports."

Inferred Mineral Resource

"An 'Inferred Mineral Resource' 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, workings and drill holes."

"Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred Mineral Resources must be excluded from estimates forming the basis of feasibility or other economic studies."

Indicated Mineral Resource

"An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics, 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 reasonably assumed."

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

Measured Mineral Resource

"A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are 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."

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

At Ixtaca the geologic continuity has been established through surface mapping and drill hole interpretation. This has resulted in a multi domain interpretation that has been used to constrain the resource estimate. The grade continuity within each domain has been quantified by semivariogram analysis. The semivariograms were used to determine the search directions and distances for each pass in the kriging procedure. Using the semivariogram range to estimate blocks would normally allow classification as follows:

- Blocks estimated in Pass 1 using ¼ of the semivariogram range might be considered Measured.
- Blocks estimated in Pass 2 using ½ of the semivariogram range might be considered Indicated
- All other blocks would be classified as Inferred.

At this time only 11% of all blocks were estimated in Pass 1 and they were still very spotty in their distribution, so as a result all blocks estimated in Pass 1 or 2 were classified as Indicated. All other blocks were classified as Inferred at this time.

The results are presented in two sets of tables. The first 14-11 and 14-12 assumes one could mine to the limits of the mineralized solids and no edge dilution is included. The second set of tables 14-13 and 14-14 assumes one would mine a total 10 x 10 x 5 m block and as a result, includes edge dilution around the outer limit of the mineralized solids.

Reality is somewhere between these two extremes as one could never mine exactly to the limits of the mineralized solids but with proper grade control one should never have to take all the edge dilution included in this size of block. In both tables, a cut-off of 0.50 g/t Au has been highlighted as a possible cut-off for open pit mining. At this time, however, no economic studies have been completed and the economic cut-off is unknown.

Table 14-11. Indicated Resource for Mineralized Portion of Blocks

Au Cut-off (g/t)	Tonnes > Cut-off (tonnes)	Grade>Cut-off		Contained Metal	
		Au (g/t)	Ag (g/t)	Au (ozs)	Ag (ozs)
0.10	117,860,000	0.35	17.01	1,319,000	64,460,000
0.20	65,640,000	0.51	22.48	1,083,000	47,440,000
0.25	52,420,000	0.59	24.80	988,000	41,800,000
0.30	42,560,000	0.66	27.13	902,000	37,120,000
0.40	29,550,000	0.80	31.57	757,000	29,990,000
0.50	**21,610,000**	**0.93**	**35.83**	**643,000**	**24,890,000**
0.60	16,450,000	1.05	39.69	553,000	20,990,000
0.70	12,900,000	1.16	43.29	479,000	17,950,000
0.80	10,260,000	1.26	46.25	416,000	15,260,000
1.00	6,510,000	1.47	53.28	308,000	11,150,000
2.00	820,000	2.50	85.79	66,000	2,260,000

Table 14-12. Inferred Resource for Mineralized Portion of Blocks

Au Cut-off (g/t)	Tonnes > Cut-off (tonnes)	Grade>Cut-off		Contained Metal	
		Au (g/t)	Ag (g/t)	Au (ozs)	Ag (ozs)
0.10	84,200,000	0.38	16.90	1,023,000	45,750,000
0.20	53,400,000	0.51	21.96	881,000	37,700,000
0.25	43,680,000	0.58	24.56	810,000	34,490,000
0.30	36,540,000	0.64	27.08	748,000	31,810,000
0.40	25,630,000	0.76	32.28	627,000	26,600,000
0.50	**18,700,000**	**0.88**	**37.27**	**527,000**	**22,410,000**
0.60	14,140,000	0.98	41.35	447,000	18,800,000
0.70	10,790,000	1.09	44.89	378,000	15,570,000
0.80	8,510,000	1.18	47.11	323,000	12,890,000
1.00	5,270,000	1.36	51.82	230,000	8,780,000
2.00	310,000	2.42	67.30	24,000	670,000

Where Mineralized Portion of Blocks means one could mine to the boundaries of the mineralized domains.

Table 14-13. Indicated Resource for Total Blocks

Au Cut-off (g/t)	Tonnes > Cut-off (tonnes)	Grade>Cut-off		Contained Metal	
		Au (g/t)	Ag (g/t)	Au (ozs)	Ag (ozs)
0.10	117,650,000	0.35	17.00	1,316,000	64,300,000
0.20	65,520,000	0.51	22.47	1,081,000	47,330,000
0.25	52,320,000	0.59	24.80	986,000	41,720,000
0.30	42,430,000	0.66	27.15	899,000	37,040,000

0.40	29,460,000	0.80	31.59	755,000	29,920,000
0.50	**21,530,000**	**0.93**	**35.89**	**641,000**	**24,840,000**
0.60	16,400,000	1.05	39.73	551,000	20,950,000
0.70	12,850,000	1.16	43.38	477,000	17,920,000
0.80	10,220,000	1.26	46.37	414,000	15,240,000
1.00	6,460,000	1.47	53.54	306,000	11,120,000
2.00	810,000	2.50	85.93	65,000	2,240,000

Table 14-14. Inferred Resource for Total Blocks

Au Cut-off (g/t)	Tonnes > Cut-off (tonnes)	Grade>Cut-off		Contained Metal	
		Au (g/t)	Ag (g/t)	Au (ozs)	Ag (ozs)
0.10	84,000,000	0.38	16.85	1,018,000	45,510,000
0.20	52,990,000	0.51	22.01	874,000	37,500,000
0.25	43,410,000	0.58	24.60	805,000	34,330,000
0.30	36,250,000	0.64	27.17	742,000	31,670,000
0.40	25,440,000	0.76	32.41	622,000	26,510,000
0.50	**18,550,000**	**0.88**	**37.48**	**524,000**	**22,350,000**
0.60	14,040,000	0.99	41.56	445,000	18,760,000
0.70	10,730,000	1.09	45.06	376,000	15,540,000
0.80	8,480,000	1.18	47.24	322,000	12,880,000
1.00	5,240,000	1.36	52.00	229,000	8,760,000
2.00	310,000	2.42	67.30	24,000	670,000

Where Total Blocks means one would mine complete 10 x 10 x 5 m blocks taking in dilution around the edges of the mineralized solids.

These same tables are shown below using gold equivalent cut-offs where:

Gold – 3 yr. trailing average price of $1500
Silver – 3 yr. trailing average price of $29

Preliminary metallurgy has shown roughly equivalent metal recoveries for Au and Ag so for now the Au Equivalent equation is:

$$AuEq = Au + (Ag * 29 / 1500)$$

Table 14-15. Indicated Resource with AuEq Cut-off for Mineralized Portion of Blocks

AuEq Cut-off (g/t)	Tonnes > Cut-off (tonnes)	Grade>Cut-off			Contained Metal x1000		
		Au (g/t)	Ag (g/t)	AuEq (g/t)	Au (ozs)	Ag (ozs)	AuEQ (ozs)
0.10	191,390,000	0.24	13.54	0.50	1,465	83,320	3,077
0.20	133,100,000	0.31	17.81	0.66	1,335	76,210	2,807
0.25	113,720,000	0.35	19.80	0.73	1,269	72,390	2,669
0.30	97,840,000	0.38	21.80	0.80	1,202	68,580	2,526
0.40	73,610,000	0.45	25.87	0.95	1,074	61,230	2,258
0.50	**56,990,000**	**0.52**	**29.91**	**1.10**	**960**	**54,800**	**2,019**
0.60	44,920,000	0.59	34.05	1.25	856	49,180	1,807
0.70	36,130,000	0.66	38.15	1.40	767	44,320	1,624
0.80	29,690,000	0.73	42.10	1.54	692	40,190	1,469
1.00	20,920,000	0.85	49.82	1.81	570	33,510	1,218
2.00	5,740,000	1.31	88.14	3.01	241	16,270	556

Table 14-16. Inferred Resource with AuEq Cut-off for Mineralized Portion of Blocks

AuEq Cut-off (g/t)	Tonnes > Cut-off (tonnes)	Grade>Cut-off			Contained Metal x1000		
		Au (g/t)	Ag (g/t)	AuEq (g/t)	Au (ozs)	Ag (ozs)	AuEQ (ozs)
0.10	121,520,000	0.28	14.32	0.56	1,098	55,950	2,180
0.20	86,290,000	0.36	18.81	0.73	1,010	52,190	2,017
0.25	75,110,000	0.40	20.86	0.80	964	50,370	1,937
0.30	65,880,000	0.43	22.93	0.88	917	48,570	1,855
0.40	51,800,000	0.50	27.12	1.02	826	45,170	1,700
0.50	**41,530,000**	**0.56**	**31.41**	**1.16**	**741**	**41,940**	**1,552**
0.60	33,450,000	0.62	35.95	1.31	662	38,660	1,410
0.70	27,370,000	0.68	40.46	1.46	595	35,600	1,283
0.80	23,200,000	0.73	44.37	1.59	544	33,100	1,183
1.00	17,830,000	0.82	50.60	1.80	469	29,010	1,030
2.00	5,080,000	1.14	83.18	2.75	186	13,590	449
3.00	1,420,000	1.49	113.47	3.68	68	5,180	168

Where Mineralized Portion of Blocks means one could mine to the boundaries of the mineralized domains.

Table 14-17. Indicated Resource with AuEq Cut-off for Total Blocks

AuEq Cut-off (g/t)	Tonnes > Cut-off (tonnes)	Grade>Cut-off			Contained Metal x1000		
		Au (g/t)	Ag (g/t)	AuEq (g/t)	Au (ozs)	Ag (ozs)	AuEQ (ozs)
0.10	191,880,000	0.24	13.49	0.50	1,462	83,220	3,072
0.20	132,830,000	0.31	17.80	0.66	1,328	76,020	2,802
0.25	113,470,000	0.35	19.79	0.73	1,266	72,200	2,660
0.30	97,580,000	0.38	21.80	0.80	1,198	68,390	2,519
0.40	73,370,000	0.45	25.88	0.95	1,071	61,050	2,250
0.50	**56,780,000**	**0.52**	**29.94**	**1.10**	**957**	**54,660**	**2,014**
0.60	44,760,000	0.59	34.09	1.25	853	49,060	1,802
0.70	36,020,000	0.66	38.20	1.40	764	44,240	1,619
0.80	29,610,000	0.73	42.14	1.54	690	40,120	1,465
1.00	20,840,000	0.85	49.92	1.81	568	33,450	1,214
2.00	5,730,000	1.31	88.21	3.01	241	16,250	555
3.00	2,170,000	1.71	118.93	4.01	119	8,300	280

Table 14-18. Inferred Resource with AuEq Cut-off for Total Blocks

AuEq Cut-off (g/t)	Tonnes > Cut-off (tonnes)	Grade>Cut-off			Contained Metal x1000		
		Au (g/t)	Ag (g/t)	AuEq (g/t)	Au (ozs)	Ag (ozs)	AuEQ (ozs)
0.10	122,790,000	0.28	14.13	0.55	1,094	55,780	2,171
0.20	85,860,000	0.36	18.77	0.73	1,002	51,810	2,004
0.25	74,610,000	0.40	20.84	0.80	957	49,990	1,924
0.30	65,310,000	0.43	22.94	0.88	909	48,170	1,842
0.40	51,310,000	0.50	27.14	1.02	820	44,770	1,686
0.50	**41,120,000**	**0.56**	**31.44**	**1.16**	**735**	**41,570**	**1,539**
0.60	33,070,000	0.62	36.03	1.31	657	38,310	1,397
0.70	27,010,000	0.68	40.56	1.46	591	35,220	1,271
0.80	22,920,000	0.73	44.46	1.59	540	32,760	1,174
1.00	17,680,000	0.82	50.63	1.80	467	28,780	1,023
2.00	5,070,000	1.14	83.15	2.75	186	13,550	448
3.00	1,420,000	1.49	113.47	3.68	68	5,180	168

Where Total Blocks means one would mine complete 10 x 10 x 5 m blocks taking in dilution around the edges of the mineralized solids.

14.8 Block Model Verification

To check the results, level plans were produced on 50 m intervals through the deposit. Estimated block grades were checked against composite grades above and below the bench level. The results matched reasonably well with no bias indicated. Example bench levels are show in Figures 14.7 to 14.11 for bench levels 2250 down to 2050.

Another check on the results was completed by comparing the average composite grade for each domain with the average kriged grades for that domain.

Table 14-19. Comparison of Composite Mean Au Grade to Block Mean Au Grade

Domain	Variable	Number of Assays	Mean Grade Composites	Number of Blocks	Mean Grade Blocks
BASH	Au (g/t)	2,305	0.007	110,686	0.011
	Ag (g/t)	2,305	0.17	110.686	0.23
LGASH	Au (g/t)	2,682	0.280	69,018	0.278
	Ag (g/t)	2,682	5.18	69,018	5.65
MHG	Au (g/t)	1,922	0.892	19,656	0.877
	Ag (g/t)	1,922	61.21	19,656	66.41
LGLS	Au (g/t)	10,366	0.154	192,923	0.158
	Ag (g/t)	10,366	9.70	192,923	8.24
LGSH	Au (g/t)	778	0.079	16,311	0.078
	Ag (g/t)	778	5.45	16,311	5.46
NEHG	Au (g/t)	419	0.771	8,781	0.857
	Ag (g/t)	419	42.00	8,781	37.71
NELGSH	Au (g/t)	3,588	0.072	126,379	0.081
	Ag (g/t)	3,588	6.42	126,379	6.28
WASTE	Au (g/t)	4,700	0.007	63,123	0.012
	Ag (g/t)	4,700	0.50	63,123	0.74

Figure 14-7. IXTACA 2250 Level Plan Showing Estimated Gold in Blocks

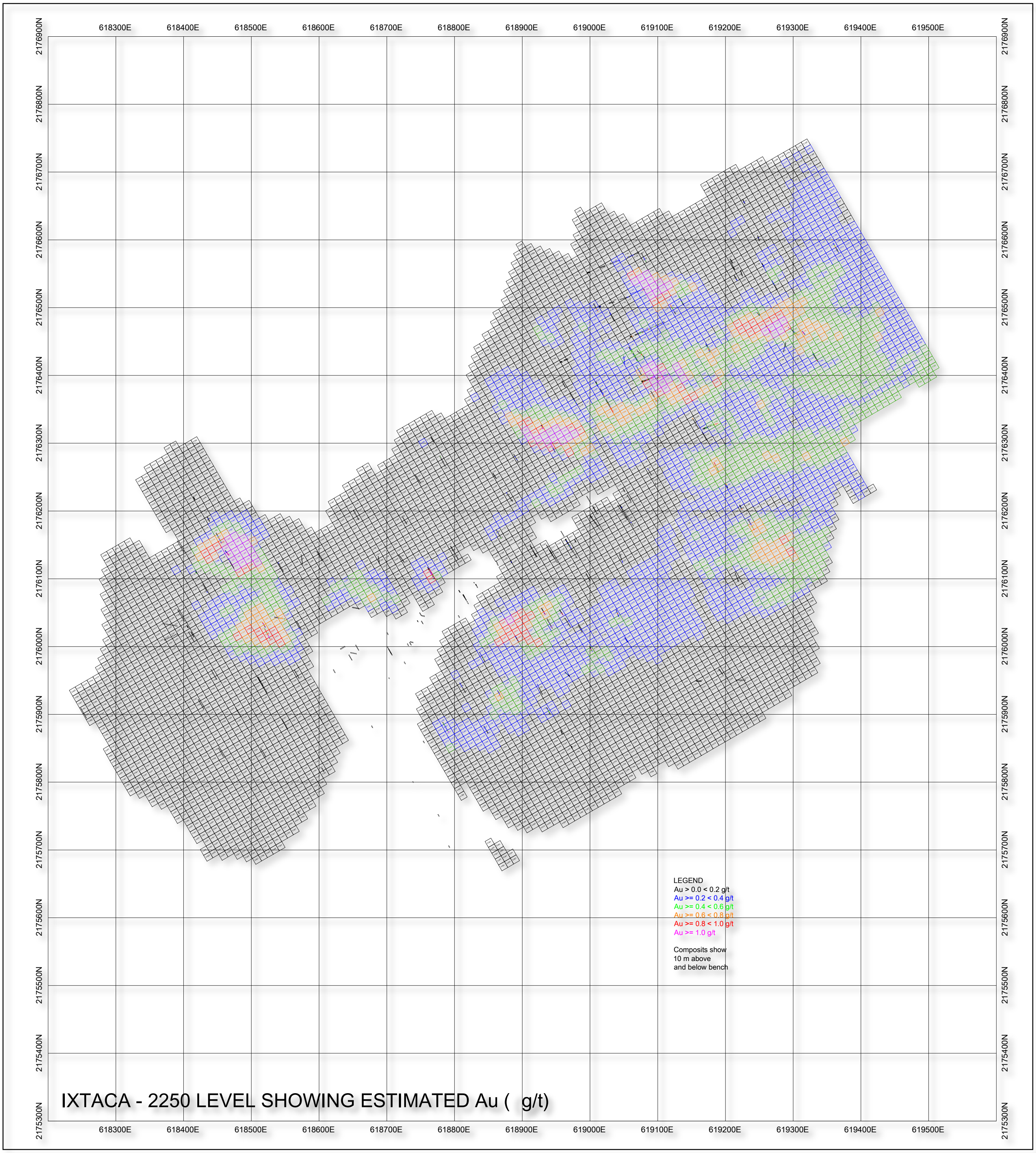


Figure 14-8. IXTACA 2200 Level Plan Showing Estimated Gold in Blocks

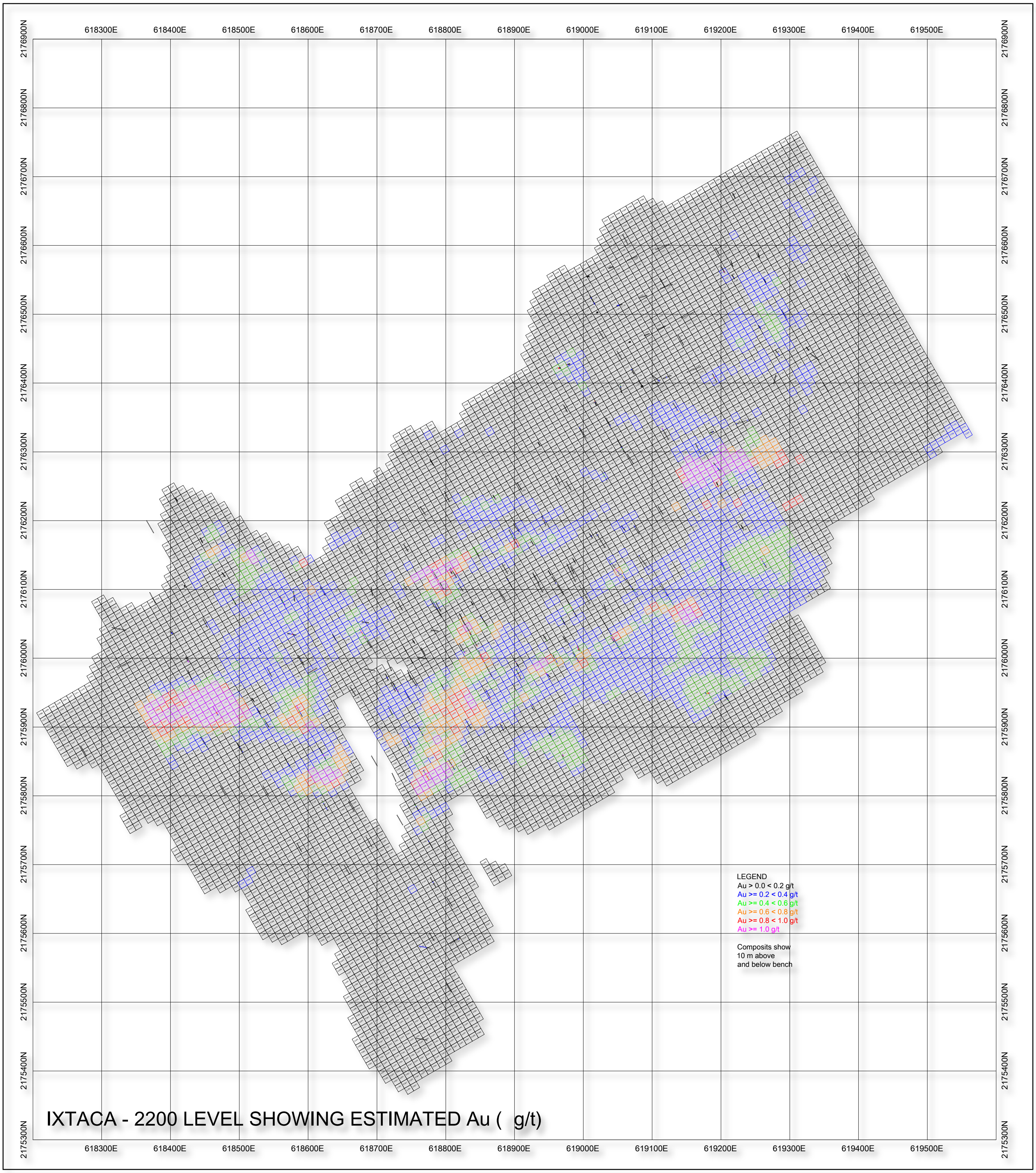


Figure 14-9. IXTACA 2150 Level Plan Showing Estimated Gold in Blocks

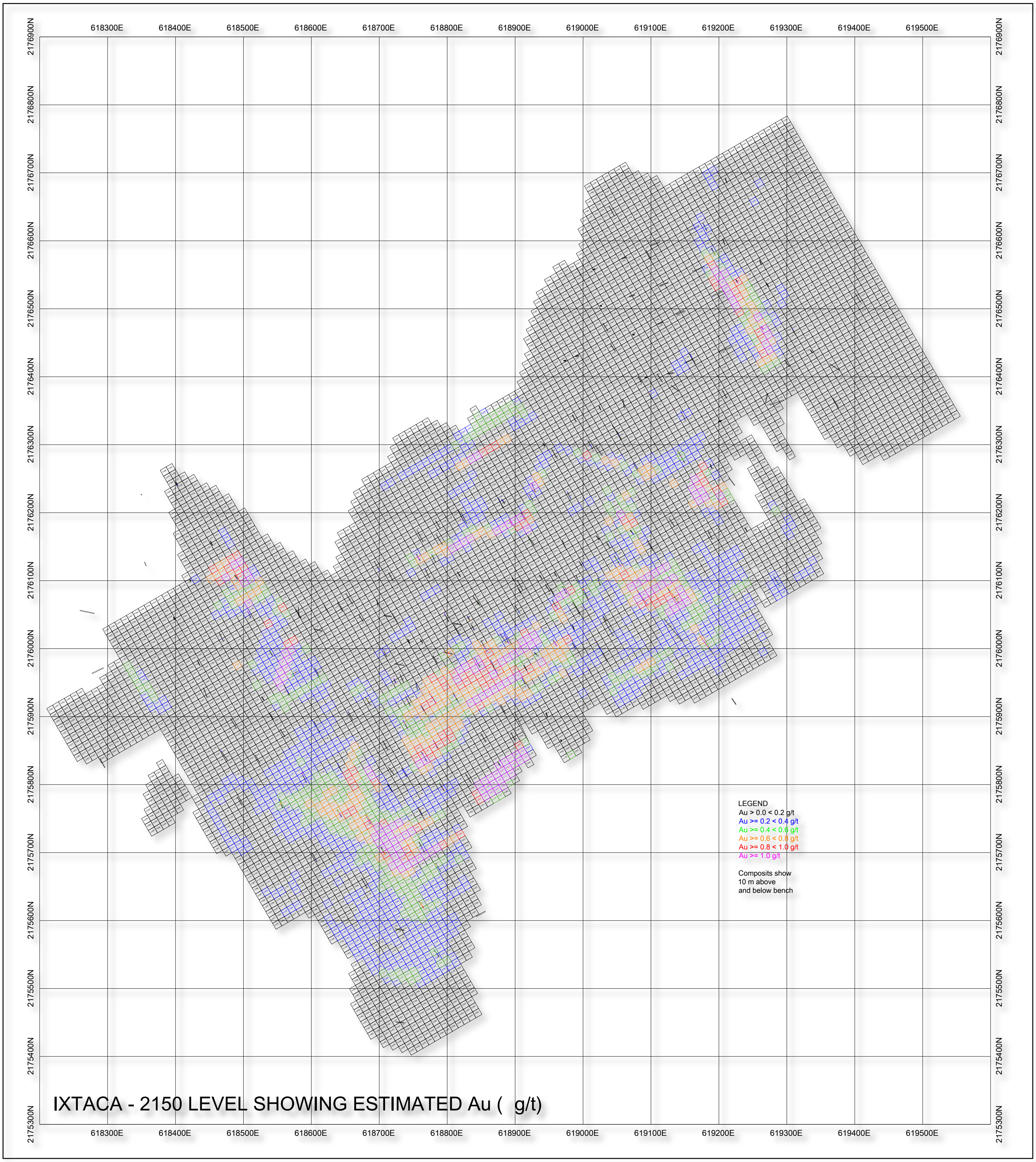


Figure 14-10. IXTACA 2100 Level Plan Showing Estimated Gold in Blocks

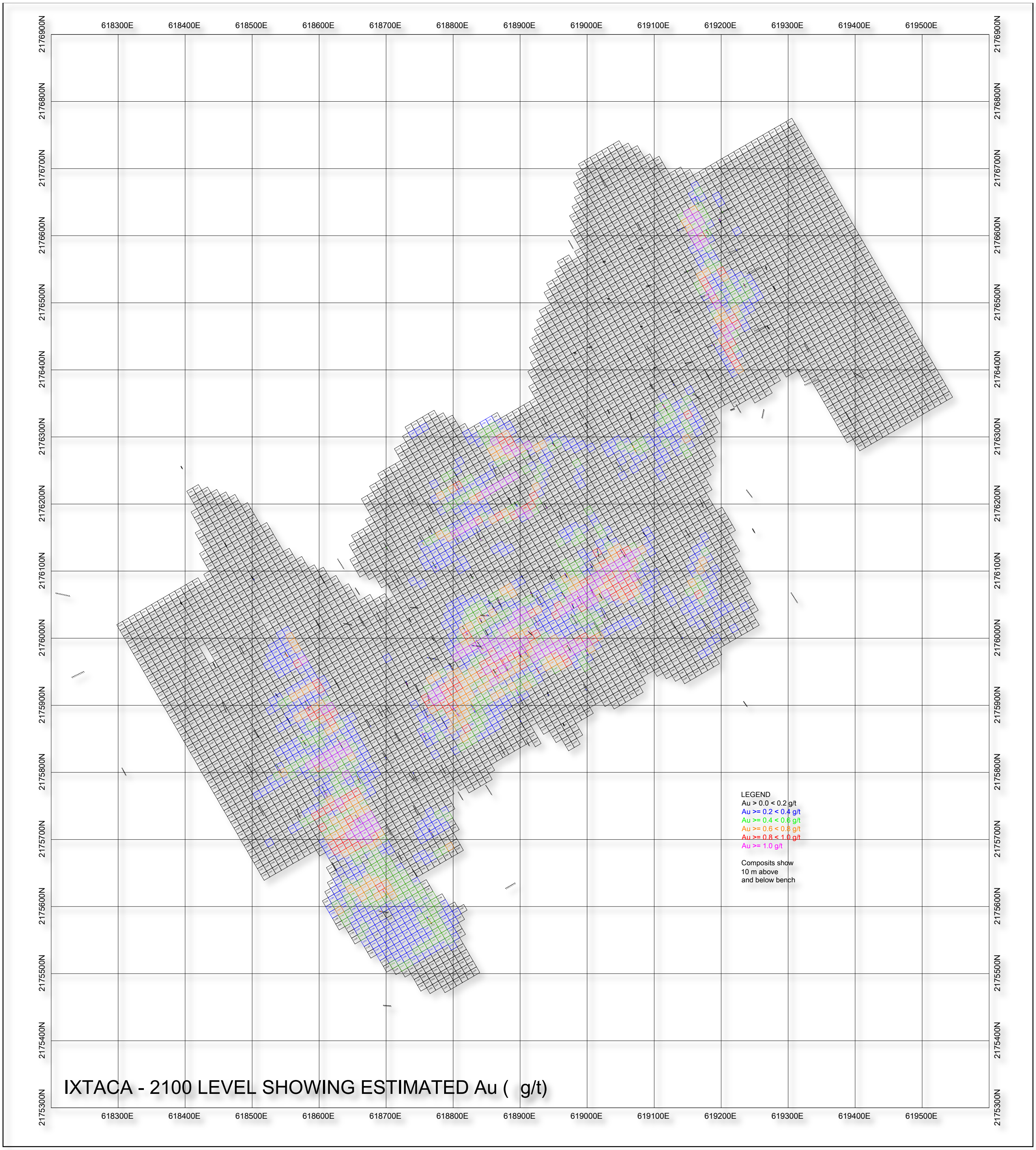
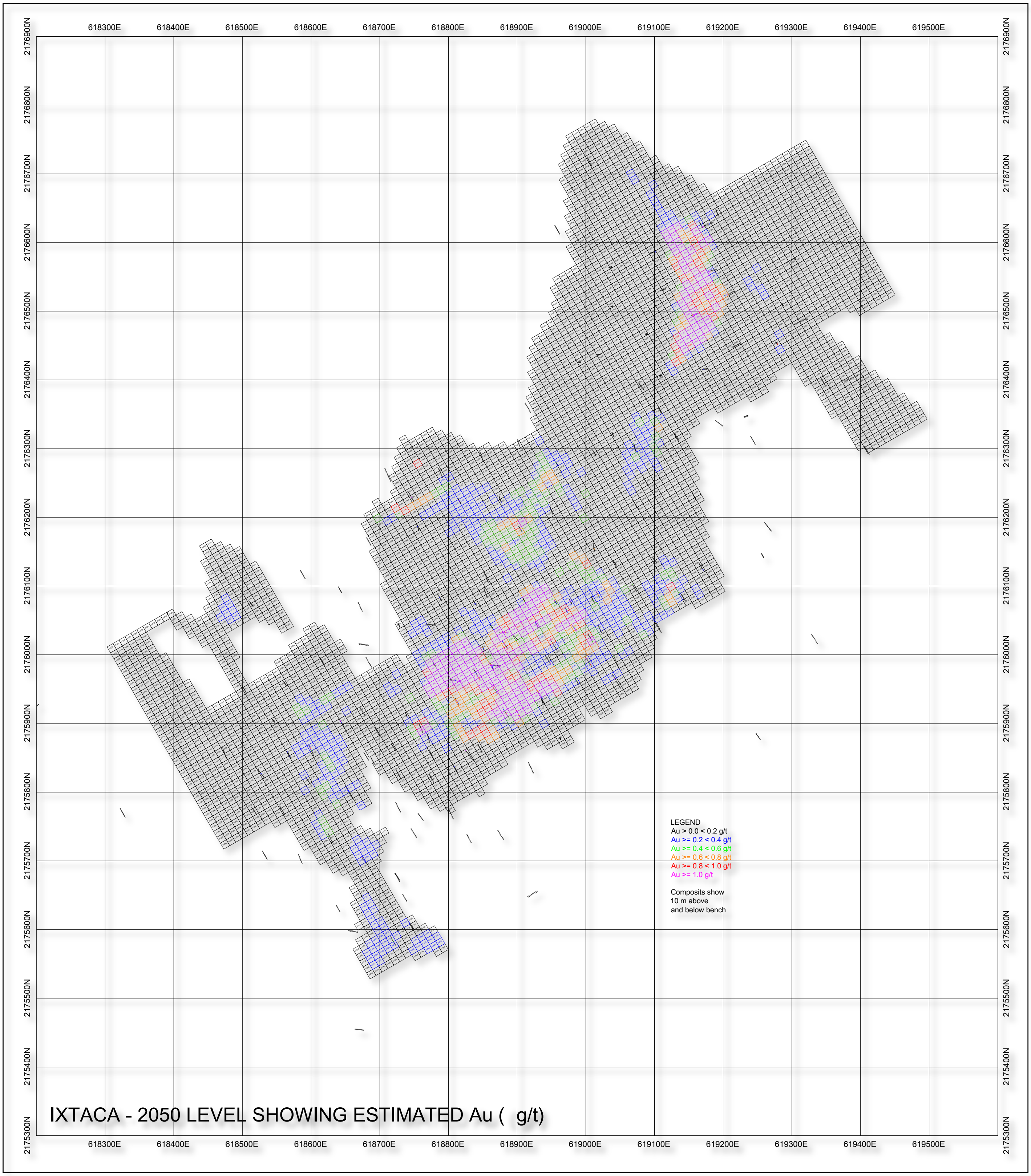


Figure 14-11. IXTACA 2050 Level Plan Showing Estimated Gold in Blocks



15 Adjacent Properties

15.1 Santa Fe Metals Corp. Cuyoaco Property

The Cuyoaco property is 100% owned by Santa Fe Metals Corp. It is located approximately 4 km south east of the Tuligitic property (Figure 4-2) and it covers 643 hectares over two mineralized targets: the Pau copper-silver-gold skarn, and the Santa Anita gold project.

15.1.1 Pau Skarn Project

The Pau Project is a copper-silver-gold skarn in Santa Fe Metals' Pau claims and in the western part of its Santa Anita claims. The claims cover an area of approximately 3 square kilometers of epidote-garnet skarn mineralization around a large granodioritic pluton.

In total there are 16 documented, historical workings on the Pau project, many of which are believed to be as old as 16th century. The largest workings include the 170 m x 200 m 'El Magistral' open pit, 3 levels of underground workings at 'California' as well as 'Lincon' (two 50 m adits), 'La Juanita' (two adits), 'La Verdiosa' and 'El Toro'.

Geology on the Pau Project is characterized by garnet-actinolite-quartz-hematite skarn style mineralization associated with two copper, silver, gold rich zones along the western and eastern margins of the granodioritic pluton. Skarn mineralization is exposed at surface in several locations and in the historical workings. Secondary (oxidized) enrichment extends for at least 10 m below surface and is characterized by malachite, azurite and chalcocite but most likely does not form the bulk of the mineralization.

Soil and rock sampling in 2008 by Oremex Silver Inc. returned high-grades of copper, silver, gold, lead and zinc from the exposed rock within workings, and mapping in 2011 found that many of the adits ended in mineralization. Soil and Rock sampling by Santa Fe Metals in 2011 focused on further exploration of the northern part of the Pau Claim and mapping skarn mineralization between known adits. Highlights include a 7.21 g/t Au, 27.7 g/t Ag skarn sample in the El Magistral zone. Low grade gold (0.32 g/t Au) was found within the granodiorite itself, and a previously unknown skarn showing was discovered in the north of the property, a further 1 km north of the La Juanita adits.

15.1.2 Santa Anita Project

Santa Anita is a historic dyke and sill hosted gold rich deposit found in the east of the Cuyoaco property. It is characterized by a zone of parallel gold rich dykes and sills approximately 1 km along and 800 meters wide. In 2011 a parallel dyke and sill system 200 m wide and 600 m in length was discovered to the north east.

The Santa Anita gold project covers a series of parallel, gold-rich dykes and sills that have intruded and altered a sedimentary sequence of limestone and mudstones. The dykes and sills are between 1 m and 10 m wide and form a 1 km by 800 m NW-SE trending zone. The dykes and sills are porphyritic dacites that contain varying amounts of feldspar and hornblende phenocrysts and in places up to 10% fine grained disseminated pyrite.

An extensive surface geochemical mapping program in 2008, delineated a large gold rich envelope called the Santa Anita zone. Mineralization was found to be coarse free metallic gold and electrum in calcite stringers associated to narrow dacitic dikes hosted in a skarn-hornfels-limestone sequence. A limited chip sampling program of the underground workings returned an average grade of 3 g/t. Fifty-eight samples were collected in total.

Drilling of five shallow holes (607 metres in total) in 2005/2006 intersected gold mineralization, with one hole intersecting 12 metres of 2.45 g/t Au and another hole intersecting 4 metres of 2.54 g/t Au.

Rock and channel samples collected by Santa Fe Metals in 2011 outline a large low grade gold anomaly that extends beyond the historical boundary of the Santa Anita gold deposit and indicates that the zone of gold rich mineralization is considerably larger than previously thought. The parallel dyke system, named ‘Santa Anita Nuevo’, has a (surface) width of 200 meters and a strike length of 600 meters. To date, Santa Fe Metals has collected 29 channel samples from dykes to the north of the property that have returned values greater than 0.1 g/t.

15.2 Minera Frisco S.A. de C.V. Espejeras

The Espejeras property is 100% owned by Minera Frisco S.A. de C.V. It is located roughly 7 km north of the Tuligtic property (Figure 4-1) and it covers a surface of 8.75 hectares. Information on the exploration work carried out in the area to date is very limited. The area is considered prospective for gold and silver and Minera Frisco’s 2011 Annual Report lists the Espejeras project among feasibility studies and implementation projects. Minera Frisco is looking to obtain environmental permits to implement an extensive diamond drilling program on the property in the near future.

16 Other Relevant Data and Information

The author is not aware of any other relevant information with respect to the Tuligtic Project that is not disclosed in the Technical Report.

17 Interpretation and Conclusions

Almaden acquired the Cero Grande claim of the Tuligtic Project in 2003 following the identification of surficial clay deposits that were interpreted to represent high-level epithermal alteration. Subsequent geologic mapping, rock, stream silt sampling and induced polarization (IP) geophysical surveys identified porphyry copper and epithermal gold targets within an approximately 5 x 5 km area of intensely altered rock. In July 2010 Almaden initiated a diamond drilling program to test epithermal alteration within the Tuligtic Property, resulting in the discovery of the Ixtaca Zone. The first hole, TU-10-001 intersected 302.42 metres of 1.01 g/t Au and 48 g/t Ag and multiple high grade intervals including 1.67 metres of 60.7 g/t Au and 2122 g/t Ag.

Within the Tuligtic Project, argillaceous limestone of the Late Jurassic to Early Cretaceous Upper Tamaulipas formation is underlain by transitional calcareous siltstone and grainstones units, calcareous shale. During the Laramide orogeny the carbonate package was intensely deformed into a series of thrust-related east verging anticlines. Calcareous shale units appear to occupy the cores of the anticlines while the thick bedded limestone/mudstone units occupy the cores of major synclines at the Ixtaca Zone.

Limestone basement units are crosscut by intensely altered intermediate dykes. The deformed Mesozoic sedimentary sequence is discordantly overlain by late Cenozoic bedded crystal tuff of the upper Coyoltepec subunit.

Between 2001 and 2012, Almaden's exploration at the Tuligtic Property included rock and soil geochemical sampling, ground magnetics, IP and resistivity, Controlled Source Audio-frequency Magnetotelluric (CSAMT), and Controlled Source Induced Polarization (CSIP) geophysical surveys.

Of the 436 rock grab samples collected, a total of 45 samples returned assays of greater than 100 parts-per-billion (ppb) gold (Au), and up to 6.14 grams-per-tonne (g/t) Au. A total of 49 rock samples returned assays of greater than 10 g/t silver (Ag) and up to 291 g/t Ag. Basement carbonate units, altered intrusive, and locally calc-silicate skarn mineralization occur as erosional windows beneath unmineralized tuff of the upper Coyoltepec subunit. Surface mineralization at the Ixtaca Zone occurs as limestone boulders containing quartz vein fragments and high level epithermal alteration within overlying volcanic rocks. Epithermal alteration and mineralization is observed overprinting earlier skarn and porphyry style alteration and mineralization. Numerous small skarn-related showings exist on the project. At the Caleva soil anomaly, a 200 x 100 m skarn zone hosts sphalerite, galena and chalcopyrite quartz vein stockwork mineralization along the contact zone between limestone and altered and mineralized intrusive rocks to the east.

The collection of 4,760 soil samples by Almaden between 2005 and 2011 resulted in the identification of five anomalous areas: the Ixtaca, Ixtaca East, Caleva, Azul, and Sol zones. Anomalous thresholds (95th percentile) for gold and silver were calculated to be 20.63 ppb Au and 0.71 ppm Ag, respectively. A total of 238 samples containing anomalous Au were found, including 120 samples with coincident Ag anomalies. The Ixtaca Zone produces the largest Au and Ag response within the Tuligtic Property. Based metals do not correlate significantly with the Ixtaca Zone, and Hg and Sb anomalies occur peripherally within altered volcanic rocks. Base metals correlate well with Au-Ag at the Caleva, Azul, and Sol zones to such an extent they are best termed Cu-Zn (Au-Ag) anomalies. Based on the distribution of soil geochemical anomalies and the mapped geology it is apparent that the overlying post mineral volcanics significantly suppress sedimentary and intrusive basement rock geochemical anomalies. Soil responses are consistnetn with these zones being prospective for both epithermal and earlier skarn mineralization.

IP and CSAMT resistivity surveys largely reflect surface geology, which is controlled by local topography. Resistivity anomalies occur where surface exposures are dominated by limestone and intrusive lithologies. The anomalies are controlled in part by topographic lows that down-cut through overlying tuff rocks and expose resistive basement lithologies. Conductive anomalies occur along local topographic high ridges and plateaus where accumulations of conductive tuff rocks remain. At the Ixtaca Zone, a northwest trending resistivity and weak chargeability anomaly is centered on the North and Main Ixtaca zones. The anomaly is coincident with the east-verging limestone-cored syncline that hosts the high-grade North and Main Ixtaca zones of mineralization.

From July, 2010 to the November 13, 2012 maiden mineral resource estimate cut-off, Almaden has drilled 225 holes totalling 81,971 m on the Main Ixtaca, Ixtaca North and

Northeast Extension zones. Diamond drilling at 25 to 50 m section spacing has defined the Main Ixtaca and Ixtaca North zones over a strike length of approximately 650 m. High-grade mineralization has been intersected to depths of 200 to 300 m vertically from surface and occurs within a broader zone of mineralization extending laterally (NNW-SSE) over 600 m and to a vertical depth of 600 m below surface. The epithermal vein system at the Main Ixtaca and Ixtaca North zones is associated with two subparallel ENE (060 degrees) trending, subvertical to steeply north dipping dyke zones.

At the Main Ixtaca Zone, a series of 2 m to over 20 m true width dykes occur within an approximately 100 m wide zone. The Ixtaca North dyke zone is narrower and comprises a steeply north-dipping zone of two or three discrete dykes ranging from 5 to 20 m in width. Epithermal vein mineralization occurs both within the dykes and sedimentary host rocks, with the highest grades often occurring within or marginal to the dykes. Vein density decreases outward to the north and south from the dyke zones resulting in the formation of two high-grade zones that lack sharp geologic boundaries. On surface, the Main Ixtaca and Ixtaca North zones are separated by a steep sided ENE trending valley.

The bulk of Main Ixtaca and Ixtaca North zone mineralization is bound within an ENE-verging asymmetric synform. The synfom is cored by a structurally thickened sequence of argillaceous limestone that grades laterally and at depth through transition units, into calcareous shale at depth. The Limestone sequence thins to the west along the rising limb of an ENE-verging antiform. The Main Ixtaca and Ixtaca North vein systems and the dykes transect the antiform sub-perpendicular to the strike of the fold axis. Vein density decreases within shale units coring the antiform and mineralization is confined near the axis of the antiform within a west dipping tabular zone of low-grade mineralization having a true thickness ranging from 150 to 200 m. Mineralized basement rocks are unconformably overlain by crystal tuff, which is also mineralized. High-grade zones of mineralization are present within the tuff vertically above the Main Ixtaca and Ixtaca North vein systems. The high-grade zones transition laterally into low grade mineralization, which together form a broad tabular zone of mineralization at the base of the tuff unit.

The Northeast Extension Zone has a strike length of approximately 350 m as defined by drilling along a series of five ENE (070 degrees) oriented sections spaced at intervals of 50 to 100 m, and near-surface oblique NNW-SSE oriented drill holes. The Northeast Extension Zone dips moderately-steeply to the WSW. High grade mineralization having a true-width ranging from less than 30 and up to 60 m has been intersected beneath approximately 30 m of tuff to a vertical depth of 550 m, or approximately 600 m down-dip. Northeast Extension Zone mineralization is interpreted to occur within the hinge zone of a shale cored antiform. Near surface along the axis of the antiform a narrow zone structurally thinned, brecciated, and mineralized limestone is unconformably overlain by mineralized tuff rocks. At a vertical depth of approximately 80 m below surface, high-grade shale-hosted mineralization dips moderately-steeply WSW sub-parallel to the interpreted axial plane of the antiform. The footwall of the high-grade zone is marked by a distinct 20 to 30 m true-thickness felsic porphyry dyke (Chemalaco Dyke), which is also mineralized. The Chamelaco Dyke has been interested in multiple drill holes ranging from 250 to 550 m vertically below surface, and its lower contact currently marks the base of Northeast Extension Zone.

Metallurgical testwork was completed on each of the Ixtaca Zone geologic domains: limestone, limestone/dyke high grade (HG), shale (Northeast Extension Zone) and volcanic tuff material. Modelling shows that a combination of grinding to a p_{80} of 100-150µm plus gravity recovery on the cyclone underflow, with recovery of gold and silver by means of bulk flotation, followed by intensive leaching of the combined gravity and flotation concentrates is a viable process route for the Ixtaca resource. A summary of metallurgical parameters for the main zones tested for this process route is presented in Table 17-1. While an acceptable economic baseline has been established, further opportunities exist for optimising the gold and silver recoveries from the resource, and a programme of metallurgical optimization, including further flotation and cyanidation work is planned.

Table 17-1. Overall Projected Gravity + Flotation + Intensive Leach Recoveries

Zone	Overall Recovery	
	Au (Wt%)	Ag (Wt%)
Dyke	96.8	85.3
Limestone	88.7	78.3
Limestone HG	94.9	87.0
Shale	95.9	81.8
Tuff (Volcanic)	54.1	61.9

Giroux Consultants Ltd. prepared the Maiden mineral resource estimate for the Ixtaca Deposit based on the results of diamond drilling completed by Almaden. Preliminary metallurgy has shown roughly equivalent metal recoveries for Au and Ag, therefore the mineral resource estimate is presented at a series of Au-equivalent (AuEq) cut-offs based on a three years trailing average price of $1,500 per-ounce Au, and $29 per-ounce Ag, and assuming one could mine to the limits of the mineralized solids and no edge dilution is included. Ixtaca Deposit mineralization has been classified as an inferred and indicated mineral resource according to the definitions from NI 43-101 and from CIM (2005). A cut-off of 0.50 g/t Au has been highlighted as a possible cut-off for open pit mining (Table 17-1 and 17-2). At this time, however, no economic studies have been completed and the economic cut-off is unknown.

Table 17-2. Indicated Resource with AuEq Cut-off for Mineralized Portion of Blocks

AuEq Cut-off (g/t)	Tonnes > Cut-off (tonnes)	Grade>Cut-off			Contained Metal x1000		
		Au (g/t)	Ag (g/t)	AuEq (g/t)	Au (ozs)	Ag (ozs)	AuEQ (ozs)
0.10	191,390,000	0.24	13.54	0.50	1,465	83,320	3,077
0.20	133,100,000	0.31	17.81	0.66	1,335	76,210	2,807
0.25	113,720,000	0.35	19.80	0.73	1,269	72,390	2,669
0.30	97,840,000	0.38	21.80	0.80	1,202	68,580	2,526
0.40	73,610,000	0.45	25.87	0.95	1,074	61,230	2,258
0.50	**56,990,000**	**0.52**	**29.91**	**1.10**	**960**	**54,800**	**2,019**
0.60	44,920,000	0.59	34.05	1.25	856	49,180	1,807
0.70	36,130,000	0.66	38.15	1.40	767	44,320	1,624
0.80	29,690,000	0.73	42.10	1.54	692	40,190	1,469
1.00	20,920,000	0.85	49.82	1.81	570	33,510	1,218
2.00	5,740,000	1.31	88.14	3.01	241	16,270	556

Table 17-3. Inferred Resource with AuEq Cut-off for Mineralized Portion of Blocks

AuEq Cut-off (g/t)	Tonnes > Cut-off (tonnes)	Grade>Cut-off			Contained Metal x1000		
		Au (g/t)	Ag (g/t)	AuEq (g/t)	Au (ozs)	Ag (ozs)	AuEQ (ozs)
0.10	121,520,000	0.28	14.32	0.56	1,098	55,950	2,180
0.20	86,290,000	0.36	18.81	0.73	1,010	52,190	2,017
0.25	75,110,000	0.40	20.86	0.80	964	50,370	1,937
0.30	65,880,000	0.43	22.93	0.88	917	48,570	1,855
0.40	51,800,000	0.50	27.12	1.02	826	45,170	1,700
0.50	**41,530,000**	**0.56**	**31.41**	**1.16**	**741**	**41,940**	**1,552**
0.60	33,450,000	0.62	35.95	1.31	662	38,660	1,410
0.70	27,370,000	0.68	40.46	1.46	595	35,600	1,283
0.80	23,200,000	0.73	44.37	1.59	544	33,100	1,183
1.00	17,830,000	0.82	50.60	1.80	469	29,010	1,030
2.00	5,080,000	1.14	83.18	2.75	186	13,590	449
3.00	1,420,000	1.49	113.47	3.68	68	5,180	168

Diamond drilling by Almaden has resulted in the identification of an indicated mineral resource of 56.99 million-tonnes, comprising 2.02 million-ounces AuEq at an average grade of 1.10 g/t AuEq; and an inferred mineral resource of 41.53 million-tonnes, comprising 1.55 million-ounces AuEq at an average grade of 1.16 g/t AuEq, each using a cut-off grade of 0.5 g/t AuEq. Roughly 90% of the deposit is hosted by the carbonate units, the remaining 10% in volcanic rocks.

Subsequent to the November 13, 2012 drilling cuttoff for the resource, Almaden announced the discovery of a new volcanic-hosted high grade area along the trend of the Main Ixtaca Zone with holes TU-12-222, 224, 225 and 227, all drilled form the same setup. These holes were drilled on section 11+000E, outside the resource shell, and located 50 m northeast of the closest drill holes that were part of the resource. For the first time in the Ixtaca drill program visible gold was identified in one of these holes, TU-12-224. Intersections in this new zone included 134.20 m of 4.1 g/t AuEq (3.76 g/t Au and 18.1 g/t Ag). This new zone is indicative of the potential for teh resource to grow in this area as well as elsewhere where mineralization has yet to be constrained.

Based upon the drilling conducted to date, the Main Ixtaca and Ixtaca North zones remain open to the west, north and south; and the Northeast Extension Zone remains open to the north, south and east. Further diamond drilling is warranted to test for the possibility of additional limestone-hosted dyke zones to the north and south of the Main Ixtaca and Ixtaca North zones. Additional diamond drilling to the north and south along the hinge of axis of shale-cored antiforms at the Northeast Extension Zone and west of the Main Ixtaca and Ixtaca North zones is also warranted.

18 Recommendations

Based on the results of diamond drilling to date and the Maiden mineral resource estimate, additional drilling is warranted to expand the Ixtaca Deposit mineral resource. Further diamond drilling is should test the possibility of additional limestone-hosted dyke zones to the north and south of the Main Ixtaca and Ixtaca North zones. Additional diamond drilling to the north and south along the hinge of axis of shale-cored antiforms at the Northeast Extension Zone and west of the Main Ixtaca and Ixtaca North zones is also warranted.

Diamond drilling should include, but not be limited to, diamond drilling of an additional 40,000 metres to expand the Ixtaca Deposit mineral resource. The estimated cost to complete additional diamond drilling is $4,400,000 (Phase 1).

Concurrent with ongoing exploration of the Ixtaca Deposit, baseline environmental, hydro-geological and open pit optimization engineering studies should be initiated towards completion of a preliminary economic assessment (PEA). The estimated cost to complete engineering studies is $500,000 (Phase 2).

Table 18-1. Budget for Proposed 2013 Exploration, Tuligtic Project

Budget Item	Estimated Cost
Additional Diamond Drilling to Expand the Ixtaca Deposit Resource PHASE 1: Diamond Drilling 40,000 m (@ $110/metre all-up)	$4,400,000.00
TOTAL PHASE 1:	**$4,400,000.00**
Completion of Baseline Environmental, Hydro-geological and Open Pit Optimization PHASE 2: Baseline Environmental and Hydro-geological Engineering Study Open Pit Optimization Engineering Study	**$250,000.00** **$250,000.00**
TOTAL PHASE 2:	**$500,000.00**
Total Project Costs, Excluding GST	**$4,900,000.00**

19 Date and Signature Page

This Technical Report was prepared to NI 43-101 standards by the following Qualified Persons. The effective date of this report is March 13, 2013.

Kristopher J. Raffle, B.Sc., P.Geo.
APEX Geoscience Ltd.
Vancouver, British Columbia, Canada
March 13, 2013

(signed) *Gary H. Giroux*

Gary H. Giroux, P.Eng., MASc.
Giroux Consultants Ltd.
Vancouver, British Columbia, Canada
March 13, 2013

(signed) *Andrew Bamber*

Andrew Bamber, B.Sc. (Mech.), Ph.D. (Mining), P.Eng.
BC Mining Research Ltd.
Vancouver, British Columbia, Canada
March 13, 2013

20 Certificate of Author

20.1 K.J. Raffle Certificate of Author

I, Kristopher J. Raffle, residing in Vancouver British Columbia, do hereby certify that:

1. I am a principal of APEX Geoscience Ltd. ("APEX"), 200, 9797 – 45 Avenue, Edmonton, Alberta, Canada.

2. I am the author and responsible for all sections, except sections 13 and 14, of this Technical Report entitled: **"Technical Report on the Tuligtic Project, Puebla State, Mexico"**, and dated March 13, 2013 (the "Technical Report").

3. I am a graduate of The University of British Columbia, Vancouver, British Columbia with a B.Sc. in Geology (2000) and have practiced my profession continuously since 2000. I have supervised numerous exploration programs specific to low sulphidation epithermal gold-silver deposits having similar geologic characteristics to the Tuligtic Project throughout British Columbia, Canada; and Jalisco, Nayarit and Puebla States, Mexico. I am a Professional Geologist registered with APEGGA (Association of Professional Engineers, Geologists and Geophysicists of Alberta), and APEGBC (Association of Professional Engineers and Geoscientists of British Columbia) and I am a 'Qualified Person' in relation to the subject matter of this Technical Report.

4. I visited the Property that is the subject of this Report on October 17th, 2011 and September 23rd, 2012. I have no prior involvement with the Property.

5. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101. I have not received, nor do I expect to receive, any interest, directly or indirectly, in Almaden Minerals. I am not aware of any other information or circumstance that could interfere with my judgment regarding the preparation of the Technical Report.

7. I have read and understand National Instrument 43-101 and Form 43-101 F1 and the Report has been prepared in compliance with the instrument.

8. 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 Technical Report not misleading.

9. 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.

Dated this March 13, 2013

Vancouver British Columbia, Canada

PROFESSIONAL GEOLOGIST KRISTOPHER J. RAFFLE ALBERTA

Kristopher J. Raffle, B.Sc., P.Geo.

20.2 G.H. Giroux Certificate of Author

I, G.H. Giroux, of 982 Broadview Drive, North Vancouver, British Columbia, do hereby certify that:

1. I am a consulting geological engineer with an office at #1215 - 675 West Hastings Street, Vancouver, British Columbia.

2. I am a graduate of the University of British Columbia in 1970 with a B.A. Sc. and in 1984 with a M.A. Sc., both in Geological Engineering.

3. I am a member in good standing of the Association of Professional Engineers and Geoscientists of the Province of British Columbia.

4. I have practiced my profession continuously since 1970. I have had over 30 years' experience calculating mineral resources. I have previously completed resource estimations on a wide variety of precious metal deposits both in B.C. and around the world, many similar to the Ixtaca project.

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, past relevant work experience and affiliation with a professional association (as defined in NI 43-101), I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the preparation of Section 14 "Mineral Resource Estimate" of the technical report titled **"Technical Report on the Tuligtic Project, Puebla State, Mexico"**, and dated March 13, 2013 (the "Technical Report"). I have not visited the property.

7. I have not had prior involvement with the property that is the subject of the Technical Report.

8. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

9. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.

10. I have read NI 43-101, and the portions of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

Dated this March 13, 2013

(signed) *G. H. Giroux* [Sealed]

G. H. Giroux, P.Eng., MASc.

20.3 A. Bamber Certificate of Author

I, Andrew Bamber Ph.D., P. Eng., of 2315 West 13th Avenue, Vancouver, British Columbia do hereby certify that:

1. I am a Principal Engineer with BC Mining Research Ltd. with a business address at 2315 West 13th Avenue, Vancouver, British Columbia

2. I am a graduate of the the University of Cape Town, B.Sc.(Hons.),Mechanical Engineering, 1993; the University of British Columbia, M.A.Sc., Mining and Mineral Process Engineering, 2005; and the University of British Columbia, Ph.D., Mining Engineering, 2008. I have practiced my profession continuously since graduation.

3. I am member in good standing of the Engineering Council of South Africa, License # 990013.

4. 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. My relevant experience with respect to the Tuligtic Project includes over 14 years of experience in mining and mineral processing projects in Southern Africa, Canada and Central Asia. I have been a principal in several pre-feasibility and feasibility studies, including the Kroondal “K2” Platinum Project, the Mimosa Phase III Platinum Expansion, the Voskhod Chrome Project in Kazakhstan, the Pipe II Nickel scoping study for INCO Thompson, as well as numerous NI 43-101 preliminary assessments with specific reference to silver/lead/zinc deposits, including Murgor’s Hudvam and Wim projects, Selkirk’s Ruddock Creek project and Silver Corp’s Silvertip project.

5. I am responsible for authoring Section 13 “Mineral Processing and Metallurgical Testing” of the technical report titled **“Technical Report on the Tuligtic Project, Puebla State, Mexico”**, and dated March 13, 2013 (the “Technical Report”). I have not visited the property.

6. 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 Technical Report not misleading.

7. I am an independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.

8. I have had no prior involvement with the Tuligtic Project that is the subject of this technical report.

9. I have read NI 43-101 and Form 43-101 F1 and the Report has been prepared in compliance therewith.

Dated this March 13, 2013

(signed) *Andrew Bamber* [Sealed]

Andrew Bamber, B.Sc. (Mech.), Ph.D. (Mining), P.Eng.

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APPENDIX 1: List of Drill Holes on the Tuligtic Project

Holes outlining the Ixtaca Main Zone are highlighted

HOLE	EASTING	NORTHING	ELEVATION	Hole Length (m)
CA-11-001	619100.90	2176535.30	2302.30	410.87
CA-11-002	619148.11	2176789.80	2402.17	597.77
CA-11-003	619147.74	2176790.16	2403.33	575.46
CA-11-004	619154.90	2176474.60	2298.50	276.76
TU-10-001	618734.70	2176006.60	2247.50	349.91
TU-10-002	618751.50	2176045.20	2248.40	377.34
TU-10-003	618726.10	2175977.20	2244.40	391.67
TU-10-004	618753.70	2176128.70	2278.70	446.60
TU-10-005	618753.70	2176128.70	2278.70	490.12
TU-10-006	618834.80	2176219.10	2323.70	529.74
TU-10-007	618777.90	2175748.90	2245.40	442.54
TU-10-008	618644.40	2175987.60	2252.10	559.61
TU-10-009	618646.40	2176057.90	2264.60	341.90
TU-10-010	618646.60	2175990.60	2252.60	611.43
TU-10-011	618790.20	2176155.60	2277.70	458.72
TU-10-012	618751.50	2176045.20	2248.40	544.98
TU-10-013	618790.20	2176155.60	2277.70	559.07
TU-10-014	618751.50	2176037.40	2246.44	361.49
TU-11-015	618916.80	2176140.30	2252.20	291.39
TU-11-016	618978.70	2175835.20	2375.70	480.36
TU-11-017	618916.80	2176140.30	2252.20	468.78
TU-11-018	618964.10	2176158.20	2253.50	302.97
TU-11-019	618978.70	2175835.20	2375.70	455.98
TU-11-020	618964.10	2176158.20	2253.50	356.86
TU-11-021	619004.50	2176206.60	2255.00	319.43
TU-11-022	619004.50	2176206.60	2255.00	392.58
TU-11-023	618793.40	2175702.98	2243.80	465.12
TU-11-024	619002.30	2176209.90	2255.10	389.53
TU-11-025	619260.60	2176009.30	2382.10	438.42
TU-11-026	619055.30	2176223.60	2253.30	319.43
TU-11-027	619092.80	2176248.00	2255.20	340.46
TU-11-028	618659.20	2175993.80	2250.50	282.24
TU-11-029	618863.25	2176122.30	2244.04	324.31
TU-11-030	618602.40	2175894.08	2246.20	230.43
TU-11-031	618806.97	2176043.89	2242.90	344.12
TU-11-032	619154.90	2176474.60	2298.50	356.01
TU-11-033	618509.50	2176044.90	2285.40	406.60
TU-11-034	618779.10	2175987.80	2243.30	316.38
TU-11-035	618700.72	2176020.35	2245.20	401.12

TU-11-036	618745.96	2175925.12	2242.21	166.73
TU-11-037	618512.46	2175852.96	2263.82	437.69
TU-11-038	618739.65	2175798.95	2241.21	285.90
TU-11-039	618962.37	2176161.65	2252.40	263.04
TU-11-040	618450.56	2176157.40	2298.56	198.12
TU-11-041	619241.11	2176587.53	2327.99	569.37
TU-11-042	618244.68	2175915.65	2269.83	639.26
TU-11-043	619311.04	2176678.66	2374.59	407.82
TU-11-044	619100.90	2176535.30	2302.30	276.76
TU-11-045	618791.29	2175575.38	2231.13	480.36
TU-11-046	619241.11	2176587.53	2327.99	301.14
TU-11-047	619161.37	2176320.10	2262.40	243.23
TU-11-048	618916.80	2176140.30	2252.20	365.15
TU-11-049	619091.07	2175947.99	2410.11	465.12
TU-11-050	619164.04	2176319.31	2263.80	304.19
TU-11-051	618914.70	2176144.40	2250.88	316.38
TU-11-052	619091.27	2176252.37	2253.45	167.03
TU-11-053	618863.70	2176122.61	2244.04	410.87
TU-11-054	619040.03	2176028.18	2392.35	471.22
TU-11-055	619052.21	2176227.51	2251.21	231.04
TU-11-056	618829.90	2176092.90	2243.06	392.58
TU-11-057	618806.97	2176043.89	2242.90	480.97
TU-11-058	619082.10	2176028.70	2385.65	187.76
TU-11-059	618979.23	2175834.90	2371.00	701.34
TU-11-060	618758.23	2175983.00	2237.90	176.17
TU-11-061	618743.77	2175929.00	2239.70	420.01
TU-11-062	618758.23	2175983.00	2237.90	292.00
TU-11-063	618795.80	2175650.00	2232.90	432.21
TU-11-064	618782.92	2175888.24	2260.66	285.90
TU-11-065	618754.18	2175860.52	2243.76	420.01
TU-11-066	618979.23	2175834.90	2371.00	630.02
TU-11-067	618730.44	2175904.32	2237.56	261.52
TU-11-068	618803.94	2175953.38	2269.96	234.09
TU-11-069	618749.80	2175736.77	2237.57	465.73
TU-11-070	618832.54	2175999.74	2271.01	319.43
TU-11-071	618820.40	2175620.41	2236.10	255.42
TU-11-072	619022.54	2175897.56	2403.24	486.46
TU-11-073	618832.51	2175901.98	2300.06	219.15
TU-11-074	618819.30	2175495.40	2234.40	288.95
TU-11-075	618792.10	2175575.61	2227.00	477.93
TU-11-076	618851.70	2175955.88	2294.90	238.66
TU-11-077	618795.50	2175440.40	2236.30	453.54
TU-11-078	618877.90	2176036.30	2312.20	309.68

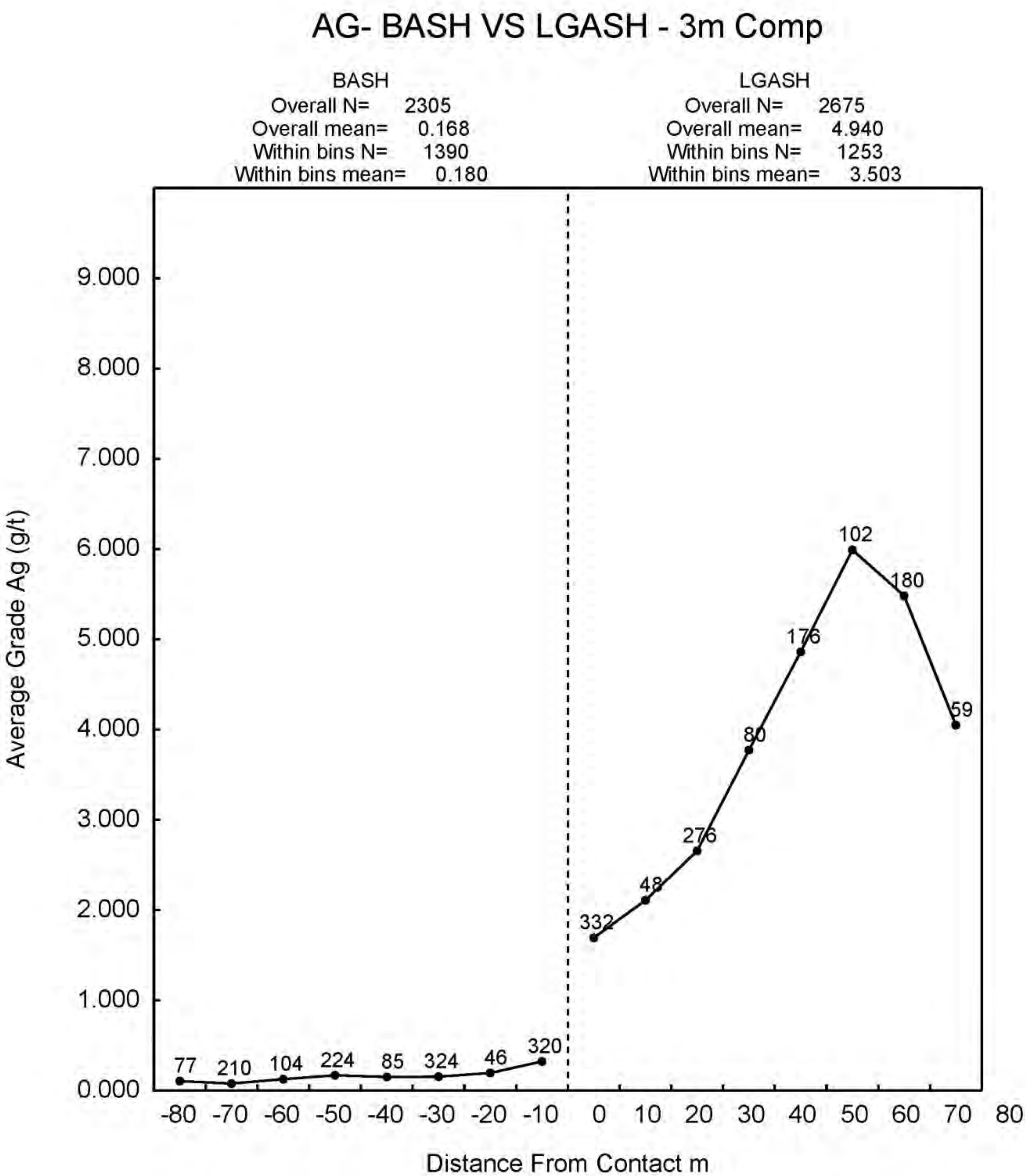
TU-11-079	619035.90	2175935.80	2409.90	359.66
TU-11-080	619795.60	2175994.20	2393.60	432.21
TU-11-081	618913.60	2176081.90	2320.80	325.53
TU-11-082	619035.70	2175937.80	2408.90	462.08
TU-11-083	618831.60	2176091.70	2247.08	365.15
TU-11-084	619302.70	2176484.90	2331.90	429.16
TU-11-085	619089.90	2175950.80	2413.90	532.18
TU-11-086	618913.60	2176081.90	2320.80	288.95
TU-11-087	619301.40	2176485.60	2330.70	298.09
TU-11-088	618831.80	2176091.40	2246.50	517.55
TU-11-089	619088.50	2175950.10	2413.10	221.28
TU-11-090	619240.50	2176626.30	2321.00	243.23
TU-11-091	618937.70	2176081.90	2322.50	274.76
TU-11-092	619091.20	2175948.70	2413.70	239.57
TU-11-093	619238.90	2176628.90	2320.70	209.70
TU-11-094	619198.10	2176586.50	2309.80	246.28
TU-11-095	618937.70	2176081.90	2322.50	224.94
TU-12-096	618883.70	2176125.60	2251.52	401.73
TU-12-097	618977.90	2176157.10	2250.00	413.92
TU-12-098	619235.90	2176510.50	2326.96	404.77
TU-12-099	619151.20	2176032.30	2396.50	474.27
TU-12-100	619235.90	2176510.50	2326.96	267.61
TU-12-101	618883.70	2176125.60	2251.52	538.89
TU-12-102	618964.10	2176158.20	2253.50	292.00
TU-12-103	619232.80	2176513.50	2325.50	401.73
TU-12-104	618964.10	2176158.20	2253.50	264.57
TU-12-105	618791.30	2175575.40	2231.13	346.25
TU-12-106	619235.90	2176510.50	2326.40	343.20
TU-12-107	618919.10	2176136.80	2254.90	465.73
TU-12-108	619040.90	2176208.50	2258.70	325.53
TU-12-109	619235.90	2176510.50	2326.40	368.20
TU-12-110	618450.80	2176157.50	2305.00	331.01
TU-12-111	619044.60	2176208.50	2254.10	295.05
TU-12-112	619000.50	2176193.30	2253.20	413.92
TU-12-113	619237.70	2176515.40	2333.40	325.53
TU-12-114	618510.00	2176047.30	2288.90	425.50
TU-12-115	619044.60	2176208.50	2254.10	365.15
TU-12-116	619299.20	2176482.80	2330.80	197.51
TU-12-117	619000.50	2176193.30	2253.20	307.24
TU-12-118	618510.00	2176047.30	2288.90	321.87
TU-12-119	618685.90	2176257.90	2374.10	615.09
TU-12-120	618940.60	2176142.30	2257.40	331.62
TU-12-121	619000.50	2176193.30	2253.20	267.61

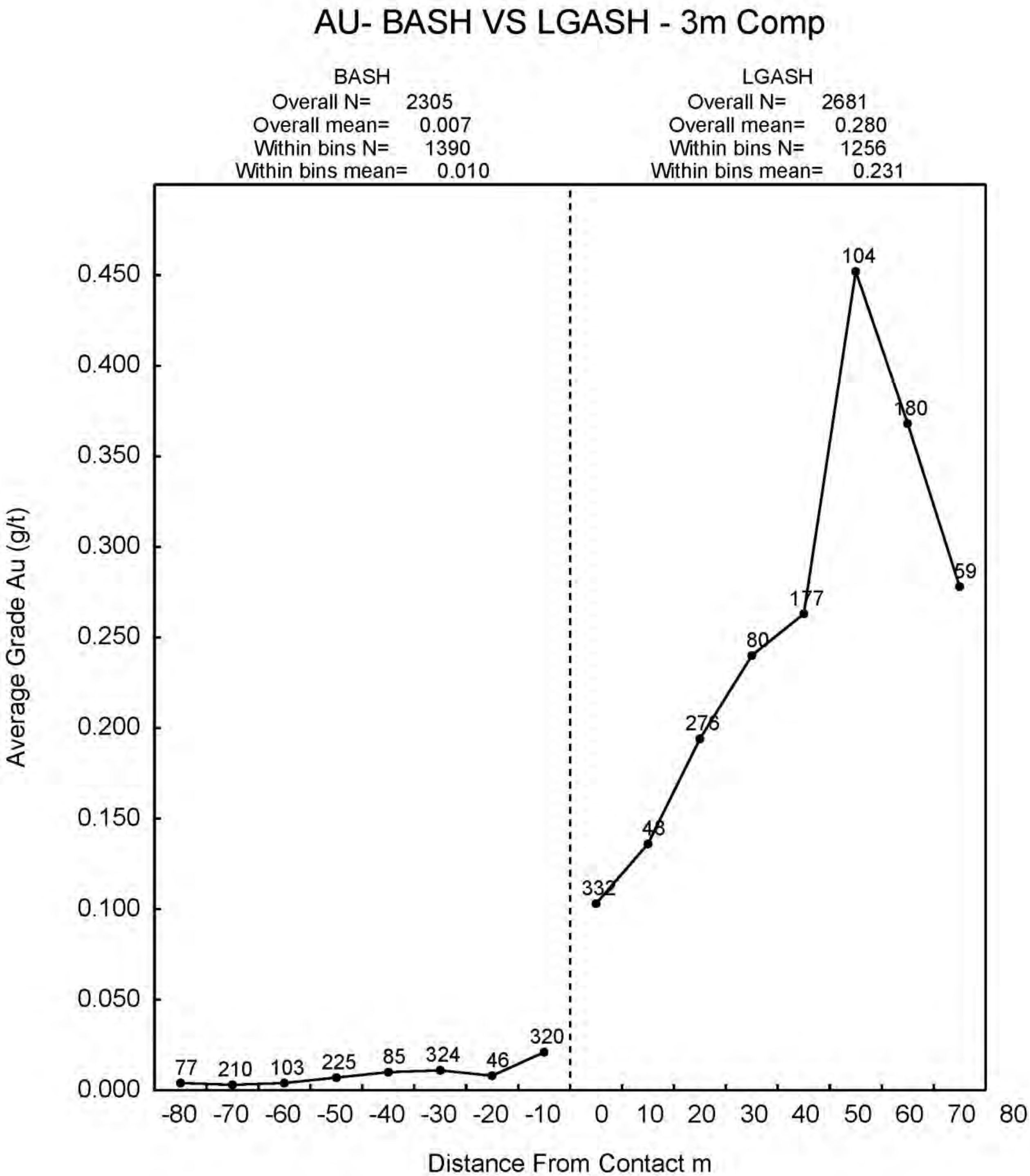
TU-12-122	618506.50	2175961.00	2283.00	395.02
TU-12-123	618813.10	2176076.20	2247.10	356.01
TU-12-124	618940.60	2176142.30	2257.40	356.01
TU-12-125	618693.04	2176334.10	2376.90	404.77
TU-12-126	618813.10	2176076.20	2247.10	393.19
TU-12-127	618940.60	2176142.30	2257.40	420.01
TU-12-128	618506.50	2175961.00	2283.00	425.50
TU-12-129	618732.40	2176365.60	2377.80	444.40
TU-12-130	618813.10	2176076.20	2247.10	288.95
TU-12-131	618506.50	2175961.00	2283.00	431.60
TU-12-132	618940.60	2176142.30	2257.40	273.71
TU-12-133	618813.10	2176076.20	2247.10	261.52
TU-12-134	618732.40	2176365.60	2377.80	438.30
TU-12-135	618813.10	2176076.20	2247.10	438.30
TU-12-136	618939.90	2176143.10	2252.90	185.32
TU-12-137	618621.50	2175965.70	2247.90	331.01
TU-12-138	618834.20	2176293.00	2358.80	404.77
TU-12-139	618705.70	2175991.60	2247.70	349.30
TU-12-140	619082.70	2176389.60	2274.40	218.85
TU-12-141	618544.70	2175894.40	2263.20	362.10
TU-12-142	618705.70	2175991.60	2247.70	443.79
TU-12-143	619082.70	2176389.60	2274.40	200.56
TU-12-144	618834.20	2176293.00	2358.80	307.24
TU-12-145	619051.20	2176453.70	2295.50	441.35
TU-12-146	618705.70	2175991.60	2247.70	248.72
TU-12-147	618564.10	2175964.80	2256.90	296.57
TU-12-148	618705.70	2175991.60	2247.70	312.72
TU-12-149	618853.10	2176343.20	2353.70	340.77
TU-12-150	618677.90	2175882.90	2245.30	294.44
TU-12-151	619051.20	2176453.70	2295.50	392.58
TU-12-152	618563.20	2176043.90	2268.10	319.43
TU-12-153	618613.80	2176265.30	2348.10	334.67
TU-12-154	618646.60	2175813.20	2239.60	259.38
TU-12-155	619051.20	2176453.70	2295.50	380.39
TU-12-156	618673.20	2175759.90	2238.70	270.05
TU-12-157	618518.50	2176161.10	2312.30	423.06
TU-12-158	618639.10	2175999.90	2252.50	145.69
TU-12-159	619051.20	2176453.20	2295.50	371.25
TU-12-160	618640.40	2175720.50	2239.40	382.83
TU-12-161	618914.70	2176351.30	2330.00	282.85
TU-12-162	619051.20	2176453.20	2295.50	395.63
TU-12-163	618469.30	2175923.20	2277.70	432.21
TU-12-164	618730.70	2176004.10	2244.50	327.96

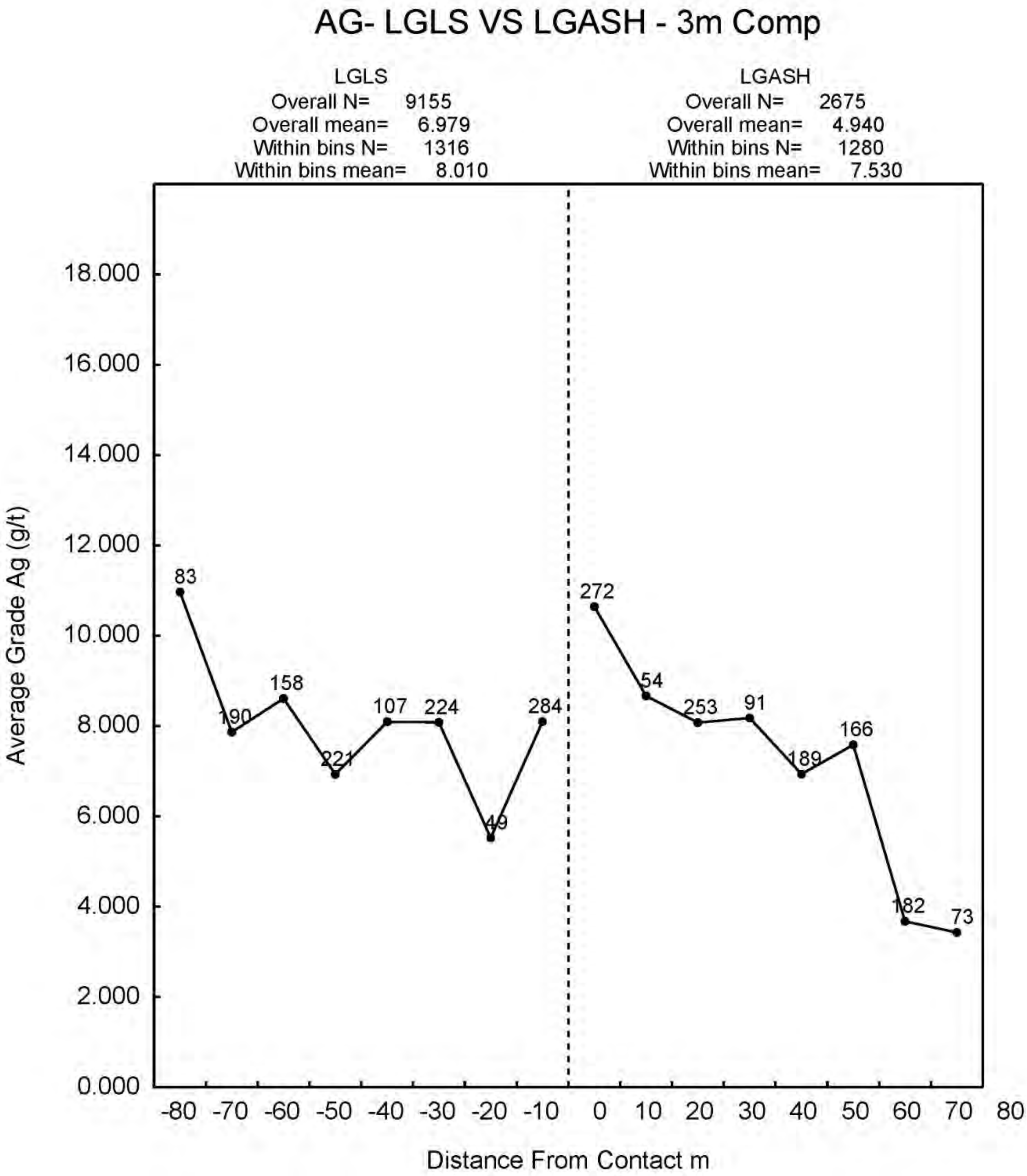
TU-12-165	618914.70	2176351.30	2330.00	407.82
TU-12-166	619051.20	2176453.20	2295.50	453.54
TU-12-167	618405.00	2176026.00	2267.90	487.07
TU-12-168	618734.10	2176005.90	2246.50	373.68
TU-12-169	618946.40	2176414.40	2308.50	413.92
TU-12-170	618984.30	2176547.10	2323.60	392.58
TU-12-171	618435.90	2175974.50	2272.00	444.40
TU-12-172	618745.60	2176037.90	2246.00	571.80
TU-12-173	618946.40	2176414.40	2308.50	416.97
TU-12-174	618984.30	2176547.10	2323.60	407.82
TU-12-175	619001.70	2176403.90	2299.00	313.33
TU-12-176	618407.50	2176026.90	2272.60	535.84
TU-12-177	618604.70	2175820.10	2247.40	416.36
TU-12-178	618984.30	2176547.10	2323.60	426.11
TU-12-179	619001.70	2176403.90	2299.00	349.91
TU-12-180	618984.30	2176547.10	2323.60	420.01
TU-12-181	619001.70	2176403.90	2299.00	224.94
TU-12-182	618569.60	2175756.10	2245.50	446.84
TU-12-183	618408.31	2176025.50	2272.60	264.57
TU-12-184	618982.70	2176546.50	2323.60	434.04
TU-12-185	618408.31	2176025.50	2272.60	167.03
TU-12-186	619166.30	2176320.60	2262.00	352.96
TU-12-187	618408.00	2176026.90	2272.60	200.56
TU-12-188	618416.10	2175932.00	2273.80	443.79
TU-12-189	618404.50	2176024.40	2270.90	490.12
TU-12-190	619006.00	2176498.30	2312.40	413.92
TU-12-191	619165.40	2176319.80	2265.30	395.63
TU-12-192	618446.00	2175860.50	2273.00	316.38
TU-12-193	618427.70	2176204.10	2302.30	130.45
TU-12-194	619006.00	2176498.30	2312.30	407.82
TU-12-195	618427.70	2176204.10	2302.30	325.53
TU-12-196	619074.90	2176389.50	2271.00	383.44
TU-12-197	618423.40	2176205.70	2302.30	215.80
TU-12-198	618417.50	2176112.00	2286.90	316.38
TU-12-199	619006.00	2176498.30	2312.30	480.97
TU-12-200	618417.50	2176112.00	2286.90	160.93
TU-12-201	619074.90	2176389.50	2271.00	413.92
TU-12-202	618568.40	2176189.60	2327.10	484.02
TU-12-203	618414.40	2176115.20	2286.90	182.27
TU-12-204	619074.90	2176389.50	2271.00	453.54
TU-12-205	619002.20	2176499.80	2312.80	368.20
TU-12-206	618675.70	2176200.30	2361.70	205.13
TU-12-207	618565.40	2176189.80	2326.70	263.96

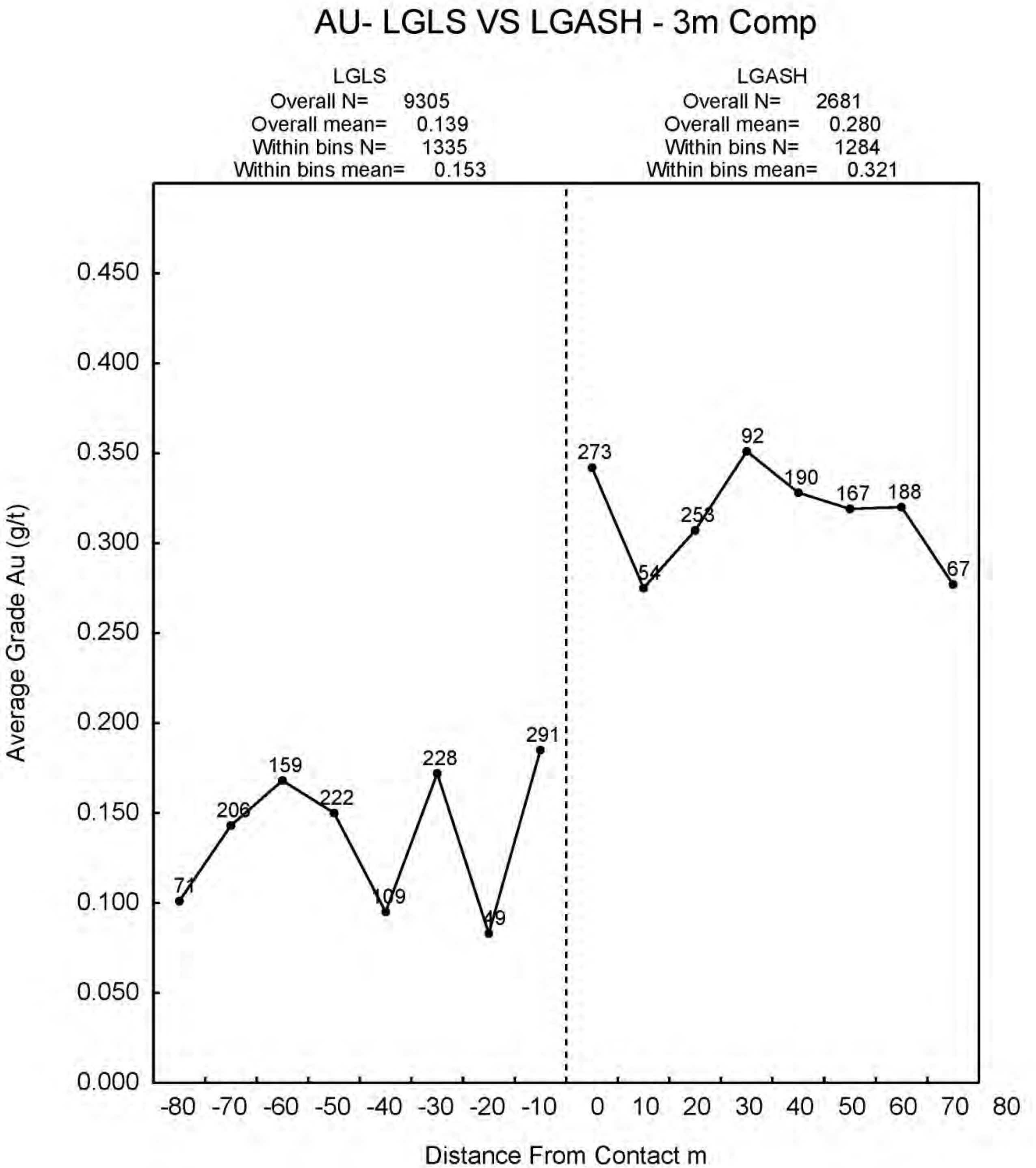
TU-12-208	619083.80	2176389.60	2271.00	368.20
TU-12-209	618675.70	2176200.30	2361.70	258.47
TU-12-210	619049.20	2176453.30	2291.60	319.43
TU-12-211	618703.40	2175953.70	2242.50	322.48
TU-12-212	618808.70	2176079.40	2244.90	313.33
TU-12-213	619214.50	2176220.80	2298.40	304.19
TU-12-214	619046.70	2176450.80	2292.50	337.72
TU-12-215	618948.30	2176416.70	2307.90	605.94
TU-12-216	619214.50	2176220.80	2298.40	404.77
TU-12-217	618808.70	2176079.40	2244.90	235.61
TU-12-218	619050.70	2176453.90	2287.90	295.05
TU-12-219	619211.60	2176220.30	2301.80	203.61
TU-12-220	619211.60	2176220.30	2301.80	282.85
TU-12-221	618948.30	2176416.70	2307.90	548.03

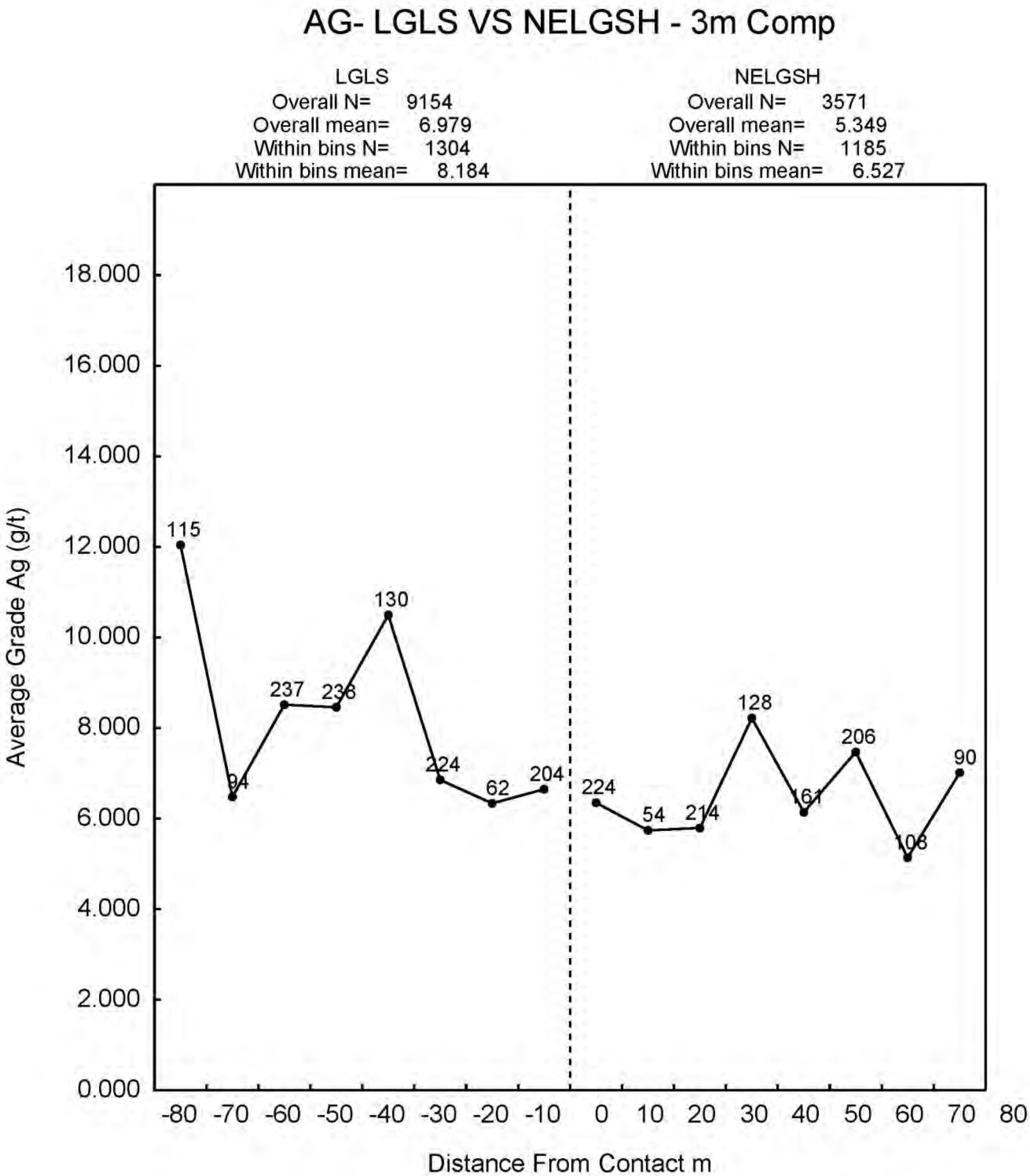
APPENDIX 2: Contact Plots

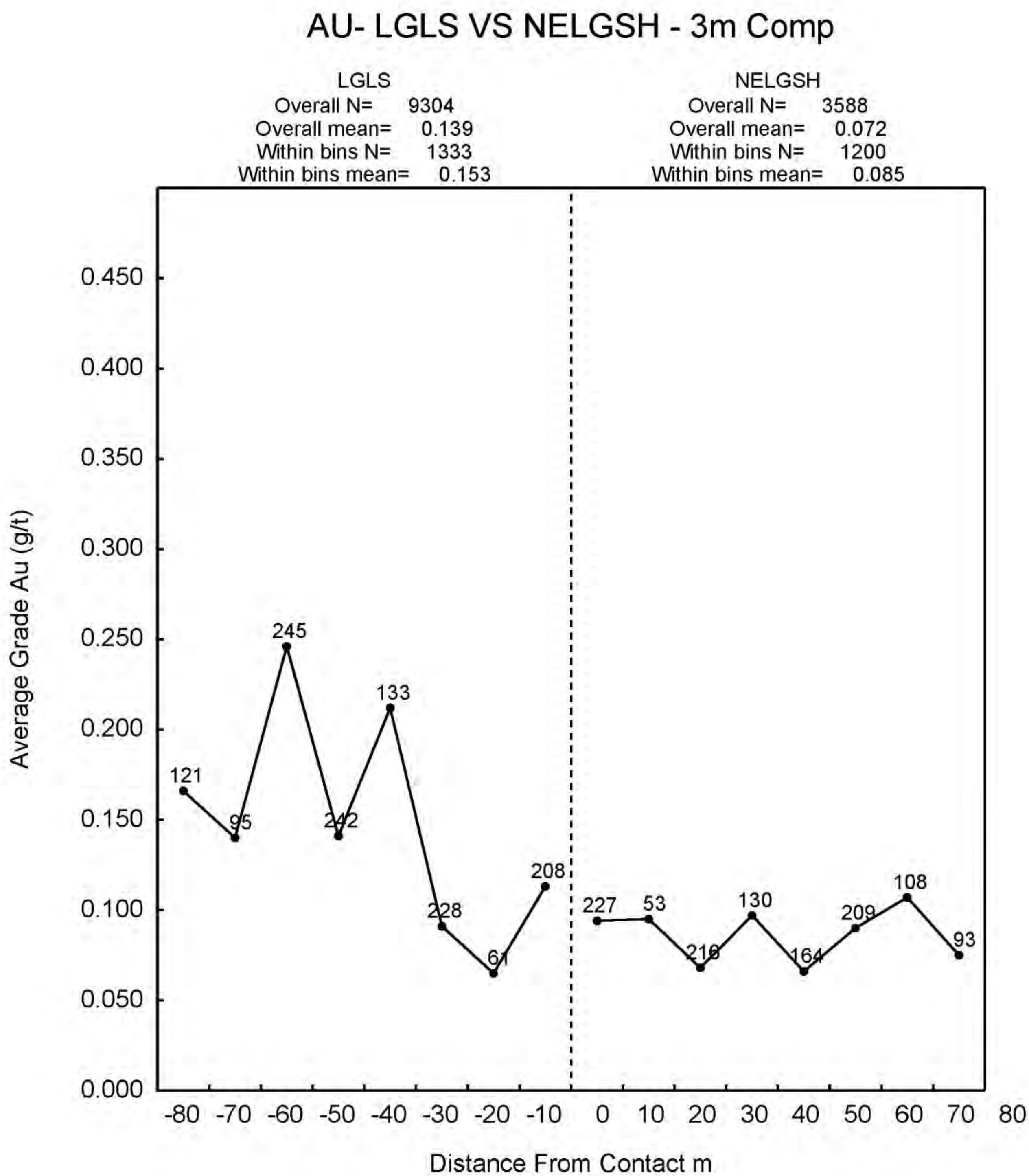

AG- BASH VS LGASH - 3m Comp
BASH
Overall N= 2305
Overall mean= 0.168
Within bins N= 1390
Within bins mean= 0.180
LGASH
Overall N= 2675
Overall mean= 4.940
Within bins N= 1253
Within bins mean= 3.503
Average Grade Ag (g/t)
0.000
1.000
2.000
3.000
4.000
5.000
6.000
7.000
8.000
9.000
77
210
104
224
85
324
46
320
332
48
276
80
176
102
180
59
-80 -70 -60 -50 -40 -30 -20 -10 0 10 20 30 40 50 60 70 80
Distance From Contact m

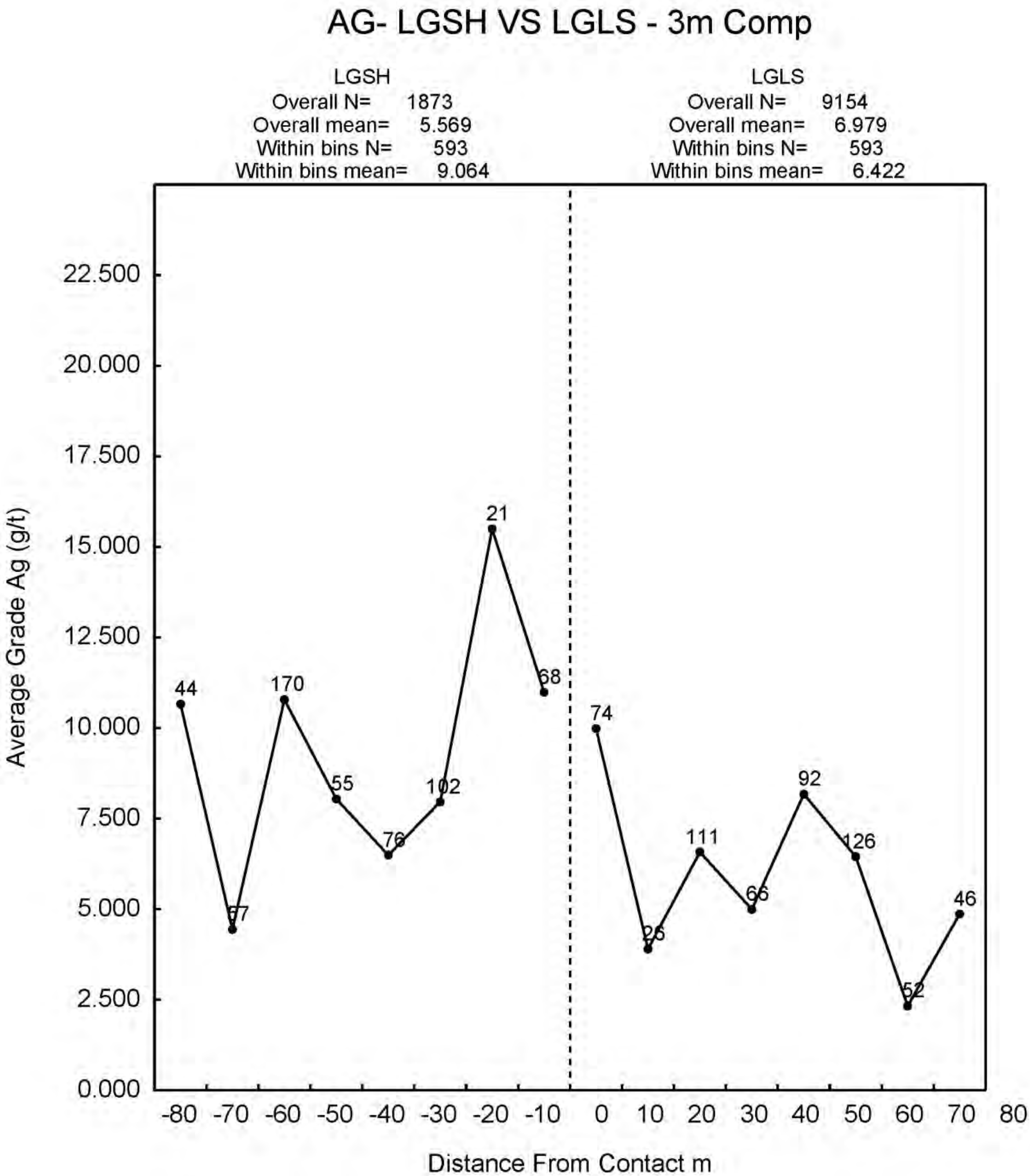

AU- BASH VS LGASH - 3m Comp
BASH
Overall N= 2305
Overall mean= 0.007
Within bins N= 1390
Within bins mean= 0.010
LGASH
Overall N= 2681
Overall mean= 0.280
Within bins N= 1256
Within bins mean= 0.231
Average Grade Au (g/t)
0.000
0.050
0.100
0.150
0.200
0.250
0.300
0.350
0.400
0.450
77
210
103
225
85
324
46
320
332
48
276
80
177
104
180
59
-80 -70 -60 -50 -40 -30 -20 -10 0 10 20 30 40 50 60 70 80
Distance From Contact m

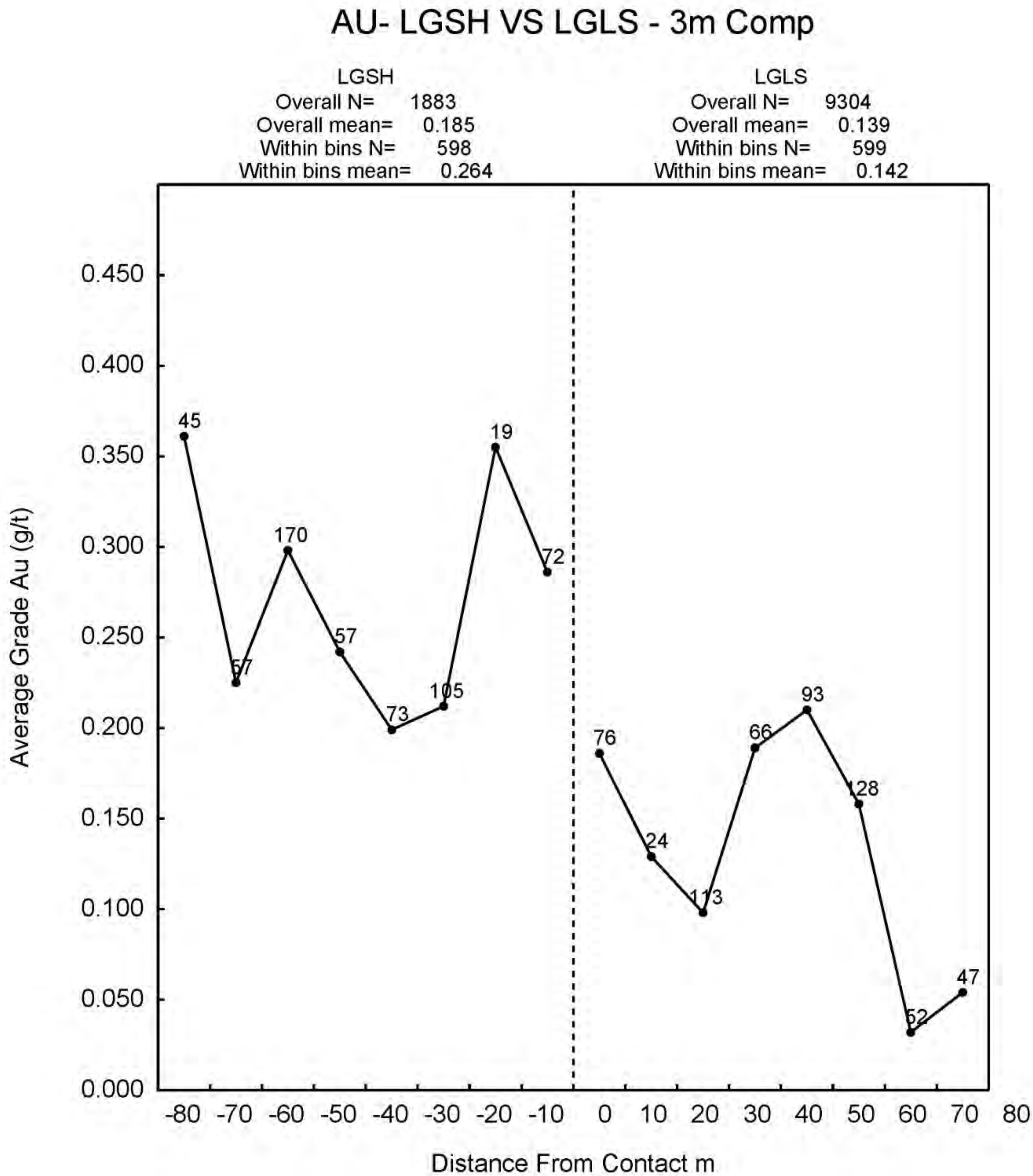

AG- LGLS VS LGASH - 3m Comp
LGLS
Overall N= 9155
Overall mean= 6.979
Within bins N= 1316
Within bins mean= 8.010
LGASH
Overall N= 2675
Overall mean= 4.940
Within bins N= 1280
Within bins mean= 7.530
Average Grade Ag (g/t)
0.000
2.000
4.000
6.000
8.000
10.000
12.000
14.000
16.000
18.000
83
190
158
221
107
224
49
284
272
54
253
91
189
166
182
73
-80 -70 -60 -50 -40 -30 -20 -10 0 10 20 30 40 50 60 70 80
Distance From Contact m


AU- LGLS VS LGASH - 3m Comp
LGLS
Overall N= 9305
Overall mean= 0.139
Within bins N= 1335
Within bins mean= 0.153
LGASH
Overall N= 2681
Overall mean= 0.280
Within bins N= 1284
Within bins mean= 0.321
Average Grade Au (g/t)
0.000
0.050
0.100
0.150
0.200
0.250
0.300
0.350
0.400
0.450
71
206
159
222
109
228
49
291
273
54
253
92
190
167
188
67
-80 -70 -60 -50 -40 -30 -20 -10 0 10 20 30 40 50 60 70 80
Distance From Contact m

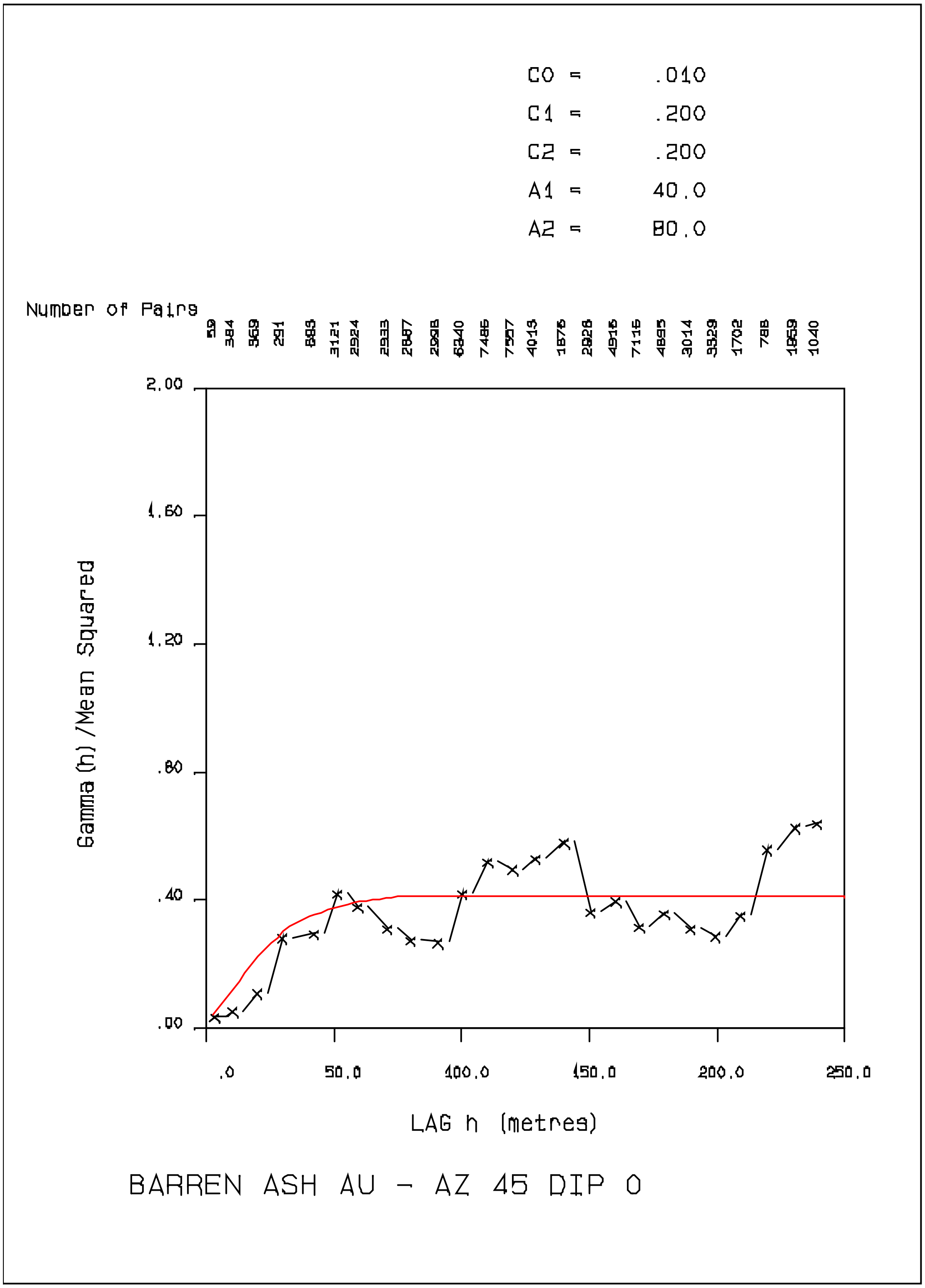

AG- LGLS VS NELGSH - 3m Comp
LGLS
Overall N= 9154
Overall mean= 6.979
Within bins N= 1304
Within bins mean= 8.184
NELGSH
Overall N= 3571
Overall mean= 5.349
Within bins N= 1185
Within bins mean= 6.527
Average Grade Ag (g/t)
18.000
16.000
14.000
12.000
10.000
8.000
6.000
4.000
2.000
0.000
115
94
237
238
130
224
62
204
224
54
214
128
161
206
108
90
-80 -70 -60 -50 -40 -30 -20 -10 0 10 20 30 40 50 60 70 80
Distance From Contact m

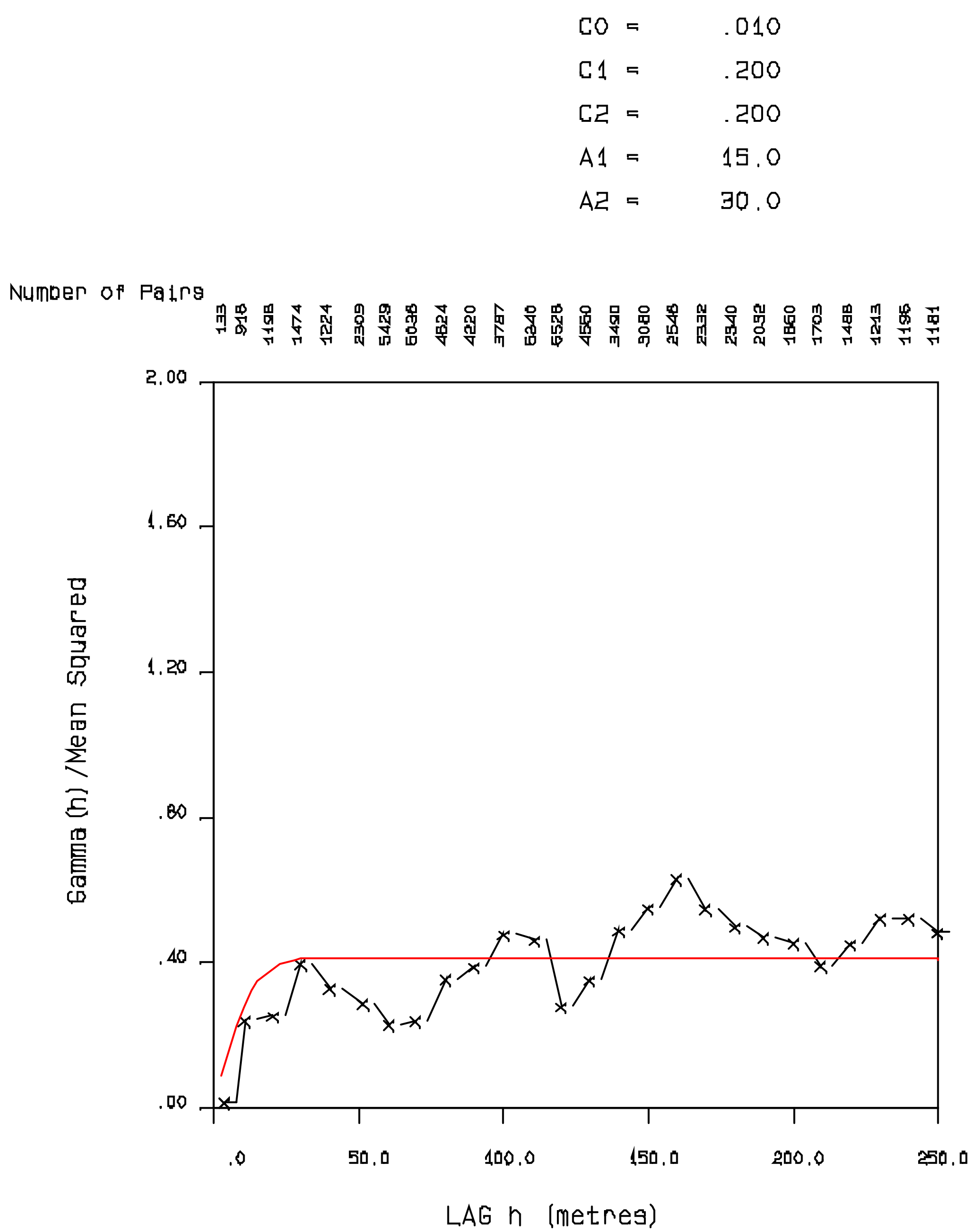

AU- LGLS VS NELGSH - 3m Comp
LGLS
Overall N= 9304
Overall mean= 0.139
Within bins N= 1333
Within bins mean= 0.153
NELGSH
Overall N= 3588
Overall mean= 0.072
Within bins N= 1200
Within bins mean= 0.085
Average Grade Au (g/t)
0.000
0.050
0.100
0.150
0.200
0.250
0.300
0.350
0.400
0.450
121
95
245
242
133
228
61
208
227
53
216
130
164
209
108
93
-80 -70 -60 -50 -40 -30 -20 -10 0 10 20 30 40 50 60 70 80
Distance From Contact m


AG- LGSH VS LGLS - 3m Comp
LGSH
Overall N= 1873
Overall mean= 5.569
Within bins N= 593
Within bins mean= 9.064
LGLS
Overall N= 9154
Overall mean= 6.979
Within bins N= 593
Within bins mean= 6.422
Average Grade Ag (g/t)
0.000
2.500
5.000
7.500
10.000
12.500
15.000
17.500
20.000
22.500
44
67
170
55
76
102
21
68
74
26
111
66
92
126
52
46
-80 -70 -60 -50 -40 -30 -20 -10 0 10 20 30 40 50 60 70 80
Distance From Contact m

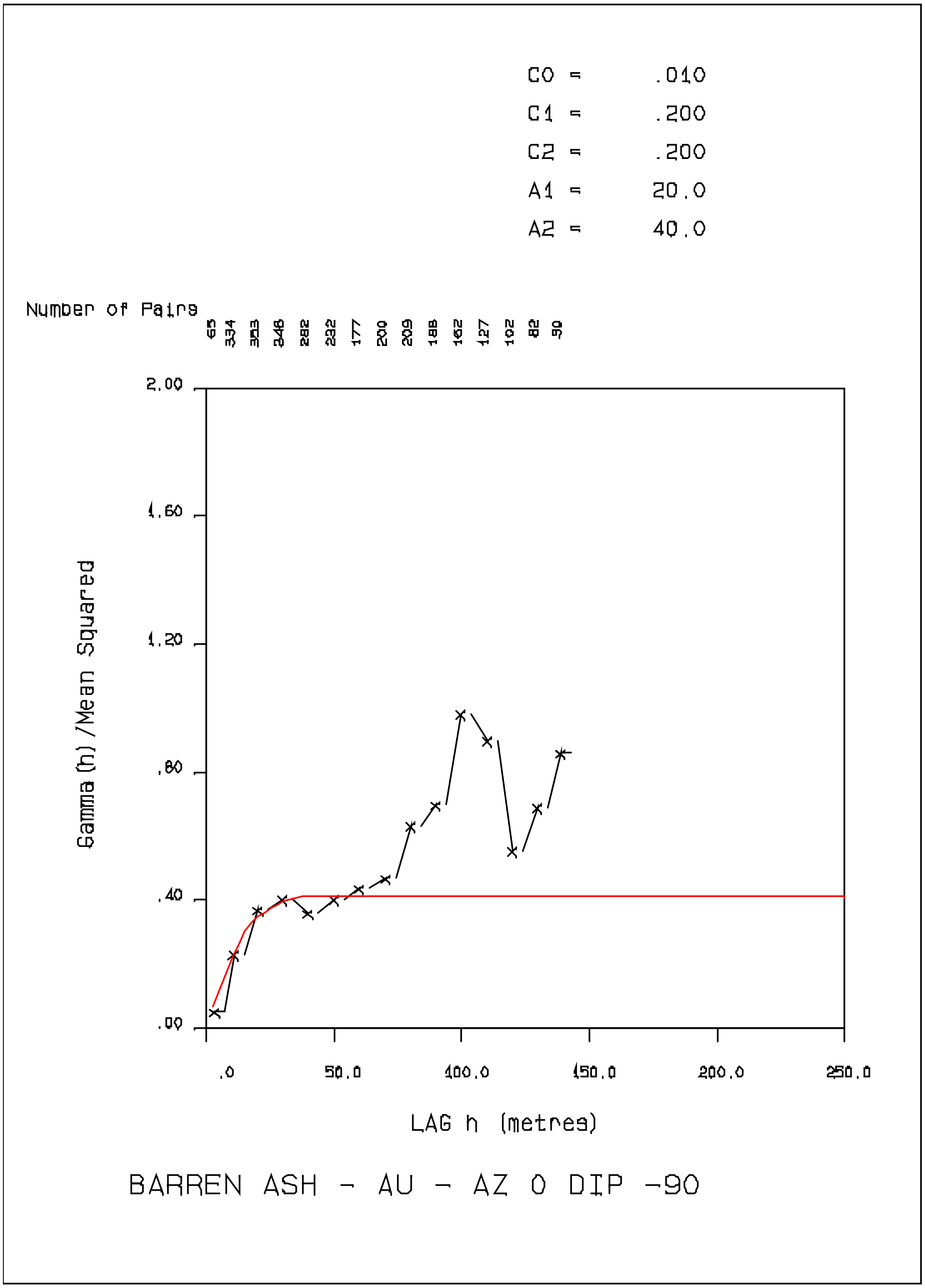

AU- LGSH VS LGLS - 3m Comp
LGSH
Overall N= 1883
Overall mean= 0.185
Within bins N= 598
Within bins mean= 0.264
LGLS
Overall N= 9304
Overall mean= 0.139
Within bins N= 599
Within bins mean= 0.142
Average Grade Au (g/t)
0.000
0.050
0.100
0.150
0.200
0.250
0.300
0.350
0.400
0.450
45
57
170
57
73
105
19
72
76
24
113
66
93
128
52
47
-80 -70 -60 -50 -40 -30 -20 -10 0 10 20 30 40 50 60 70 80
Distance From Contact m

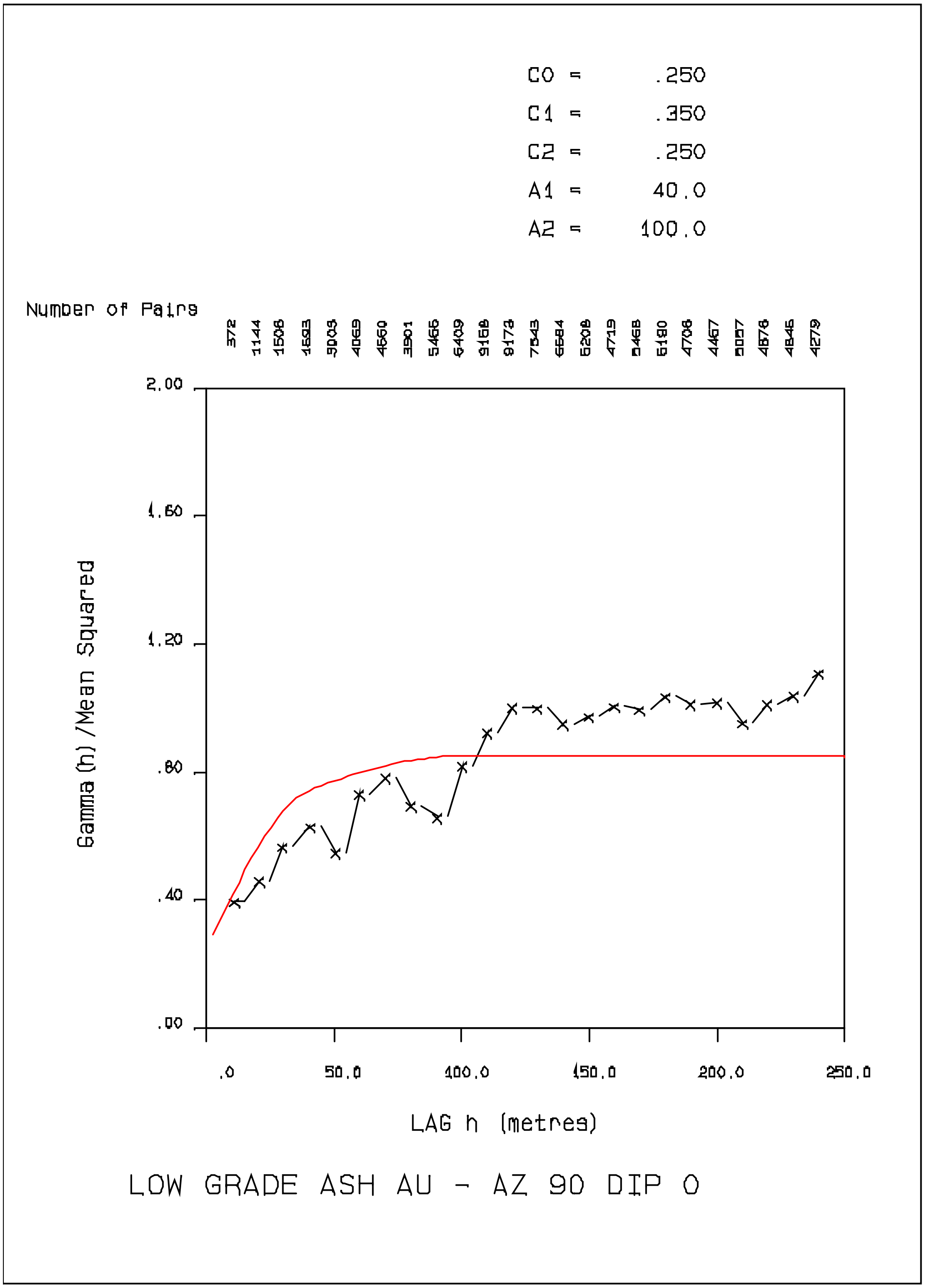
APPENDIX 3: Semivariogram Models for Gold in Each Domain

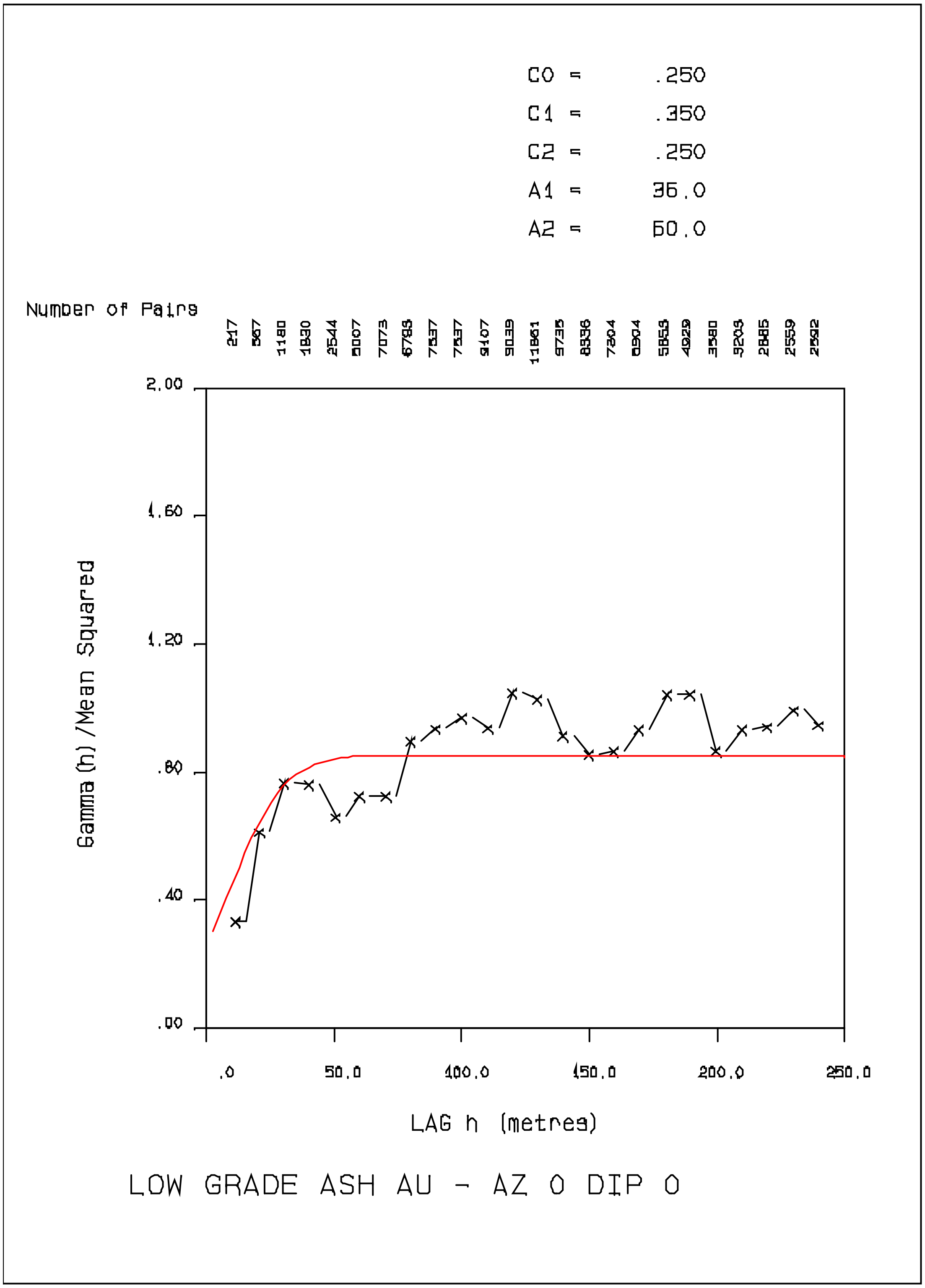

C0 = .010
C1 = .200
C2 = .200
A1 = 40.0
A2 = 80.0
Number of Pairs
384
3121
2924
2933
7486
4013
3014
1702
788
1040
2.00
1.60
1.20
.80
.40
.00
Gamma (h) /Mean Squared
.0
50.0
100.0
150.0
200.0
250.0
LAG h (metres)
BARREN ASH AU - AZ 45 DIP 0

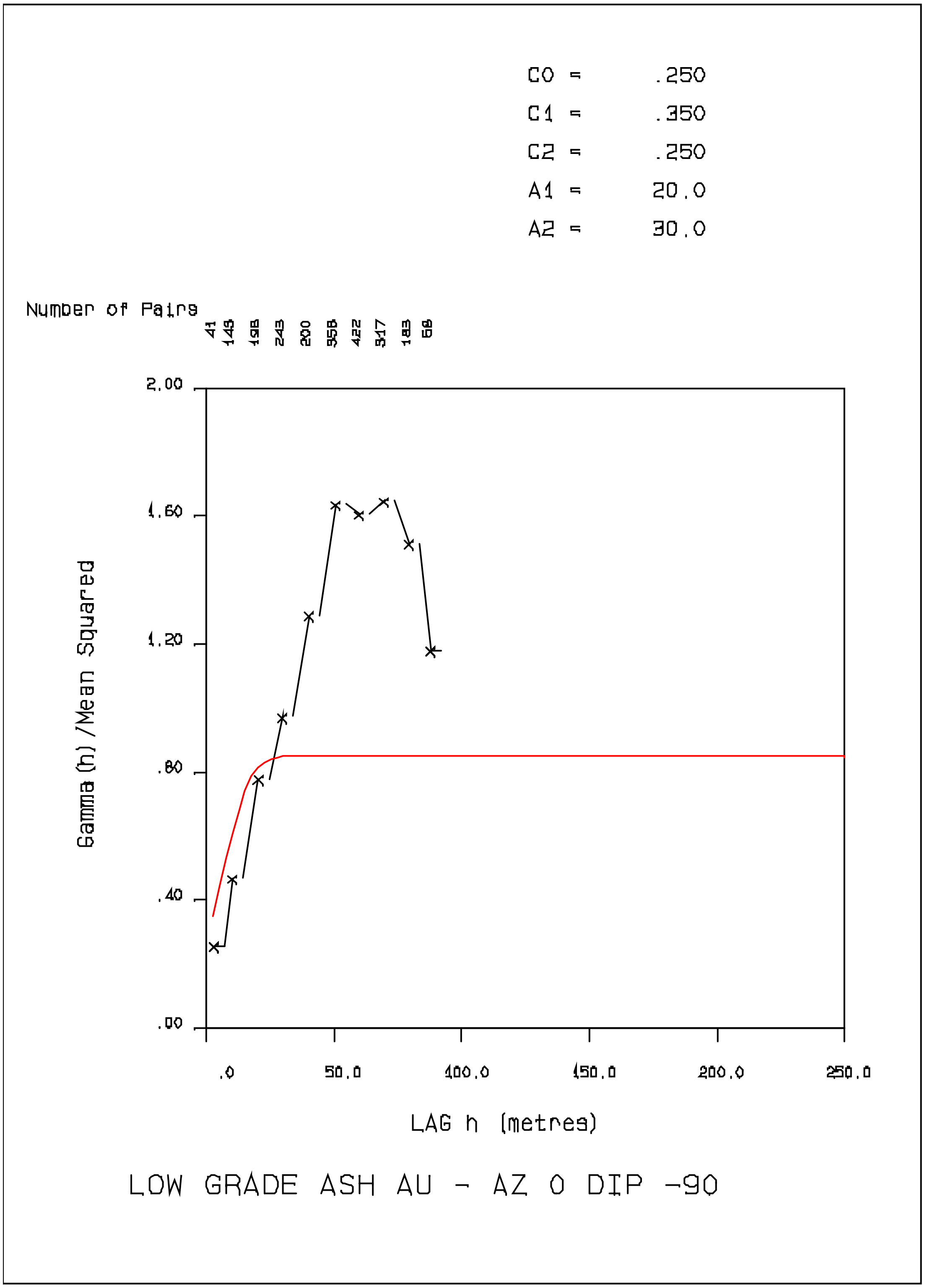


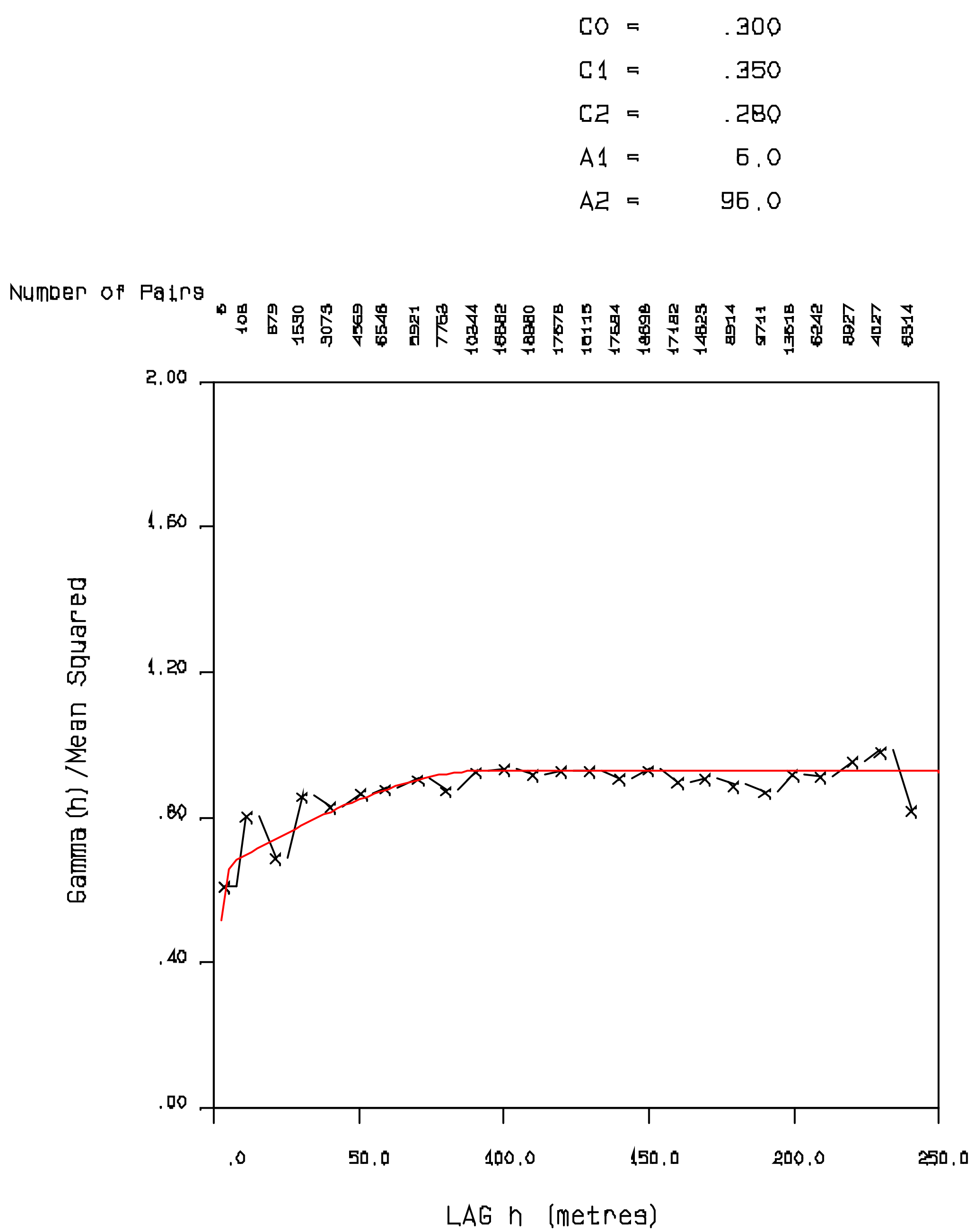
BARREN ASH AU - AZ 135 DIP 0


C0 = .010
C1 = .200
C2 = .200
A1 = 20.0
A2 = 40.0
Number of Pairs
65 334 353 346 282 232 177 200 209 188 162 127 102 82 39
2.00
1.60
1.20
.80
.40
.00
Gamma (h) /Mean Squared
.0
50.0
100.0
150.0
200.0
250.0
LAG h (metres)
BARREN ASH - AU - AZ 0 DIP -90

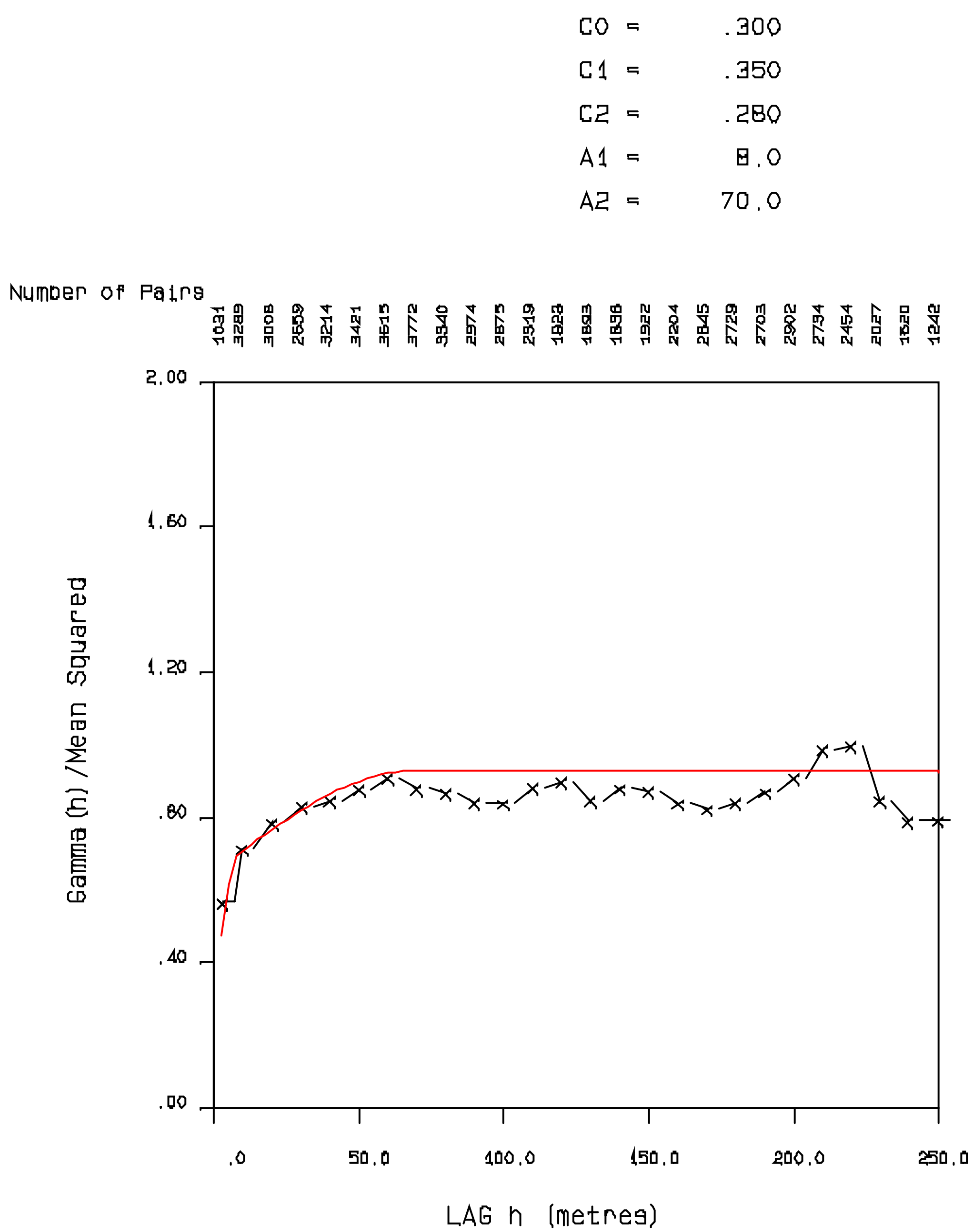

C0 = .250
C1 = .350
C2 = .250
A1 = 40.0
A2 = 100.0
Number of Pairs
2.00
1.60
1.20
.80
.40
.00
Gamma (h) /Mean Squared
.0
50.0
100.0
150.0
200.0
250.0
LAG h (metres)
LOW GRADE ASH AU - AZ 90 DIP 0


C0 = .250
C1 = .350
C2 = .250
A1 = 36.0
A2 = 60.0
Number of Pairs
217 567 1180 1830 2544 5007 7073 6793 7537 7537 9407 9039 11861 9735 8336 7304 6904 5853 4929 3580 3203 2885 2559 2592
2.00
1.60
1.20
.80
.40
.00
Gamma (h) /Mean Squared
.0
50.0
100.0
150.0
200.0
250.0
LAG h (metres)
LOW GRADE ASH AU - AZ 0 DIP 0

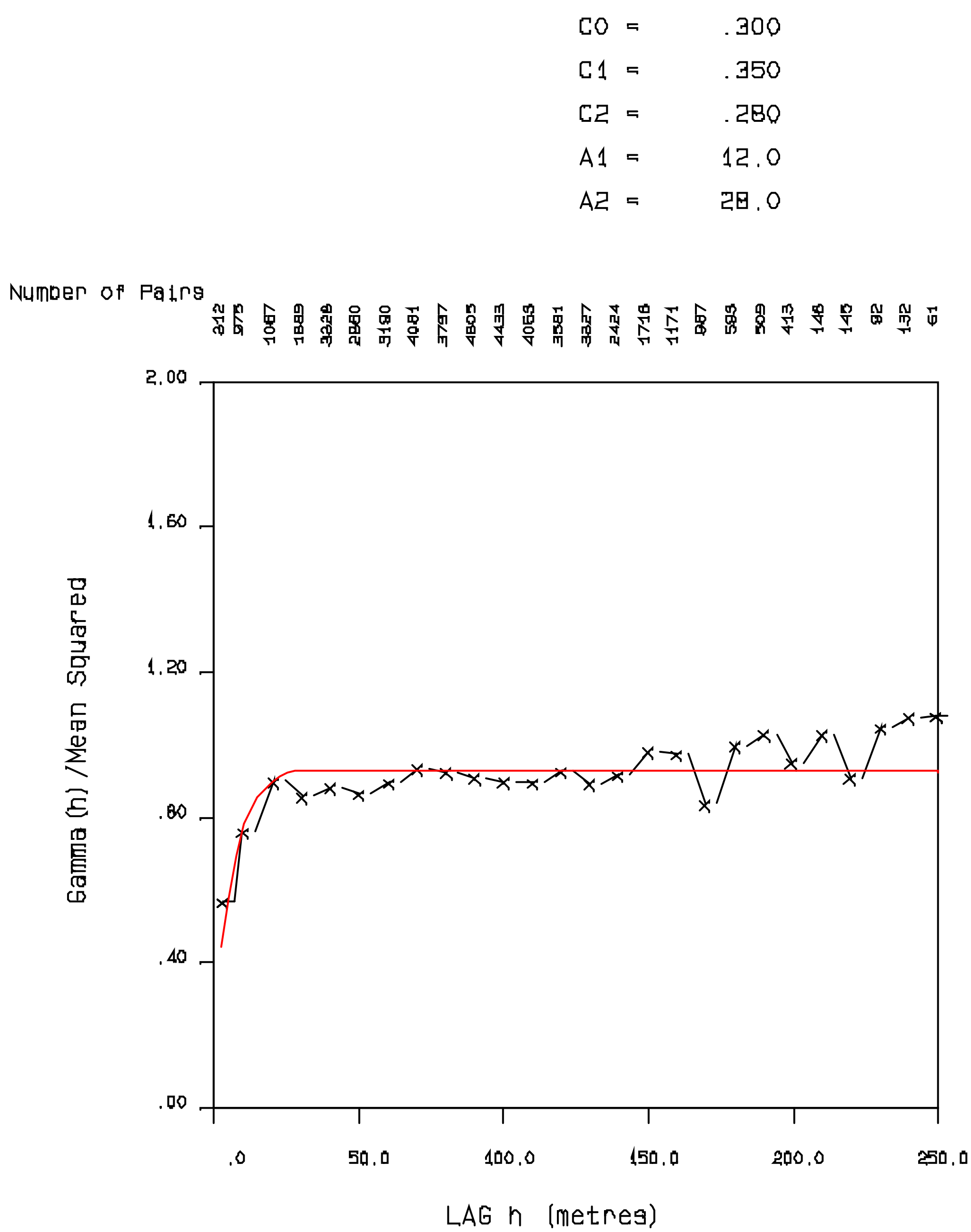

C0 = .250
C1 = .350
C2 = .250
A1 = 20.0
A2 = 30.0
Number of Pairs
41
145
198
243
200
558
422
317
183
58
2.00
1.60
1.20
.80
.40
.00
Gamma (h) /Mean Squared
.0
50.0
100.0
150.0
200.0
250.0
LAG h (metres)
LOW GRADE ASH AU - AZ 0 DIP -90



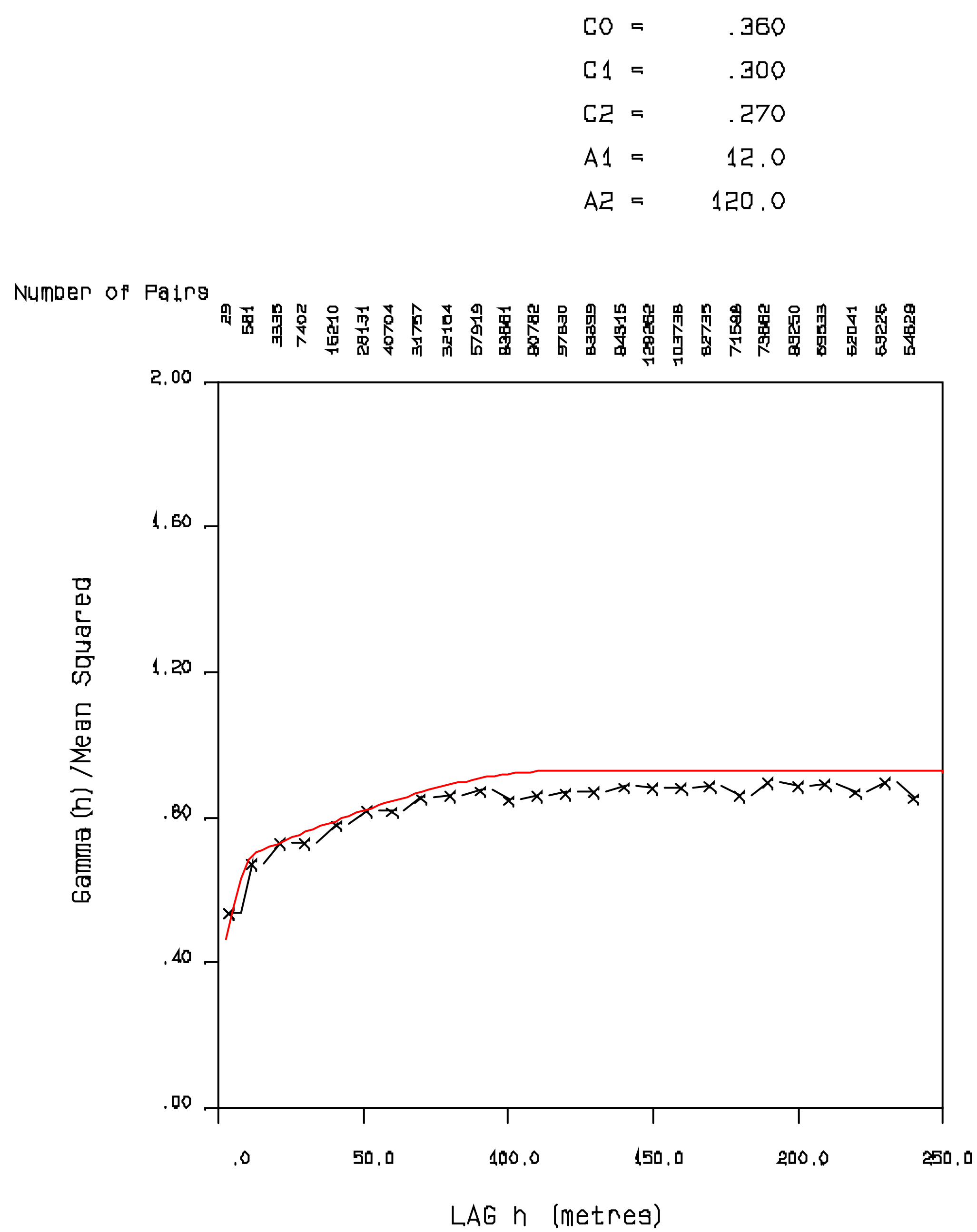
MAIN ZONE HG AU – AZ 60 DIP 0



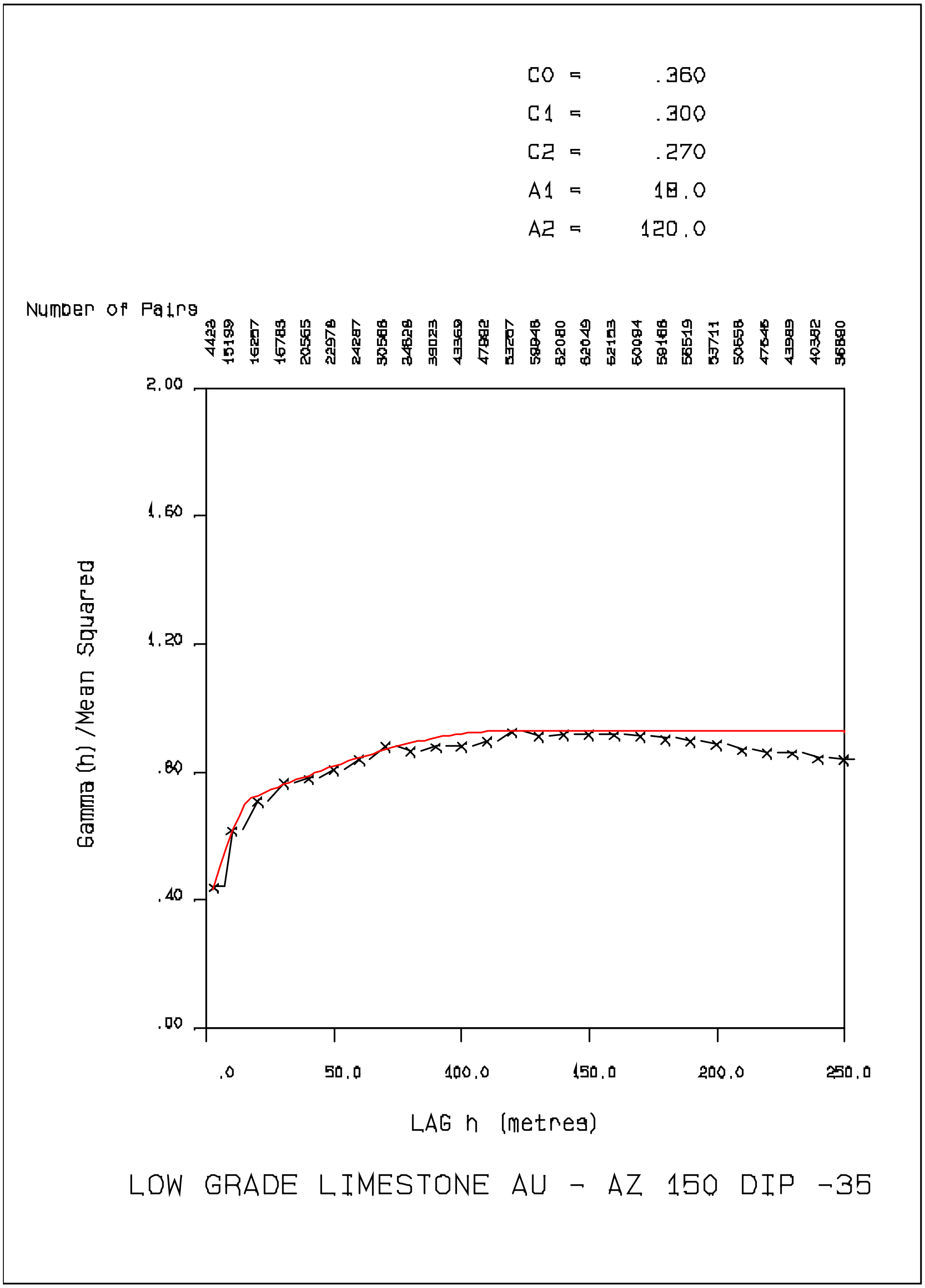
MAIN ZONE HG AU - AZ 150 DIP -35

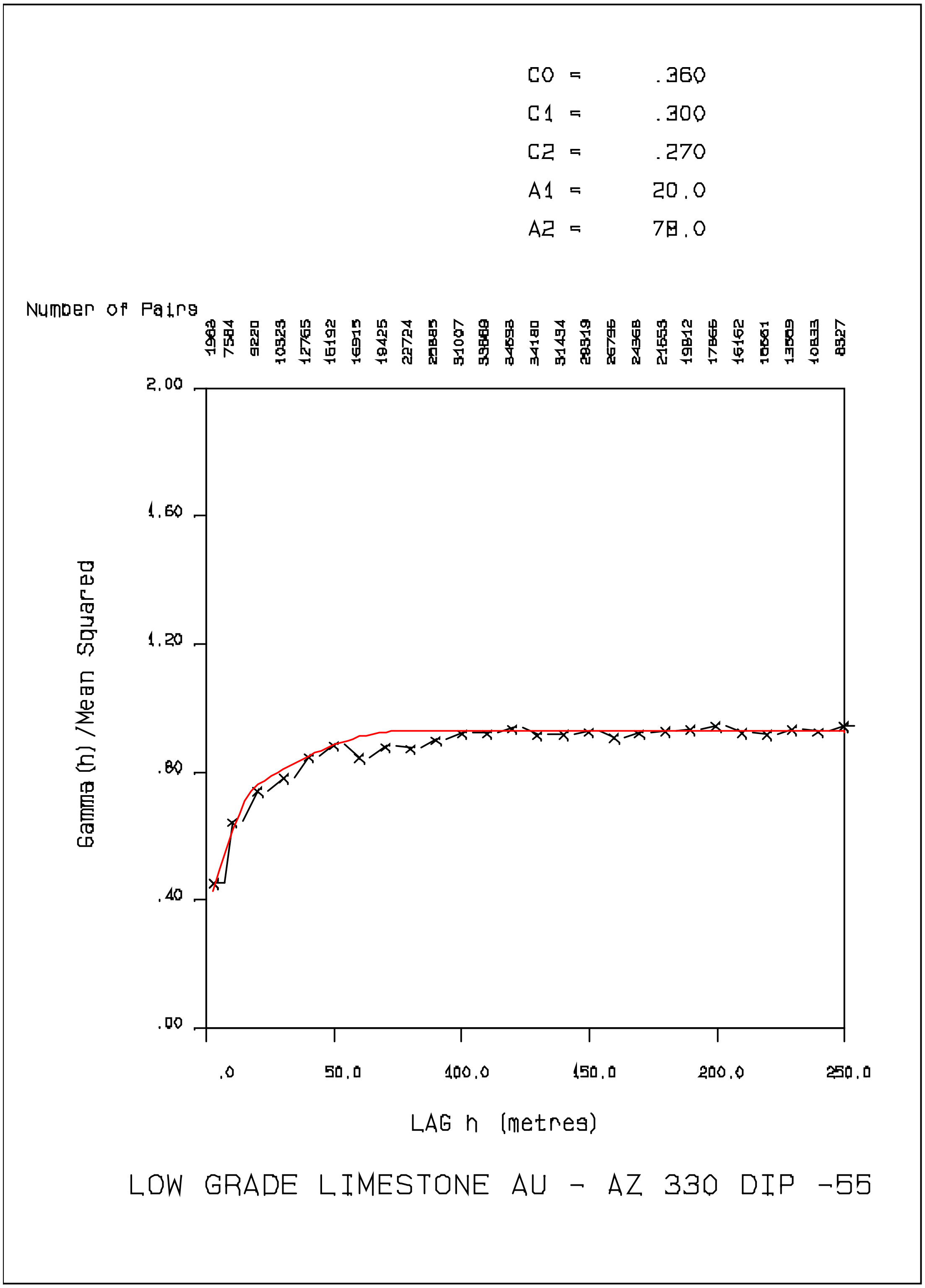


MAIN ZONE HG AU – AZ 330 DIP -55

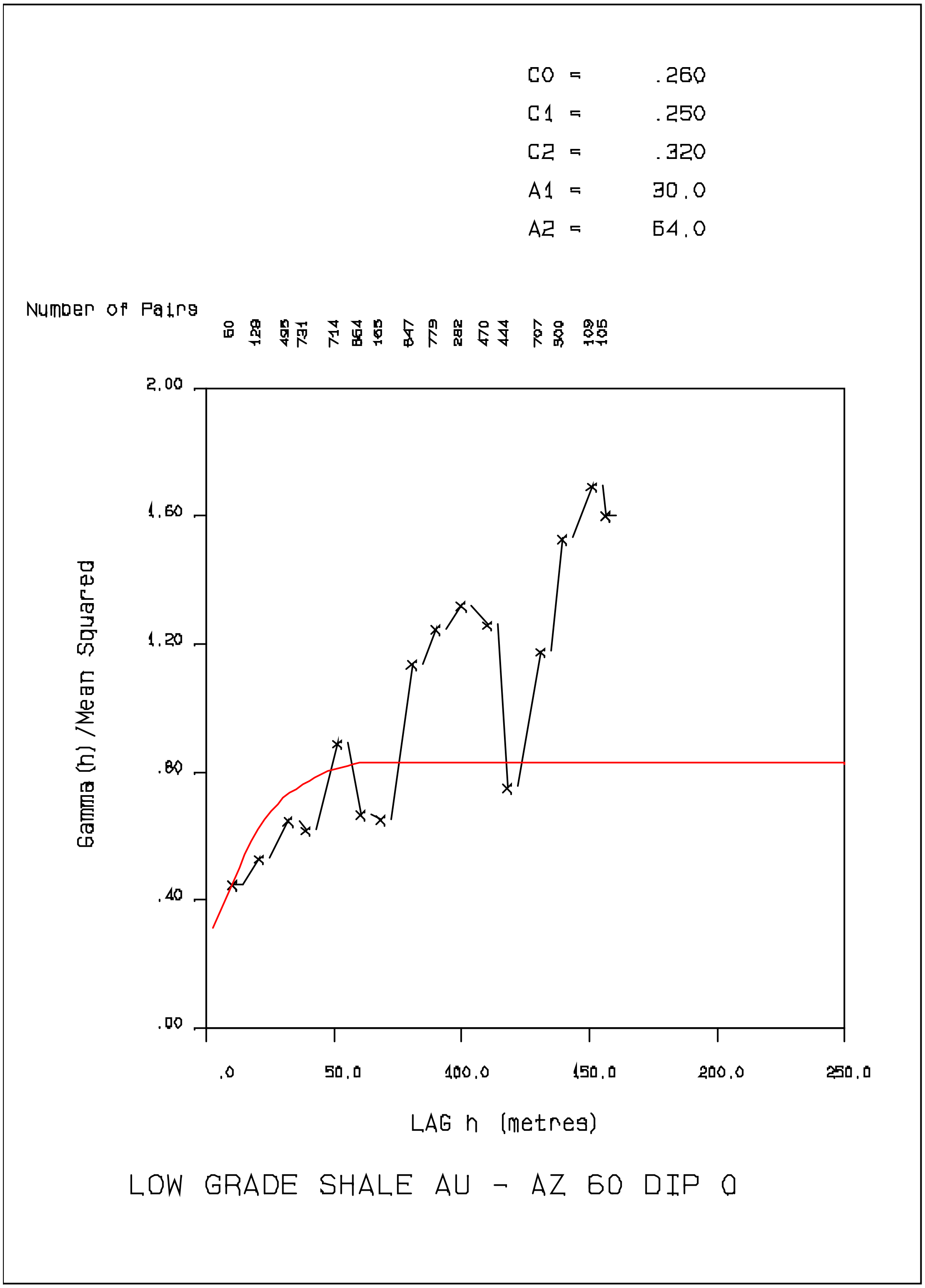


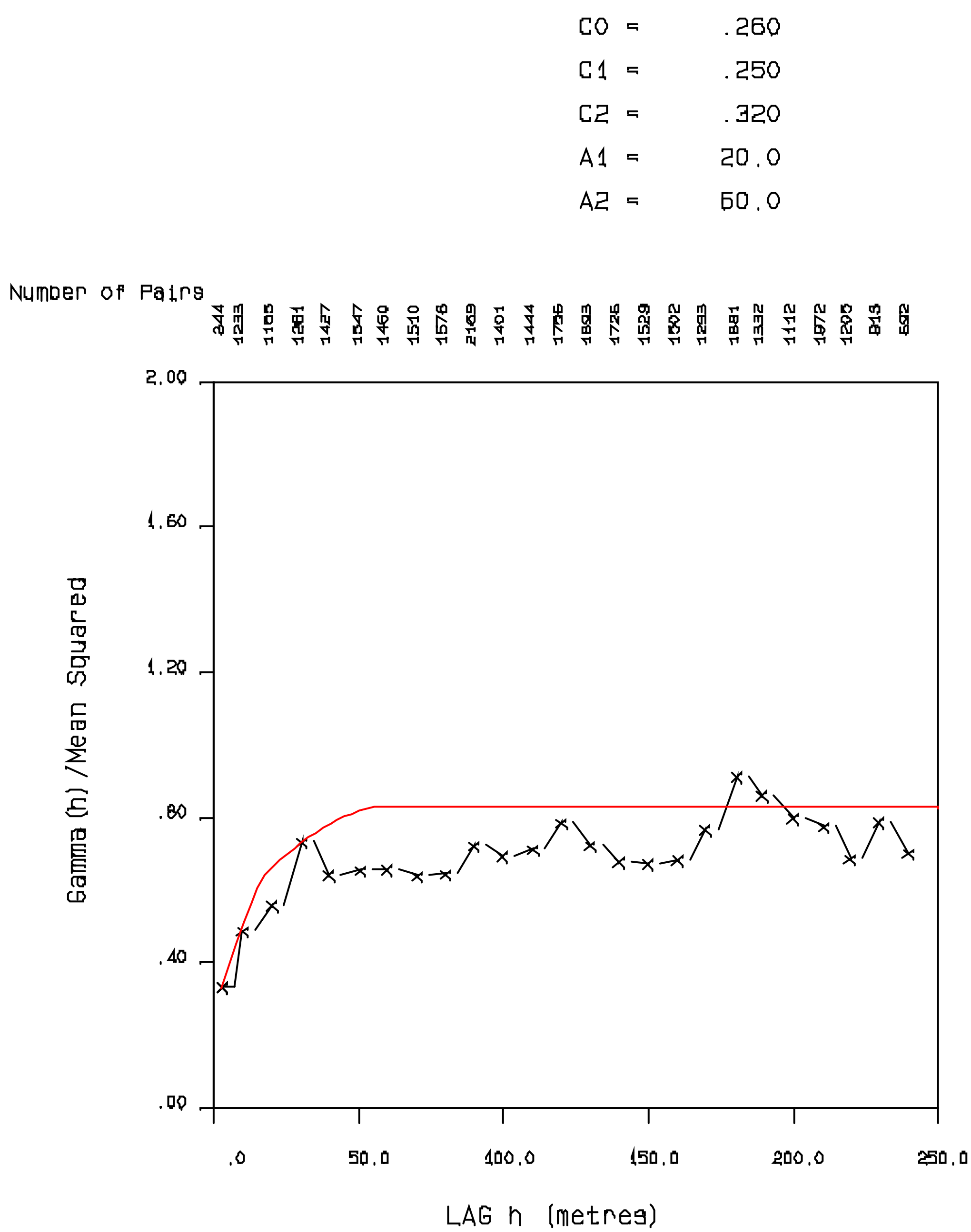
LOW GRADE LIMESTONE AU - AZ 60 DIP 0


C0 = .360
C1 = .300
C2 = .270
A1 = 18.0
A2 = 120.0
Number of Pairs
2.00
1.60
1.20
.80
.40
.00
Gamma (h) /Mean Squared
.0
50.0
100.0
150.0
200.0
250.0
LAG h (metres)
LOW GRADE LIMESTONE AU - AZ 150 DIP -35

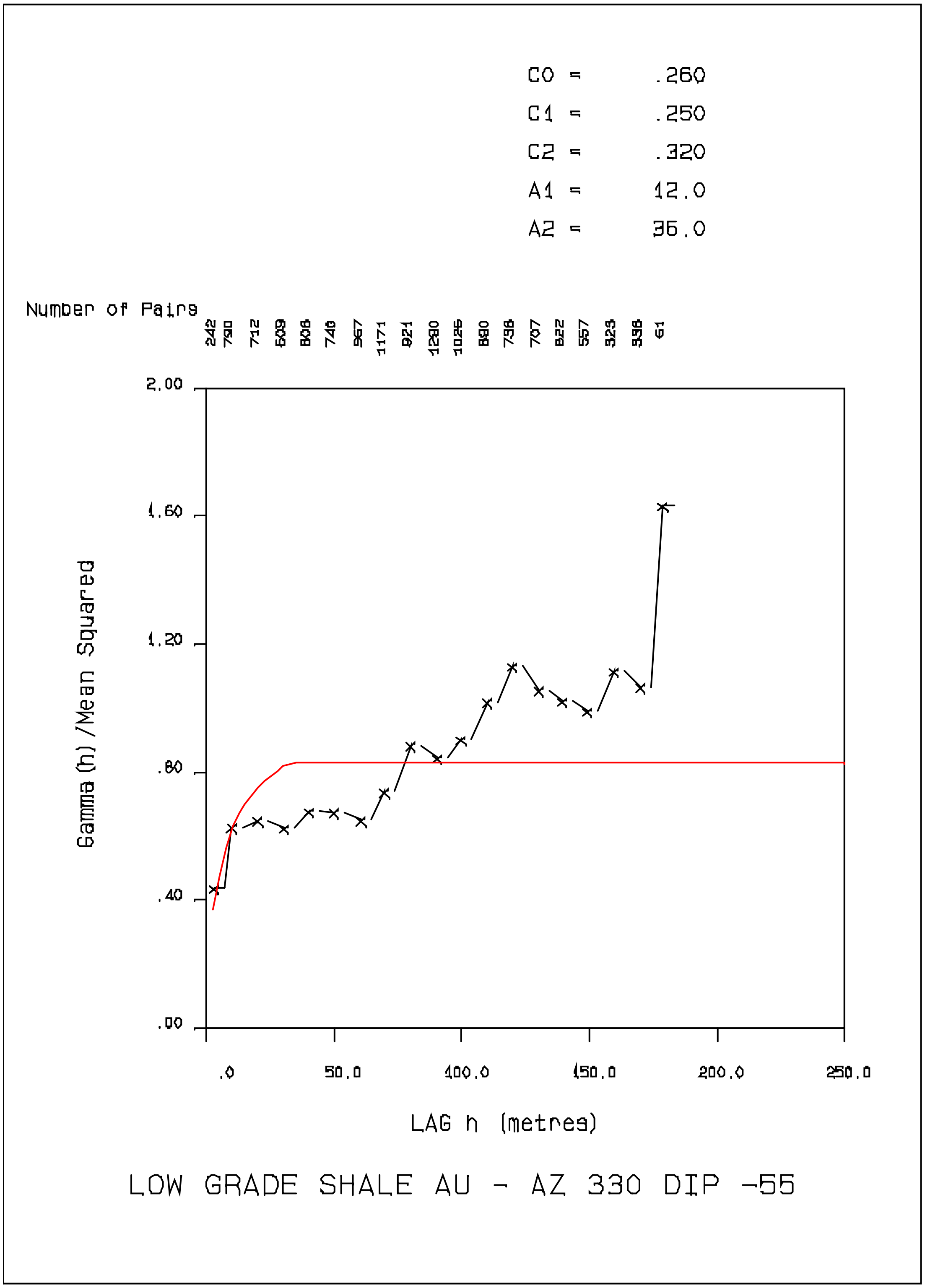


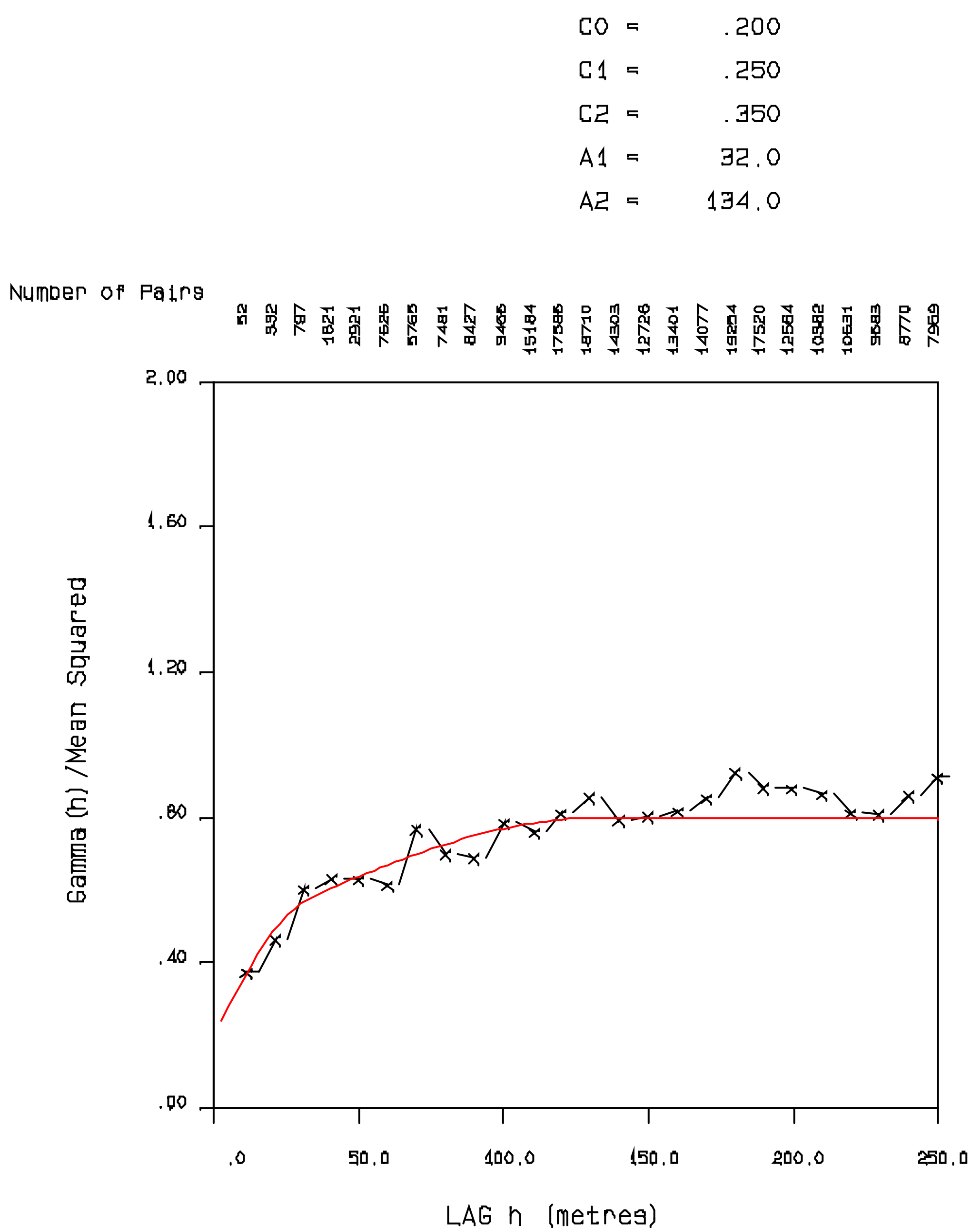
C0 = .360
C1 = .300
C2 = .270
A1 = 20.0
A2 = 78.0
Number of Pairs
2.00
1.60
1.20
.80
.40
.00
Gamma (h) /Mean Squared
.0
50.0
100.0
150.0
200.0
250.0
LAG h (metres)
LOW GRADE LIMESTONE AU - AZ 330 DIP -55


C0 = .260
C1 = .250
C2 = .320
A1 = 30.0
A2 = 64.0
Number of Pairs
60
129
495
731
714
864
165
847
779
282
470
444
707
500
109
105
2.00
1.60
1.20
.80
.40
.00
Gamma (h) /Mean Squared
.0
50.0
100.0
150.0
200.0
250.0
LAG h (metres)
LOW GRADE SHALE AU - AZ 60 DIP 0

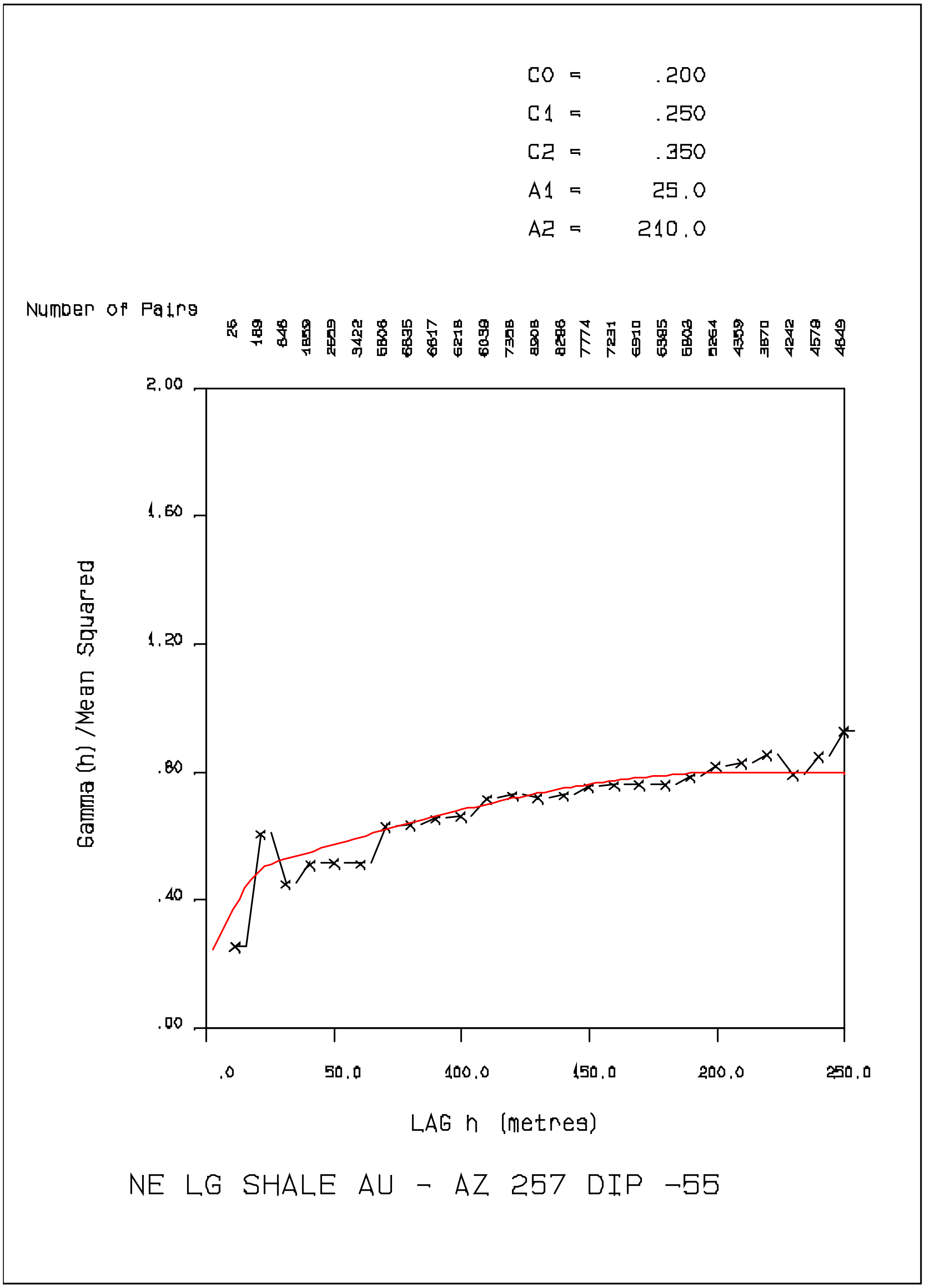


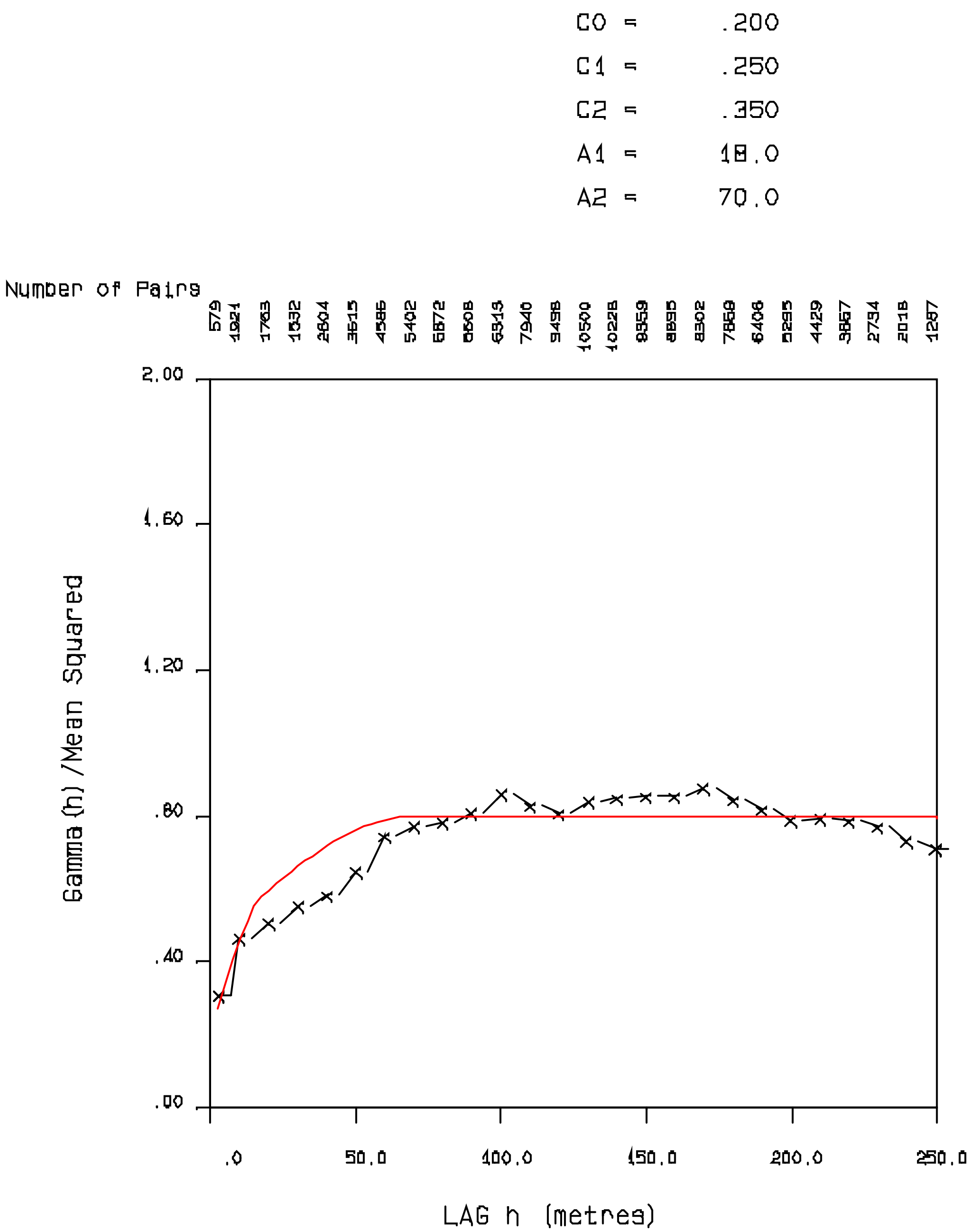
LOW GRADE SHALE AU - AZ 150 DIP -35


C0 = .260
C1 = .250
C2 = .320
A1 = 12.0
A2 = 36.0
Number of Pairs
242
790
712
609
808
746
967
1171
921
1280
1026
890
738
707
822
557
323
336
61
2.00
1.60
1.20
.80
.40
.00
Gamma (h) /Mean Squared
.0
50.0
100.0
150.0
200.0
250.0
LAG h (metres)
LOW GRADE SHALE AU - AZ 330 DIP -55

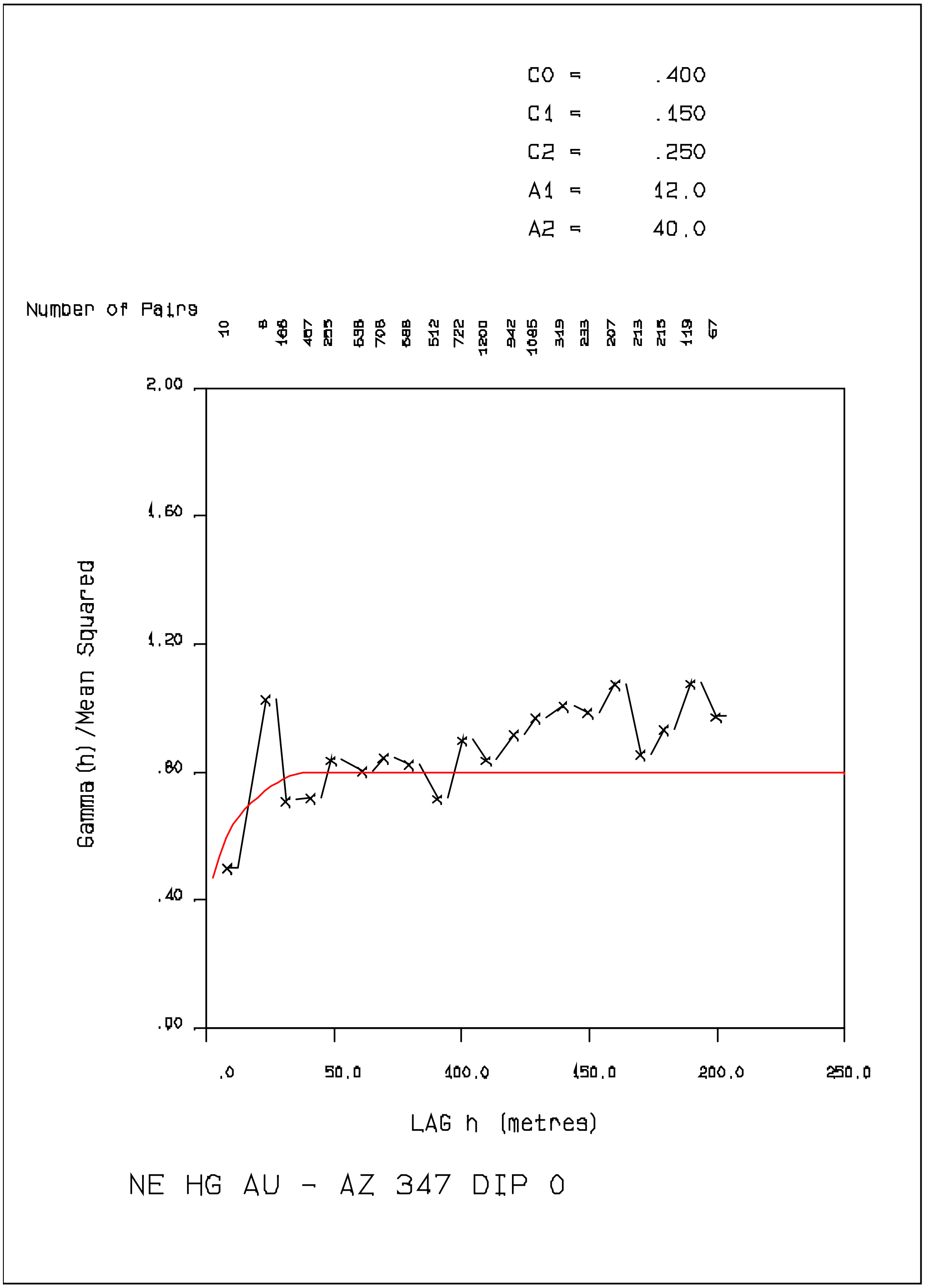


NE LG SHALE AU - AZ 347 DIP 0

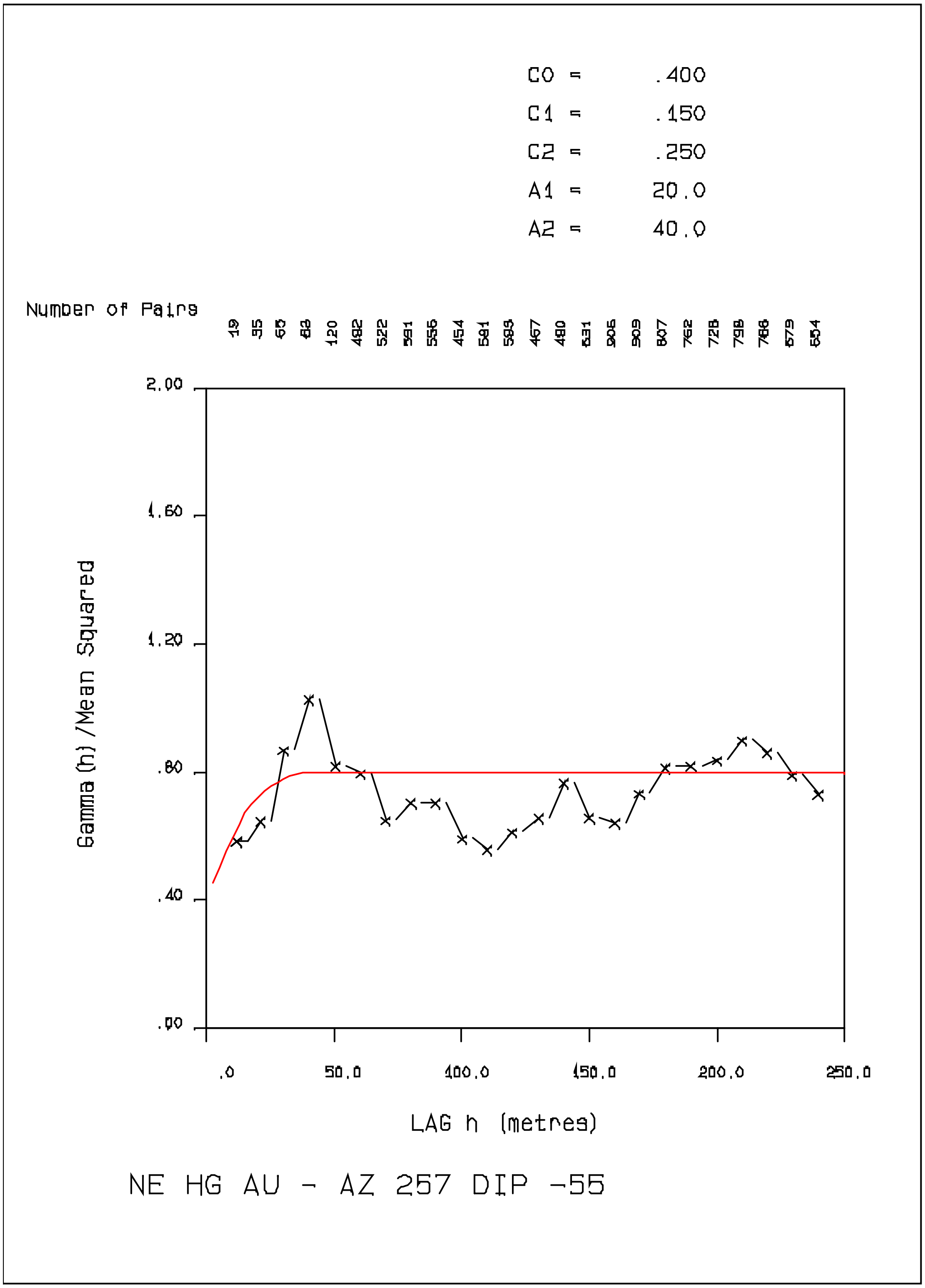

C0 = .200
C1 = .250
C2 = .350
A1 = 25.0
A2 = 210.0
Number of Pairs
25 189 646 1859 2629 3422 5806 6835 6617 6218 6039 7358 8908 8286 7774 7231 6910 6585 5803 5264 4359 3870 4242 4578 4849
2.00
1.60
1.20
.80
.40
.00
Gamma (h) /Mean Squared
.0
50.0
100.0
150.0
200.0
250.0
LAG h (metres)
NE LG SHALE AU - AZ 257 DIP -55

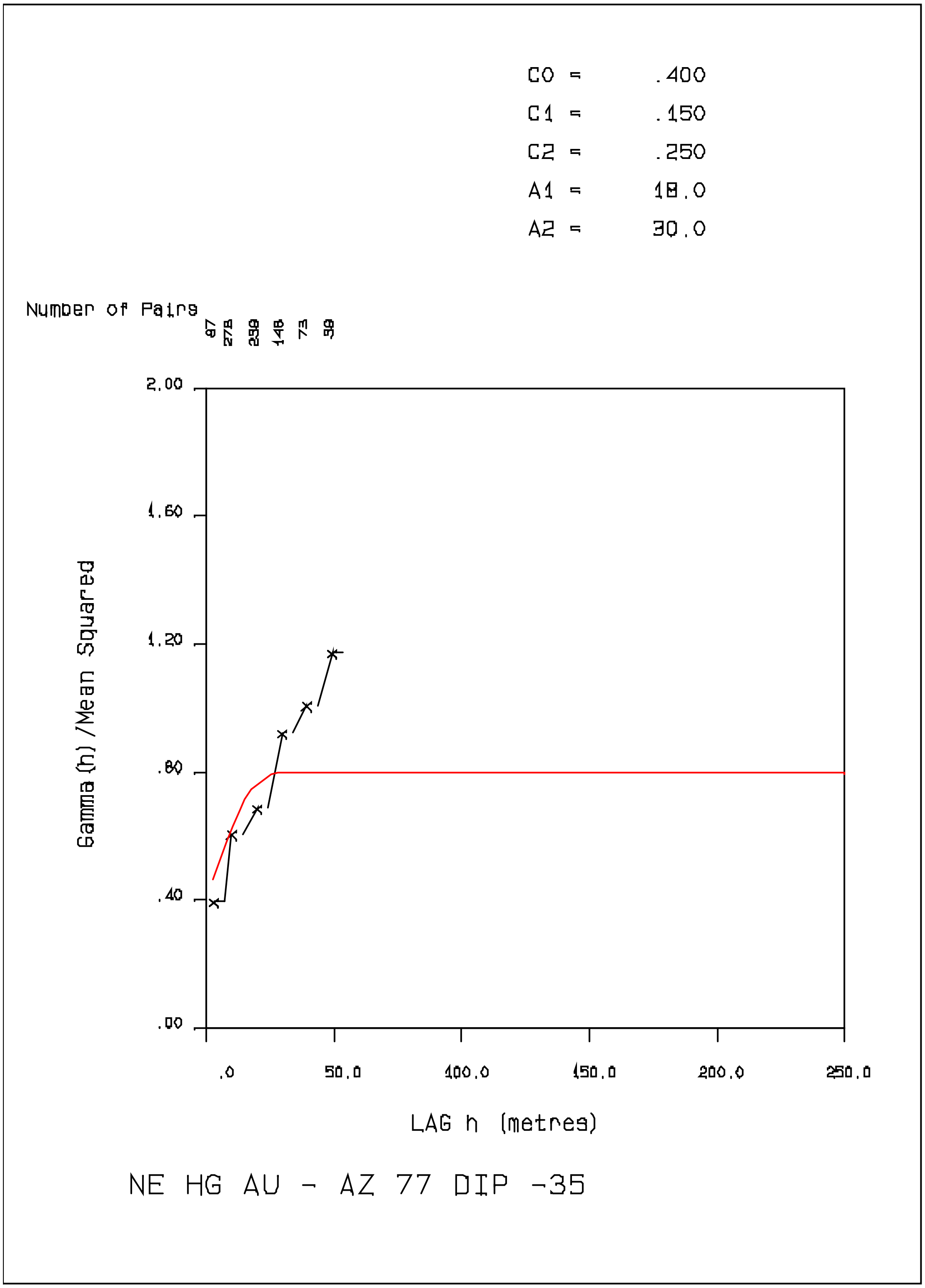


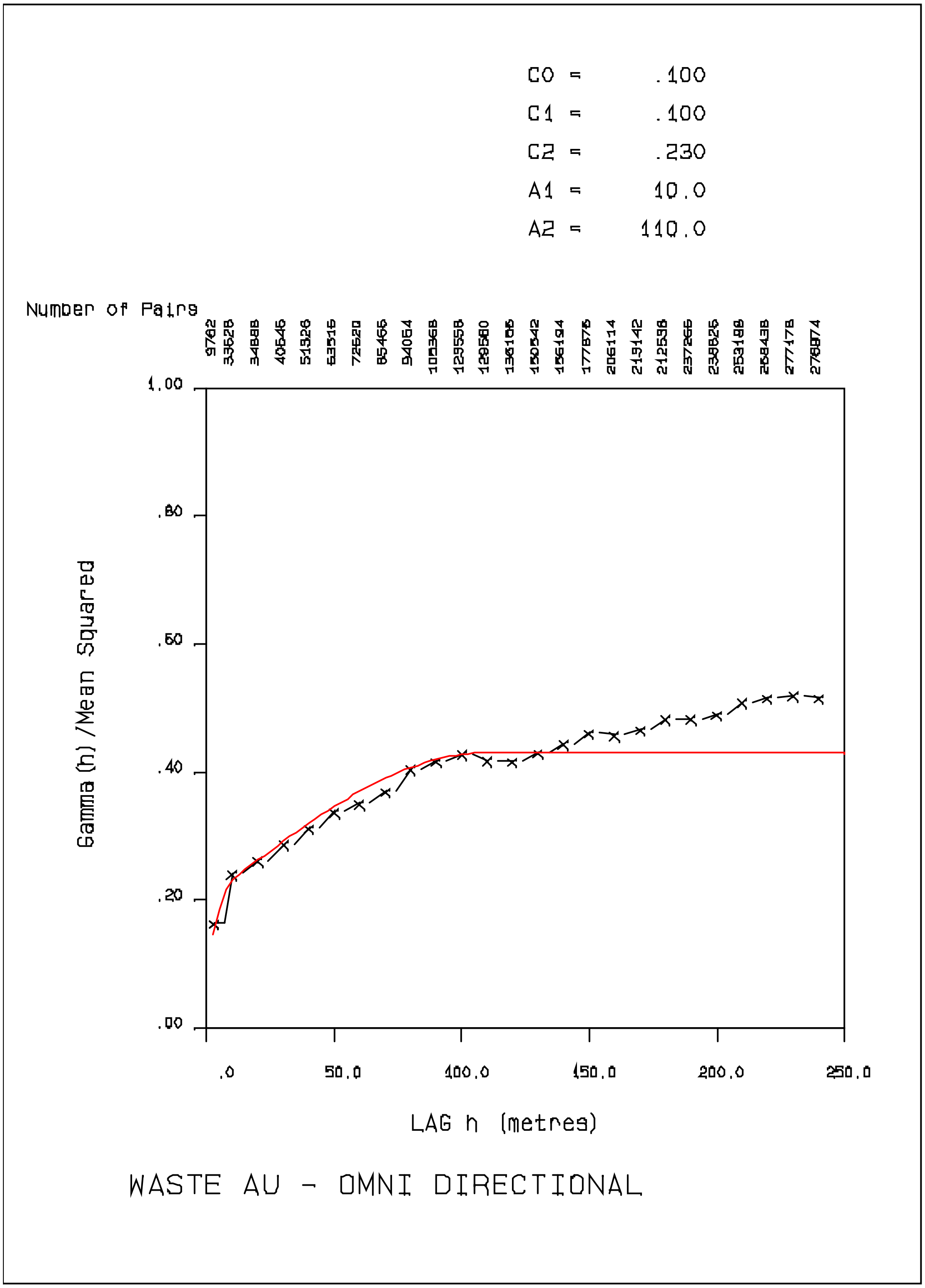
NE LG SHALE AU - AZ 77 DIP -35



C0 = .400
C1 = .150
C2 = .250
A1 = 12.0
A2 = 40.0
Number of Pairs
10
8
168
457
255
538
708
588
512
722
1200
942
1085
319
233
207
213
215
119
67
2.00
1.60
1.20
.80
.40
.00
Gamma (h) /Mean Squared
.0
50.0
100.0
150.0
200.0
250.0
LAG h (metres)
NE HG AU - AZ 347 DIP 0


C0 = .400
C1 = .150
C2 = .250
A1 = 20.0
A2 = 40.0
Number of Pairs
Gamma (h) /Mean Squared
2.00
1.60
1.20
.80
.40
.00
.0
50.0
100.0
150.0
200.0
250.0
LAG h (metres)
NE HG AU - AZ 257 DIP -55


C0 = .400
C1 = .150
C2 = .250
A1 = 18.0
A2 = 30.0
Number of Pairs
97
278
339
148
73
39
2.00
1.60
1.20
.80
.40
.00
Gamma (h) /Mean Squared
.0
50.0
100.0
150.0
200.0
250.0
LAG h (metres)
NE HG AU - AZ 77 DIP -35



C0 = .100
C1 = .100
C2 = .230
A1 = 10.0
A2 = 110.0
Number of Pairs
9762
33628
34888
40646
51326
63516
72620
85466
94064
105368
129558
129580
136166
150542
156194
177876
206114
219142
212536
237266
238826
253188
268438
277178
278874
1.00
.80
.60
.40
.20
.00
Gamma (h) /Mean Squared
.0
50.0
100.0
150.0
200.0
250.0
LAG h (metres)
WASTE AU - OMNI DIRECTIONAL

APPENDIX 4: Blue Coast Research Ltd. Metallurgical Test Report

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Almaden Ixtaca Project

PEA Metallurgical Testwork Report

Almaden Minerals Ltd.

Vancouver, British Columbia, Canada

Date: 7 March 2013

Authors: Dave Middleditch B.Eng, Marjorie Colebrook M.ASc

Experimental Testwork: Marjorie Colebrook, Jerry Chang, Owen Martin

DISCLAIMER

The data provided in this report and the associated interpretations offered are based on the laboratory testwork performed by Blue Coast Research Ltd.   No assurances can be made by Blue Coast Research Ltd on the representivity of the samples tested.

EXECUTIVE SUMMARY

Samples from four main domains (Dyke, Limestone, Black Shale and TUFF) from Almaden's Ixtaca project were subjected to a series of amenability metallurgical tests at Blue Coast Research Ltd. An additional High Grade sample comprising of high grade sections of Limestone and Dyke material was also tested.

A combination of gravity and cyanidation testwork (EGRG of the whole ore followed by cyanide leaching of the tails) indicates that the following gold recoveries can be achieved from these domain samples:

Table 1.1 – Summary of Combined Gravity and Cyanidation Gold Recoveries

Sample ID	EGRG Au Rec (%)	Cyanidation Au Rec (%)	Total Au Rec (%)
Dyke	48.4	61.9	**80.3**
Limestone	58.7	61.1	**83.9**
Black Shale	54.9	25.6	**66.4**
TUFF	15.1	41.5	**50.3**

Flotation also offers a viable alternative at potentially lower operating cost and less environmental impact. Grades and recoveries to bulk rougher concentrate for each domain are summarised below:

Table 1.2 – Summary of Flotation Only Gold and Silver Recoveries

Domain	Bulk Conc Grade (g/t)		Recovery (%)	
	Ag	Au	Ag	Au
Dyke	**225**	**4.21**	**87.0**	**94.4**
Limestone	**656**	**8.75**	**72.6**	**76.8**
Black Shale	**196**	**4.13**	**83.5**	**93.2**
TUFF	**78**	**3.74**	**63.2**	**52.3**
High Grade	**1001**	**13.94**	**91.2**	**93.2**

Hardness testwork was completed in the form of a Bond BWi test on each domain. Testing suggests that the TUFF domain is the softest at 10.5kwh/t and the Black Shale is the hardest at 18.6kwh/t. The Dyke and Limestone both exhibit similar hardness characteristics at 14.6 and 13.2kwh/t respectively.

Further opportunities exist for optimising the gold and silver recoveries for each of the domain samples. This report communicates the methods employed in the testwork program, the results achieved and provides conclusions and recommendations for future testwork.

CONTENTS

LIST OF FIGURES

LIST OF TABLES

1. INTRODUCTION

Blue Coast Research Ltd. (BCR) was approached by Almaden Minerals to perform a series of amenability tests on samples from its Ixtaca gold-silver deposit in Mexico. The Ixtaca zone is part of Almaden's 100% owned Tuligitic Project (www.almadenminerals.com). The Tuligitic project is 95km North of Puebla City and 120km South East of the prolific Pachuca deposit which had historical silver and gold production of 1.4 billion ounces and 7 million ounces respectively.

This testwork program was executed by Blue Coast Research personnel under the direction of Dr Andrew Bamber of BC Mining Research. Dr Bamber is also the Qualified Person (QP) for the Preliminary Economic Assessment of the Ixtaca Zone.

Metallurgical samples were received from four distinctive domains within the deposit:

- Dyke
- Limestone
- Black Shale
- Tuff

The following testwork was undertaken on each of the four domain samples:

- Head Assay characterisation of each domain sample.
- Bond Ball Work Index test
- E- GRG (Gravity Recoverable Gold) test
- Cyanidation on the EGRG tails
- Baseline rougher flotation tests

All testwork and assays were performed at the Blue Coast Research metallurgical testwork facility located in Parksville, British Columbia unless otherwise stated. This report communicates the methods employed in the testwork program, the results achieved and provides conclusions and recommendations for future testwork.

2. SAMPLE SELECTION & PREPARATION OF ORIGINAL SAMPLES

Initially eight samples were received by Blue Coast Research in early September 2012. The samples were divided into two types – Hardness (drill core) and MET samples. The samples represented four domains in the deposit:

- Dyke, a quartz vein;
- Limestone, a carbonate
- Black Shale, a shale;
- TUFF, a brecciated pumice (volcanic)

The Dyke, Limestone and TUFF MET samples were comprised of coarse assay rejects which are not considered to be ideal from a flotation testwork perspective but are perfectly adequate for cyanidation and gravity testwork where surface oxidation of the sample is not detrimental to metallurgical performance. The Black Shale MET sample was comprised of fresh drillcore which is considered to be perfectly acceptable for hardness and all types of metallurgical testwork including flotation. As hardness testwork requires samples to be stage crushed to 100% passing 6 mesh (3.35mm) without the over generation of fines, the Dyke, Limestone and TUFF zones all had accompanying samples comprised of drillcore only.

A fifth high-grade sample was also received. This sample was a blend of higher grade samples of Dyke and Limestone and was comprised of coarse assay rejects. When this sample arrived, it was inspected and deemed too fine for meaningful flotation testwork i.e. it had been over crushed by the assay prep laboratory potentially leading to an excessively fine flotation feed size distribution, and as the samples had not been freezer stored there was significant risk of oxidation of the minerals. Therefore, it was agreed by Almaden Minerals, Dr Andrew Bamber and the Blue Coast Research team to ship a replacement half drillcore High Grade MET sample. This sample was received by BCR at the end of September 2012.

The following tables summarise the sample section IDs, partial ICP geochem assays and weights for each sample as received by BCR.

Table 2.1 – Limestone MET Sample Coarse Assay Rejects Sample Inventory

Sample No.	Description	Domain	Mass (g)	Zn (ppm)	Pb (ppm)	Cu (ppm)	Fe (%)	S (%)	Au (g/t)	Ag (g/t)	Mn (ppm)
51608	Met Sample	Limestone	**652**	105	33	11	3.98	2.92	0.18	7.20	5140
51609	Met Sample	Limestone	**736**	215	87	61	2.91	2.44	1.81	29.70	76300
51611	Met Sample	Limestone	**3695**	59	25	20	1.80	1.12	0.74	95.80	7980
51612	Met Sample	Limestone	**2090**	25	18	6	1.27	0.98	0.19	21.80	6170
51613	Met Sample	Limestone	**232**	202	117	37	0.95	0.81	1.19	100.00	43500
51614	Met Sample	Limestone	**903**	8	6	4	0.97	0.59	0.07	2.80	2630
51616	Met Sample	Limestone	**311**	83	33	15	0.98	0.71	0.20	19.90	8860
78721	Met Sample	Limestone	**1866**	23	10	9	1.57	0.98	0.24	12.50	18500
78722	Met Sample	Limestone	**1081**	35	10	29	1.59	0.96	4.37	20.70	12000
78723	Met Sample	Limestone	**1005**	62	28	21	1.82	1.34	1.55	34.20	16250
78724	Met Sample	Limestone	**1181**	150	11	12	0.85	0.73	0.13	2.00	3950
78726	Met Sample	Limestone	**1455**	117	25	22	0.92	0.82	1.71	7.80	3480
78727	Met Sample	Limestone	**1148**	265	96	64	0.90	0.84	0.40	50.50	4150
78728	Met Sample	Limestone	**1182**	38	12	16	0.96	0.97	0.13	3.50	2980
298280	Met Sample	Limestone	**1057**	618	223	84	0.43	0.31	2.65	100.00	26500
298281	Met Sample	Limestone	**3000**	71	12	28	0.46	0.25	0.13	6.30	1990
298282	Met Sample	Limestone	**1751**	28	7	6	0.24	0.09	0.15	11.80	5490
298283	Met Sample	Limestone	**1139**	175	102	16	0.33	0.18	0.71	85.50	30100
298284	Met Sample	Limestone	**1232**	740	279	26	0.55	0.41	1.26	69.20	35700
298286	Met Sample	Limestone	**1941**	624	263	43	1.65	1.98	2.11	100.00	82700
84372	Met Sample	Limestone	**1225**	394	110	31	0.24	0.14	1.44	93.90	73800
84373	Met Sample	Limestone	**971**	73	40	16	0.09	0.06	1.05	52.10	12300
84374	Met Sample	Limestone	**1056**	57	17	9	0.11	0.09	0.26	27.10	6380
84376	Met Sample	Limestone	**921**	350	212	54	0.39	0.49	0.96	100.00	14300
84377	Met Sample	Limestone	**1124**	41	11	20	0.14	0.12	0.32	22.10	12750
84378	Met Sample	Limestone	**1850**	66	11	31	0.23	0.20	0.09	13.90	3990
84379	Met Sample	Limestone	**1892**	50	8	7	0.36	0.33	0.02	1.10	1780
85813	Met Sample	Limestone	**1469**	94	38	31	1.88	1.03	1.49	70.70	10550
85814	Met Sample	Limestone	**1097**	32	7	9	2.90	3.19	0.37	11.10	7140
85816	Met Sample	Limestone	**1355**	49	2	9	3.06	1.94	0.44	9.80	2460
85817	Met Sample	Limestone	**1089**	80	16	16	1.88	0.64	0.58	28.40	4000
85818	Met Sample	Limestone	**1867**	70	21	14	2.02	1.49	0.26	37.10	4890
85819	Met Sample	Limestone	**2025**	29	6	12	1.76	2.17	0.24	22.20	4260
TOTAL			**45600**	140	51	22	1.20	0.93	0.75	38.36	14856

Table 2.2 – Dyke MET Sample Coarse Assay Rejects Sample Inventory

Sample No.	Description	Domain	Mass (g)	Zn (ppm)	Pb (ppm)	Cu (ppm)	Fe (%)	S (%)	Au (g/t)	Ag (g/t)	Mn (ppm)
51736	Met Sample	Dyke	**1026**	978	634	25	3.91	4.29	0.83	100.00	7070
51737	Met Sample	Dyke	**1122**	1500	338	28	5.44	6.09	1.53	100.00	8640
51738	Met Sample	Dyke	**1024**	300	192	14	3.88	4.22	0.39	54.80	6340
51739	Met Sample	Dyke	**914**	662	394	27	3.67	3.88	1.89	100.00	8850
51740	Met Sample	Dyke	**1107**	1015	199	38	3.62	2.36	1.37	80.10	6320
51741	Met Sample	Dyke	**967**	413	64	32	3.65	1.64	0.62	17.10	6210
51742	Met Sample	Dyke	**1207**	149	103	49	3.56	2.14	0.51	21.10	6170
51743	Met Sample	Dyke	**771**	57	8	16	3.47	2.09	0.19	3.10	6110
56247	Met Sample	Dyke	**752**	653	183	34	5.11	2.46	0.90	57.20	6050
56248	Met Sample	Dyke	**898**	239	99	25	5.16	3.61	0.51	33.30	6860
56249	Met Sample	Dyke	**872**	270	119	17	5.39	3.65	0.60	33.90	4530
56251	Met Sample	Dyke	**918**	133	46	13	5.51	3.37	0.32	14.70	5930
56252	Met Sample	Dyke	**894**	1065	408	35	5.30	2.57	0.53	40.60	5630
56253	Met Sample	Dyke	**747**	856	268	125	4.90	3.16	1.05	85.80	6020
56254	Met Sample	Dyke	**822**	1555	211	81	6.09	4.07	1.51	71.50	6870
56256	Met Sample	Dyke	**897**	1190	364	56	5.33	3.13	0.72	81.80	5780
56257	Met Sample	Dyke	**844**	475	157	32	4.77	2.86	1.07	67.80	7050
62732	Met Sample	Dyke	**1077**	214	90	27	3.18	3.68	0.21	24.70	18050
62733	Met Sample	Dyke	**763**	186	52	5	2.85	3.27	0.57	17.30	6000
62734	Met Sample	Dyke	**843**	485	185	12	3.13	3.62	0.56	35.80	5710
62736	Met Sample	Dyke	**712**	451	164	20	3.72	4.39	0.65	100.00	9650
62737	Met Sample	Dyke	**518**	275	136	5	5.11	5.87	0.10	16.50	5810
62738	Met Sample	Dyke	**1048**	261	65	22	4.25	3.95	1.12	10.40	10050
62739	Met Sample	Dyke	**873**	877	153	21	4.33	3.76	2.37	19.60	13500
62740	Met Sample	Dyke	**921**	779	218	19	3.91	3.85	0.68	15.40	9130
62741	Met Sample	Dyke	**944**	91	19	20	4.49	3.59	0.65	3.30	13300
62742	Met Sample	Dyke	**909**	195	46	17	4.65	4.57	0.62	4.40	10900
298338	Met Sample	Dyke	**989**	79	20	70	3.23	2.19	0.88	50.80	15100
298339	Met Sample	Dyke	**1479**	61	23	17	3.24	2.70	0.80	20.80	11150
298340	Met Sample	Dyke	**1070**	809	168	19	3.36	3.13	0.71	39.20	10600
298341	Met Sample	Dyke	**870**	180	45	16	3.21	2.09	0.29	20.80	5310
298342	Met Sample	Dyke	**1014**	74	79	8	3.06	2.07	0.44	29.30	15900
298343	Met Sample	Dyke	**1176**	43	18	17	2.83	2.04	1.09	28.90	11950
298344	Met Sample	Dyke	**941**	192	48	45	3.34	2.66	1.49	25.60	17650
298346	Met Sample	Dyke	**831**	185	63	71	3.47	2.36	1.00	17.20	12650
85749	Met Sample	Dyke	**925**	72	25	24	5.14	5.94	0.25	45.70	11150
85751	Met Sample	Dyke	**798**	37	7	17	4.98	5.74	0.13	20.30	15800
85752	Met Sample	Dyke	**866**	52	31	42	3.37	4.02	0.16	46.70	27700
85753	Met Sample	Dyke	**741**	182	34	57	5.58	6.37	0.43	65.40	5790
85754	Met Sample	Dyke	**766**	168	13	57	4.95	5.75	0.42	44.50	13350
85756	Met Sample	Dyke	**765**	104	9	47	5.89	7.01	0.40	24.80	12700
85757	Met Sample	Dyke	**798**	68	13	39	6.12	7.22	0.45	29.70	10850
85758	Met Sample	Dyke	**924**	113	38	36	6.07	7.23	0.43	57.00	12050
85759	Met Sample	Dyke	**897**	103	58	72	4.23	5.04	0.62	72.60	19600
85760	Met Sample	Dyke	**869**	40	18	59	3.04	3.58	0.77	64.30	24200
TOTAL			**41111**	407	127	33	4.11	3.69	0.74	42.39	10172

Table 2.3 – TUFF MET Sample Coarse Assay Rejects Sample Inventory

Sample No.	Description	Domain	Mass (g)	Zn (ppm)	Pb (ppm)	Cu (ppm)	Fe (%)	S (%)	Au (g/t)	Ag (g/t)	Mn (ppm)
96560	Met Sample	TUFF	**1640**	25	1	3	3.14	0.98	0.24	1.00	856
96561	Met Sample	TUFF	**1319**	76	3	5	3.93	1.12	0.47	0.25	4140
96562	Met Sample	TUFF	**1696**	18	1	11	2.37	0.74	0.89	1.00	1060
96563	Met Sample	TUFF	**1621**	66	1	4	2.72	0.63	1.64	2.50	3070
96564	Met Sample	TUFF	**1287**	144	1	13	2.94	0.42	1.29	2.60	630
96566	Met Sample	TUFF	**1901**	131	1	34	2.70	0.43	1.81	2.30	484
89722	Met Sample	TUFF	**1020**	47	7	22	1.76	2.04	1.78	8.30	9560
89723	Met Sample	TUFF	**1050**	44	4	17	2.08	2.42	1.01	8.60	4250
89724	Met Sample	TUFF	**1151**	39	9	15	1.64	1.91	0.78	7.30	17100
89726	Met Sample	TUFF	**1437**	46	10	39	1.96	2.22	1.03	8.90	10800
89727	Met Sample	TUFF	**794**	70	7	19	2.11	2.39	0.24	4.10	7050
89728	Met Sample	TUFF	**648**	59	1	16	2.33	2.60	0.17	3.80	1530
89729	Met Sample	TUFF	**796**	59	6	52	2.62	2.91	0.27	4.60	2320
89731	Met Sample	TUFF	**738**	67	11	44	2.50	2.84	0.32	4.70	5920
89732	Met Sample	TUFF	**919**	55	15	30	2.49	2.81	0.38	7.00	4810
57327	Met Sample	TUFF	**1489**	54	15	16	2.45	0.02	0.15	12.80	13650
57328	Met Sample	TUFF	**1511**	67	11	17	2.52	0.01	0.18	13.40	12600
57329	Met Sample	TUFF	**1472**	114	24	16	2.51	0.07	0.42	28.70	26200
57331	Met Sample	TUFF	**1910**	63	7	12	3.53	0.09	0.19	9.90	3710
77426	Met Sample	TUFF	**1982**	1120	27	21	4.32	2.79	1.27	15.00	39800
77427	Met Sample	TUFF	**3185**	245	9	37	2.63	2.19	0.76	6.30	9750
77428	Met Sample	TUFF	**1924**	124	6	31	1.97	1.93	0.65	4.90	3540
77429	Met Sample	TUFF	**1373**	97	7	36	1.91	1.93	0.88	3.90	2910
77431	Met Sample	TUFF	**1473**	119	8	47	2.66	2.79	1.27	5.80	5800
74636	Met Sample	TUFF	**1213**	124	21	30	2.78	3.04	1.02	31.50	18250
74637	Met Sample	TUFF	**1157**	78	21	29	3.76	4.20	1.00	28.40	4420
74638	Met Sample	TUFF	**1100**	95	26	46	3.64	4.00	1.39	49.30	8650
74639	Met Sample	TUFF	**1596**	110	36	86	3.66	4.15	1.48	57.60	10100
74640	Met Sample	TUFF	**1117**	128	35	32	3.63	4.16	1.00	36.80	9900
74641	Met Sample	TUFF	**1495**	56	32	22	4.05	4.50	0.36	19.30	2240
74642	Met Sample	TUFF	**1094**	72	27	69	3.96	4.33	0.47	16.60	1840
TOTAL			**43104**	135	12	27	2.79	2.02	0.83	12.53	8168

Table 2.4 – Black Shale MET Sample Drillcore Inventory

Sample No.	Description	Domain	Mass (g)	Zn (ppm)	Pb (ppm)	Cu (ppm)	Fe (%)	S (%)	Au (g/t)	Ag (g/t)	Mn (ppm)
99279	Met Sample	Black Shale	**1434**	3770	2250	88	4.21	2.63	0.77	37.60	22400
99280	Met Sample	Black Shale	**1626**	1665	3530	34	7.15	4.76	1.02	35.50	37000
99281	Met Sample	Black Shale	**1555**	4670	1985	56	6.11	3.08	1.22	27.30	49700
99282	Met Sample	Black Shale	**1591**	3110	6820	119	5.49	3.08	2.11	41.10	36200
99283	Met Sample	Black Shale	**1363**	5240	2550	105	5.15	3.42	0.72	26.60	31900
99284	Met Sample	Black Shale	**1508**	4710	3780	38	4.26	2.89	0.51	28.70	43600
99286	Met Sample	Black Shale	**1444**	10000	9460	113	7.33	6.34	2.21	84.50	97400
98273	Met Sample	Black Shale	**1181**	2540	1335	133	3.19	3.25	1.24	16.90	20100
98274	Met Sample	Black Shale	**568**	3550	1880	76	3.66	3.55	2.52	12.60	48300
98276	Met Sample	Black Shale	**1270**	1260	1315	33	4.20	3.38	0.87	9.50	63300
98277	Met Sample	Black Shale	**1286**	800	266	38	2.35	1.28	1.41	45.30	7460
98278	Met Sample	Black Shale	**1186**	1815	608	35	2.59	2.36	0.50	5.90	7560
98279	Met Sample	Black Shale	**1285**	2340	1280	159	4.32	3.54	0.88	11.10	30200
98280	Met Sample	Black Shale	**1265**	2450	1480	21	5.06	3.02	0.53	8.30	71700
98281	Met Sample	Black Shale	**1223**	1410	771	27	2.94	1.88	0.48	5.30	14800
90797	Met Sample	Black Shale	**1062**	3120	1280	86	4.89	5.22	1.26	95.10	4640
90798	Met Sample	Black Shale	**1325**	921	289	18	1.60	1.65	0.25	10.50	6020
90799	Met Sample	Black Shale	**1180**	1675	1090	37	3.76	4.07	0.91	71.80	14300
90800	Met Sample	Black Shale	**1138**	757	328	37	3.05	3.15	0.20	27.20	5690
90801	Met Sample	Black Shale	**1273**	1545	648	50	6.04	6.46	0.34	33.30	7750
90802	Met Sample	Black Shale	**1351**	2230	938	49	5.95	6.39	0.44	47.40	18450
90803	Met Sample	Black Shale	**1268**	2410	3590	55	5.94	6.47	1.16	100.00	44400
90804	Met Sample	Black Shale	**1255**	10000	7590	321	3.70	4.85	2.27	100.00	25800
67198	Met Sample	Black Shale	**1613**	1190	405	72	2.39	2.63	1.35	93.10	28000
67199	Met Sample	Black Shale	**1372**	678	423	40	1.72	1.60	4.14	33.90	100000
67200	Met Sample	Black Shale	**1253**	1810	673	38	2.56	2.98	1.39	64.40	33100
67201	Met Sample	Black Shale	**1475**	1150	355	36	2.47	2.86	0.27	26.00	9860
67202	Met Sample	Black Shale	**1950**	746	114	43	2.24	2.39	0.11	10.90	3380
67203	Met Sample	Black Shale	**2400**	150	29	30	1.66	1.80	0.08	6.70	2550
67204	Met Sample	Black Shale	**1413**	522	126	33	1.88	2.07	0.12	10.60	3630
67206	Met Sample	Black Shale	**1386**	476	124	33	1.48	1.57	0.17	16.00	6230
69041	Met Sample	Black Shale	**1432**	4660	2000	47	3.39	3.73	1.12	31.50	17600
69042	Met Sample	Black Shale	**1242**	4550	1620	62	3.82	4.42	0.51	23.80	21000
69043	Met Sample	Black Shale	**1116**	1600	522	77	2.16	2.43	0.63	25.40	6550
69044	Met Sample	Black Shale	**2515**	1590	520	46	2.33	2.45	0.29	18.80	18750
69046	Met Sample	Black Shale	**1439**	24200	10850	400	6.58	8.63	1.37	100.00	51900
69047	Met Sample	Black Shale	**1353**	4080	1705	89	2.60	3.01	0.38	38.10	14250
69048	Met Sample	Black Shale	**1127**	26500	7970	453	5.96	8.49	1.48	100.00	15750
TOTAL			**52725**	4250	2384	90	3.89	3.69	0.97	40.06	27959

Table 2.5 – High Grade MET Sample Coarse Assay Rejects Sample Inventory

Sample No.	Description	Domain	Mass (g)	Zn (ppm)	Pb (ppm)	Cu (ppm)	Fe (%)	S (%)	Au (g/t)	Ag (g/t)	Mn (ppm)
94667	Limestone Met Sample	LIMESTONE-DYKE	**1447**	2340	1295	37	3.41	4.16	3.31	100.00	40000
94668	Limestone Met Sample	LIMESTONE-DYKE	**774**	466	263	111	3.81	4.08	2.64	100.00	13300
94669	Limestone Met Sample	LIMESTONE-DYKE	**914**	6070	2130	358	4.55	4.95	0.96	78.40	22500
94671	Limestone Met Sample	LIMESTONE-DYKE	**1197**	176	56	42	4.75	4.10	0.81	8.90	11700
94672	Limestone Met Sample	LIMESTONE-DYKE	**919**	249	102	64	4.02	4.02	1.84	90.30	14700
94673	Limestone Met Sample	LIMESTONE-DYKE	**1117**	473	358	71	2.79	2.94	0.79	73.90	12300
94674	Limestone Met Sample	LIMESTONE-DYKE	**1170**	591	583	28	5.22	6.55	0.81	97.80	32600
94676	Limestone Met Sample	LIMESTONE-DYKE	**983**	1175	557	85	2.21	4.90	1.51	100.00	54800
94677	Limestone Met Sample	LIMESTONE-DYKE	**855**	231	106	11	0.52	0.49	0.13	45.80	6690
84932	Limestone Met Sample	LIMESTONE-DYKE	**1485**	98100	70700	2790	4.43	10.00	1.83	100.00	67900
84933	Limestone Met Sample	LIMESTONE-DYKE	**1293**	2420	929	76	0.42	0.50	0.15	10.40	3150
84934	Limestone Met Sample	LIMESTONE-DYKE	**905**	2080	1110	54	0.50	0.58	0.90	63.80	73400
84936	Limestone Met Sample	LIMESTONE-DYKE	**1132**	5720	3030	60	1.19	1.71	2.86	100.00	100000
84937	Limestone Met Sample	LIMESTONE-DYKE	**1067**	11750	3680	139	2.75	3.55	5.85	100.00	68600
84938	Limestone Met Sample	LIMESTONE-DYKE	**1142**	89	61	6	0.39	0.34	0.20	9.40	11950
84939	Limestone Met Sample	LIMESTONE-DYKE	**1448**	1080	645	52	1.66	2.04	1.62	54.10	23900
84940	Limestone Met Sample	LIMESTONE-DYKE	**713**	269	127	14	0.62	0.57	2.86	71.00	50600
84941	Limestone Met Sample	LIMESTONE-DYKE	**1069**	99	38	5	0.17	0.10	0.13	14.40	2730
84942	Limestone Met Sample	LIMESTONE-DYKE	**1098**	319	133	11	0.80	0.75	2.66	64.90	62800
60034	Limestone Met Sample	LIMESTONE-DYKE	**1573**	316	215	16	0.31	0.21	1.55	100.00	61600
69467	Limestone Met Sample	LIMESTONE-DYKE	**1039**	54	40	9	0.26	0.13	0.55	55.40	46900
69468	Limestone Met Sample	LIMESTONE-DYKE	**1831**	462	198	25	0.57	0.55	1.59	100.00	43100
69469	Limestone Met Sample	LIMESTONE-DYKE	**1155**	965	335	39	1.90	2.10	5.88	100.00	78800
69471	Limestone Met Sample	LIMESTONE-DYKE	**1214**	300	117	19	1.25	1.27	0.77	82.50	39700
69472	Limestone Met Sample	LIMESTONE-DYKE	**1209**	507	180	76	0.73	0.62	1.36	100.00	9570
69473	Limestone Met Sample	LIMESTONE-DYKE	**1282**	163	37	43	0.80	0.44	0.30	13.10	5420
69474	Limestone Met Sample	LIMESTONE-DYKE	**964**	159	69	14	0.29	0.14	0.31	39.70	12000
50327	Dyke Met Sample	LIMESTONE-DYKE	**785**	314	186	31	4.37	3.75	1.39	45.00	9870
50328	Dyke Met Sample	LIMESTONE-DYKE	**1109**	124	116	78	5.55	5.00	1.33	86.70	11550
50329	Dyke Met Sample	LIMESTONE-DYKE	**1806**	1410	662	66	5.90	4.86	6.02	100.00	12200
50331	Dyke Met Sample	LIMESTONE-DYKE	**809**	119	47	37	4.56	2.84	0.97	34.20	15150
50332	Dyke Met Sample	LIMESTONE-DYKE	**492**	180	93	24	2.69	2.16	0.56	22.80	9130
50333	Dyke Met Sample	LIMESTONE-DYKE	**938**	61	39	10	5.15	3.92	0.56	16.80	13100
50334	Dyke Met Sample	LIMESTONE-DYKE	**830**	151	72	57	4.72	3.67	0.86	24.50	12450
43851	Dyke Met Sample	LIMESTONE-DYKE	**1568**	1610	977	154	5.10	5.75	2.89	100.00	18800
43852	Dyke Met Sample	LIMESTONE-DYKE	**1733**	299	160	65	5.15	5.65	2.39	56.80	10550
43853	Dyke Met Sample	LIMESTONE-DYKE	**1676**	327	128	276	3.68	4.13	2.07	83.50	11700
43854	Dyke Met Sample	LIMESTONE-DYKE	**615**	248	67	47	0.81	0.85	0.81	70.40	21000
43856	Dyke Met Sample	LIMESTONE-DYKE	**2296**	77	15	6	0.43	0.45	0.09	7.00	3280
TOTAL			**45652**	4119	2693	127	1.48	1.75	1.22	48.76	26045

Table 2.6 – High Grade MET Replacement Drillcore Sample Inventory

Sample No.	Description	Mass (g)	Pb (%)	Zn (%)	Fe (%)	Ag (g/t)	Au (g/t)	S (%)
94667	Limestone Met Sample	**1442**	1295	2340	3.41	148	3.31	4.16
94668	Limestone Met Sample	**1150**	263	466	3.81	114	2.64	4.08
94669	Limestone Met Sample	**1109**	2130	6070	4.55	78	0.96	4.95
94671	Limestone Met Sample	**1406**	56	176	4.75	9	0.81	4.10
94672	Limestone Met Sample	**872**	102	249	4.02	90	1.84	4.02
94673	Limestone Met Sample	**1363**	358	473	2.79	74	0.79	2.94
94674	Limestone Met Sample	**508**	583	591	5.22	98	0.81	6.55
94676	Limestone Met Sample	**1077**	557	1175	2.21	460	1.51	4.90
94677	Limestone Met Sample	**1359**	106	231	0.52	46	0.13	0.49
60034	Limestone Met Sample	**790**	215	316	0.31	194	1.55	0.21
69467	Limestone Met Sample	**1746**	40	54	0.26	55	0.55	0.13
69468	Limestone Met Sample	**1866**	198	462	0.57	176	1.59	0.55
69469	Limestone Met Sample	**1261**	335	965	1.90	241	5.88	2.10
69471	Limestone Met Sample	**1073**	117	300	1.25	83	0.77	1.27
69472	Limestone Met Sample	**1377**	180	507	0.73	115	1.36	0.62
69473	Limestone Met Sample	**1314**	37	163	0.80	13	0.30	0.44
69474	Limestone Met Sample	**572**	69	159	0.29	40	0.31	0.14
50327	Dyke Met Sample	**1003**	186	314	4.37	45	1.39	3.75
50328	Dyke Met Sample	**1283**	116	124	5.55	87	1.33	5.00
50329	Dyke Met Sample	**1984**	662	1410	5.90	336	6.02	4.86
50331	Dyke Met Sample	**1092**	47	119	4.56	34	0.97	2.84
50332	Dyke Met Sample	**684**	93	180	2.69	23	0.56	2.16
50333	Dyke Met Sample	**1178**	39	61	5.15	17	0.56	3.92
50334	Dyke Met Sample	**431**	72	151	4.72	25	0.86	3.67
43851	Dyke Met Sample	**1764**	977	1610	5.10	312	2.89	5.75
43852	Dyke Met Sample	**1939**	160	299	5.15	57	2.39	5.65
43853	Dyke Met Sample	**1508**	128	327	3.68	84	2.07	4.13
43854	Dyke Met Sample	**401**	67	248	0.81	70	0.81	0.85
43856	Dyke Met Sample	**2228**	15	77	0.43	7	0.09	0.45
N298383	Not Available	**1162**	Not Available					
N298386		**1290**						
N298384		**1484**						
N298391		**1663**						
N298389		**1288**						
N298382		**1175**						
N298387		**1227**						
N298388		**1234**						
TOTAL		**35780**	331	705	2.96	116	1.72	2.93

Table 2.7 – Dyke Hardness Drillcore Sample Inventory

Sample No.	Description	Domain	Mass (g)	Zn (ppm)	Pb (ppm)	Cu (ppm)	Fe (%)	S (%)	Au (g/t)	Ag (g/t)	Mn (ppm)
51738	Hardness Sample	DYKE	**1260**	300	192	14	3.88	4.22	0.39	54.80	6340
51740	Hardness Sample	DYKE	**1133**	1015	199	38	3.62	2.36	1.37	80.10	6320
51743	Hardness Sample	DYKE	**1110**	57	8	16	3.47	2.09	0.19	3.10	6110
56247	Hardness Sample	DYKE	**1006**	653	183	34	5.11	2.46	0.90	57.20	6050
56248	Hardness Sample	DYKE	**1127**	239	99	25	5.16	3.61	0.51	33.30	6860
56252	Hardness Sample	DYKE	**1155**	1065	408	35	5.30	2.57	0.53	40.60	5630
62729	Hardness Sample	DYKE	**1266**	74	18	18	3.36	3.74	0.20	4.00	10400
62736	Hardness Sample	DYKE	**1152**	451	164	20	3.72	4.39	0.65	100.00	9650
62738	Hardness Sample	DYKE	**1392**	261	65	22	4.25	3.95	1.12	10.40	10050
298338	Hardness Sample	DYKE	**1092**	79	20	70	3.23	2.19	0.88	50.80	15100
298340	Hardness Sample	DYKE	**1203**	809	168	19	3.36	3.13	0.71	39.20	10600
85753	Hardness Sample	DYKE	**1253**	182	34	57	5.58	6.37	0.43	65.40	5790
85754	Hardness Sample	DYKE	**668**	168	13	57	4.95	5.75	0.42	44.50	13350
85758	Hardness Sample	DYKE	**1236**	113	38	36	6.07	7.23	0.43	57.00	12050
TOTAL			**14794**	400	109	34	4.35	3.78	0.65	43.89	9045

Table 2.8 – Limestone Hardness Drillcore Sample Inventory

Sample No.	Description	Domain	Mass (g)	Zn (ppm)	Pb (ppm)	Cu (ppm)	Fe (%)	S (%)	Au (g/t)	Ag (g/t)	Mn (ppm)
51609	Hardness Sample	LIMESTONE	**1059**	215	87	61	2.91	2.44	1.81	29.70	76300
51616	Hardness Sample	LIMESTONE	**635**	83	33	15	0.98	0.71	0.20	19.90	8860
78726	Hardness Sample	LIMESTONE	**1411**	62	28	21	1.82	1.34	1.55	34.20	16250
78727	Hardness Sample	LIMESTONE	**1030**	117	25	22	0.92	0.82	1.71	7.80	3480
78723	Hardness Sample	LIMESTONE	**1193**	265	96	64	0.90	0.84	0.40	50.50	4150
298281	Hardness Sample	LIMESTONE	**2685**	71	12	28	0.46	0.25	0.13	6.30	1990
298282	Hardness Sample	LIMESTONE	**1743**	28	7	6	0.24	0.09	0.15	11.80	5490
298283	Hardness Sample	LIMESTONE	**1276**	175	102	16	0.33	0.18	0.71	85.50	30100
298284	Hardness Sample	LIMESTONE	**1257**	740	279	26	0.55	0.41	1.26	69.20	35700
84372	Hardness Sample	LIMESTONE	**1425**	394	110	31	0.24	0.14	1.44	93.90	73800
84378	Hardness Sample	LIMESTONE	**2068**	66	11	31	0.23	0.20	0.09	13.90	3990
85813	Hardness Sample	LIMESTONE	**1287**	94	38	31	1.88	1.03	1.49	70.70	10550
85817	Hardness Sample	LIMESTONE	**1454**	80	16	16	1.88	0.64	0.58	28.40	4000
TOTAL			**18523**	168	58	28	0.90	0.60	0.78	36.94	18800

Table 2.9 – TUFF Hardness Drillcore Sample Inventory

Sample No.	Description	Domain	Mass (g)	Zn (ppm)	Pb (ppm)	Cu (ppm)	Fe (%)	S (%)	Au (g/t)	Ag (g/t)	Mn (ppm)
77426	Hardness Sample	TUFF	**2045**	1120	27	21	4.32	2.79	1.27	15.00	39800
77427	Hardness Sample	TUFF	**3270**	245	9	37	2.63	2.19	0.76	6.30	9750
96560	Hardness Sample	TUFF	**1839**	25	1	3	3.14	0.98	0.24	1.00	856
96562	Hardness Sample	TUFF	**1756**	18	1	11	2.37	0.74	0.89	1.00	1060
89724	Hardness Sample	TUFF	**1642**	39	9	15	1.64	1.91	0.78	7.30	17100
89726	Hardness Sample	TUFF	**1889**	46	10	39	1.96	2.22	1.03	8.90	10800
74640	Hardness Sample	TUFF	**1419**	128	35	32	3.63	4.16	1.00	36.80	9900
74643	Hardness Sample	TUFF	**1796**	66	22	32	3.13	3.52	0.57	25.40	4010
57329	Hardness Sample	TUFF	**1583**	114	24	16	2.51	0.07	0.42	28.70	26200
57331	Hardness Sample	TUFF	**2145**	63	7	12	3.53	0.09	0.19	9.90	3710
TOTAL			**19384**	203	14	23	2.89	1.85	0.71	12.80	12169

Following the sample inventory process, each of the hardness testwork drillcore composites was individually stage crushed to 100% passing 6 mesh (3.35mm). Each composite was then thoroughly blended via rotary splitter and set aside for Bond Ball Work Index (BWi) testing. The Black Shale drillcore MET Sample was also stage crushed to 100% passing 6 mesh (3.35mm) and thoroughly blended. Once blended, a 15kg subsample was taken and set aside for Bond BWi testwork. The remaining ~38kg of Black Shale composite material was then further stage crushed to 100% passing 10 mesh (1.7mm), reblended and split into 2.0kg testwork charges ahead of gravity, flotation and cyanidation testwork. All samples were freezer stored once crushed to mitigate further risk of sample oxidation.

The Limestone, Dyke and TUFF MET samples were all individually stage crushed to 100% passing 10 mesh (1.7mm) and thoroughly blended via rotary splitter. Once blended, each composite was split into replicate 2.0kg testwork charges ahead of metallurgical testwork. Once at 10 mesh, representative subsamples were taken from each composite and submitted for Pb, Zn, Fe, Ag and Au head assay at Blue Coast Research. S and C assays were also undertaken and these analyses were subcontracted to SGS Minerals Services in Vancouver, BC. The following table summarises the measured head assays for each composite. For completeness, the calculated composite head assays from the drillcore geochem assays are shown where available. Generally good agreement between the expected and actual head assays was achieved.

Au and Ag head grades ranged from 0.7 to 1.7g/t and 13 to 116g/t respectively with the highest grades naturally observed in the High Grade sample and the lowest grades observed in the TUFF sample for silver and Dyke sample for gold. Pb and Zn grades for all the composites were low (<0.1% combined) with the exception of the Black Shale composite where Pb and Zn grades of 0.24% and 0.43% respectively were observed.

Sulphur grades were variable and ranged from a low of 0.77% (Limestone) to 3.64% (Dyke) which is interesting as these domains are considered to be highly intermixed and it will be almost impossible to mine them selectively form each other (pers. Comm. M. Poliquin, Almaden Minerals, September 2012). Therefore, it can be concluded that the sulphide content is variable and this may have an impact on the flotation strategies employed for each of these domains.

Carbon content ranged from 1.45% (Limestone) to 7.69% (Dyke) further highlighting the differences between each of these two domains. However, this analysis does not discriminate between graphitic carbon and carbon in carbonates.

Table 2.10 - Metallurgical Composite Head Assays

Sample ID	Pb % Pb-AR-AA	Zn % Zn-AR-AA	Fe % Fe-AR-AA	Ag g/t Ag-AR-AA	Au g/t Au-FA-AA	C % LECO	S % LECO
Dyke Met Head 1	0.02	0.05	3.89	40	0.49	1.46	3.62
Dyke Met Head 2	0.02	0.03	3.79	40	0.60	1.43	3.62
Dyke Met Head 3	0.01	0.05	3.91	40	1.04	1.45	3.69
Dyke Met Average	**0.02**	**0.04**	**3.86**	**40**	**0.71**	**1.45**	**3.64**
Drillcore Assay	**0.01**	**0.04**	**4.11**	**42**	**0.74**	**N/A**	**3.69**

Sample ID	Pb % Pb-AR-AA	Zn % Zn-AR-AA	Fe % Fe-AR-AA	Ag g/t Ag-AR-AA	Au g/t Au-FA-AA	C % LECO	S % LECO
Limestone Met Head 1	0.02	0.02	0.98	40	0.55	7.8	0.74
Limestone Met Head 2	0.01	0.02	1.00	38	0.52	7.66	0.78
Limestone Met Head 3	0.01	0.02	0.97	44	0.67	7.62	0.79
Limestone Met Head	**0.01**	**0.02**	**0.98**	**41**	**0.58**	**7.69**	**0.77**
Drillcore Assay	**0.01**	**0.01**	**1.20**	**38**	**0.75**	**N/A**	**0.93**

Sample ID	Pb % Pb-AR-AA	Zn % Zn-AR-AA	Fe % Fe-AR-AA	Ag g/t Ag-AR-AA	Au g/t Au-FA-AA	C % LECO	S % LECO
Limestone/Dyke HG Head 1	0.05	0.07	2.26	112	1.98	5.07	2.53
Limestone/Dyke HG Head 2	0.04	0.06	2.27	138	2.00	5.04	2.4
Limestone/Dyke HG Head 3	0.04	0.06	2.32	130	2.73	4.99	2.34
High Grade Average	**0.04**	**0.06**	**2.28**	**127**	**2.24**	**5.03**	**2.42**
Drillcore Assay	**0.03**	**0.07**	**2.96**	**116**	**1.72**	**N/A**	**2.93**

Sample ID	Pb % Pb-AR-AA	Zn % Zn-AR-AA	Fe % Fe-AR-AA	Ag g/t Ag-AR-AA	Au g/t Au-FA-AA	C % LECO	S % LECO
Black Shale Head 1	0.23	0.42	3.10	44	0.96	3.7	3.32
Black Shale Head 2	0.24	0.45	3.29	42	1.03	3.66	3.4
Black Shale Head 3	0.23	0.43	3.20	48	0.94	3.67	3.41
Black Shale Average	**0.23**	**0.43**	**3.20**	**45**	**0.98**	**3.68**	**3.38**
Drillcore Assay	**0.24**	**0.43**	**3.89**	**40**	**0.97**	**N/A**	**3.69**

Sample ID	Pb % Pb-AR-AA	Zn % Zn-AR-AA	Fe % Fe-AR-AA	Ag g/t Ag-AR-AA	Au g/t Au-FA-AA	C % LECO	S % LECO
Tuff Met Head 1	0.01	0.02	2.57	10	0.85	1.06	1.98
Tuff Met Head 2	0.01	0.02	2.50	8	0.87	1.04	1.9
Tuff Met Head 3	0.01	0.02	2.51	8	0.85	1.01	1.97
Tuff Met Average	**0.01**	**0.02**	**2.53**	**9**	**0.86**	**1.04**	**1.95**
Drillcore Assay	**0.001**	**0.01**	**2.79**	**13**	**0.83**	**N/A**	**2.02**

The density each of the sample was also measured via volumetric flask method.

Table 2.11 - Density Results

Sample	Density (g/cc)
Dyke	2.59
Limestone	2.65
Black Shale	2.63
TUFF (volcanic)	2.33

3. SAMPLE SELECTION & PREPARATION OF SUPPLEMENTARY SAMPLES

Midway through the project, it was decided that flotation testwork would only be conducted on "fresh" drillcore samples. Therefore drillcore from the Dyke, Limestone and TUFF zones was shipped to BCR in December 2012. For completeness, additional Black Shale drillcore was also shipped by Almaden but this sample was not earmarked for further testing as the original Black Shale comprised of fresh drillcore. The following tables summarise the weights and sample IDs of the supplementary MET samples.

Table 3.1 – Supplementary Black Shale Drillcore MET Sample

Sample No.	Description	Domain	Mass (kg)
122284	Met Sample	Blackshale	**1.40**
112114	Met Sample	Blackshale	**0.71**
122287	Met Sample	Blackshale	**1.16**
112117	Met Sample	Blackshale	**1.49**
112116	Met Sample	Blackshale	**1.51**
122283	Met Sample	Blackshale	**1.41**
122288	Met Sample	Blackshale	**1.26**
122286	Met Sample	Blackshale	**1.37**
112120	Met Sample	Blackshale	**1.57**
112118	Met Sample	Blackshale	**1.65**
112119	Met Sample	Blackshale	**1.61**
101949	Met Sample	Blackshale	**0.93**
101948	Met Sample	Blackshale	**1.26**
101953	Met Sample	Blackshale	**1.86**
101952	Met Sample	Blackshale	**1.29**
101954	Met Sample	Blackshale	**0.83**
101956	Met Sample	Blackshale	**1.22**
122281	Met Sample	Blackshale	**1.33**
106266	Met Sample	Blackshale	**1.10**
122282	Met Sample	Blackshale	**1.36**
122280	Met Sample	Blackshale	**1.36**
106267	Met Sample	Blackshale	**1.07**
106261	Met Sample	Blackshale	**1.16**
106262	Met Sample	Blackshale	**1.03**
106263	Met Sample	Blackshale	**1.09**
106264	Met Sample	Blackshale	**1.26**
106260	Met Sample	Blackshale	**1.34**
112479	Met Sample	Blackshale	**1.55**
112477	Met Sample	Blackshale	**1.52**
112476	Met Sample	Blackshale	**1.66**
112474	Met Sample	Blackshale	**0.82**
112478	Met Sample	Blackshale	**1.42**
112473	Met Sample	Blackshale	**1.66**
TOTAL			**43.19**

Table 3.2 – Supplementary Dyke Drillcore Met Sample Inventory

Sample No.	Description	Domain	Mass (kg)
85760	Met Sample	Dyke	**1.15**
85756	Met Sample	Dyke	**1.15**
85759	Met Sample	Dyke	**1.27**
85854	Met Sample	Dyke	**0.64**
85714	Met Sample	Dyke	**0.64**
85751	Met Sample	Dyke	**1.32**
85757	Met Sample	Dyke	**1.07**
85708	Met Sample	Dyke	**1.48**
85752	Met Sample	Dyke	**1.25**
N298341	Met Sample	Dyke	**1.23**
N298344	Met Sample	Dyke	**1.04**
N298356	Met Sample	Dyke	**1.04**
N298346	Met Sample	Dyke	**1.03**
N298342	Met Sample	Dyke	**1.15**
85749	Met Sample	Dyke	**1.28**
N298339	Met Sample	Dyke	**1.44**
N298343	Met Sample	Dyke	**1.17**
N298318	Met Sample	Dyke	**1.14**
62778	Met Sample	Dyke	**0.61**
62740	Met Sample	Dyke	**1.16**
62741	Met Sample	Dyke	**1.23**
56256	Met Sample	Dyke	**1.17**
56253	Met Sample	Dyke	**1.08**
62737	Met Sample	Dyke	**0.66**
62732	Met Sample	Dyke	**1.28**
62739	Met Sample	Dyke	**1.19**
62734	Met Sample	Dyke	**0.51**
62786	Met Sample	Dyke	**1.17**
56257	Met Sample	Dyke	**1.14**
62733	Met Sample	Dyke	**1.22**
62742	Met Sample	Dyke	**1.24**
56194	Met Sample	Dyke	**0.83**
56254	Met Sample	Dyke	**0.55**
56251	Met Sample	Dyke	**1.21**
56249	Met Sample	Dyke	**1.07**
51687	Met Sample	Dyke	**1.31**
56184	Met Sample	Dyke	**1.05**
51742	Met Sample	Dyke	**1.03**
56196	Met Sample	Dyke	**0.92**
51766	Met Sample	Dyke	**1.11**
51741	Met Sample	Dyke	**1.15**
51737	Met Sample	Dyke	**1.14**
51692	Met Sample	Dyke	**1.04**
51739	Met Sample	Dyke	**1.12**
51736	Met Sample	Dyke	**1.13**
TOTAL			**48.71**

Table 3.3 - Supplementary TUFF Drillcore Met Sample Inventory

Sample No.	Description	Domain	Mass (kg)
74642	Met Sample	Tuff	**1.08**
89732	Met Sample	Tuff	**1.06**
89731	Met Sample	Tuff	**1.13**
89701	Met Sample	Tuff	**2.16**
89728	Met Sample	Tuff	**1.03**
89723	Met Sample	Tuff	**1.39**
89729	Met Sample	Tuff	**1.15**
89727	Met Sample	Tuff	**1.15**
96566	Met Sample	Tuff	**1.75**
89722	Met Sample	Tuff	**1.12**
96563	Met Sample	Tuff	**1.66**
89700	Met Sample	Tuff	**1.90**
96569	Met Sample	Tuff	**1.63**
96561	Met Sample	Tuff	**1.52**
96577	Met Sample	Tuff	**2.01**
96564	Met Sample	Tuff	**1.68**
74547	Met Sample	Tuff	**0.88**
74641	Met Sample	Tuff	**1.78**
74638	Met Sample	Tuff	**1.26**
74639	Met Sample	Tuff	**1.71**
74637	Met Sample	Tuff	**1.33**
77429	Met Sample	Tuff	**1.63**
774636	Met Sample	Tuff	**1.48**
77431	Met Sample	Tuff	**1.62**
77428	Met Sample	Tuff	**2.37**
77461	Met Sample	Tuff	**1.32**
77433	Met Sample	Tuff	**3.23**
57327	Met Sample	Tuff	**1.61**
57328	Met Sample	Tuff	**1.58**
57336	Met Sample	Tuff	**1.60**
57321	Met Sample	Tuff	**0.83**
TOTAL			**47.57**

Table 3.4 - Supplementary Limestone Drillcore Met Sample Inventory

Sample No.	Description	Domain	Mass (kg)
85833	Met Sample	Limestone	**1.74**
85818	Met Sample	Limestone	**2.19**
85819	Met Sample	Limestone	**2.76**
85716	Met Sample	Limestone	**1.02**
84379	Met Sample	Limestone	**2.37**
85714	Met Sample	Limestone	**0.78**
85712	Met Sample	Limestone	**1.47**
84376	Met Sample	Limestone	**1.32**
84359	Met Sample	Limestone	**2.82**
84374	Met Sample	Limestone	**0.72**
84377	Met Sample	Limestone	**1.59**
84373	Met Sample	Limestone	**1.38**
N298243	Met Sample	Limestone	**1.47**
N298236	Met Sample	Limestone	**1.11**
84366	Met Sample	Limestone	**1.89**
N298286	Met Sample	Limestone	**2.10**
N298231	Met Sample	Limestone	**1.31**
N298280	Met Sample	Limestone	**1.19**
N298263	Met Sample	Limestone	**1.13**
78724	Met Sample	Limestone	**1.33**
78728	Met Sample	Limestone	**1.25**
78769	Met Sample	Limestone	**1.19**
78721	Met Sample	Limestone	**1.80**
78789	Met Sample	Limestone	**1.94**
51540	Met Sample	Limestone	**1.08**
51613	Met Sample	Limestone	**0.55**
51614	Met Sample	Limestone	**0.58**
78722	Met Sample	Limestone	**1.10**
78699	Met Sample	Limestone	**1.19**
51569	Met Sample	Limestone	**1.65**
51611	Met Sample	Limestone	**3.96**
51608	Met Sample	Limestone	**1.02**
51612	Met Sample	Limestone	**2.32**
TOTAL			51.24

Each of the supplementary samples was stage crushed to 100% passing 10 mesh, thoroughly blended via rotary splitter and split into replicate 2.0kg charges ahead of the flotation testwork. All samples were freezer stored once crushed to mitigate further risk of sample oxidation. Head assays for each composite were also measured.

Table 3.5 – Supplementary MET Sample Head Assays

Sample ID	Pb % Pb-AR-AA	Zn % Zn-AR-AA	Fe % Fe-AR-AA	Ag g/t Ag-AR-AA	Au g/t Au-FA-AA
Dyke Supp. Head	0.02	0.03	3.60	38	0.68
Limestone Supp. Head	0.01	0.01	0.84	44	0.67
TUFF Supp. Head	0.01	0.01	2.54	12	0.78

4. METALLURGICAL TESTWORK RESULTS

This testwork program was subdivided into four main areas of focus as described below:

- Bond Ball Work Index test
- E-GRG (Gravity Recoverable Gold) test
- Cyanidation on E-GRG tails
- Rougher flotation tests

This testwork program was intended to be an amenability study and no previous metallurgical testwork data was available for review. Therefore, the testwork conducted was undertaken using benchmark conditions and parameters. The Bond Ball Work Index test is a standard test that provides an estimate of the amount of energy required to achieve a given ball mill product size. The E-GRG test is a standard test intended to provide an indication of the quantity of gravity recoverable gold in a given sample. Often in production scale plants, the gravity tails is treated via cyanidation to recover non GRG gold therefore, cyanidation tests were conducted on the E-GRG test tails at standard conditions. Flotation was conducted on the whole ore using a conventional bulk sulphide flotation process tailored for gold and silver recovery. Due to the higher grades of lead and zinc in the Black Shale sample, flotation focused on differential lead/zinc rougher flotation using a flowsheet known to be successful on a similar deposit in the region.

The following sections of the report communicate the metallurgical testwork results for this study.

4.1. Bond Ball Work Index (BWi)

A Bond Ball Work Index (BWi) test was undertaken on the samples with a closing size of 100 mesh (150 microns). The full datasheets can be found in the appendices.

Table 4.1 - Bond Work Index Results

Sample	Bond Ball Work Index		Closing Size (µm)
	kwh/ton	kwh/tonne	
Dyke	13.2	**14.6**	150
Limestone	12.0	**13.2**	150
Black Shale	16.8	**18.6**	150
TUFF (volcanic)	9.5	**10.5**	150

4.2. Gravity Recoverable Gold Testwork

Gravity gold recovery can be of significant economic importance for gold ores (A. Laplante, 2000). It is often beneficial to recover gold at the earliest possible opportunity in the concentrator flowsheet as overgrinding of gold in the grinding circuit and tarnishing of gold surfaces in downstream flotation circuits can have a detrimental effect on overall gold recovery. Recovering coarse, free gold at the moment of liberation is critical when achieving maximum gold recovery and it can often allow for the production of high grade, low mass gold products that may be intensively leached onsite or sold directly to gold refineries.

Gravity Recoverable Gold (GRG) testwork is typically performed using a laboratory scale centrifugal gravity concentrating machine such as a Knelson MD-3 or Falcon L-40 concentrator. For this study, a Knelson MD-3 concentrator was employed.

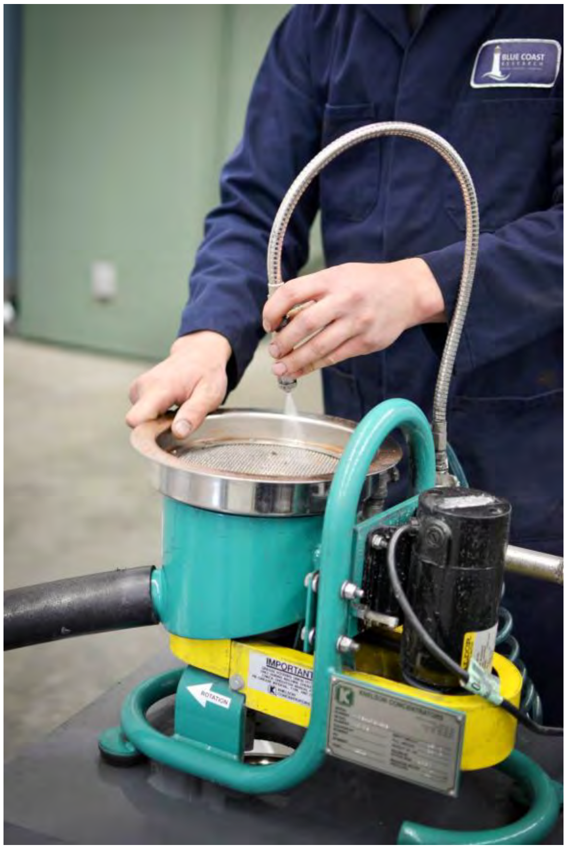


Figure 4.1 - Knelson MD-3 Concentrator at Blue Coast Research

An ERG test was completed on the four domains as follows.

- Dyke
- Limestone
- Black Shale
- TUFF (volcanic)

20kg of material from each of the above ore type composites was passed through the Knelson MD-3 concentrator at p80s of ~850µm, 180µm and 75µm. Detailed results from each test can be found in the appendix of this report and the summary tables from each test are included as follows for simplicity.

Table 4.2 – Dyke MET Sample EGRG Results Summary Table

Grind Size	Product	Mass		Assay	Metal Units	Distribution
		grams	wt %	g/t	Au	%
P_{80} = 792 microns	Stage 1 Concentrate	87.1	0.4	24.28	2,115.4	**13.4**
	Stage 1 Tails	19,912.9	99.6	0.69	13,670.1	86.6
P_{80} = 267 microns	Stage 2 Concentrate	98.9	0.5	19.32	1,911.3	**12.1**
	Stage 2 Tails	19,813.9	99.1	0.59	11,758.8	74.5
P_{80} = 74 microns	Stage 3 Concentrate	96.4	0.5	37.42	3,607.1	**22.9**
	Stage 3 Tails Sample	512.3	2.6	0.44	225.0	1.4
	Final Tails	18,044.6	90.2	0.44	7,926.6	50.2
	Head	20,000.0	100.0	0.79	15,785.5	100.0
	Total Concentrate	**282.5**	**1.4**	**27.02**	**7,633.8**	**48.4**
	Total Tailings	18,556.9	92.8	0.44	8,151.6	51.6

E-GRG Number = 48.4

Table 4.3 – Limestone MET EGRG Results Summary Table

Grind Size	Product	Mass		Assay	Metal Units	Distribution
		grams	wt %	g/t	Au	%
P_{80} = 956 microns	Stage 1 Concentrate	74.0	0.4	41.49	3,070.2	**19.6**
	Stage 1 Tails	19,926.0	99.6	0.63	12,590.1	80.4
P_{80} = 250 microns	Stage 2 Concentrate	85.1	0.4	34.30	2,919.1	**18.6**
	Stage 2 Tails	19,840.9	99.2	0.62	12,274.8	78.4
P_{80} = 75 microns	Stage 3 Concentrate	74.7	0.4	42.90	3,206.0	**20.5**
	Stage 3 Tails Sample	508.2	2.5	0.34	174.5	1.1
	Final Tails	18,320.0	91.6	0.34	6,290.6	40.2
	Head	20,000.0	100.0	0.78	15,660.3	100.0
	Total Concentrate	**233.9**	**1.2**	**39.32**	**9,195.2**	**58.7**
	Total Tailings	18,828.2	94.1	0.34	6,465.1	41.3

E-GRG Number = 58.7

Table 4.4 – Black Shale MET Sample EGRG Results Summary Table

Grind Size	Product	Mass		Assay	Metal Units	Distribution
		grams	wt %	g/t	Au	%
P_{80} = 747 microns	Stage 1 Concentrate	90.5	0.5	65.41	5,919.3	**24.2**
	Stage 1 Tails	19,909.5	99.5	0.93	18,510.0	75.8
P_{80} = 194 microns	Stage 2 Concentrate	91.3	0.5	47.75	4,357.6	**17.8**
	Stage 2 Tails	19,818.3	99.1	0.93	18,379.1	75.2
P_{80} = 70 microns	Stage 3 Concentrate	86.1	0.4	36.31	3,126.0	**12.8**
	Stage 3 Tails Sample	354.0	1.8	0.60	211.9	0.9
	Final Tails	18,065.0	90.3	0.60	10,814.5	44.3
	Head	20,000.0	100.0	1.22	24,429.3	100.0
	Total Concentrate	**267.9**	**1.3**	**50.04**	**13,402.9**	**54.9**
	Total Tailings	18,419.0	92.1	0.60	11,026.4	45.1

Table 4.5 – TUFF MET Sample EGRG Results Summary Table

Grind Size	Product	Mass		Assay	Metal Units	Distribution
		grams	wt %	g/t	Au	%
P_{80} = 825 microns	Stage 1 Concentrate	77.1	0.4	11.88	915.4	**5.4**
	Stage 1 Tails	19,922.9	99.6	0.81	16,101.1	94.6
P_{80} = 226 microns	Stage 2 Concentrate	73.6	0.4	10.73	790.5	**4.6**
	Stage 2 Tails	19,849.3	99.2	0.77	15,310.5	90.0
P_{80} = 85 microns	Stage 3 Concentrate	77.3	0.4	11.26	870.3	**5.1**
	Stage 3 Tails Sample	642.9	3.2	0.76	491.6	2.9
	Final Tails	18,240.6	91.2	0.76	13,948.6	82.0
	Head	20,000.0	100.0	0.85	17,016.4	100.0
	Total Concentrate	**228.0**	**1.1**	**11.30**	**2,576.2**	**15.1**
	Total Tailings	18,883.5	94.4	0.76	14,440.3	84.9

E-GRG Number = 15.1

The combined gold recovery (EGRG Number) at each of the three grind sizes for each composite ranged from a low of 15.1% (TUFF) to a high of 58.7% (Limestone). The Dyke and Black Shale samples both exhibited gold recoveries similar to the Limestone composite at 48.4% and 54.9% respectively. These three composites would be considered amenable to gravity concentration i.e they contain a significant portion of gravity recoverable gold whereas the TUFF sample is not considered to be amenable.

Table 4.6 - Overview of EGRG Results

Sample	E-GRG Number (%)
Dyke	48.4
Limestone	58.7
Black Shale	54.9
TUFF (volcanic)	15.1

The gold grades to GRG concentrate ranged from 11.3g/t Au (TUFF) to 50g/t (Black Shale).

4.3. Cyanidation of GRG Tails

The EGRG tails from each of the domains were filtered, dried and individually blended prior to splitting into charges ahead of cyanidation testwork. Three cyanidation bottle roll tests were undertaken on each of the three domain composite GRG tails to determine whether the remaining non GRG gold could be extracted into a pregnant leach solution (PLS), further increasing overall gold recovery.

For each domain, two levels on cyanide concentration were tested (3.0g/L and 5.0g/L) and a third test was undertaken at 5.0g/L with a nominal 25 minute regrind on each of the GRG tails samples. All leach tests were conducted at 33% solids, and the pH was maintained at 10.5-11.0 with lime throughout. Cyanide concentration was also maintained over the 48hr leach residence time via titration at regular intervals.

Table 4.7 – Summary of EGRG Tails Cyanidation Conditions and Results

Sample	Test ID	Test Conditions					NaCN Consumption (kg/t CN feed)	Calculated Head (g/t)		Recovery %	
		Residence Time (hrs)	CN Conc	Pulp Density	pH	Regrind		Au	Ag	Au	Ag
Dyke	CN-1	48	3.0 g/L	33.3	10.5 - 11.0	No	5.04	0.54	50.31	60.80	72.17
Dyke	CN-2	48	5.0 g/L	33.3	10.5 - 11.0	No	6.20	0.55	52.80	61.89	81.82
Dyke	CN-9	48	5.0 g/L	33.3	10.5 - 11.0	Yes -25 min	15.99	0.43	46.19	60.86	87.01
Limestone	CN-3	48	3.0 g/L	33.3	10.5 - 11.0	No	9.89	0.47	37.51	61.13	82.94
Limestone	CN-4	48	5.0 g/L	33.3	10.5 - 11.0	No	16.62	0.46	43.47	60.29	86.20
Limestone	CN-10	48	5.0 g/L	33.3	10.5 - 11.0	Yes -25 min	5.90	0.47	52.84	57.63	77.67
Blackshale	CN-5	48	3.0 g/L	33.3	10.5 - 11.0	No	6.40	0.60	41.14	25.55	7.63
Blackshale	CN-6	48	5.0 g/L	33.3	10.5 - 11.0	No	9.49	0.62	41.40	25.32	10.15
Blackshale	CN-11	48	5.0 g/L	33.3	10.5 - 11.0	Yes -25 min	13.58	0.66	55.11	23.01	56.45
TUFF	CN-7	48	3.0 g/L	33.3	10.5 - 11.0	No	10.90	0.85	13.19	43.44	46.92
TUFF	CN-8	48	5.0 g/L	33.3	10.5 - 11.0	No	15.99	0.67	12.82	37.35	47.73
TUFF	CN-12	48	5.0 g/L	33.3	10.5 - 11.0	Yes -25 min	21.32	0.75	13.28	41.49	58.57

The feed size distribution for each of the non-regrind test was determined by the stage 3 grind p80 of the EGRG test which is nominally 75 microns. Particle Size Distributions (PSDs) were performed on tests CN-9, 10, 11 and 12 and the p80s were consistently between 40 and 45 microns.

The results in the above table indicate that:

- The Limestone and Dyke domains exhibited the best overall response to cyanidation of the GRG tails. 60-62% of the non GRG gold was extracted into the PLS. It appears that regrinding or cyanidation at increased cyanide concentration had little or no effect on gold extraction.
- The Black Shale gold extractions were low at 25% regardless of the cyanide concentration employed. Regrinding to a p80 of 45 microns appeared to have no positive effect on gold extraction.
- TUFF gold extractions were consistently low at 37-43%. Regrinding and increased cyanide concentration had no positive effect of gold extraction. Silver extraction was 47% and was increased ~11% by regrinding to 45 microns.
- Overall, silver extractions were variable at 81-82% for the Limestone/Dyke composites. Regrinding had a positive effect of silver extractions for the Black shale and TUFF composites increasing them to 56% and 59% respectively.

If one takes the EGRG results and combines them with the best cyanidation results (including a regrind on the EGRG tails where it showed a positive benefit), the following overall gold recoveries can be calculated:

Table 4.8 – Projected Combined Gravity and Cyanidation Gold Recovery

Sample ID	EGRG Au Rec (%)	Cyanidation Au Rec (%)	Total Au Rec (%)
Dyke	48.4	61.9	**80.3**
Limestone	58.7	61.1	**83.9**
Black Shale	54.9	25.6	**66.4**
TUFF	15.1	41.5	**50.3**

The Dyke and Limestone domains exhibit the highest overall gold recoveries of 80% and 84% respectively. The Black Shale and TUFF domains return somewhat lower overall gold recoveries at 66% and 50% respectively.

4.4. Bulk Rougher Flotation

As an alternative to gravity recovery and cyanidation, flotation of the whole ore was investigated as an alternative. The initial flotation program consisted of bulk flotation tests on the four domain samples in addition to bulk flotation on the High Grade sample.

All bulk flotation tests were conducted at natural pH with 300g/t copper sulphate, between 150-200g/t SIPX, 45g/t 3418A and F-140 frother as needed to produce a stable froth phase. Total rougher flotation residence time was fixed at 11 minutes and flotation was conducted over three rougher stages. The primary grind was the main variable tested with the majority of the tests conducted at a nominal 100-120 micron p80. Coarser and finer extremes were tested on the High Grade composite and Limestone/Dyke domains.

Table 4.9 - Bulk Flotation Conditions

Test ID	Grind Time (min)	p80	Charge kg	Cell Size L	pH	Rougher Reagents (g/tonne) CuSO4	SIPX	3418A	F-140	Float Time (min)
BS F-4	16.00	71	1.0	2.0	Natural	300	200	45	57.5	11
HG F-1	16.00	116	2.0	4.0	Natural	300	200	45	34.5	11
HG F-2	25.50	88	2.0	4.0	Natural	300	200	45	34.5	11
HG F-3	10.00	313	2.0	4.0	Natural	300	150	45	34.5	11
HG F-4 *Con for leaching	16.00	116	2.0	4.0	Natural	300	200	45	34.5	11
Dyke F-1	16.00	154	2.0	4.0	Natural	300	200	45	11.5	11
Dyke F-2	21.00	106	2.0	4.0	Natural	300	200	45	11.5	11
Limestone F-1	16.00	156	2.0	4.0	Natural	300	150	45	34.5	11
Limestone F-2	21.00	105	2.0	4.0	Natural	300	150	45	46.0	11
TUFF F-1	16.00	93	2.0	4.0	Natural	300	150	45	34.5	11
TUFF F-2	15.00	98	2.0	8.0	Natural	300	150	45	34.5	11

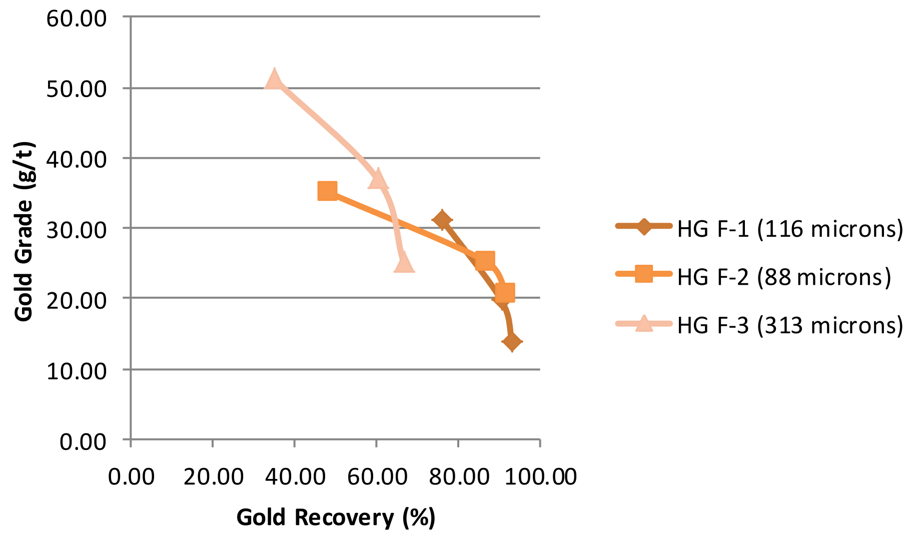
All Pb, Zn, Fe, Ag and Au assays were performed at Blue Coast Research. The S assays were subcontracted to SGS Minerals Services in Vancouver, BC. A summary of the flotation testwork results is included below and the full results can be found in the appendices of this report.

Table 4.10 - Bulk Flotation Results

Test ID	Ro Mass Pull %	Bulk Concentrate Grade (%, g/t) Pb	Zn	Fe	Ag	Au	S	Recovery to Bulk Conc. (%) Pb	Zn	Fe	Ag	Au	S
BS F-4	18.93	1.07	1.81	11.2	**196**	**4.13**	16.0	91.6	98.4	65.3	**83.5**	**93.2**	89.9
HG F-1	12.05	0.24	0.49	14.5	**1001**	**13.94**	18.6	62.3	86.9	66.8	**91.2**	**93.2**	93.4
HG F-2	9.68	0.29	0.57	17.2	**1270**	**23.14**	23.3	86.0	98.4	66.6	**90.3**	**92.5**	92.6
HG F-3	6.15	0.39	0.69	20.9	**1616**	**25.24**	27.2	81.0	81.9	54.6	**75.8**	**66.6**	71.0
Dyke F-1	13.17	0.08	0.17	20.0	**307**	**4.64**	22.8	68.0	70.5	62.9	**88.3**	**89.3**	89.7
Dyke F-2	17.63	0.08	0.12	15.2	**225**	**4.21**	17.7	72.9	84.1	65.1	**87.0**	**94.4**	95.2
Limestone F-1	3.77	0.08	0.17	7.6	**822**	**13.12**	9.0	32.0	37.6	26.5	**62.9**	**54.3**	48.1
Limestone F-2	6.00	0.09	0.12	6.1	**656**	**8.75**	8.6	43.9	88.1	36.9	**72.6**	**76.8**	67.9
TUFF F-1	19.10	0.01	0.03	5.1	**33**	**1.76**	5.0	14.4	42.0	36.0	**49.3**	**42.6**	50.5
TUFF F-2	10.59	0.02	0.04	11.0	**78**	**3.74**	12.7	9.0	80.6	49.7	**63.2**	**52.3**	70.8

The Black Shale bulk flotation test conducted at a primary grind p80 of 71 microns recovered 93% of the gold and 83.5% of the silver into a bulk concentrate grading 4.1g/t Au, 196g/t Ag, 1.8% Zn and 1.1% Pb. Despite the excellent precious metals recoveries, the lead and zinc grades of this concentrate may limit the ability for this concentrate to be directly leached. The lead and zinc recoveries were higher than the gold and silver recoveries, which was not expected considering the objective of the test to produce a bulk concentrate. It is proposed that sequential lead zinc flotation be assessed on this sample and this is discussed in the following section of the report.

The High Grade MET sample showed excellent amenability to bulk rougher flotation. Comprised of high grade intersections of core from the Limestone and Dyke domains, this composite is not necessarily representative of the grade of the deposit but it does give an indication on the potential upside of selective mining and processing of high grade material form the Ixtaca deposit. Three rougher flotation tests were completed on the High Grade composite. All conditions were maintained constant except for the primary grind p80 which was tested at 88 microns (HG F-2), 116 microns (HG F-1) and 313 microns (HG F-3).



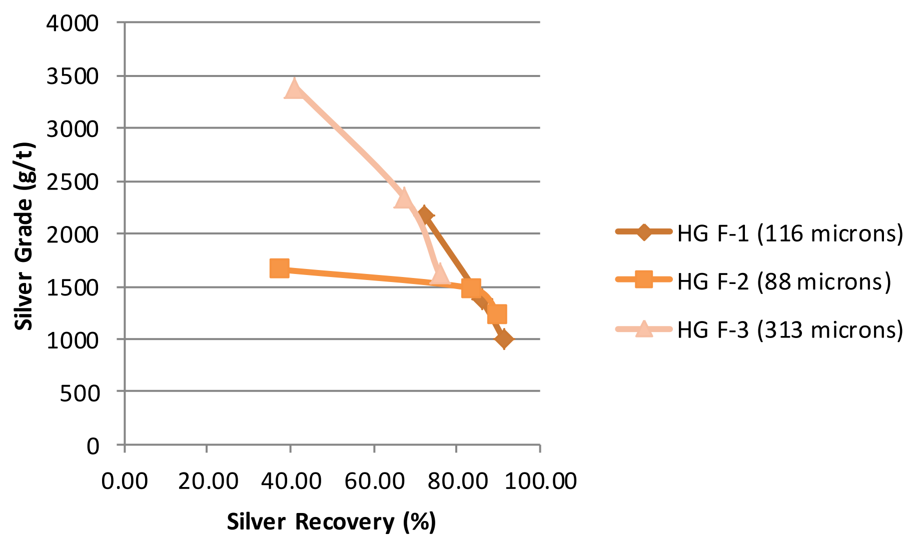
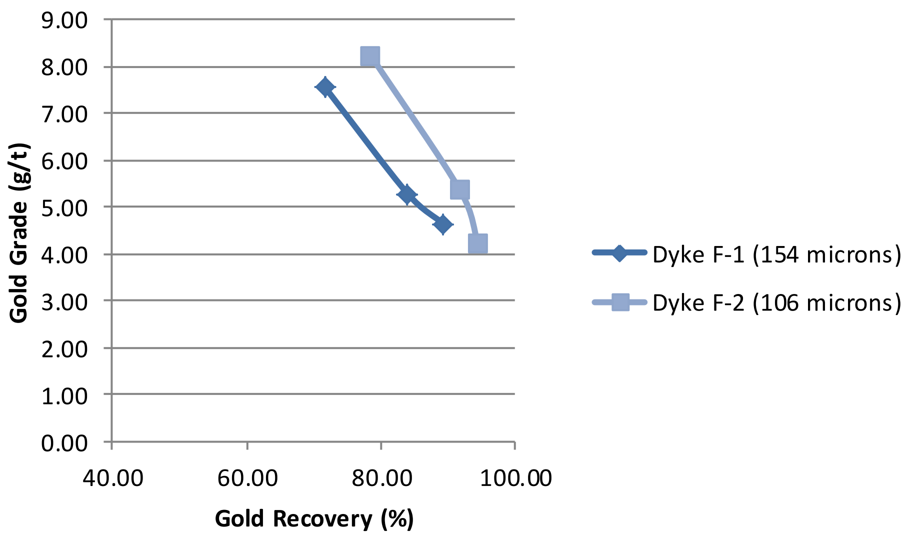


Figure 4.2 – Gold and Silver High Grade Bulk Rougher Flotation Grade Recovery Curves

The coarser primary grind appears to yield higher grade gold and silver bulk rougher concentrates although overall gold and silver recovery is limited to 67% and 76% respectively. There appears to be little benefit in grinding finer than ~115 microns as the 88 micron test (HG F-2) grade recovery curves both reach the same endpoint for gold and silver grade recovery. Test HG F-2 produced a bulk rougher concentrate grading 21g/t Au and 1220g/t Ag. Gold and silver recoveries were an impressive 92% and 90% respectively.

The Dyke bulk rougher grade recovery curves are included below.



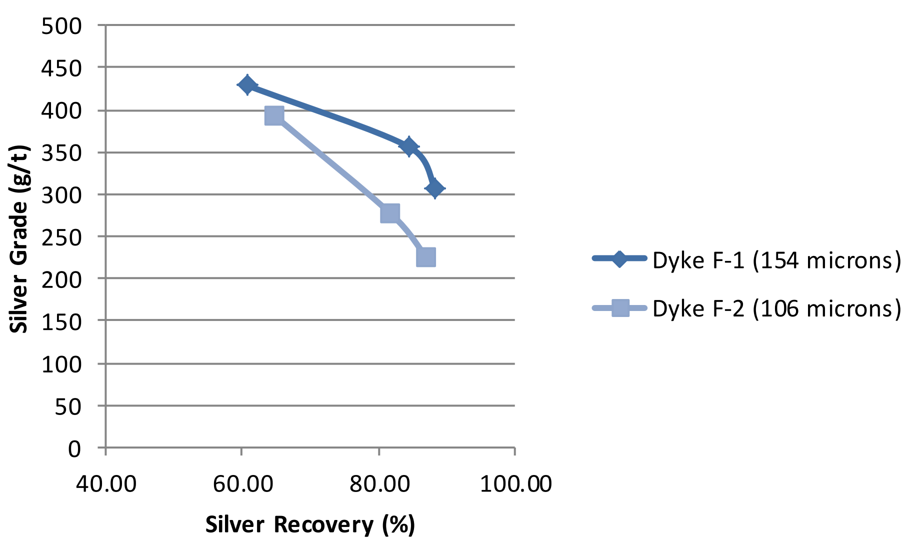
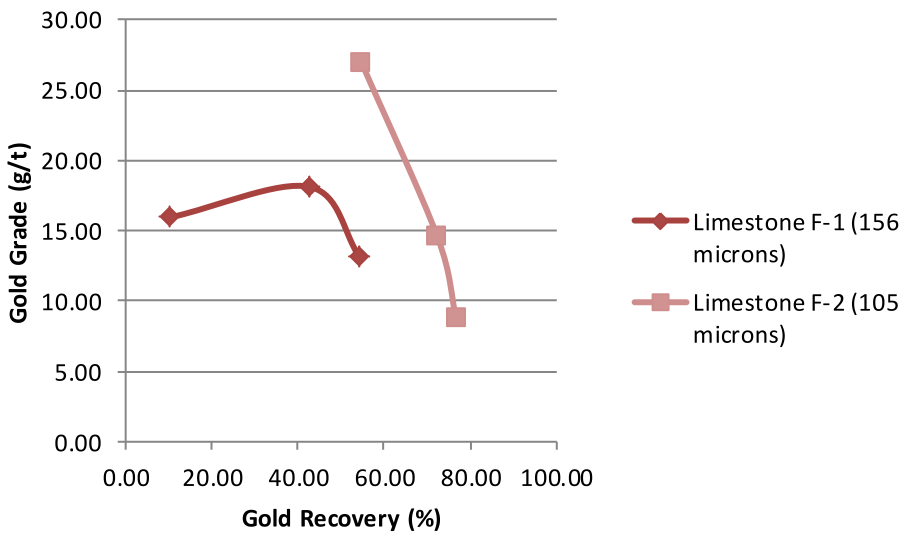


Figure 4.3 – Gold and Silver Dyke Bulk Rougher Flotation Grade Recovery Curves

Although bulk rougher concentrate grades for these samples were considerably lower compared to the High Grade tests, the recoveries were still excellent. The finer grind of 106 microns appears to be beneficial to gold recovery but does not increase silver recovery. Dyke flotation Test F-1 produced a bulk rougher concentrate grading 4.6g/t Au and 307g/t Ag at gold and silver recoveries of 89% and 88% respectively.

The Limestone bulk rougher grade recovery curves are included below.



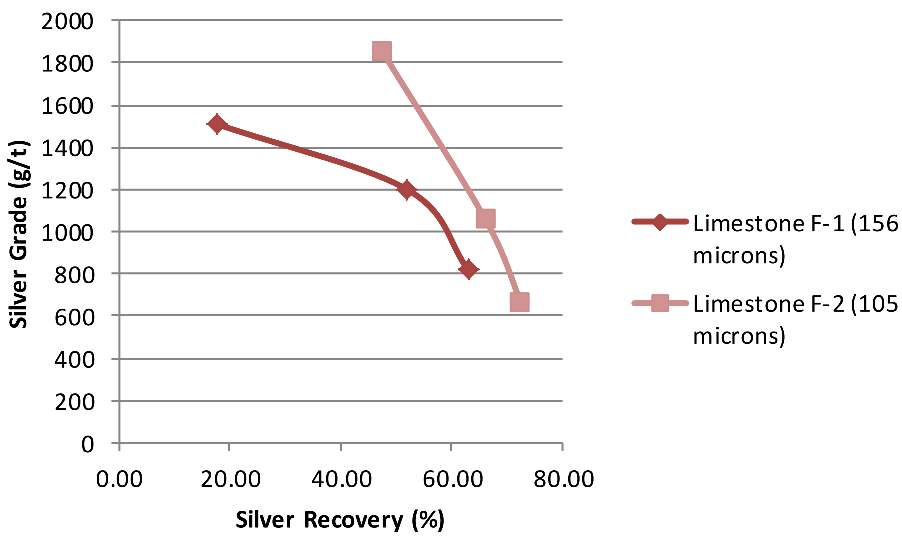
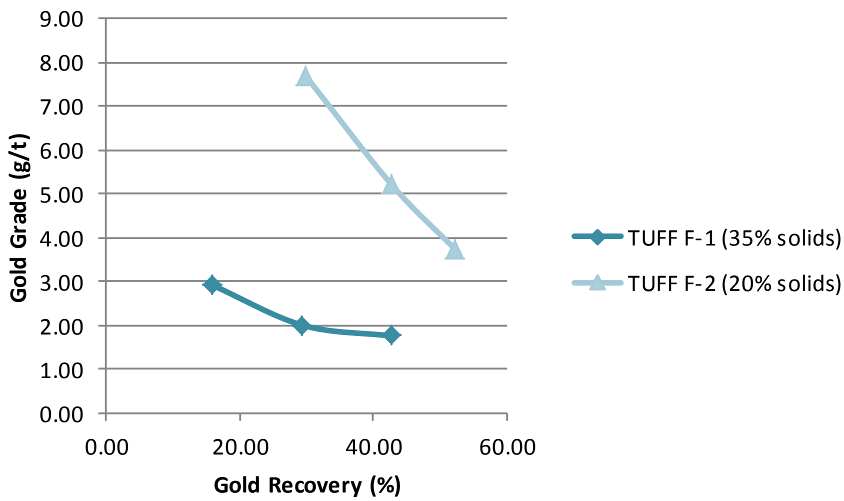


Figure 4.4 - Gold and Silver Limestone Bulk Rougher Flotation Grade Recovery Curves

The two Limestone bulk flotation tests indicate that this particular sample was very sensitive to primary grind p80. A coarse grind of 156 microns resulted in significantly lower gold and silver recoveries compared to a finer primary grind p80 of 105 microns. Compared to the Dyke domain sample, gold and silver recoveries at a primary grind of 105 microns were lower at 77% and 73% respectively. The bulk rougher concentrate graded 9g/t Au and 660g/t Ag which is higher than the best Dyke test. Therefore, the Limestone domain sample appears to produce a higher grade, lower recovery concentrate compared to the Dyke domain sample at the same primary grind p80. This suggests that the Limestone domain sample may be liberation limited and could benefit from an even finer primary grind.



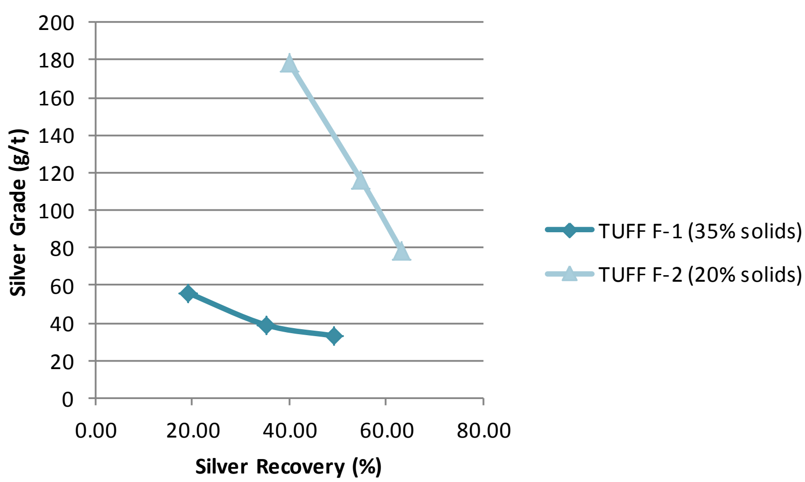


Figure 4.5 - Gold and Silver TUFF Bulk Rougher Flotation Grade Recovery Curves

Two bulk rougher flotation tests were completed on the TUFF MET sample. The first test (TUFF F-1) was conducted at the standard 35% solids pulp density. It was observed during the test that the pulp appeared extremely viscous indicating a rheology/viscosity issue potentially due to the presence of weathered/altered minerals in the TUFF domain. It was therefore decided to repeat the test at a lower pulp density of ~20 (2kg in an 8 litre cell). Although significantly poorer compared to the other domain samples flotation response the test at a lower pulp density improved both recovery and grade for both gold and silver. Bulk rougher flotation of TUFF F-2 at a lower pulp density produced a concentrate grading 4g/t Au and 78g/t Ag at gold and silver recoveries of 52% and 63% respectively.

The graph below summarises the bulk flotation gold results for all four domains plus the High Grade sample.

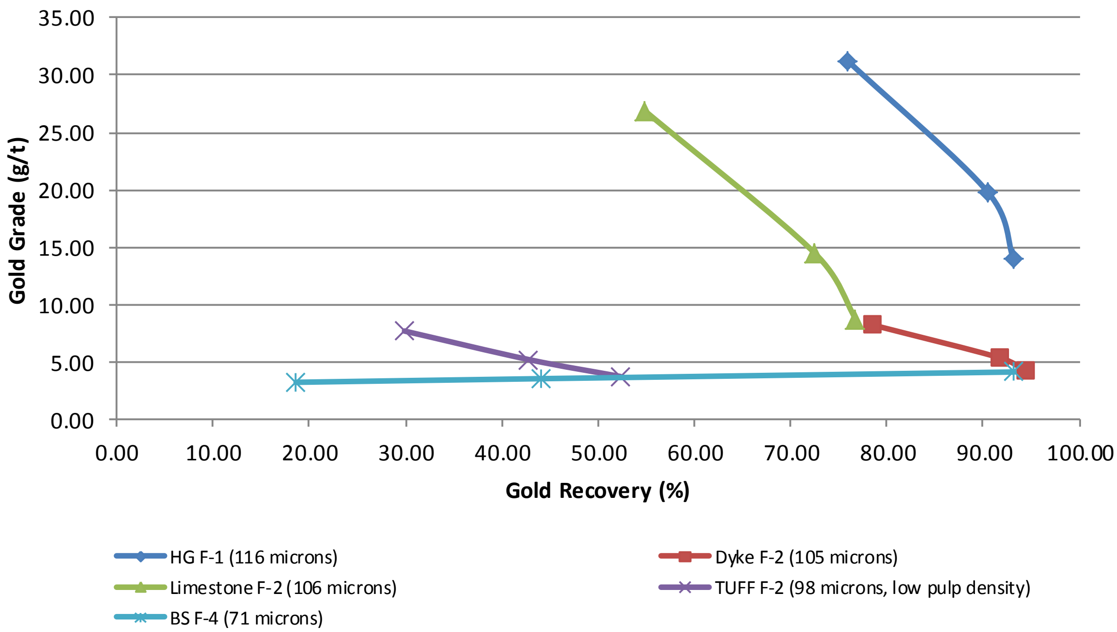


Figure 4.6 – Summary of the Domain Bulk Flotation Results

Clearly some variability exists between the various domains and unsurprisingly, the High Grade MET sample yielded the highest grade concentrate at the +90% gold recovery. Both the Dyke and Black Shale composites produce high gold recoveries +90% albeit at lower bulk concentrate grades. The TUFF appears to behave differently to all other domains (as was observed in the gravity and cyanidation testwork) and yielded a much lower grade concentrate and lower recovery to said concentrate.

4.5. Cyanidation of High Grade Bulk Concentrate

To demonstrate whether the bulk rougher concentrate from the High Grade MET sample could be intensively leached, a bulk flotation test (HG F-4) was conducted to produce bulk concentrate for an intensive cyanidation bottle roll test.

The sample was repulped to 33% solids and leached for 48 hours in the presence of a 20g/L cyanide solution. The pH was maintained between pH10.5-11.0. A gold extraction of 88% and silver extraction of 93% was achieved and the ratio of silver to gold in the PLS was 72:1 suggesting that a Merrill Crowe process would be most suitable for producing doré. The cyanide consumption was extremely high at 92kg/t and gold extraction was somewhat disappointing for an intensive leach. More investigation would be needed to determine what caused the high cyanide consumption and relatively low extraction rate. The presence of cyanide consuming sulphide minerals may be the culprit and it is believed that these samples contain the manganese sulphide alabandite in significant quantities (pers. comm. M Poliquin, February 2013)

4.6. Pb/Zn Differential Flotation

Due to the relatively high grade of lead and zinc in the Black Shale composite, it was decided to assess whether sequential lead zinc rougher flotation would be appropriate for this domain. Three rougher flotation tests were completed as per the following conditions:

- BS F-1 – Primary grind p80 = 168 microns with 500g/t lime, 20g/t NaCN and 60g/t $ZnSO_4$. 30g/t 3418A in the lead circuit, pH 9. 100g/t copper sulphate, 30g/t SIPX and pH 11 in zinc circuit. F-140 frother used throughout.
- Repeat of BS F-1 but with carbon prefloat (no collector no frother)
- Repeat of BS F-1 but at a primary grind p80 of 88 microns.

The lead and zinc grade recovery curves for the tests are summarised below. The data shows that separation of the lead and zinc was achieved however, the lead rougher circuit would require some optimisation to increase both concentrate grade and recovery. The lead grade recovery curves below show delayed lead flotation kinetics suggesting that the zinc-cyanide complex dosage is too high or not needed. Zinc rougher performance was generally good with BS F-2 producing a zinc grade recovery point of 88% zinc recovery to a 9% zinc rougher concentrate grade. Further optimisation of these flotation conditions would be required to enhance the lead zinc separation and increase the grades of the concentrates.

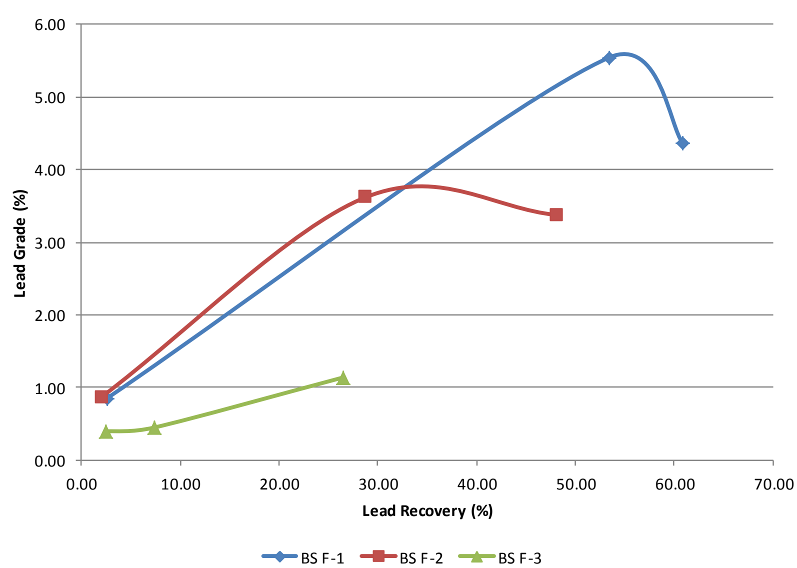


Figure 4.7 – Lead Grade Recovery Curve

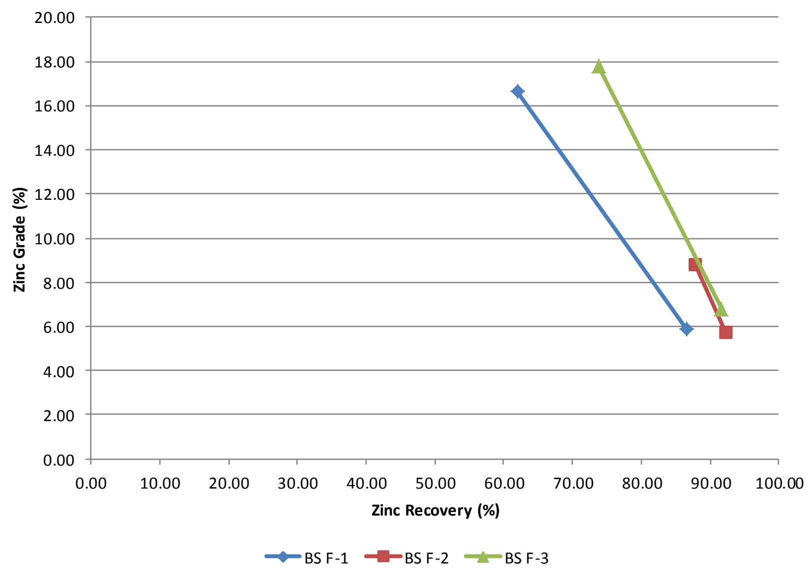


Figure 4.8 – Zinc Grade Recovery Curve

Without mineralogical information, it is challenging to optimise the primary grind; however, the data above does show little or no benefit in grinding finer (88 vs. 160 microns) which suggests that the lead and zinc sulphide minerals are quite coarse and liberate well as coarse grind sizes.

Test BS F-2 also indicated that a carbon prefloat is not required. Indeed, carbon reports overwhelmingly to the tails in all tests undertaken suggesting that it is rather benign and should not pose a problem.

Test BS F-1 indicates that ~30% of the gold and silver reports to the lead concentrate. ~45% of the gold and silver reports to the zinc concentrate and the balance reports to the rougher tails. In a flowsheet such as this it is desirable to "push" as much of the precious metals as possible into the lead circuit where it is ultimately payable. Gold is seldom payable in zinc concentrates. Unfortunately, it appears that the gold has a tendency to report to the zinc concentrate with the flotation conditions tested. If sequential lead/zinc flotation is to be explored further, some optimisation would be required to divert more gold to the lead rougher concentrate.

5. DISCUSSION AND CONCLUSIONS

The following conclusions are drawn from the metallurgical testwork performed on the Almaden Ixtaca domain samples:

- Au and Ag head grades ranged from 0.7 to 1.7g/t and 13 to 116g/t respectively with the highest grades naturally observed in the High Grade sample and the lowest grades observed in the TUFF sample for silver and Dyke sample for gold. Pb and Zn grades for all the composites were low (<0.1% combined) with the exception of the Black Shale composite where Pb and Zn grades of 0.24% and 0.43% respectively were observed.
- Sulphur grades were variable and ranged from a low of 0.77% (Limestone) to 3.64% (Dyke).
- Carbon content ranged from 1.45% (Limestone) to 7.69% (Dyke) further highlighting the differences between these two domains. However, this analysis does not discriminate between graphitic carbon and carbon in carbonates.
- Bond BWi hardness testing suggests that the TUFF domain is the softest at 10.5kwh/t and the Black Shale is the hardest at 18.6kwh/t. The Dyke and Limestone both exhibit similar hardness characteristics at 14.6 and 13.2kwh/t respectively.
- Standard E-GRG tests showed that the Dyke, Limestone and Black Shale domains are all quite amenable to gravity recovery of coarse gold. These domains achieved gold recoveries to concentrate of 48%, 59% and 55% respectively suggesting a significant amount of the gold is present as coarse, liberated gold.
- The TUFF sample exhibited poor amenability to the standard E-GRG test with only 15% of the gold reporting to concentrate.
- Cyanidation of the E-GRG tails can provide additional gold recovery. The combined gravity + cyanidation gold recoveries for the Dyke, Limestone, Black Shale and TUFF were 80%, 84%, 66% and 50% respectively.
- Flotation appears to be an appropriate method of gold recovery. Bulk flotation of the Dyke, Limestone, Black Shale and TUFF domains produced gold recoveries of 89%, 77%, 93% and 52% respectively. Silver recoveries were 88%, 73% 83%, 63%. Though further work would have to be conducted to determine whether these concentrates could be intensively leached effectively.
- Test HG F-2 produced a bulk rougher concentrate grading 21g/t Au and 1220g/t Ag. Gold and silver recoveries were an impressive 92% and 90% respectively.
- Intensive leaching of the High Grade bulk rougher concentrate was conducted. A gold extraction of 88% and silver extraction of 93% was achieved and the ratio of silver to gold in the PLS was 72:1 suggesting that a Merrill Crowe process would be most suitable for producing dore. The cyanide

consumption was high at 92kg/t and gold extraction was somewhat disappointing for an intensive leach. More investigation would be needed to determine what caused the high cyanide consumption and relatively low extraction rate.

- Differential flotation of lead and zinc for the Black Shale domain was assessed and the three amenability tests indicate that there is some potential to treat this material in this way and produce separate lead and zinc concentrates. Further optimisation would be required if this is to be considered an option by the project team.

6. RECOMMENDATIONS

The testwork communicated in this report was conducted as part of an amenability study to assess various processing options for the Almaden Ixtaca domains. The testwork has shown that flotation and gravity show the most promise in achieving acceptable precious metals recoveries; however cyanidation has shown that extra gold and silver recovery can be gained from gravity tails. With this in mind, we recommend that the next phase of metallurgical testwork should include:

- Gravity followed by bulk rougher flotation to determine the combined effect of both with respect to precious metals recovery.
- Cyanidation of the whole ore for completeness.
- It has been noted that the Ixtaca zone contains significant occurances of alabandite (MnS). Although Mn was not tracked in this program of work it could be the source of the high cyanide consumption and this warrants further investigation. If indeed alabandite is abundant in these samples, it may be beneficial to assess whether a sulphurous preleach could be employed to recover Mn into a saleable by product while reducing the cyanide consumption in the gold-silver leach process.
- Further optimisation of differential lead/zinc flotation if this domain is of large enough tonnage to warrant it.
- Intensive cyanidation of all bulk flotation concentrates.
- Flotation optimisation of the TUFF zone would demonstrate the most upside. This zone must be mined in order to access the other, less problematic zones so any improvements in metallurgical performance in the TUFF zone could significantly improve project economics. It seems that the presence of clays is the main driver for lower metallurgical performance in this zone. Desliming to remove the clays may be of benefit as well as the addition of dispersants.
- Detailed mineralogical analysis of the domains to determine modal mineralogy as well as gold deportment.

APPENDIX A –BOND BWI WORKSHEETS

Bond Ball Mill Grindability Test Report

Project No.:	PJ# 124 Company: Almaden Date: 10/16/12
Sample.:	Blackshale
Purpose:	To determine the ball mill grindability of the sample in terms of a Bond work index number.
Procedure:	The equipment and procedure duplicate the Bond method for determining ball mill work indices.
Test Conditions:	Screen size: 150 microns Test feed weight (700 mL): 1232.18 grams Equivalent to : 1760 kg/m³ at Minus 6 mesh Weight % of the undersize material in the ball mill feed: 14.0 % Weight of undersize product for 250% circulating load: 352.1 grams
Results:	**Average for Last Three Stages = 1.15g. 249% Circulating load**

BWI =	**16.8**	kwh/ton (imperial)
BWI =	**18.6**	kwh/tonne (metric)

Bond Ball Work Index Calculation

$$BWI = \frac{44.5}{P1^{0.23} \times Grp^{0.82} \times \left\{\frac{10}{\sqrt{P}} - \frac{10}{\sqrt{F}}\right\}}$$

P1 = 100% passing size of the product	**150 microns**
Grp = Grams per revolution	**1.15 grams**
P80 = 80% passing size of product	**111 microns**
F80 = 80% passing size of the feed	**2417 microns**

Bond Ball Mill Grindability Test Report Project No.: PJ# 124 Blackshale

Stage No.	Revs	New Feed (grams)	Undersize In Feed (grams)	To Be Ground (grams)	Total Product (grams)	Product Produced (grams)	Per Mill Rev (grams)
1	100	1,232	172	180	277	105	1.05
2	298	279	39	313	370	331	1.11
3	271	371	52	300	359	307	1.13
4	266	361	50	302	355	304	1.14
5	265	355	50	302	351	302	1.14
6	266	353	49	303	358	308	1.16
7	260	359	50	302	349	299	1.15
8	264	352	49	303	0	-49	-0.19

Average for Last Three Stages = 353g. 1.15g.

Feed Size Distribution: µm		Weight grams	% Retained Individual	Cumulative	% Passing Cumulative
	2360	62.7	10.4	10.4	89.6
	1,700	127.3	21.0	31.4	68.6
	1,180	104.6	17.3	48.7	51.3
	850	67.7	11.2	59.9	40.1
	600	49.0	8.1	68.0	32.0
	425	39.0	6.4	74.5	25.5
	300	31.9	5.3	79.7	20.3
	212	19.9	3.3	83.0	17.0
	150	18.2	3.0	86.0	14.0
	106	14.3	2.4	88.4	11.6
	75	13.0	2.2	90.6	9.4
	53	9.9	1.6	92.2	7.8
	38	8.2	1.4	93.6	6.4
Pan	-38	39.0	6.4	100.0	0.0
Total	-	**604.7**	**100.0**	-	-

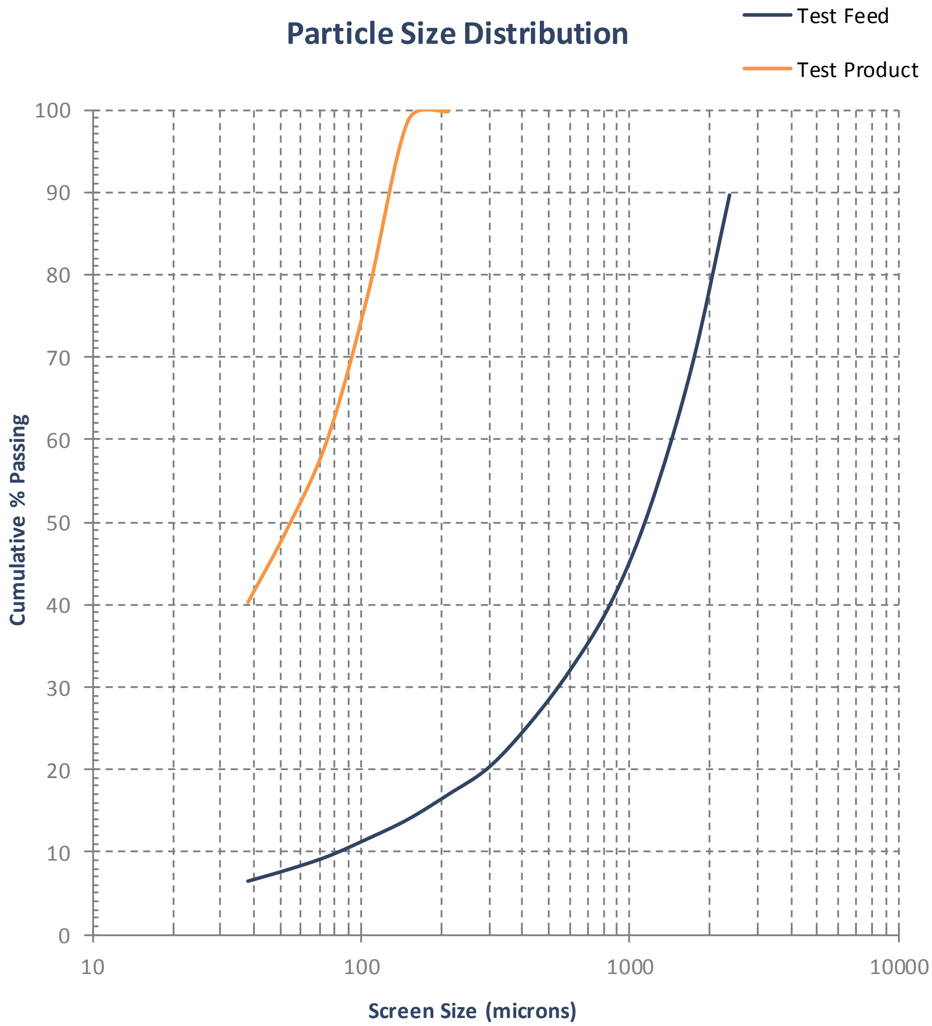
K80	**2,417**	**microns**

Product Size Distribution: µm		Weight grams	% Retained Individual	Cumulative	% Passing Cumulative
	212	0.7	0.2	0.2	99.8
	150	3.7	1.1	1.3	98.7
	106	69.3	21.2	22.6	77.4
	75	57.0	17.4	40.0	60.0
	53	35.7	10.9	51.0	49.0
	38	28.7	8.8	59.7	40.3
Pan	-38	131.5	40.3	100.0	0.0
Total	-	326.7	**100.0**	-	-

P80	**111**	**microns**

BLUE COAST RESEARCH

Bond Ball Mill Grindability Test Report Project No.: PJ# 124 Blackshale



Bond Ball Mill Grindability Test Report

Project No.:	PJ# 124 Company: Almaden Date: 10/09/2012
Sample.:	Dyke (Quartz Vein)
Purpose:	To determine the ball mill grindability of the sample in terms of a Bond work index number.
Procedure:	The equipment and procedure duplicate the Bond method for determining ball mill work indices.
Test Conditions:	Screen size: 150 microns Test feed weight (700 mL): 1216.79 grams Equivalent to : 1738 kg/m³ at Minus 6 mesh Weight % of the undersize material in the ball mill feed: 17.9 % Weight of undersize product for 250% circulating load: 347.7 grams
Results:	**Average for Last Three Stages = 1.53g. 249% Circulating load**

BWI =	**13.2**	kwh/ton (imperial)
BWI =	**14.6**	kwh/tonne (metric)

Bond Ball Work Index Calculation

$$BWI = \frac{44.5}{P1^{0.23} \times Grp^{0.82} \times \left\{ \frac{10}{\sqrt{P}} - \frac{10}{\sqrt{F}} \right\}}$$

P1 = 100% passing size of the product	**150 microns**
Grp = Grams per revolution	**1.53 grams**
P80 = 80% passing size of product	**111 microns**
F80 = 80% passing size of the feed	**2485 microns**

Bond Ball Mill Grindability Test Report   Project No.: PJ# 124   Dyke (Quartz Vein)

Stage No.	Revs	New Feed (grams)	Undersize In Feed (grams)	To Be Ground (grams)	Total Product (grams)	Product Produced (grams)	Per Mill Rev (grams)
1	100	1,217	218	130	350	132	1.32
2	215	354	63	284	376	313	1.45
3	193	377	68	280	351	284	1.47
4	193	354	63	284	355	292	1.51
5	187	362	65	283	344	279	1.49
6	192	342	61	286	354	293	1.52
7	187	356	64	284	348	284	1.52
8	187	349	63	285	352	290	1.55
9	183	354	63	284	344	281	1.53
		Average for Last Three Stages =			348g.		1.53g.

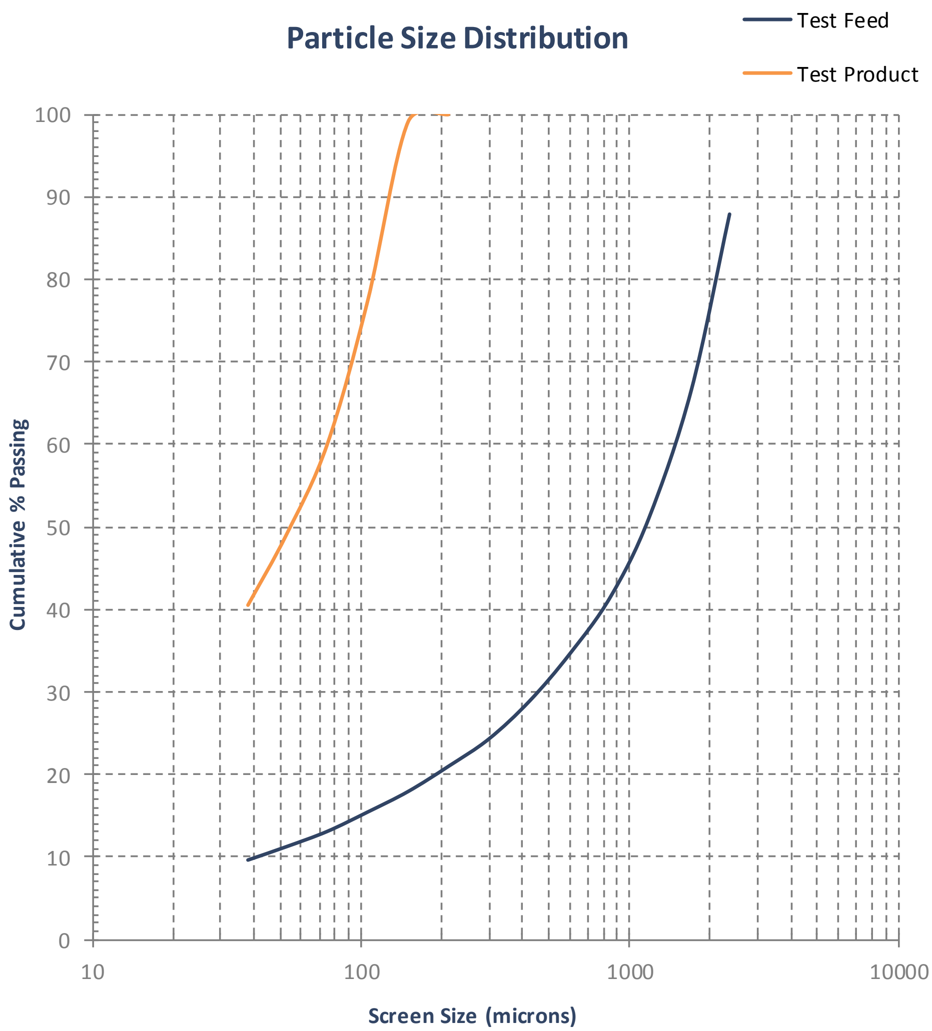
Feed Size Distribution: µm		Weight grams	% Retained Individual	Cumulative	% Passing Cumulative
	2360	72.3	12.1	12.1	87.9
	1,700	127.3	21.3	33.5	66.5
	1,180	92.8	15.6	49.0	51.0
	850	56.7	9.5	58.5	41.5
	600	41.1	6.9	65.4	34.6
	425	34.1	5.7	71.1	28.9
	300	27.4	4.6	75.7	24.3
	212	19.8	3.3	79.0	21.0
	150	18.2	3.1	82.1	17.9
	106	14.9	2.5	84.6	15.4
	75	14.0	2.3	86.9	13.1
	53	11.0	1.8	88.8	11.2
	38	9.7	1.6	90.4	9.6
Pan	-38	57.3	9.6	100.0	0.0
Total	-	596.6	100.0	-	-

K80	**2,485**	**microns**

Product Size Distribution: µm		Weight grams	% Retained Individual	Cumulative	% Passing Cumulative
	212	0.2	0.0	0.0	100.0
	150	2.8	0.9	0.9	99.1
	106	69.4	21.7	22.7	77.3
	75	55.4	17.4	40.0	60.0
	53	34.7	10.9	50.9	49.1
	38	27.5	8.6	59.5	40.5
Pan	-38	129.4	40.5	100.0	0.0
Total	-	319.4	100.0	-	-

P80	**111**	**microns**

Bond Ball Mill Grindability Test Report Project No.: PJ# 124 Dyke (Quartz Vein)



Bond Ball Mill Grindability Test Report

Project No.:	PJ# 124 Company: Almaden Date: 10/18/2012
Sample.:	Limestone
Purpose:	To determine the ball mill grindability of the sample in terms of a Bond work index number.
Procedure:	The equipment and procedure duplicate the Bond method for determining ball mill work indices.
Test Conditions:	Screen size: 150 microns Test feed weight (700 mL): 1240.87 grams Equivalent to : 1773 kg/m³ at Minus 6 mesh Weight % of the undersize material in the ball mill feed: 8.6 % Weight of undersize product for 250% circulating load: 354.5 grams
Results:	**Average for Last Three Stages = 1.69g. 251% Circulating load**

BWI =	**12.0**	kwh/ton (imperial)
BWI =	**13.2**	kwh/tonne (metric)

Bond Ball Work Index Calculation

$$\mathrm{BWI} = \frac{44.5}{P1^{0.23} \times Grp^{0.82} \times \left\{\frac{10}{\sqrt{P}} - \frac{10}{\sqrt{F}}\right\}}$$

P1 = 100% passing size of the product	**150 microns**
Grp = Grams per revolution	**1.69 grams**
P80 = 80% passing size of product	**111 microns**
F80 = 80% passing size of the feed	**2789 microns**

Bond Ball Mill Grindability Test Report Project No.: PJ# 124 Limestone

Stage No.	Revs	New Feed (grams)	Undersize In Feed (grams)	To Be Ground (grams)	Total Product (grams)	Product Produced (grams)	Per Mill Rev (grams)
1	100	1,241	106	248	249	143	1.43
2	233	253	22	333	399	377	1.62
3	198	401	34	320	371	336	1.70
4	190	373	32	323	357	325	1.71
5	189	359	31	324	355	324	1.72
6	189	357	31	324	353	322	1.71
7	190	355	30	324	354	324	1.71
8	190	355	30	324	353	323	1.70
9	191	355	30	324	353	322	1.69
10	192	354	30	324	354	323	1.69
		Average for Last Three Stages =			353g.		1.69g.

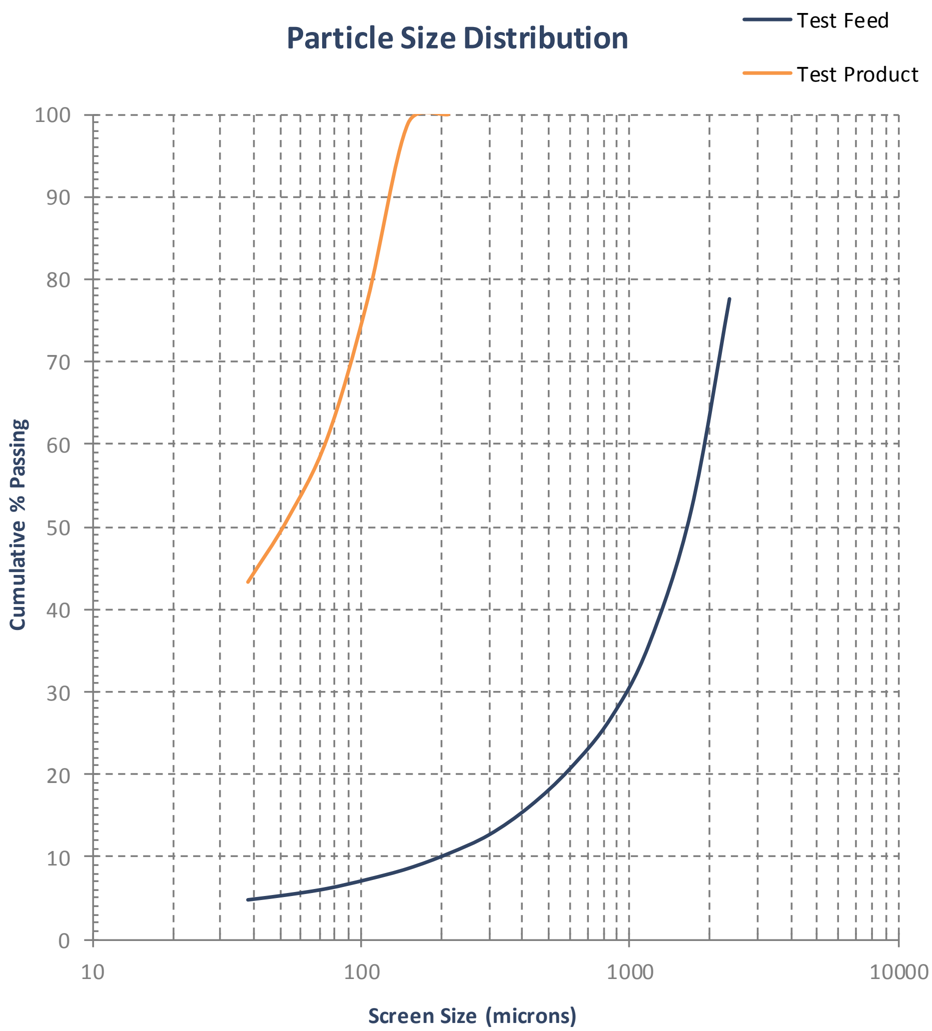
Feed Size Distribution:	µm	Weight grams	% Retained Individual	Cumulative	% Passing Cumulative
	2360	127.3	22.4	22.4	77.6
	1,700	145.7	25.6	48.0	52.0
	1,180	93.1	16.4	64.4	35.6
	850	50.6	8.9	73.3	26.7
	600	34.6	6.1	79.4	20.6
	425	25.9	4.5	83.9	16.1
	300	19.6	3.4	87.4	12.6
	212	12.9	2.3	89.6	10.4
	150	10.3	1.8	91.4	8.6
	106	7.5	1.3	92.8	7.2
	75	6.3	1.1	93.9	6.1
	53	4.4	0.8	94.6	5.4
	38	3.4	0.6	95.2	4.8
Pan	-38	27.1	4.8	100.0	0.0
Total	-	**568.6**	**100.0**	-	-

K80	**2,789**	**microns**

Product Size Distribution:	µm	Weight grams	% Retained Individual	Cumulative	% Passing Cumulative
	212	0.1	0.0	0.0	100.0
	150	3.4	1.1	1.1	98.9
	106	67.6	21.4	22.5	77.5
	75	52.8	16.7	39.3	60.7
	53	31.6	10.0	49.3	50.7
	38	23.5	7.4	56.7	43.3
Pan	-38	136.6	43.3	100.0	0.0
Total	-	315.5	**100.0**	-	-

P80	**111**	**microns**

Bond Ball Mill Grindability Test Report Project No.: PJ# 124 Limestone



Bond Ball Mill Grindability Test Report

Project No.:	PJ# 124 Company: Almaden Date: 10/15/2012
Sample.:	TUFF (Brecciated Pumice)
Purpose:	To determine the ball mill grindability of the sample in terms of a Bond work index number.
Procedure:	The equipment and procedure duplicate the Bond method for determining ball mill work indices.
Test Conditions:	Screen size: 150 microns Test feed weight (700 mL): 1093.74 grams Equivalent to : 1562 kg/m³ at Minus 6 mesh Weight % of the undersize material in the ball mill feed: 29.3 % Weight of undersize product for 250% circulating load: 312.5 grams
Results:	**Average for Last Three Stages = 2.23g. 251% Circulating load**

BWI =	**9.5**	kwh/ton (imperial)
BWI =	**10.5**	kwh/tonne (metric)

Bond Ball Work Index Calculation

$$BWI = \frac{44.5}{P1^{0.23} \times Grp^{0.82} \times \left\{\frac{10}{\sqrt{P}} - \frac{10}{\sqrt{F}}\right\}}$$

P1 = 100% passing size of the product	**150**	**microns**
Grp = Grams per revolution	**2.23**	**grams**
P80 = 80% passing size of product	**105**	**microns**
F80 = 80% passing size of the feed	**2316**	**microns**

Bond Ball Mill Grindability Test Report Project No.: PJ# 124 TUFF (Brecciated Pumice)

Stage No.	Revs	New Feed (grams)	Undersize In Feed (grams)	To Be Ground (grams)	Total Product (grams)	Product Produced (grams)	Per Mill Rev (grams)
1	100	1,094	320	-8	455	134	1.34
2	133	455	133	179	410	277	2.07
3	93	409	120	193	318	198	2.13
4	103	318	93	219	325	232	2.25
5	97	326	95	217	310	214	2.22
6	100	311	91	221	316	225	2.25
7	98	317	93	220	310	217	2.22
		Average for Last Three Stages =			312g.		2.23g.

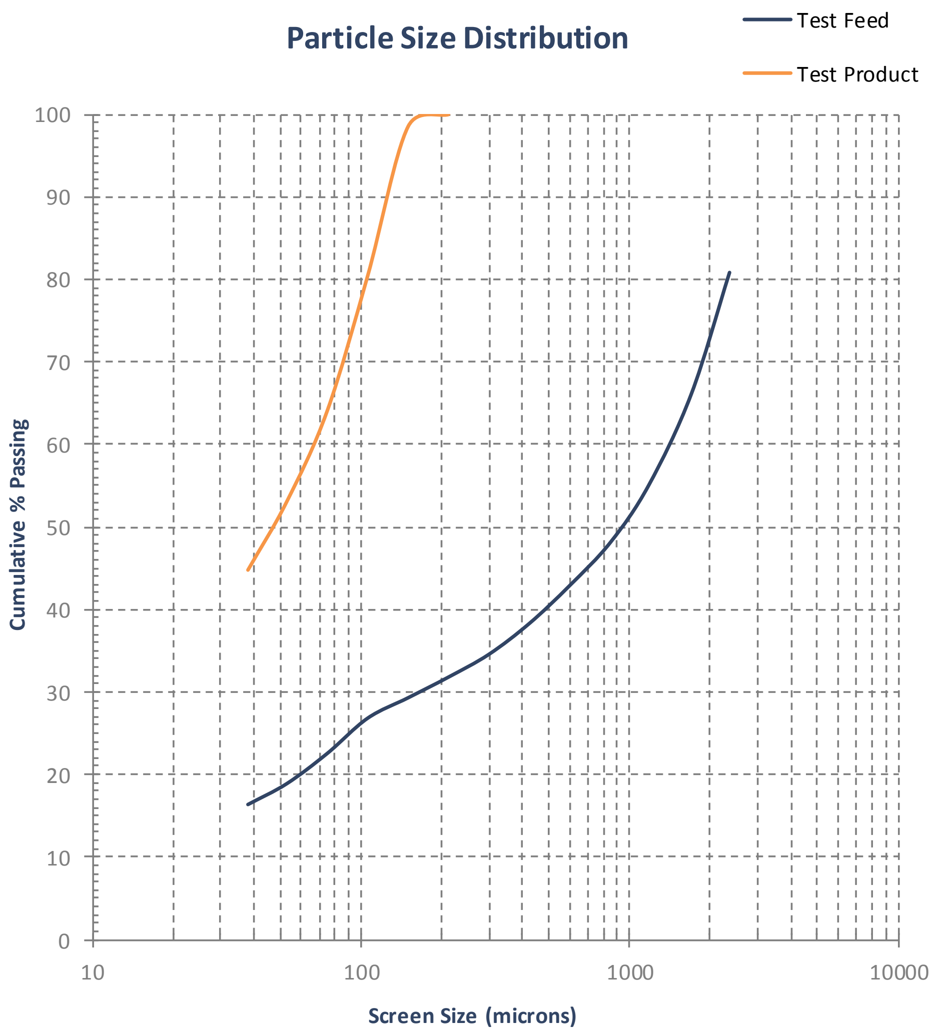
Feed Size Distribution: µm		Weight grams	% Retained Individual	Cumulative	% Passing Cumulative
	2360	125.4	19.2	19.2	80.8
	1,700	96.3	14.7	33.9	66.1
	1,180	72.2	11.0	45.0	55.0
	850	45.2	6.9	51.9	48.1
	600	34.1	5.2	57.1	42.9
	425	30.0	4.6	61.7	38.3
	300	24.6	3.8	65.4	34.6
	212	18.1	2.8	68.2	31.8
	150	16.4	2.5	70.7	29.3
	106	16.2	2.5	73.2	26.8
	75	27.9	4.3	77.5	22.5
	53	24.0	3.7	81.1	18.9
	38	16.5	2.5	83.6	16.4
Pan	-38	106.9	16.4	100.0	0.0
Total	-	**653.7**	**100.0**	-	-

K80	**2,316**	**microns**

Product Size Distribution: µm		Weight grams	% Retained Individual	Cumulative	% Passing Cumulative
	212	0.1	0.0	0.0	100.0
	150	4.8	1.5	1.5	98.5
	106	59.6	18.1	19.6	80.4
	75	53.6	16.3	35.9	64.1
	53	36.2	11.0	47.0	53.0
	38	27.2	8.3	55.2	44.8
Pan	-38	147.1	44.8	100.0	0.0
Total	-	328.5	**100.0**	-	-

P80	**105**	**microns**

Bond Ball Mill Grindability Test Report Project No.: PJ# 124 TUFF (Brecciated Pumice)



APPENDIX B –EGRG TEST WORKSHEETS

E-GRG Test Description:

Test #:	Blackshale
Project #:	PJ124- Almaden
Operator:	KJ/JC
Date:	Oct-12
Purpose:	Determine Gravity recoverable gold and E-GRG Number
Procedure:	As outlined below.
Feed:	Blackshale- 2 x 10 kg charges of minus 12 mesh

Grind:					
	Stage 1:	5 mins @ 60% solids in rod mill	p80 =	747	microns
	Stage 2:	20 mins @ 60% solids in rod mill	p80 =	194	microns
	Stage 3:	50 mins @ 60% solids in rod mill	p80 =	70	microns

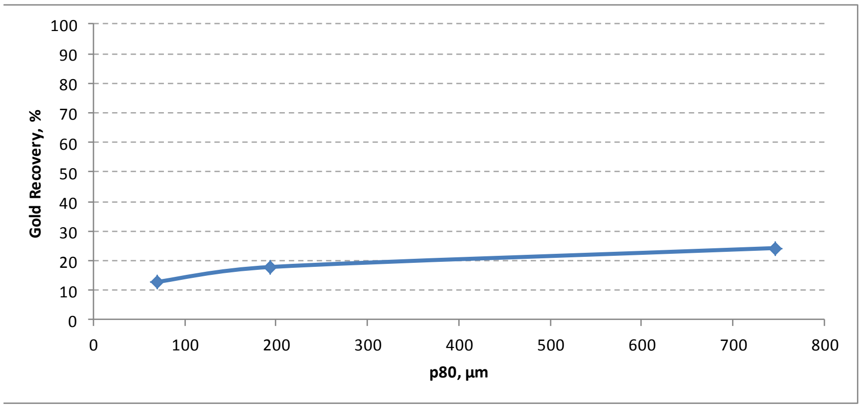
Target p80
850 microns
250 microns
75 microns

Data entry Required

Metallurgical Balance:

Grind Size	Product	Mass grams	Mass wt %	Assay g/t	Metal Units Au	Distribution %
P_{80} = 747 microns	Stage 1 Concentrate	90.5	0.5	65.41	5,919.3	**24.2**
	Stage 1 Tails	19,909.5	99.5	0.93	18,510.0	75.8
P_{80} = 194 microns	Stage 2 Concentrate	91.3	0.5	47.75	4,357.6	**17.8**
	Stage 2 Tails	19,818.3	99.1	0.93	18,379.1	75.2
P_{80} = 70 microns	Stage 3 Concentrate	86.1	0.4	36.31	3,126.0	**12.8**
	Stage 3 Tails Sample	354.0	1.8	0.60	211.9	0.9
	Final Tails	18,065.0	90.3	0.60	10,814.5	44.3
	Head	20,000.0	100.0	1.22	24,429.3	100.0
	Total Concentrate	**267.9**	**1.3**	**50.04**	**13,402.9**	**54.9**
	Total Tailings	18,419.0	92.1	0.60	11,026.4	45.1

E-GRG Number = 54.9



E-GRG Test Description:

Test #:	Limestone
Project #:	PJ124- Almaden
Operator:	KJ/JC
Date:	Oct-12
Purpose:	Determine Gravity recoverable gold and E-GRG Number
Procedure:	As outlined below.
Feed:	Limestone- 2 x 10 kg charges of minus 12 mesh

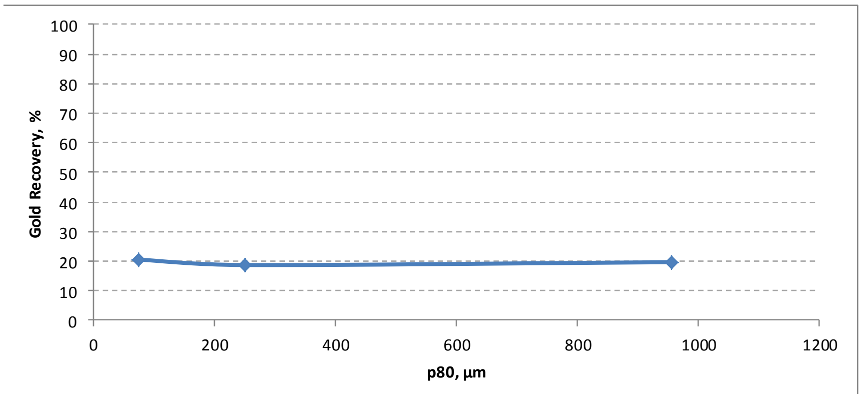
					Target p80
Grind:	Stage 1:	2 mins @ 60% solids in rod mill	p80 =	956 microns	850 microns
	Stage 2:	11 mins @ 60% solids in rod mill	p80 =	250 microns	250 microns
	Stage 3:	35 mins @ 60% solids in rod mill	p80 =	75 microns	75 microns

Data entry Required

Metallurgical Balance:

Grind Size	Product	Mass		Assay	Metal Units	Distribution
		grams	wt %	g/t	Au	%
P_{80} = 956 microns	Stage 1 Concentrate	74.0	0.4	41.49	3,070.2	**19.6**
	Stage 1 Tails	19,926.0	99.6	0.63	12,590.1	80.4
P_{80} = 250 microns	Stage 2 Concentrate	85.1	0.4	34.30	2,919.1	**18.6**
	Stage 2 Tails	19,840.9	99.2	0.62	12,274.8	78.4
P_{80} = 75 microns	Stage 3 Concentrate	74.7	0.4	42.90	3,206.0	**20.5**
	Stage 3 Tails Sample	508.2	2.5	0.34	174.5	1.1
	Final Tails	18,320.0	91.6	0.34	6,290.6	40.2
	Head	20,000.0	100.0	0.78	15,660.3	100.0
	Total Concentrate	**233.9**	**1.2**	**39.32**	**9,195.2**	**58.7**
	Total Tailings	18,828.2	94.1	0.34	6,465.1	41.3

E-GRG Number = 58.7



E-GRG Test Description:

Test #:	TUFF
Project #:	PJ124- Almaden
Operator:	KJ/JC
Date:	Oct-12
Purpose:	Determine Gravity recoverable gold and E-GRG Number
Procedure:	As outlined below.
Feed:	TUFF- 2 x 10 kg charges of minus 12 mesh

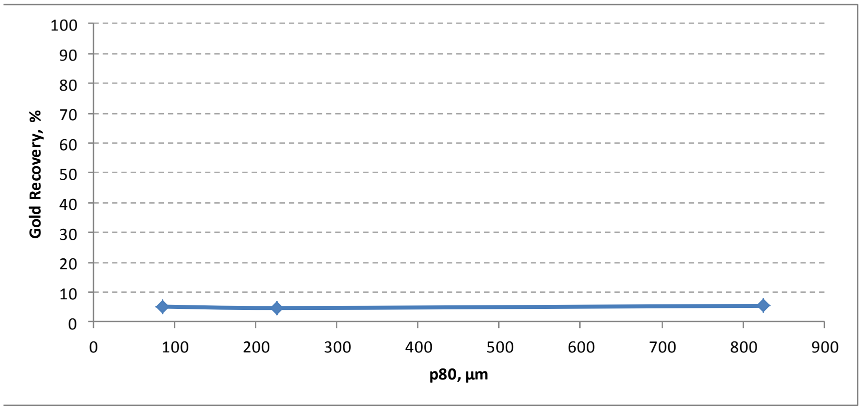
				Target p80
Grind:	Stage 1:	No grind	p80 = 825 microns	850 microns
	Stage 2:	7 mins @ 60% solids in rod mill	p80 = 226 microns	250 microns
	Stage 3:	25.5 mins @ 60% solids in rod mill	p80 = 85 microns	75 microns

Data entry Required

Metallurgical Balance:

Grind Size	Product	Mass grams	Mass wt %	Assay g/t	Metal Units Au	Distribution %
P_{80} = 825 microns	Stage 1 Concentrate	77.1	0.4	11.88	915.4	**5.4**
	Stage 1 Tails	19,922.9	99.6	0.81	16,101.1	94.6
P_{80} = 226 microns	Stage 2 Concentrate	73.6	0.4	10.73	790.5	**4.6**
	Stage 2 Tails	19,849.3	99.2	0.77	15,310.5	90.0
P_{80} = 85 microns	Stage 3 Concentrate	77.3	0.4	11.26	870.3	**5.1**
	Stage 3 Tails Sample	642.9	3.2	0.76	491.6	2.9
	Final Tails	18,240.6	91.2	0.76	13,948.6	82.0
	Head	20,000.0	100.0	0.85	17,016.4	100.0
	Total Concentrate	**228.0**	**1.1**	**11.30**	**2,576.2**	**15.1**
	Total Tailings	18,883.5	94.4	0.76	14,440.3	84.9

E-GRG Number = 15.1



E-GRG Test Description:

Test #:	Limestone
Project #:	PJ124- Almaden
Operator:	KJ/JC
Date:	Oct-12
Purpose:	Determine Gravity recoverable gold and E-GRG Number
Procedure:	As outlined below.
Feed:	Limestone- 2 x 10 kg charges of minus 12 mesh

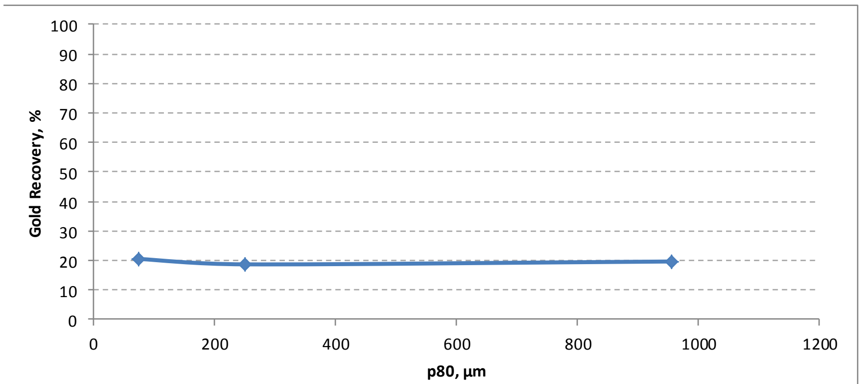
Grind:				Target p80
	Stage 1:	2 mins @ 60% solids in rod mill	p80 = 956 microns	850 microns
	Stage 2:	11 mins @ 60% solids in rod mill	p80 = 250 microns	250 microns
	Stage 3:	35 mins @ 60% solids in rod mill	p80 = 75 microns	75 microns

Metallurgical Balance:

Data entry Required

Grind Size	Product	Mass grams	Mass wt %	Assay g/t	Metal Units Au	Distribution %
P_{80} = 956 microns	Stage 1 Concentrate	74.0	0.4	41.49	3,070.2	**19.6**
	Stage 1 Tails	19,926.0	99.6	0.63	12,590.1	80.4
P_{80} = 250 microns	Stage 2 Concentrate	85.1	0.4	34.30	2,919.1	**18.6**
	Stage 2 Tails	19,840.9	99.2	0.62	12,274.8	78.4
P_{80} = 75 microns	Stage 3 Concentrate	74.7	0.4	42.90	3,206.0	**20.5**
	Stage 3 Tails Sample	508.2	2.5	0.34	174.5	1.1
	Final Tails	18,320.0	91.6	0.34	6,290.6	40.2
	Head	20,000.0	100.0	0.78	15,660.3	100.0
	Total Concentrate	**233.9**	**1.2**	**39.32**	**9,195.2**	**58.7**
	Total Tailings	18,828.2	94.1	0.34	6,465.1	41.3

E-GRG Number = 58.7



APPENDIX B –CYANIDATION TEST WORKSHEETS

Test Description:

Test #:	CN-1 (Dyke)
Project #:	PJ124 Almaden
Operator:	JC
Date:	November 20th, 2012
Purpose:	Standard batch cyanidation test
Procedure:	48 hour milled sample leach

Primary Grind:	36	minutes @ 60% solids	**Feed:**	500	g Knelson tails
Actual K_{80}:	74	µm	**Solution Volume:**	1000	mL (tap water)
NaCN Addition:	3.06	g	**Pulp Density:**	33.3	% Solids
Tare Mass:	1154.1	g	**Solution Composition:**	3.0	g/L NaCN (maintained)
Initial Gross Mass:	2665.9	g	**pH Range:**	10.5 - 11.0	maintained with lime
Final Gross Mass:	2659.0	g			
			24 hr Gold Recovery =	**60.8**	**%**
			24 hr Silver Recovery =	**72.2**	**%**

Cyanidation Schedule:

Reagent addition (kg/t of cyanide feed)	NaCN:	8.98	CaO:	1.14
Reagent consumption (kg/t of cyanide feed)	NaCN:	5.04	CaO:	0.76

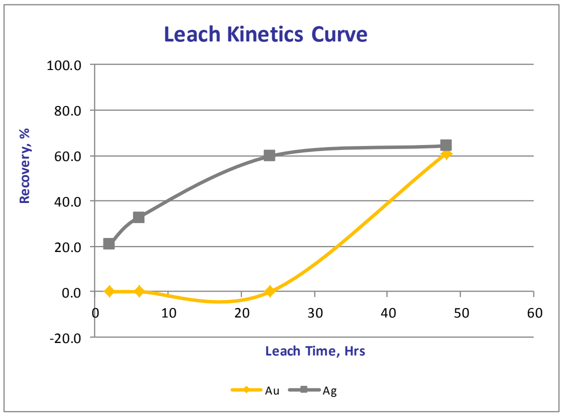
Start Time: 10:20	**Added Grammes**				**Residual**		**Consumed**						
	Actual		**Equivalent**		**Grams**		**Grams**		**pH**	**DO_2**	**Bottle Mass g**	**mL aliquot**	**mL $AgNO_3$**
Time Hours	**NaCN**	**$Ca(OH)_2$**	**NaCN**	**CaO**	**NaCN**	**CaO**	**NaCN**	**CaO**					
0	**3.06**	**0.77**	3.00	0.57					**7.74-10.46**	**7.4**	**2665.9**	-	-
0 - 2	**1.32**	**0.00**	1.29	0.00	1.70		1.30		**10.46-10.84**	**7.5**	**2655.7**	**5**	**1.7**
2 - 6	**0.00**	**0.00**	0.00	0.00	3.10		-0.10		**10.84-10.95**	**7.4**	**2654.2**	**10**	**6.2**
6 - 24	**0.20**	**0.00**	0.20	0.00	2.76		0.23		**10.95-10.88**	**7.1**	**2641.4**	**10**	**5.6**
24 - 48	**0.00**	**0.00**	0.00	0.00	1.91	0.19	1.09		**10.88-10.93**	**7.6**	**2659.0**	**10**	**3.8**
Total	**4.58**	**0.77**	**4.49**	**0.57**	**1.91**	**0.19**	**2.52**	**0.38**					

Observations:	color of indication was light orange, making it difficult to determine end point

mL aliquot	**mL Oxalic**
10	**1.9**

Results:

Product		**Amount g, mL**	**Assay mg/L, g/t**		**Distribution %**	
			Au	**Ag**	**Au**	**Ag**
2	**Hr PLS**	1001.6		5.2	0.0	20.7
6	**Hr PLS**	1000.1		8.2	0.0	32.7
24	**Hr PLS**	987.3		15.1	0.0	59.7
48	**Hr PLS**	**1004.9**	**0.11**	**14.4**	**43.8**	**64.3**
Wash Solution		**1516.9**	**0.03**	**1.3**	**17.0**	**7.9**
Residue		499.2	0.21	14.0	39.2	27.8
Calculated Head			0.54	50.3	100.0	100.0
ERD Head			0.44	40		
Accountability			121.8	125.8		



Test Description:

Test #:	CN-2 (Dyke)
Project #:	PJ124 Almaden
Operator:	JC
Date:	November 20th, 2012
Purpose:	Standard batch cyanidation test
Procedure:	48 hour milled sample leach

Primary Grind:	36	minutes @ 60% solids	Feed:	500	g Knelson tails
Actual K_{80}:	74	µm	Solution Volume:	1000	mL (tap water)
NaCN Addition:	5.10	g	Pulp Density:	33.3	% Solids
Tare Mass:	1163.8	g	Solution Composition:	5.0	g/L NaCN (maintained)
Initial Gross Mass:	2663.9	g	pH Range:	10.5 - 11.0	maintained with lime
Final Gross Mass:	2663.2	g			
			24 hr Gold Recovery =	61.9	%
			24 hr Silver Recovery =	81.8	%

Cyanidation Schedule:

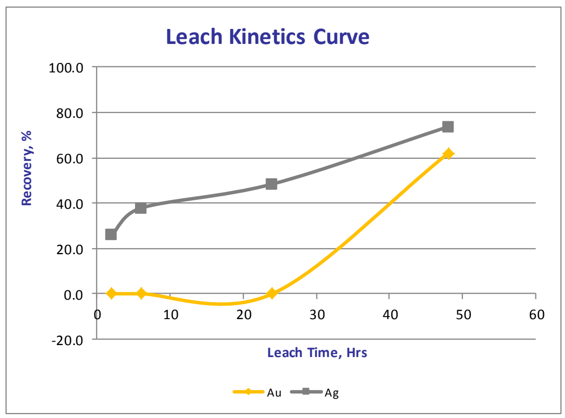
Reagent addition (kg/t of cyanide feed)	NaCN:	14.48	CaO:	1.17
Reagent consumption (kg/t of cyanide feed)	NaCN:	6.20	CaO:	-0.03

Start Time: 10:40	Added Grammes				Residual		Consumed		pH	DO_2	Bottle Mass g	mL aliquot	mL $AgNO_3$
	Actual		Equivalent		Grams		Grams						
Time Hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO					
0	5.10	0.79	5.00	0.59					7.76-10.51	5.9	2663.9	-	-
0 - 2	0.41	0.00	0.40	0.00	4.60		0.40		10.51-11.31	7.5	2663.5	10	9.2
2 - 6	0.71	0.00	0.70	0.00	4.30		0.70		11.31-11.12	7.4	2663.6	10	8.6
6 - 24	1.17	0.00	1.15	0.00	3.85		1.15		11.12-11.14	6.7	2663.4	10	7.7
24 - 48	0.00	0.00	0.00	0.00	4.15	0.60	0.85		11.14-11.2	7.0	2663.2	10	8.3
Total	7.39	0.79	7.24	0.59	4.15	0.60	3.10	-0.01					

Observations:	color of indication was light orange, making it difficult to determine end point	mL aliquot	mL Oxalic
		10	6

Results:

Product	Amount g, mL	Assay mg/L, g/t		Distribution %	
		Au	Ag	Au	Ag
2 Hr PLS	999.7		6.8	0.0	25.7
6 Hr PLS	999.8		9.9	0.0	37.7
24 Hr PLS	999.6		12.6	0.0	48.4
48 Hr PLS	999.4	0.12	17.4	46.4	73.6
Wash Solution	1421.6	0.03	1.5	15.5	8.2
Residue	499.0	0.21	9.6	38.1	18.2
Calculated Head		0.55	52.8	100.0	100.0
ERD Head		0.44	40		
Accountability		125.2	132.0		



Test Description:

Test #:	CN-3 (Limestone)
Project #:	PJ124 Almaden
Operator:	JC
Date:	November 20th, 2012
Purpose:	Standard batch cyanidation test
Procedure:	48 hour milled sample leach

Primary Grind:	35	minutes @ 60% solids	Feed:	500	g Knelson tails
Actual K_{80}:	75	µm	Solution Volume:	1000	mL (tap water)
NaCN Addition:	3.06	g	Pulp Density:	33.3	% Solids
Tare Mass:	1160.2	g	Solution Composition:	3.0	g/L NaCN (maintained)
Initial Gross Mass:	2660.7	g	pH Range:	10.5 - 11.0	maintained with lime
Final Gross Mass:	2660.4	g			
			24 hr Gold Recovery =	61.1	%
			24 hr Silver Recovery =	82.9	%

Cyanidation Schedule:

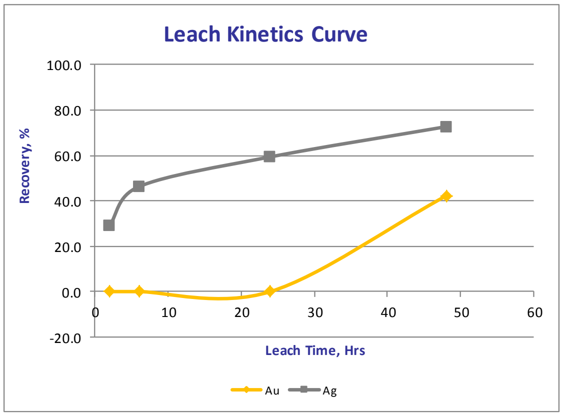
Reagent addition (kg/t of cyanide feed)	NaCN:	14.39	CaO:	0.67
Reagent consumption (kg/t of cyanide feed)	NaCN:	9.89	CaO:	-0.01

Start Time: 10:10	Added Grammes				Residual		Consumed		pH	DO_2	Bottle Mass g	mL aliquot	mL $AgNO_3$
	Actual		Equivalent		Grams		Grams						
Time Hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO					
0	3.06	0.45	3.00	0.33					7.52-10.49	6.1	2660.7	-	-
0 - 2	1.07	0.00	1.05	0.00	1.95		1.05		10.49-10.89	7.6	2660.5	10	3.9
2 - 6	1.27	0.00	1.24	0.00	1.75		1.25		10.89-11.06	7.5	2660.4	10	3.5
6 - 24	1.94	0.00	1.90	0.00	1.10		1.90		11.06-11.6	8.2	2660.1	10	2.2
24 - 48	0.00	0.00	0.00	0.00	2.25	0.34	0.75		11.6-11.91	8.4	2660.4	10	4.5
Total	7.34	0.45	7.19	0.33	2.25	0.34	4.94	-0.01					

Observations:	color of indication was light orange, making it difficult to determine end point	mL aliquot	mL Oxalic
		10	3.4

Results:

Product		Amount g, mL	Assay mg/L, g/t		Distribution %	
			Au	Ag	Au	Ag
2	Hr PLS	1000.3		5.4	0.0	29.1
6	Hr PLS	1000.2		8.6	0.0	46.2
24	Hr PLS	999.9		11.0	0.0	59.4
48	Hr PLS	1000.2	0.09	12.1	42.2	72.6
Wash Solution		1485.4	0.03	1.3	19.0	10.3
Residue		498.9	0.18	6.4	38.9	17.1
Calculated Head			0.47	37.5	100.0	100.0
ERD Head			0.34	30.2		
Accountability			138.5	124.2		



Test Description:

Test #:	CN-4 (Limestone)
Project #:	PJ124 Almaden
Operator:	JC
Date:	November 20th, 2012
Purpose:	Standard batch cyanidation test
Procedure:	48 hour milled sample leach

Primary Grind:	35	minutes @ 60% solids	Feed:	500	g Knelson tails
Actual K_{80}:	75	µm	Solution Volume:	1000	mL (tap water)
NaCN Addition:	5.10	g	Pulp Density:	33.3	% Solids
Tare Mass:	1168.3	g	Solution Composition:	5.0	g/L NaCN (maintained)
Initial Gross Mass:	2671.1	g	pH Range:	10.5 - 11.0	maintained with lime
Final Gross Mass:	2671.8	g			
			24 hr Gold Recovery =	60.3	%
			24 hr Silver Recovery =	86.2	%

Cyanidation Schedule:

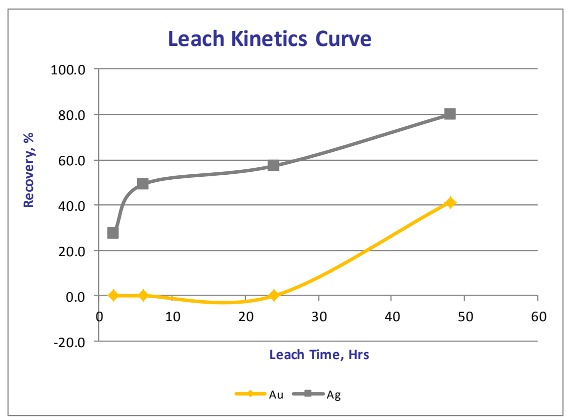
Reagent addition (kg/t of cyanide feed)	NaCN:	20.21	CaO:	0.68
Reagent consumption (kg/t of cyanide feed)	NaCN:	16.62	CaO:	-3.61

Start Time: 10:35	Added Grammes				Residual		Consumed		pH	DO_2	Bottle Mass g	mL aliquot	mL $AgNO_3$
	Actual		Equivalent		Grams		Grams						
Time Hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO					
0	5.10	0.46	5.00	0.34					7.51-10.7	7.6	2671.1	-	-
0 - 2	1.89	0.00	1.85	0.00	3.16		1.84		10.7-11.09	7.5	2670.8	10	6.3
2 - 6	0.97	0.00	0.95	0.00	4.06		0.93		11.09-11.32	7.4	2671.9	10	8.1
6 - 24	2.35	0.00	2.30	0.00	2.70		2.30		11.32-11.87	7.8	2668.7	10	5.4
24 - 48	0.00	0.00	0.00	0.00	1.76	2.15	3.24		11.87-12.11	7.9	2671.8	10	3.5
Total	10.31	0.46	10.10	0.34	1.76	2.15	8.31	-1.81					

		mL aliquot	mL Oxalic
Observations:	color of indication was light orange, making it difficult to determine end point	10	21.4

Results:

Product		Amount g, mL	Assay mg/L, g/t Au	Assay mg/L, g/t Ag	Distribution % Au	Distribution % Ag
2	Hr PLS	1002.5		5.9	0.0	27.3
6	Hr PLS	1003.6		10.6	0.0	49.1
24	Hr PLS	1000.4		12.3	0.0	57.3
48	Hr PLS	1003.5	0.09	15.4	41.3	79.9
Wash Solution		1458.2	0.03	0.9	19.0	6.3
Residue		499.0	0.18	6.0	39.7	13.8
Calculated Head			0.46	43.5	100.0	100.0
ERD Head			0.34	30.2		
Accountability			135.6	143.9		



Test Description:

Test #:	CN-5 (Black shale)
Project #:	PJ124 Almaden
Operator:	JC
Date:	November 20th, 2012
Purpose:	Standard batch cyanidation test
Procedure:	48 hour milled sample leach

Primary Grind:	50	minutes @ 60% solids	Feed:	500	g Knelson tails
Actual K_{80}:	70	µm	Solution Volume:	1000	mL (tap water)
NaCN Addition:	3.06	g	Pulp Density:	33.3	% Solids
Tare Mass:	1029.5	g	Solution Composition:	3.0	g/L NaCN (maintained)
Initial Gross Mass:	2529.6	g	pH Range:	10.5 - 11.0	maintained with lime
Final Gross Mass:	2528.8	g			

24 hr Gold Recovery =	25.6	%
24 hr Silver Recovery =	7.6	%

Cyanidation Schedule:

Reagent addition (kg/t of cyanide feed)	NaCN:	10.80	CaO:	2.06
Reagent consumption (kg/t of cyanide feed)	NaCN:	6.40	CaO:	0.58

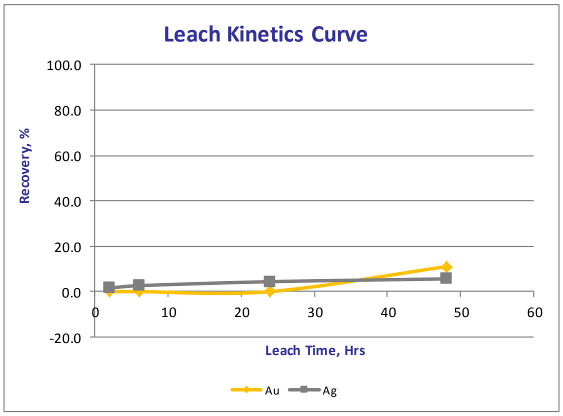
Start Time: 10:10	Added Grammes				Residual		Consumed		pH	DO_2	Bottle Mass g	mL aliquot	mL $AgNO_3$
	Actual		Equivalent		Grams		Grams						
Time Hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO					
0	3.06	1.39	3.00	1.03					7.29-10.36	3.1	2529.6	-	-
0 - 2	1.38	0.00	1.35	0.00	1.90		1.10		10.66-10.94	7.8	2529.5	10	3.8
2 - 6	0.97	0.00	0.95	0.00	2.05		0.95		10.94-11.05	8.8	2529.4	10	4.1
6 - 24	0.10	0.00	0.10	0.00	2.90		0.10		11.05-11.21	8.0	2529.0	10	5.8
24 - 48	0.00	0.00	0.00	0.00	1.95	0.74	1.05		11.21-11.04	8.0	2528.8	10	3.9
Total	5.51	1.39	5.40	1.03	1.95	0.74	3.20	0.29					

Observations:

mL aliquot	mL Oxalic
10	7.4

Results:

Product		Amount g, mL	Assay mg/L, g/t		Distribution %	
			Au	Ag	Au	Ag
2	Hr PLS	1000.0		0.3	0.0	1.5
6	Hr PLS	999.9		0.5	0.0	2.6
24	Hr PLS	999.5		0.9	0.0	4.4
48	Hr PLS	999.3	0.03	1.0	11.0	5.6
Wash Solution		1460.0	0.03	0.3	14.6	2.0
Residue		497.2	0.45	38.0	74.4	92.4
Calculated Head			0.60	41.1	100.0	100.0
ERD Head			0.60	40.6		
Accountability			100.7	101.3		



Test Description:

Test #:	CN-6 (Black shale)
Project #:	PJ124 Almaden
Operator:	JC
Date:	November 20th, 2012
Purpose:	Standard batch cyanidation test
Procedure:	48 hour milled sample leach

Primary Grind:	50	minutes @ 60% solids	Feed:	500	g Knelson tails
Actual K_{80}:	70	µm	Solution Volume:	1000	mL (tap water)
NaCN Addition:	5.10	g	Pulp Density:	33.3	% Solids
Tare Mass:	1022.9	g	Solution Composition:	5.0	g/L NaCN (maintained)
Initial Gross Mass:	2523.0	g	pH Range:	10.5 - 11.0	maintained with lime
Final Gross Mass:	2522.4	g			
			24 hr Gold Recovery =	25.3	%
			24 hr Silver Recovery =	10.2	%

Cyanidation Schedule:

Reagent addition (kg/t of cyanide feed)	NaCN:	17.19	CaO:	2.24
Reagent consumption (kg/t of cyanide feed)	NaCN:	9.49	CaO:	0.22

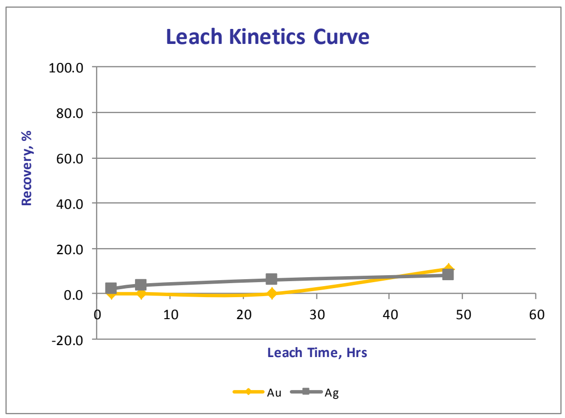
Start Time: 10:25	Added Grammes				Residual		Consumed		pH	DO_2	Bottle Mass g	mL aliquot	mL $AgNO_3$
	Actual		Equivalent		Grams		Grams						
Time Hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO					
0	5.10	1.51	5.00	1.12					7.25-10.43	6.0	2523.0	-	-
0 - 2	1.28	0.00	1.25	0.00	3.75		1.25		10.43-11.16	7.5	2522.7	10	7.5
2 - 6	1.17	0.00	1.15	0.00	3.85		1.15		11.16-11.24	7.4	2522.7	10	7.7
6 - 24	1.22	0.00	1.20	0.00	3.80		1.20		11.24-11.4	7.9	2522.5	10	7.6
24 - 48	0.00	0.00	0.00	0.00	3.85	1.01	1.15		11.4-11.29	8.0	2522.4	10	7.7
Total	8.77	1.51	8.59	1.12	3.85	1.01	4.75	0.11					

Observations:

mL aliquot	mL Oxalic
10	10.1

Results:

Product	Amount g, mL	Assay mg/L, g/t Au	Assay mg/L, g/t Ag	Distribution % Au	Distribution % Ag
2 Hr PLS	999.8		0.4	0.0	2.2
6 Hr PLS	999.8		0.7	0.0	3.7
24 Hr PLS	999.6		1.2	0.0	6.1
48 Hr PLS	999.5	0.03	1.5	10.9	8.1
Wash Solution	1452.4	0.03	0.3	14.4	2.1
Residue	490.9	0.46	37.2	74.7	89.8
Calculated Head		0.62	41.4	100.0	100.0
ERD Head		0.60	40.6		
Accountability		102.7	102.0		



Test Description:

Test #:	CN-7 (TUFF)
Project #:	PJ124 Almaden
Operator:	JC
Date:	November 20th, 2012
Purpose:	Standard batch cyanidation test
Procedure:	48 hour milled sample leach

Primary Grind:	25.5	minutes @ 60% solids	Feed:	500	g Knelson tails
Actual K_{80}:	85	µm	Solution Volume:	1000	mL (tap water)
NaCN Addition:	3.06	g	Pulp Density:	33.3	% Solids
Tare Mass:	1169.1	g	Solution Composition:	3.0	g/L NaCN (maintained)
Initial Gross Mass:	2669.0	g	pH Range:	10.5 - 11.0	maintained with lime
Final Gross Mass:	2667.5	g			
			24 hr Gold Recovery =	43.4	%
			24 hr Silver Recovery =	46.9	%

Cyanidation Schedule:

Reagent addition (kg/t of cyanide feed)	NaCN:	10.90	CaO:	0.49
Reagent consumption (kg/t of cyanide feed)	NaCN:	2.31	CaO:	0.49

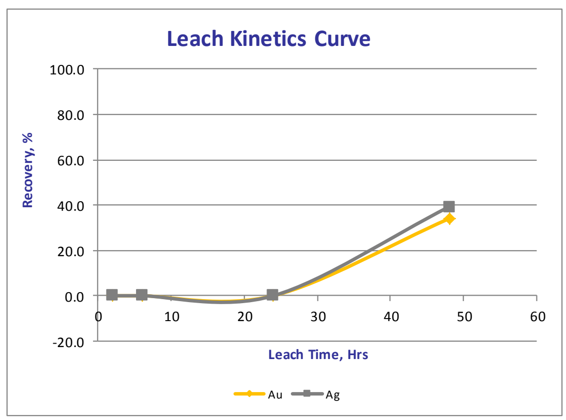
Start Time: 10:20	Added Grammes				Residual		Consumed		pH	DO_2	Bottle Mass g	mL aliquot	mL $AgNO_3$
	Actual		Equivalent		Grams		Grams						
Time Hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO					
0	3.06	0.33	3.00	0.24					7.78-10.7	8.2	2669.0	-	-
0 - 2	0.92	0.00	0.90	0.00	2.10		0.90		10.7-10.64	5.6	2667.7	5	2.1
2 - 6	0.00	0.00	0.00	0.00	3.01		-0.01		10.6-10.6	6.3	2671.7	5	3.0
6 - 24	1.58	0.00	1.55	0.00	1.44		1.56		10.6-10.49	5.5	2664.1	10	2.9
24 - 48	0.00	0.00	0.00	0.00	4.29	0.00	-1.29		10.49-10.75	5.9	2667.5	10	8.6
Total	5.56	0.33	5.45	0.24	4.29	0.00	1.15	0.24					

Observations:	

mL aliquot	mL Oxalic
10	0

Results:

Product		Amount g, mL	Assay mg/L, g/t		Distribution %	
			Au	Ag	Au	Ag
2	Hr PLS	998.6			0.0	0.0
6	Hr PLS	1002.6			0.0	0.0
24	Hr PLS	995.0			0.0	0.0
48	Hr PLS	998.4	0.13	2.3	34.0	39.2
Wash Solution		1322.6	0.03	0.4	9.4	7.7
Residue		494.9	0.48	7.00	56.6	53.1
Calculated Head			0.85	13.2	100.0	100.0
ERD Head			0.77	10.7		
Accountability			110.2	123.2		



Test Description:

Test #:	CN-8 (TUFF)
Project #:	PJ124 Almaden
Operator:	JC
Date:	November 20th, 2012
Purpose:	Standard batch cyanidation test
Procedure:	48 hour milled sample leach

Primary Grind:	25.5	minutes @ 60% solids	Feed:	500	g Knelson tails
Actual K_{80}:	85	µm	Solution Volume:	1000	mL (tap water)
NaCN Addition:	5.10	g	Pulp Density:	33.3	% Solids
Tare Mass:	1164.3	g	Solution Composition:	5.0	g/L NaCN (maintained)
Initial Gross Mass:	2664.3	g	pH Range:	10.5 - 11.0	maintained with lime
Final Gross Mass:	2661.9	g			
			24 hr Gold Recovery =	37.4	%
			24 hr Silver Recovery =	47.7	%

Cyanidation Schedule:

Reagent addition (kg/t of cyanide feed)	NaCN:	15.99	CaO:	1.04
Reagent consumption (kg/t of cyanide feed)	NaCN:	4.49	CaO:	1.04

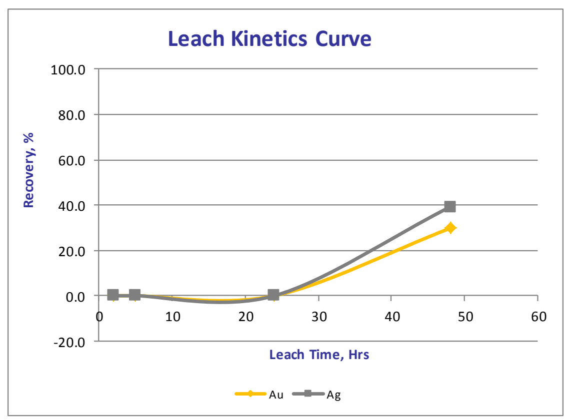
Start Time: 0.46	Added Grammes				Residual		Consumed		pH	DO_2	Bottle Mass g	mL aliquot	mL $AgNO_3$
	Actual		Equivalent		Grams		Grams						
Time Hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO					
0	5.10	0.70	5.00	0.52					7.53-10.16	5.3	2664.3	-	-
0 - 2	1.02	0.00	1.00	0.00	4.00		1.00		10.16-10.84	5.2	2664	5	4.0
2 - 6	2.04	0.00	2.00	0.00	3.01		1.99		10.8-10.9	6.1	2667.3	5	3.0
6 - 24	0.00	0.00	0.00	0.00	5.60		-0.61		10.9-10.84	4.6	2664.9	5	5.6
24 - 48	0.00	0.00	0.00	0.00	5.14	0.00	-0.14		10.84-10.99	5.0	2661.9	10	10.3
Total	8.16	0.70	8.00	0.52	5.14	0.00	2.24	0.52					

Observations:

mL aliquot	mL Oxalic
10	0

Results:

Product		Amount g, mL	Assay mg/L, g/t Au	Assay mg/L, g/t Ag	Distribution % Au	Distribution % Ag
2	Hr PLS	999.7			0.0	0.0
5	Hr PLS	1003.0			0.0	0.0
24	Hr PLS	1000.6			0.0	0.0
48	Hr PLS	997.6	0.09	2.3	29.7	39.2
Wash Solution		1264.9	0.02	0.4	7.6	8.6
Residue		495.5	0.42	6.70	62.6	52.3
Calculated Head			0.67	12.8	100.0	100.0
ERD Head			0.7	10.7		
Accountability			95.8	119.8		



Test Description:

Test #:	CN-9 (Dyke with regrind)
Project #:	PJ124 Almaden
Operator:	JC
Date:	November 20th, 2012
Purpose:	Standard batch cyanidation test
Procedure:	48 hour milled sample leach

Regrind Grind:	25	minutes @ 60% solids	Feed:	500	g Knelson tails
Actual K_{80}:	45	µm	Solution Volume:	1000	mL (tap water)
NaCN Addition:	5.10	g	Pulp Density:	33.3	% Solids
Tare Mass:	1162.2	g	Solution Composition:	5.0	g/L NaCN (maintained)
Initial Gross Mass:	2662.2	g	pH Range:	10.5 - 11.0	maintained with lime
Final Gross Mass:	2661.7	g			
			24 hr Gold Recovery =	60.9	%
			24 hr Silver Recovery =	87.0	%

Cyanidation Schedule:

Reagent addition (kg/t of cyanide feed)	NaCN:	23.30	CaO:	0.34
Reagent consumption (kg/t of cyanide feed)	NaCN:	15.99	CaO:	-0.04

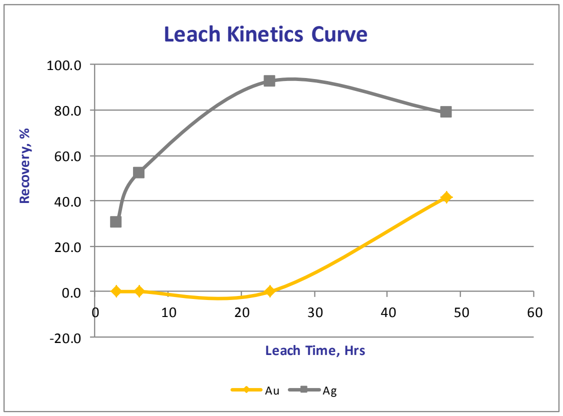
Start Time: 0.43	Added Grammes				Residual		Consumed		pH	DO_2	Bottle Mass g	mL aliquot	mL $AgNO_3$
	Actual		Equivalent		Grams		Grams						
Time Hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO					
0	5.10	0.23	5.00	0.17					7.84-10.9	7.6	2662.2	-	-
0 - 2	1.94	0.00	1.90	0.00	3.10		1.90		10.9-11.09	7.5	2661.9	10	6.2
2 - 6	2.45	0.00	2.40	0.00	2.60		2.40		11.09-11.45	7.2	2661.9	10	5.2
6 - 24	2.40	0.00	2.35	0.00	2.65		2.35		11.45-12.01	7.8	2662.0	10	5.3
24 - 48			0.00	0.00	3.65	0.19	1.35		12.01-12.08	7.6	2661.7	10	7.3
Total	11.89	0.23	11.65	0.17	3.65	0.19	8.00	-0.02					

Observations:

mL aliquot	mL Oxalic
10	1.9

Results:

Product		Amount g, mL	Assay mg/L, g/t		Distribution %	
			Au	Ag	Au	Ag
3	Hr PLS	999.7		6.9	0.0	30.3
6	Hr PLS	999.7		11.9	0.0	52.4
24	Hr PLS	999.8		21.0	0.0	92.7
48	Hr PLS	999.5	0.08	16.1	41.4	78.9
Wash Solution		1393.9	0.03	1.3	19.4	8.1
Residue		495.2	0.17	6.0	39.1	13.0
Calculated Head			0.43	46.2	100.0	100.0
ERD Head			0.44	40.0		
Accountability			98.7	115.5		



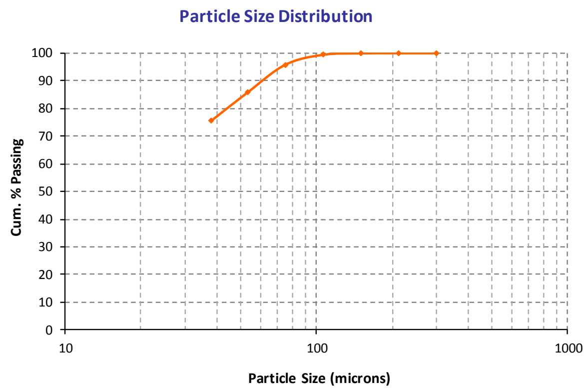
Size Distribution Determination Worksheet

Sample Tracking	
Sample ID:	Dyke Knelson Tails with 25 min regrind (CN-9)
Project No.:	PJ124
Project Name:	Almaden
Date:	December 6th, 2012
Technician:	LH
Objective:	Confirm Grind

Screen Size (µm)	Sample Dry Wt (g)	Weight (%)	Cum. Weight (%)	Cum. Weight (%) Passing
300	**0.0**	0.00	0.00	100.00
212	**0.0**	0.00	0.00	100.00
150	**0.1**	0.04	0.04	99.96
106	**1.3**	0.52	0.57	99.43
75	**9.0**	3.62	4.18	95.82
53	**25.1**	10.08	14.27	85.73
38	**25.5**	10.24	24.51	75.49
-38 pan	**2.6**	1.04		
-38 Total	187.9	75.49	100.00	0.00
Total	248.9	100.00		

Mass Accountability	
Start Mass	**249.1**
+38µm wet screen	**63.8**
-38µm wet screen	**185.3**
Mass Rec. (%)	99.92

p 80	45 µm



Test Description:

Test #:	CN-10 (Limestone with regrind)
Project #:	PJ124 Almaden
Operator:	JC
Date:	November 20th, 2012
Purpose:	Standard batch cyanidation test
Procedure:	48 hour milled sample leach

Regrind Grind:	25	minutes @ 60% solids	Feed:	500	g Knelson tails
Actual K_{80}:	48	µm	Solution Volume:	1000	mL (tap water)
NaCN Addition:	5.10	g	Pulp Density:	33.3	% Solids
Tare Mass:	1158.1	g	Solution Composition:	5.0	g/L NaCN (maintained)
Initial Gross Mass:	2658.1	g	pH Range:	10.5 - 11.0	maintained with lime
Final Gross Mass:	2657.6	g			
			24 hr Gold Recovery =	57.6	%
			24 hr Silver Recovery =	77.7	%

Cyanidation Schedule:

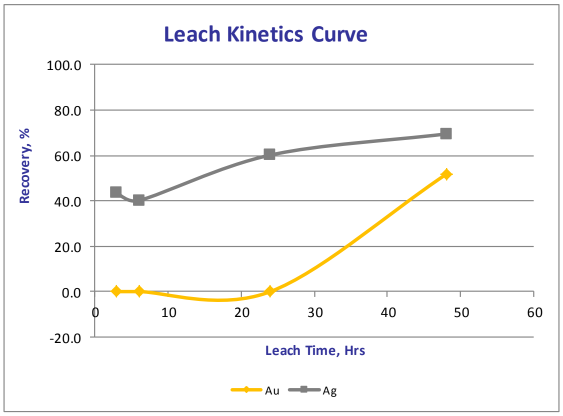
Reagent addition (kg/t of cyanide feed)	NaCN:	15.72	CaO:	0.30
Reagent consumption (kg/t of cyanide feed)	NaCN:	5.90	CaO:	0.04

Start Time:	Added Grammes				Residual		Consumed		pH	DO_2	Bottle Mass g	mL aliquot	mL $AgNO_3$
	Actual		Equivalent		Grams		Grams						
Time Hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO					
0	5.10	0.20	5.00	0.15					7.74-10.03	8.4	2658.1	-	-
0 - 3	1.33	0.00	1.30	0.00	3.70		1.30		10.03-10.68	7.6	2657.8	10	7.4
3 - 6	1.28	0.00	1.25	0.00	3.75		1.25		10.68-10.82	7.4	2657.9	10	7.5
6 - 24	0.31	0.00	0.30	0.00	4.70		0.30		10.82-11.04	7.7	2657.8	10	9.4
24 - 48	0.00	0.00	0.00	0.00	4.90	0.13	0.10		11.04-10.92	6.6	2657.6	10	9.8
Total	8.02	0.20	7.86	0.15	4.90	0.13	2.95	0.02					

Observations:		mL aliquot	mL Oxalic
		10	1.3

Results:

Product		Amount g, mL	Assay mg/L, g/t		Distribution %	
			Au	Ag	Au	Ag
3	Hr PLS	999.7		11.4	0.0	43.4
6	Hr PLS	999.8		10.4	0.0	40.1
24	Hr PLS	999.7		15.6	0.0	60.2
48	Hr PLS	999.5	0.11	16.2	51.6	69.4
Wash Solution		1403.6	0.01	1.5	6.0	8.2
Residue		496.2	0.20	11.80	42.4	22.3
Calculated Head			0.47	52.8	100.0	100.0
ERD Head			0.34	30.2		
Accountability			138.8	175.0		



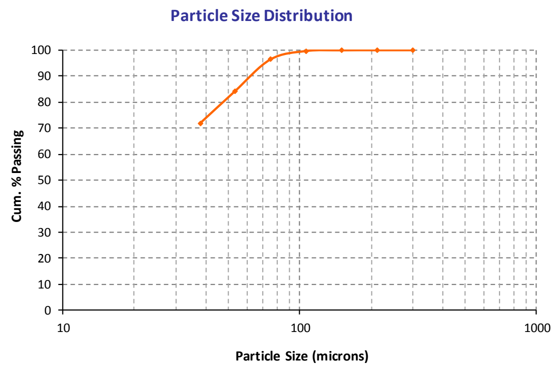
Size Distribution Determination Worksheet

Sample Tracking	
Sample ID:	Limestone Knelson Tails with 25 min regrind (CN-10)
Project No.:	PJ124
Project Name:	Almaden
Date:	December 6th, 2012
Technician:	LH
Objective:	Confirm Grind

Screen Size (µm)	Sample Dry Wt (g)	Weight (%)	Cum. Weight (%)	Cum. Weight (%) Passing
300	**0.0**	0.00	0.00	100.00
212	**0.0**	0.00	0.00	100.00
150	**0.1**	0.03	0.04	99.96
106	**0.7**	0.29	0.32	99.68
75	**8.3**	3.26	3.58	96.42
53	**31.2**	12.31	15.90	84.10
38	**30.6**	12.09	27.99	72.01
-38 pan	**2.3**	0.89		
-38 Total	182.2	72.01	100.00	0.00
Total	253.1	100.00		

Mass Accountability	
Start Mass	**253.1**
+38µm wet screen	**73.1**
-38µm wet screen	**180.0**
Mass Rec. (%)	99.98

p 80	48 µm



Test Description:

Test #:	CN-11 (Black shale with regrind)
Project #:	PJ124 Almaden
Operator:	JC
Date:	November 20th, 2012
Purpose:	Standard batch cyanidation test
Procedure:	48 hour milled sample leach

Regrind Grind:	25	minutes @ 60% solids	Feed:	500	g Knelson tails
Actual K_{80}:	46	µm	Solution Volume:	1000	mL (tap water)
NaCN Addition:	5.10	g	Pulp Density:	33.3	% Solids
Tare Mass:	1154.1	g	Solution Composition:	5.0	g/L NaCN (maintained)
Initial Gross Mass:	2654.0	g	pH Range:	10.5 - 11.0	maintained with lime
Final Gross Mass:	2653.5	g			
			24 hr Gold Recovery =	23.0	%
			24 hr Silver Recovery =	56.5	%

Cyanidation Schedule:

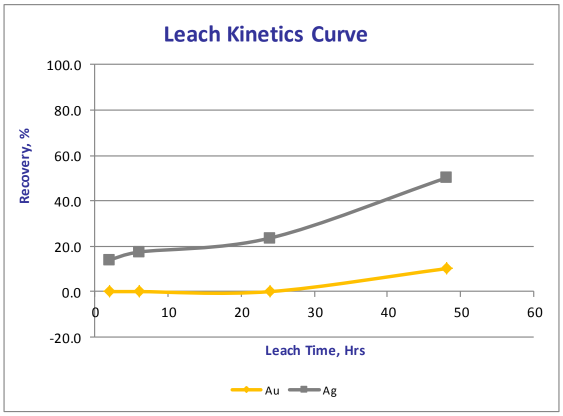
Reagent addition (kg/t of cyanide feed)	NaCN:	21.50	CaO:	0.76
Reagent consumption (kg/t of cyanide feed)	NaCN:	13.58	CaO:	0.45

Start Time: 10:00	Added Grammes				Residual		Consumed		pH	DO_2	Bottle Mass g	mL aliquot	mL $AgNO_3$
	Actual		Equivalent		Grams		Grams						
Time Hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO					
0	5.10	0.51	5.00	0.38					7.6-10.23	5.0	2654.0	-	-
0 - 2	2.65	0.00	2.60	0.00	2.40		2.60		10.23-11.27	8.5	2653.8	10	4.8
2 - 6	0.92	0.00	0.90	0.00	4.10		0.90		11.27-11.51	8.4	2654.2	10	8.2
6 - 24	2.30	0.00	2.25	0.00	2.70		2.29		11.51-11.47	8.0	2637.1	10	5.5
24 - 48			0.00	0.00	4.00	0.15	1.00		11.47-11.15	8.1	2653.5	10	8.0
Total	10.97	0.51	10.75	0.38	4.00	0.15	6.79	0.22					

		mL aliquot	mL Oxalic
Observations:	color of indication was light orange, making it difficult to determine end point	10	1.55

Results:

Product		Amount g, mL	Assay mg/L, g/t		Distribution %	
			Au	Ag	Au	Ag
2	Hr PLS	999.7		3.8	0.0	13.9
6	Hr PLS	1000.1		4.7	0.0	17.4
24	Hr PLS	983.0		6.4	0.0	23.6
48	Hr PLS	999.4	0.03	12.2	10.1	50.1
Wash Solution		1408.0	0.03	1.2	12.9	6.4
Residue		492.0	0.51	24.00	77.0	43.5
Calculated Head			0.66	55.1	100.0	100.0
ERD Head			0.60	40.6		
Accountability			110.7	135.7		



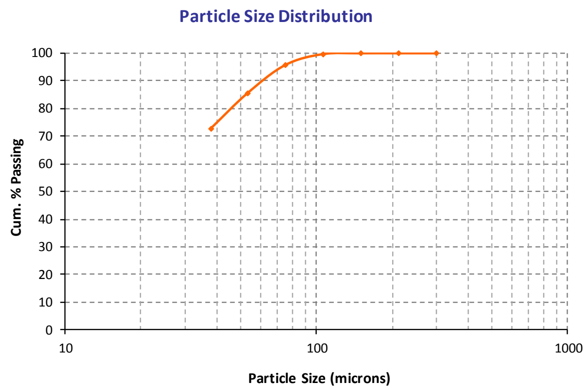
Size Distribution Determination Worksheet

Sample Tracking	
Sample ID:	Blackshale Knelson Tails with 25 min regrind (CN-11)
Project No.:	PJ124
Project Name:	Almaden
Date:	December 7th, 2012
Technician:	LH
Objective:	Confirm Grind

Screen Size (µm)	Sample Dry Wt (g)	Weight (%)	Cum. Weight (%)	Cum. Weight (%) Passing
300	**0.0**	0.00	0.00	100.00
212	**0.0**	0.00	0.01	99.99
150	**0.0**	0.01	0.02	99.98
106	**0.8**	0.29	0.31	99.69
75	**10.1**	3.95	4.26	95.74
53	**26.1**	10.23	14.49	85.51
38	**32.2**	12.63	27.11	72.89
-38 pan	**3.5**	1.37		
-38 Total	185.6	72.89	100.00	0.00
Total	254.6	100.00		

Mass Accountability	
Start Mass	**256.5**
+38µm wet screen	**72.7**
-38µm wet screen	**182.1**
Mass Rec. (%)	99.27

p 80	46 µm



Test Description:

Test #:	CN-12 (TUFF with regrind)
Project #:	PJ124 Almaden
Operator:	JC
Date:	November 20th, 2012
Purpose:	Standard batch cyanidation test
Procedure:	48 hour milled sample leach

Regrind Grind:	25	minutes @ 60% solids	Feed:	500	g Knelson tails
Actual K_{80}:	48	µm	Solution Volume:	1000	mL (tap water)
NaCN Addition:	5.10	g	Pulp Density:	33.3	% Solids
Tare Mass:	1161.2	g	Solution Composition:	5.0	g/L NaCN (maintained)
Initial Gross Mass:	2661.1	g	pH Range:	10.5 - 11.0	maintained with lime
Final Gross Mass:	2670.3	g			
			24 hr Gold Recovery =	41.5	%
			24 hr Silver Recovery =	58.6	%

Cyanidation Schedule:

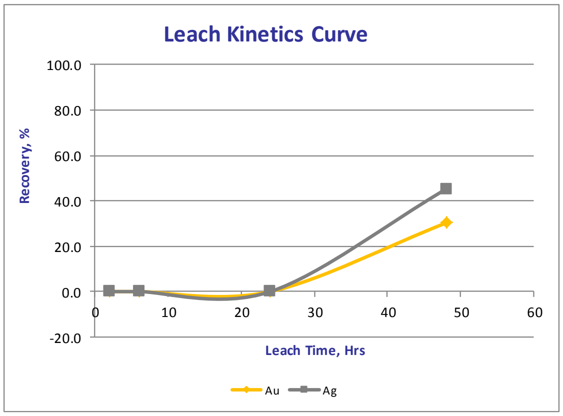
Reagent addition (kg/t of cyanide feed)	NaCN:	21.32	CaO:	0.28
Reagent consumption (kg/t of cyanide feed)	NaCN:	11.34	CaO:	-0.10

Start Time:	Added Grammes				Residual		Consumed		pH	DO_2	Bottle Mass g	mL aliquot	mL $AgNO_3$
	Actual		Equivalent		Grams		Grams						
Time Hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO					
0	5.10	0.19	5.00	0.14					7.96-10.14	6.2	2661.1	-	-
0 - 2	1.48	0.00	1.45	0.00	3.55		1.45		10.14-11.02	6.4	2661	10	7.1
2 - 6	1.49	0.00	1.46	0.00	3.29		1.71		11.02-11.02	5.5	2674.3	10	6.5
6 - 24	1.79	0.00	1.75	0.00	4.05		0.95		11.02-10.91	5.1	2674.0	5	4.0
24 - 48	1.02	0.00	1.00	0.00	3.43	0.19	1.57		10.91-10.5	5.2	2670.3	10	6.8
Total	10.88	0.19	10.66	0.14	3.43	0.19	5.67	-0.05					

		mL aliquot	mL Oxalic
Observations:	color of indication was light orange, making it difficult to determine end point	10	1.9

Results:

Product		Amount g, mL	Assay mg/L, g/t Au	Assay mg/L, g/t Ag	Distribution % Au	Distribution % Ag
2	Hr PLS	999.8			0.0	0.0
6	Hr PLS	1013.1			0.0	0.0
24	Hr PLS	1012.8			0.0	0.0
48	Hr PLS	1009.1	0.10	2.6	30.2	45.1
Wash Solution		1377.8	0.03	0.6	11.3	13.4
Residue		487.9	0.44	5.50	58.5	41.4
Calculated Head			0.75	13.3	100.0	100.0
ERD Head			0.7	10.7		
Accountability			107.4	124.1		

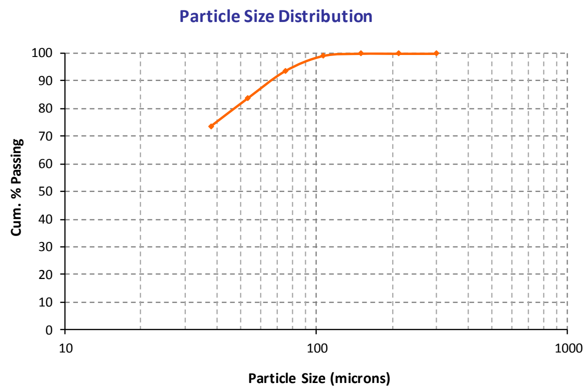


Size Distribution Determination Worksheet

Sample Tracking	
Sample ID:	TUFF Knelson Tails with 25 min regrind (CN-12)
Project No.:	PJ124
Project Name:	Almaden
Date:	December 10th, 2012
Technician:	LH
Objective:	Confirm Grind

p 80	48 µm

Screen Size (µm)	Sample Dry Wt (g)	Weight (%)	Cum. Weight (%)	Cum. Weight (%) Passing
300	**0.5**	0.20	0.20	99.80
212	**0.0**	0.00	0.20	99.80
150	**0.0**	0.00	0.20	99.80
106	**2.0**	0.81	1.02	98.98
75	**13.4**	5.46	6.48	93.52
53	**24.4**	9.94	16.42	83.58
38	**24.4**	9.94	26.37	73.63
-38 pan	**2.2**	0.90		
-38 Total	180.7	73.63	100.00	0.00
Total	**245.4**	**100.00**		



Mass Accountability	
Start Mass	**249.7**
+38µm wet screen	**67.5**
-38µm wet screen	**178.5**
Mass Rec. (%)	98.28

Test Description:

Test #:	CN-13 (High grade flotation con)
Project #:	PJ124 Almaden
Operator:	JC
Date:	November 20th, 2012
Purpose:	Standard batch cyanidation test
Procedure:	48 hour milled sample leach

Primary Grind:	16	minutes @ 60% solids	Feed:	203	g flotation con
Actual K_{80}:	92	µm	Solution Volume:	407	mL (tap water)
NaCN Addition:	8.30	g	Pulp Density:	33.3	% Solids
Tare Mass:	1019.6	g	Solution Composition:	20.0	g/L NaCN (maintained)
Initial Gross Mass:	1629.3	g	pH Range:	10.5 - 11.0	maintained with lime
Final Gross Mass:	1613.5	g			
			24 hr Gold Recovery =	88.0	%
			24 hr Silver Recovery =	93.1	%

Cyanidation Schedule:

Reagent addition (kg/t of cyanide feed)	NaCN:	92.30	CaO:	1.50
Reagent consumption (kg/t of cyanide feed)	NaCN:	123.87	CaO:	0.41

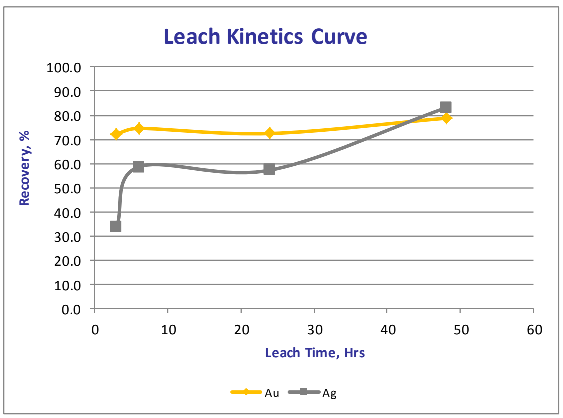
Start Time: 11:20	Added Grammes				Residual		Consumed		pH	DO_2	Bottle Mass g	mL aliquot	mL $AgNO_3$
	Actual		Equivalent		Grams		Grams						
Time Hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO					
0	8.14	0.41	7.98	0.30					6.72-9.98	0.2	1629.3	-	-
0 - 2	5.24	0.00	5.14	0.00	1.23		6.74		9.98-13.38	0.2	1633.4	10	6.0
2 - 6	1.67	0.00	1.64	0.00	2.66		5.32		13.31-13.43	0.1	1631.5	10	13.0
6 - 24	4.07	0.00	3.99	0.00	1.64		6.34		13.43-12.81	0.3	1618.0	10	8.3
24 - 48			0.00	0.00	1.23	0.22	6.75		12.81-11.5	1.4	1613.5	10	6.3
Total	19.12	0.41	18.74	0.30	1.23	0.22	25.15	0.08					

Observations:	

mL aliquot	mL Oxalic
10	5.66

Results:

Product		Amount g, mL	Assay mg/L, g/t		Distribution %	
			Au	Ag	Au	Ag
3	Hr PLS	410.8	6.93	217.7	72.1	33.6
6	Hr PLS	408.9	7.03	376.4	74.6	58.6
24	Hr PLS	395.4	6.88	370.9	72.4	57.3
48	Hr PLS	390.9	5.91	430.9	78.8	83.0
Wash Solution		1417.5	0.26	18.9	9.3	10.0
Residue		196.5	2.41	93.80	12.0	6.9
Calculated Head			20.1	1355.6	100.0	100.0
ERD Head			18.9	920.2		
Accountability			106.3	147.3		

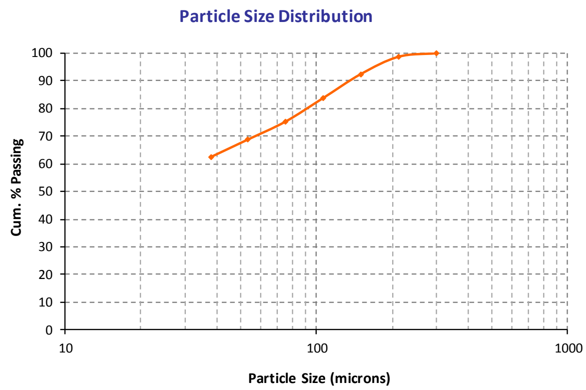


Size Distribution Determination Worksheet

Sample Tracking	
Sample ID:	High Grade Conc Leach Residue (CN-13)
Project No.:	PJ124
Project Name:	Almaden
Date:	December 7th, 2012
Technician:	LH
Objective:	Confirm Grind

p 80	92 µm

Screen Size (µm)	Sample Dry Wt (g)	Weight (%)	Cum. Weight (%)	Cum. Weight (%) Passing
300	**0.0**	0.00	0.00	100.00
212	**1.9**	1.26	1.26	98.74
150	**9.4**	6.26	7.52	92.48
106	**13.1**	8.72	16.25	83.75
75	**12.8**	8.52	24.77	75.23
53	**9.8**	6.52	31.29	68.71
38	**9.3**	6.19	37.48	62.52
-38 pan	**0.6**	0.40		
-38 Total	93.9	62.52	100.00	0.00
Total	150.2	100.00		



Mass Accountability	
Start Mass	**150.3**
+38µm wet screen	**56.9**
-38µm wet screen	**93.3**
Mass Rec. (%)	99.93

APPENDIX C –FLOTATION TEST WORKSHEETS

Test Description:

Test #:	Black Shale F-1
Project #:	PJ124 - Almaden Ixtaca
Operator:	Marjorie Colebrook
Date:	October-31-12
Purpose:	Baseline Pb/Zn Rougher Flotation - No Carbon Prefloat
Procedure:	As outlined below.
Feed:	1 kg of minus 10 mesh Black Shale Comp
Grind:	p80 = 168 microns. 1 kg @ 60% solids in a laboratory rod mill.
Regrind:	N/A
Comments:	

Flotation Schedule:

	Reagents (g/tonne)					Reagents (ml or g)					Time, minutes				
Stage	Lime	NaCN	$ZnSO_4$	3418A	F140	Lime	NaCN	ZnSO4	3418A	F140	Grind	Cond.	Froth	pH	Ep
Primary Grind	500	20	60			0.50		3.0			8.5			8.2	-70.4
Pb Rougher 1	680			20	23	0.68			10	0.046		1	1	9.0	-110.9
Pb Rougher 2	0			5	23	0.00			2.5	0.046		1	2	9.0	-111.3
Pb Rougher 3	10			5	0	0.01			2.5	0.000		1	2	9.0	-111.5
Total	1190	20	60	30	46	1.19		3.0	15.0	0.092	8.5	3	5		
	Reagents (g/tonne)					Reagents (ml or g)					Time, minutes				
Stage	Lime		$CuSO_4$	SIPX	F140	Lime		CuSO4	SIPX	F140		Cond.	Froth	pH	Ep
Zn Conditioner	1580		100			1.58		5				3		10.9	-216.3
Zn Rougher 1	0			20	11.5				10	0.023		1	2	10.7	-203
Zn Rougher 2	610			10	11.5	0.61			5	0.023		1	3	11.0	-222.3
Total	2190		100	30	23.0	2.19		5	15	0.046		5	5		

Stage	Rougher
Flotation Cell	2 litre cell
Speed: rpm	1200

Observations:	

Product	Weight g
Pb Rougher Conc 1	6.7
Pb Rougher Conc 2	14.8
Pb Rougher Conc 3	9.6
Zn Roughr Conc 1	17.6
Zn Rougher Conc 2	52.1
Rougher Tail	895.0

Reagent Strength:

Lime	100	%
NaCN ZnSO4 Compl	2	%
3418A	0.2	%
SIPX	0.2	%
CuSO4	2	%
F140	2	%
Charge	1000.00	g

*3 Parts Zinc Sulphate, 1 part sodium cyanide mixed
2.00g ZNSO4 + 0.66g NACN in 100ml water

Prep/Assay Instructions:	**Assay for Pb, Zn, Fe, Ag by AA at BCA. Assay Au by FA at BCA. Assay C, S by Leco at SGS** **Do not pulverise** **PSD on tails**

Mass Balance:

Product	Weight		Assays, %, g/t							% Distribution						
	g	%	Pb	Zn	Fe	Ag	Au	C	S	Pb	Zn	Fe	Ag	Au	C	S
Pb Rougher 1	**6.7**	0.68	**0.85**	**0.40**	**3.48**	**101**	**2.60**	**4.79**	**2.73**	2.56	0.58	0.61	1.46	1.97	0.87	0.55
Pb Rougher 2	**14.8**	1.48	**7.68**	**0.72**	**4.94**	**606**	**14.52**	**7.22**	**5.30**	50.84	2.25	1.89	19.17	24.17	2.87	2.33
Pb Rougher 3	**9.6**	0.96	**1.74**	**0.53**	**4.27**	**420**	**3.24**	**6.69**	**3.72**	7.49	1.07	1.06	8.64	3.50	1.73	1.06
Zn Rougher 1	**17.6**	1.76	**0.58**	**16.64**	**14.19**	**442**	**2.68**	**4.35**	**11.90**	4.57	61.96	6.44	16.62	5.30	2.06	6.21
Zn Rougher 2	**52.1**	5.23	**0.62**	**2.23**	**25.07**	**255**	**7.14**	**2.54**	**29.00**	14.51	24.66	33.77	28.44	41.88	3.56	44.90
Rougher Tails	**895.0**	89.88	**0.05**	**0.05**	**2.43**	**13**	**0.23**	**3.69**	**1.69**	20.04	9.49	56.24	25.67	23.18	88.91	44.96
Calculated Head	995.7	100.00	0.22	0.47	3.88	47	0.9	3.73	3.38	100.00	100.00	100.00	100.00	100.00	100.00	100.00
ERD Head	1000.0	100.00	0.23	0.43	3.20	45	1.0	3.68	3.38	-	-	-	-	-	-	-
Call Factor	99.6	-	96.1	109.2	121.5	105.0	91.3	101.5	100.1	-	-	-	-	-	-	-

Combined Products:

Pb Rougher Conc 1	6.73	0.68	0.85	0.40	3.48	101	2.60	3.81	4.79	2.56	0.58	0.61	1.46	1.97	0.87	0.55
Pb Rougher Conc 1-2	21.51	2.16	5.54	0.62	4.48	448	10.79	2.52	6.46	53.39	2.83	2.49	20.63	26.14	3.74	2.87
Pb Rougher Conc 1-3	31.11	3.12	4.37	0.59	4.42	440	8.46	4.67	6.53	60.88	3.89	3.55	29.27	29.64	5.47	3.94
Zn Rougher Conc 1	17.55	1.76	0.58	16.64	14.19	442	2.68	13.20	4.35	4.57	61.96	6.44	16.62	5.30	2.06	6.21
Zn Rougher Conc 1-2	69.64	6.99	0.61	5.86	22.33	302	6.02	34.26	3.00	19.08	86.61	40.21	45.06	47.18	5.62	51.11

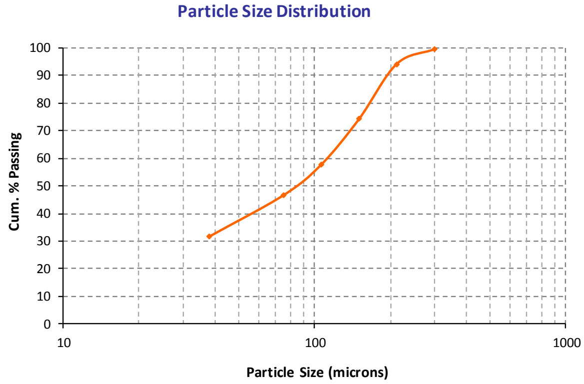
Size Distribution Determination Worksheet

Sample Tracking	
Sample ID:	BS-F1 Ro Tails
Project No.:	PJ124
Project Name:	Almaden
Date:	November 2nd, 2012
Technician:	PD
Objective:	Confirm Grind at 8.5min (Target p80=106microns)

Screen Size (µm)	Sample Dry Wt (g)	Weight (%)	Cum. Weight (%)	Cum. Weight (%) Passing
300	**0.9**	0.43	0.43	99.57
212	**11.3**	5.38	5.80	94.20
150	**42.0**	19.99	25.80	74.20
106	**34.8**	16.55	42.35	57.65
75	**23.2**	11.06	53.40	46.60
38	**31.5**	14.96	68.37	31.63
-38 pan	**2.4**	1.14		
-38 Total	66.5	31.63	100.00	0.00
Total	210.2	100.00		

Mass Accountability	
Start Mass	**213.2**
+38µm wet screen	
-38µm wet screen	**64.1**
Mass Rec. (%)	98.56

p 80 168 µm



Test Description:

Test #:	Black Shale F-2
Project #:	PJ124 - Almaden Ixtaca
Operator:	Marjorie Colebrook
Date:	November 1,2012
Purpose:	Baseline Pb/Zn Rougher Flotation - with Carbon Prefloat assessment
Procedure:	As outlined below.
Feed:	1 kg of minus 10 mesh Black Shale Comp
Grind:	p80 = 164 microns. 1 kg @ 60% solids in a laboratory rod mill.
Regrind:	N/A
Comments:	Visually assess need for C prefloat

Flotation Schedule:

	Reagents (g/tonne)					Reagents (ml or g)					Time, minutes				
Stage	Lime	NaCN	$ZnSO_4$	3418A	F140	Lime	NaCN	ZnSO4	3418A	F140	Grind	Cond.	Froth	pH	Ep
Primary Grind	500	20	60			0.50	3.0				8.5			Record	
C Prefloat													1	8.9	-118.1
Pb Rougher 1	40			20	46	0.04			10	0.092		1	1	9.0	-119
Pb Rougher 2	20			5	11.5	0.02			2.5	0.023		1	2	9.0	-120
Pb Rougher 3	20			5	11.5	0.02			2.5	0.023		1	2	9.0	-120.8
Total	**580**	**20**	**60**	**30**	**69.0**	**0.58**	**3.0**		**15.0**	**0.138**	**8.5**	**3**	**6**		
	Reagents (g/tonne)					Reagents (ml or g)					Time, minutes				
Stage	Lime		$CuSO_4$	SIPX	F140	Lime		CuSO4	SIPX	F140		Cond.	Froth	pH	Ep
Zn Conditioner	590		100			0.59		5				3		11.0	-227.8
Zn Rougher 1	0			20	11.5	0.00			10	0.023		1	2	10.8	-220
Zn Rougher 2	140			10	11.5	0.14			5	0.023		1	3	11.0	-230.5
Total	**730**		**100**	**30**	**23.0**	**0.59**		**5**	**15**	**0.046**		**5**	**5**		

Stage	Rougher
Flotation Cell	2 litre cell
Speed: rpm	1200

Observations:	

Product	Weight g
C Prefloat	**10.1**
Pb Rougher Conc 1	**4.7**
Pb Rougher Conc 2	**11.3**
Pb Rougher Conc 3	**12.7**
Zn Roughr Conc 1	**61.5**
Zn Rougher Conc 2	**38.3**
Rougher Tail	**883.4**

Reagent Strength:

Lime	100	%
NaCN ZnS(	2	%
3418A	0.2	%
SIPX	0.2	%
CuSO4	2	%
F140	2	%

Charge	1000.00	g

*3 Parts Zinc Sulphate, 1 part sodium cyanide mixed
2.00g ZNSO4 + 0.66g NACN in 100ml water

Prep/Assay	**Assay for Pb, Zn, Fe, Ag by AA at BCA. Assay Au by FA at BCA. Assay C, S by Leco at SGS** **Do not pulverise** **PSD on tails**

Mass Balance:

Product	Weight		Assays, %, g/t							% Distribution						
	g	%	Pb	Zn	Fe	Ag	Au	C	S	Pb	Zn	Fe	Ag	Au	C	S
C Prefloat	**10.1**	0.99	**0.15**	**0.23**	**2.32**	**85**	**1.32**	**8.80**	**2.59**	0.77	0.38	0.56	1.63	1.12	2.33	0.62
Pb Rougher 1	**4.7**	0.46	**0.87**	**0.24**	**31.62**	**791**	**10.00**	**7.35**	**2.95**	2.04	0.18	3.56	7.02	3.95	0.90	0.33
Pb Rougher 2	**11.3**	1.10	**4.77**	**0.45**	**34.96**	**946**	**12.64**	**7.13**	**4.42**	26.76	0.83	9.45	20.14	11.99	2.10	1.18
Pb Rougher 3	**12.7**	1.24	**3.07**	**0.53**	**6.55**	**542**	**17.67**	**7.06**	**6.12**	19.36	1.08	1.99	12.99	18.86	2.34	1.83
Zn Rougher 1	**61.5**	6.02	**0.66**	**8.78**	**22.81**	**226**	**7.37**	**2.43**	**30.00**	20.25	87.89	33.70	26.29	38.21	3.92	43.68
Zn Rougher 2	**38.3**	3.75	**0.32**	**0.71**	**19.65**	**131**	**3.16**	**2.64**	**23.80**	6.14	4.46	18.10	9.53	10.22	2.66	21.61
Rougher Tails	**883.4**	86.45	**0.06**	**0.04**	**1.54**	**13**	**0.21**	**3.70**	**1.47**	24.68	5.18	32.63	22.40	15.64	85.74	30.75
Calculated Head	1021.9	100.00	0.20	0.60	4.07	52	1.16	3.73	4.13	100.00	100.00	100.00	100.00	100.00	100.00	100.00
ERD Head	1000.0	100.00	0.23	0.43	3.20	45	0.98	3.68	3.38	-	-	-	-	-	-	-
Call Factor	102.2	-	84.1	138.7	127.4	115.8	118.8	101.5	122.4	-	-	-	-	-	-	-

Combined Products:

Pb Rougher Conc 1	4.69	0.46	0.87	0.24	31.62	791	10	4	7.35	2.04	0.18	3.56	7.02	3.95	0.90	0.33
Pb Rougher Conc 1-2	15.94	1.56	3.62	0.39	33.98	901	12	3	7.19	28.80	1.01	13.01	27.16	15.95	3.01	1.51
Pb Rougher Conc 1-3	28.60	2.80	3.37	0.45	21.84	742	14	6	7.14	48.16	2.09	15.01	40.15	34.81	5.35	3.34
Zn Rougher Conc 1	61.48	6.02	0.66	8.78	22.81	226	7	22	2.43	20.25	87.89	33.70	26.29	38.21	3.92	43.68
Zn Rougher Conc 1-2	99.82	9.77	0.53	5.68	21.60	190	6	46	2.51	26.39	92.34	51.80	35.82	48.42	6.57	65.29

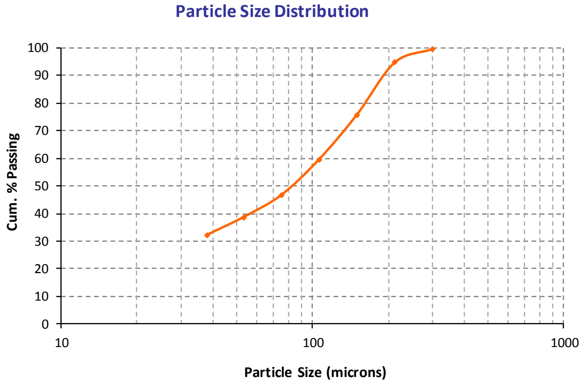
Size Distribution Determination Worksheet

Sample Tracking	
Sample ID:	BS-F2 Ro Tails
Project No.:	PJ124
Project Name:	Almaden
Date:	November 14th, 2012
Technician:	LH
Objective:	Confirm Grind at 8.5min (Target p80=106microns)

Screen Size (µm)	Sample Dry Wt (g)	Weight (%)	Cum. Weight (%)	Cum. Weight (%) Passing
300	**0.9**	0.40	0.40	99.60
212	**10.9**	4.89	5.29	94.71
150	**41.9**	18.86	24.15	75.85
106	**36.0**	16.21	40.36	59.64
75	**28.7**	12.90	53.26	46.74
53	**17.8**	8.01	61.26	38.74
38	**14.2**	6.40	67.67	32.33
-38 pan	**1.6**	0.73		
-38 Total	71.9	32.33	100.00	0.00
Total	222.3	100.00		

Mass Accountability	
Start Mass	**223.8**
+38µm wet screen	**152.0**
-38µm wet screen	**70.2**
Mass Rec. (%)	99.32

p 80 164 µm



Test Description:

Test #:	Black Shale F-3
Project #:	PJ124 - Almaden Ixtaca
Operator:	Marjorie Colebrook
Date:	November 20th, 2012
Purpose:	Baseline Pb/Zn Rougher Flotation - No Carbon Prefloat, finer grind
Procedure:	As outlined below.
Feed:	1 kg of minus 10 mesh Black Shale Comp
Grind:	p80 = 81 microns. 1 kg @ 60% solids in a laboratory rod mill.
Regrind:	N/A
Comments:	

Flotation Schedule:

	Reagents (g/tonne)					Reagents (ml or g)					Time, minutes				
Stage	Lime	NaCN	$ZnSO_4$	3418A	F140	Lime	NaCN	ZnSO4	3418A	F140	Grind	Cond.	Froth	pH	Ep
Primary Grind	500	20	60			0.50		3.0			15			8.9	-111.5
Pb Rougher 1	40			20	34.5	0.04			10	0.069		1	1	9.0	-116.9
Pb Rougher 2	20			5	11.5	0.02			2.5	0.023		1	2	9.0	-117.6
Pb Rougher 3	30			5	11.5	0.03			2.5	0.023		1	2	9.0	-116.8
Total	590	20	60	30	57.5	0.59		3.0	15.0	0.115	15.0	3	5		
	Reagents (g/tonne)					Reagents (ml or g)					Time, minutes				
Stage	Lime		$CuSO_4$	SIPX	F140	Lime		CuSO4	SIPX	F140		Cond.	Froth	pH	Ep
Zn Conditioner	650		100			0.65		5				3		11.0	-225.9
Zn Rougher 1	160			20	11.5	0.16			10	0.023		1	2	11.0	-227.3
Zn Rougher 2	0			10	11.5				5	0.023		1	3	11.0	-227.2
Total	810		100	30	23.0	0.81		5	15	0.046		5	5		

Stage	Rougher
Flotation Cell	2 litre cell
Speed: rpm	1200

Observations:	very poor froth

Product	Weight g
Pb Rougher Conc 1	12.6
Pb Rougher Conc 2	19.7
Pb Rougher Conc 3	13.6
Zn Roughr Conc 1	17.6
Zn Rougher Conc 2	39.4
Rougher Tail	894.8

Reagent Strength:

Lime	100	%
NaCN ZnSO4 Con	2	%
3418A	0.2	%
SIPX	0.2	%
CuSO4	2	%
F140	2	%
Charge	1000.00	g

*3 Parts Zinc Sulphate, 1 part sodium cyanide mixed
2.00g ZNSO4 + 0.66g NACN in 100ml water

Prep/Assay Instructions:	**Assay for Pb, Zn, Fe, Ag by AA at BCA. Assay Au by FA at BCA. Assay C, S by Leco at SGS** **Do not pulverise**

Mass Balance:

Product	Weight		Assays, %, g/t					% Distribution				
	g	%	Pb	Zn	Fe	Ag	Au	Pb	Zn	Fe	Ag	Au
Pb Rougher 1	12.6	1.26	0.39	0.38	3.21	233.6	5.89	2.53	1.12	1.18	5.75	7.65
Pb Rougher 2	19.7	1.97	0.48	0.36	3.08	218.6	1.34	4.80	1.68	1.76	8.39	2.71
Pb Rougher 3	13.6	1.36	2.78	0.39	3.27	482.4	4.96	19.15	1.26	1.29	12.77	6.93
Zn Rougher 1	17.6	1.76	1.39	17.78	4.68	479.0	6.12	12.40	73.79	2.39	16.42	11.07
Zn Rougher 2	39.4	3.95	1.32	1.92	12.46	259.4	10.82	26.55	17.91	14.28	19.98	43.97
Rougher Tails	894.8	89.70	0.08	0.02	3.04	21.0	0.30	34.58	4.23	79.11	36.70	27.66
Calculated Head	997.5	100.00	0.20	0.42	3.45	51.3	1.0	100.00	100.00	100.00	100.00	100.00
ERD Head	1000.0	100.00	0.23	0.43	3.20	45	1.0	-	-	-	-	-
Call Factor	99.8	-	84.5	97.8	107.9	114.9	99.6	-	-	-	-	-

Combined Products:

Pb Rougher Conc 1	12.61	1.26	0.39	0.38	3.21	233.6	5.9	2.53	1.12	1.18	5.75	7.65
Pb Rougher Conc 1-2	32.26	3.23	0.45	0.37	3.13	224.5	3.1	7.32	2.80	2.93	14.14	10.37
Pb Rougher Conc 1-3	45.81	4.59	1.14	0.38	3.17	300.8	3.7	26.47	4.06	4.22	26.91	17.30
Zn Rougher Conc 1	17.55	1.76	1.39	17.78	4.68	479.0	6.1	12.40	73.79	2.39	16.42	11.07
Zn Rougher Conc 1-2	56.98	5.71	1.34	6.81	10.06	327.0	9.4	38.95	91.71	16.67	36.39	55.04

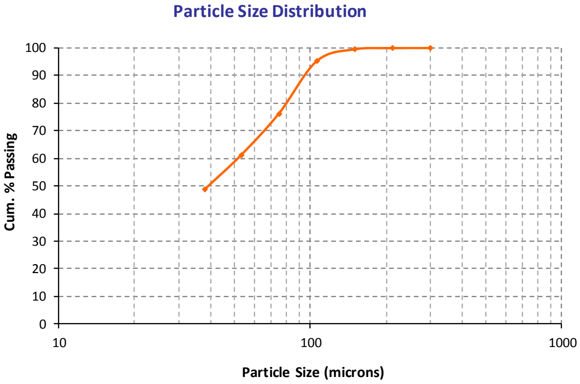
Size Distribution Determination Worksheet

Sample Tracking	
Sample ID:	BS-F3 Ro Tails
Project No.:	PJ124
Project Name:	Almaden
Date:	November 28th, 2012
Technician:	LH
Objective:	Confirm Grind at 15min

Screen Size (µm)	Sample Dry Wt (g)	Weight (%)	Cum. Weight (%)	Cum. Weight (%) Passing
300	**0.0**	0.00	0.00	100.00
212	**0.1**	0.04	0.04	99.96
150	**1.1**	0.43	0.47	99.53
106	**10.8**	4.26	4.73	95.27
75	**48.1**	18.97	23.70	76.30
53	**38.3**	15.10	38.80	61.20
38	**31.4**	12.38	51.18	48.82
-38 pan	**9.6**	3.79		
-38 Total	123.8	48.82	100.00	0.00
Total	253.6	100.00		

Mass Accountability	
Start Mass	**253.6**
+38µm wet screen	**139.4**
-38µm wet screen	**114.2**
Mass Rec. (%)	100.00

p 80	81 µm



Test Description:

Test #:	Black Shale F-4
Project #:	PJ124 - Almaden Ixtaca
Operator:	Marjorie Colebrook
Date:	November 27th, 2012
Purpose:	Baseline Bulk Flotation using HG F-1 Conditions
Procedure:	Natural pH, 300g/t CuSO4 and SIPX
Feed:	1kg of minus 1.7 mm Black Shale Met Sample
Grind:	p80 = 71 microns. 1000g @ 60% solids in lab rod mill
Comments:	

Flotation Schedule:

	Reagents (g/tonne)				Reagents (ml or g)				Time, minutes				
Stage	$CuSO_4$	SIPX	3418A	F-140	CuSO4	SIPX	3418A	F-140	Grind	Cond.	Froth	pH	Ep
Primary Grind	300				30				16.00			7.9	-57.8
Rougher 1		100	15	23.0		50	7.5	0.046		1	2	7.9	-57.8
Rougher 2		50	15	23.0		25	7.5	0.046		1	4	8.0	-64.8
Rougher 3		50	15	11.5		25	7.5	0.023		1	5	8.2	-74.2
Total	300	200	45	57.5	30	100	22.5	0.115	16.00	3	11		

Stage	Rougher
Flotation Cell	2 litre cell
Speed: rpm	1200

Observations:	

Product	Weight g
Conc 1	48.2
Conc 2	55.9
Con 3	84.7
Rougher Tail	808.1

Reagents:

CuSO4	1.0	%
SIPX	0.2	%
3418A	0.2	%
F-140	100	%
Charge	1000	g

Prep/Assay Instructions:	Assay for Pb, Zn, Fe, Ag by AA at BCA Priority. Assay Au by FA at BCA Priority. Assay S by Leco at SGS Pulverise Tails PSD on Tails

Mass Balance:

Product	Weight		Assays, %, g/t						% Distribution					
	g	%	Pb	Zn	Fe	Ag	Au	S	Pb	Zn	Fe	Ag	Au	S
Rougher Conc 1	48.2	4.83	1.46	2.91	14.27	189.0	3.23	21.60	31.86	40.32	21.26	20.59	18.62	31.02
Rougher Conc 2	55.9	5.61	1.75	2.77	16.27	268.4	3.82	23.30	44.11	44.52	28.12	33.93	25.55	38.83
Rougher Conc 3	84.7	8.50	0.41	0.56	6.06	151.4	4.84	7.93	15.62	13.54	15.87	29.01	49.06	20.03
Rougher Tails	808.1	81.07	0.02	0.01	1.39	9.0	0.07	0.42	8.40	1.63	34.75	16.46	6.77	10.12
Calculated Head	996.9	100.00	0.22	0.35	3.24	44.3	0.84	3.36	100.00	100.00	100.00	100.00	100.00	100.00
ERD Head	1000.0	100.00	0.23	0.43	3.20	44.7	0.98	3.68	3.38	-	-	-	-	-
Call Factor	99.7	-	95.1	80.5	101.4	99.3	85.8	91.5	-	-	-	-	-	-

Combined Products:

Rougher Conc 1	48.2	4.83	1.46	2.91	14.27	189	3.23	21.60	31.86	40.32	21.26	20.59	18.62	31.02
Rougher Conc 1-2	104.0	10.44	1.61	2.84	15.34	232	3.55	22.51	75.97	84.83	49.38	54.53	44.17	69.85
Rougher Conc 1-3	188.7	18.93	1.07	1.81	11.18	196	4.13	15.97	91.60	98.37	65.25	83.54	93.23	89.88

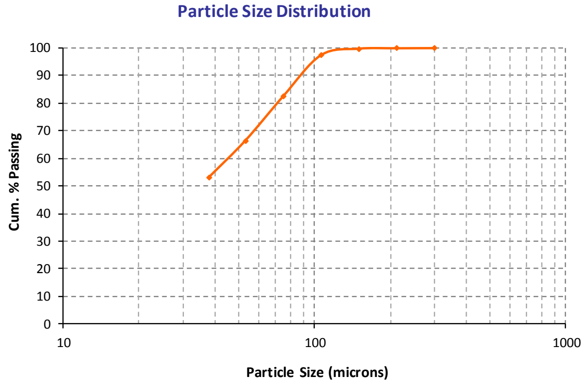
Size Distribution Determination Worksheet

Sample Tracking	
Sample ID:	BS-F4 Ro Tails
Project No.:	PJ124
Project Name:	Almaden
Date:	November 30th, 2012
Technician:	LH
Objective:	Confirm Grind

Screen Size (µm)	Sample Dry Wt (g)	Weight (%)	Cum. Weight (%)	Cum. Weight (%) Passing
300	**0.1**	0.05	0.05	99.95
212	**0.1**	0.05	0.09	99.91
150	**0.3**	0.14	0.23	99.77
106	**5.0**	2.34	2.58	97.42
75	**31.7**	14.86	17.44	82.56
53	**34.3**	16.08	33.52	66.48
38	**28.2**	13.22	46.74	53.26
-38 pan	**3.1**	1.45		
-38 Total	113.6	53.26	100.00	0.00
Total	213.3	100.00		

Mass Accountability	
Start Mass	**213.5**
+38µm wet screen	**103.0**
-38µm wet screen	**110.5**
Mass Rec. (%)	99.91

p 80 71 µm



Test Description:

Test #:	High Grade F-1
Project #:	PJ124 - Almaden Ixtaca
Operator:	Marjorie Colebrook
Date:	26-Oct-12
Purpose:	Baseline Bulk Flotation
Procedure:	Natural pH, 300g/t CuSO4 and SIPX
Feed:	2kg of minus 1.7 mm High Grade Met Sample
Grind:	p80 = 116 microns. 2000g @ 60% solids in lab rod mill
Comments:	

Flotation Schedule:

	Reagents (g/tonne)				Reagents (ml or g)				Time, minutes				
Stage	$CuSO_4$	SIPX	3418A	F-140	CuSO4	SIPX	3418A	F-140	Grind	Cond.	Froth	pH	Ep
Primary Grind	300				60				16.00				
Rougher 1		100	15	11.5		100	15	0.023		1	2	7.8	-47.7
Rougher 2		50	15	11.5		50	15	0.023		1	4	8.1	-60.6
Rougher 3		50	15	11.5		50	15	0.023		1	5	8.2	-66.3
Total	300	200	45	34.5	60	200	45.0	0.069	16.00	3	11		

Stage	Rougher
Flotation Cell	4 litre cell
Speed: rpm	1400

Observations:	viscous - increased speed to 1400 rpm

Product	Weight g
Conc 1	86.4
Conc 2	76.0
Con 3	74.6
Rougher Tail	1729.1

Reagents:

CuSO4	1.0	%
SIPX	0.2	%
3418A	0.2	%
F-140	100	%
Charge	2000	g

Prep/Assay Instructions:	**Assay for Pb, Zn, Fe, Ag by AA at BCA. Assay Au by FA at BCA. Assay S by Leco at SGS** **Do not pulverise** **PSD on Tails**

Mass Balance:

Product	Weight		Assays, %, g/t						% Distribution					
	g	%	Pb	Zn	Fe	Ag	Au	S	Pb	Zn	Fe	Ag	Au	S
Rougher Conc 1	86.4	4.39	0.53	1.08	29.03	2174.0	31.20	38.20	49.92	70.39	48.61	72.18	75.99	69.87
Rougher Conc 2	76.0	3.87	0.10	0.22	8.49	468.8	6.78	10.80	8.29	12.62	12.51	13.70	14.54	17.39
Rougher Conc 3	74.6	3.79	0.05	0.07	3.93	186.4	1.26	3.89	4.07	3.94	5.68	5.35	2.65	6.15
Rougher Tails	1729.1	87.95	0.02	0.01	0.99	13.2	0.14	0.18	37.72	13.05	33.19	8.78	6.83	6.59
Calculated Head	1966.1	100.00	0.05	0.07	2.62	132.3	1.80	2.40	100.00	100.00	100.00	100.00	100.00	100.00
ERD Head	2000.0	100.00	0.04	0.06	2.28	126.7	2.24	2.42	-	-	-	-	-	-
Call Factor	98.3	-	107.6	106.4	114.9	104.4	80.6	99.2	-	-	-	-	-	-

Combined Products:

Rougher Conc 1	86.4	4.39	0.53	1.08	29.03	2174	31.20	38.20	49.92	70.39	48.61	72.18	75.99	69.87
Rougher Conc 1-2	162.4	8.26	0.33	0.68	19.41	1376	19.77	25.37	58.21	83.01	61.12	85.88	90.52	87.26
Rougher Conc 1-3	237.0	12.05	0.24	0.49	14.54	1001	13.94	18.61	62.28	86.95	66.81	91.22	93.17	93.41

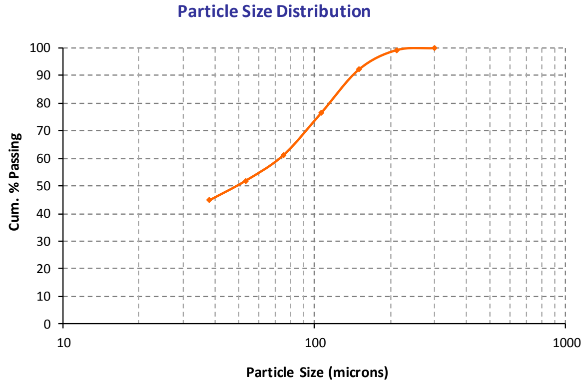
Size Distribution Determination Worksheet

Sample Tracking	
Sample ID:	HG-F1 Ro Tails
Project No.:	PJ124
Project Name:	Almaden
Date:	Oct. 24, 2012
Technician:	PD
Objective:	Confirm Grind at 16min (Target p80=106microns)

Screen Size (µm)	Sample Dry Wt (g)	Weight (%)	Cum. Weight (%)	Cum. Weight (%) Passing
300	**0.1**	0.03	0.03	99.97
212	**2.2**	0.72	0.75	99.25
150	**21.1**	7.04	7.79	92.21
106	**47.5**	15.82	23.61	76.39
75	**45.8**	15.25	38.86	61.14
53	**27.9**	9.29	48.15	51.85
38	**21.3**	7.09	55.24	44.76
-38 pan	**3.5**	1.18		
-38 Total	134.3	44.76	100.00	0.00
Total	300.0	100.00		

Mass Accountability	
Start Mass	**300.7**
+38µm wet screen	**169.6**
-38µm wet screen	**130.8**
Mass Rec. (%)	99.80

p 80 116 µm



Test Description:

Test #:	High Grade F-2
Project #:	PJ124 - Almaden Ixtaca
Operator:	Marjorie Colebrook
Date:	November 15th, 2012
Purpose:	Baseline Bulk Flotation - finer grind
Procedure:	Natural pH, 300g/t CuSO4 and SIPX
Feed:	2kg of minus 1.7 mm High Grade Met Sample
Grind:	p80 = 88 microns. 2000g @ 60% solids in lab rod mill
Comments:	

Flotation Schedule:

	Reagents (g/tonne)				Reagents (ml or g)				Time, minutes				
Stage	**$CuSO_4$**	**SIPX**	**3418A**	**F-140**	**CuSO4**	**SIPX**	**3418A**	**F-140**	**Grind**	**Cond.**	**Froth**	**pH**	**Ep**
Primary Grind	300				30				25.50				
Rougher 1		100	15	11.5		20	15	0.023		1	2	8.0	-73.3
Rougher 2		50	15	11.5		10	15	0.023		1	4	8.1	-77.1
Rougher 3		50	15	11.5		10	15	0.023		1	5	8.2	-80.5
Total	**300**	**200**	**45**	**34.5**	**30**	**40**	**45.0**	**0.069**	**25.50**	**3**	**11**		

Stage	**Rougher**
Flotation Cell	4 litre cell
Speed: rpm	1400

Observations:	viscous - increased speed to 1400 rpm

Product	**Weight g**
Conc 1	**59.5**
Conc 2	**89.0**
Con 3	**43.7**
Rougher Tail	**1793.0**

Reagents:

CuSO4	2.0	%
SIPX	1	%
3418A	0.2	%
F-140	100	%
Charge	2000	g

Prep/Assay Instructions:	**Assay for Pb, Zn, Fe, Ag by AA at BCA. Assay Au by FA at BCA. Assay S by Leco at SGS** **Do not pulverise** **PSD on Tails**

Mass Balance:

Product	**Weight**		**Assays, %, g/t**						**% Distribution**					
	g	**%**	**Pb**	**Zn**	**Fe**	**Ag**	**Au**	**S**	**Pb**	**Zn**	**Fe**	**Ag**	**Au**	**S**
Rougher Conc 1	**59.5**	2.99	**0.45**	**0.96**	**22.77**	**1657.8**	**35.05**	**30.00**	42.19	51.40	27.29	37.71	48.21	36.85
Rougher Conc 2	**89.0**	4.48	**0.28**	**0.55**	**19.51**	**1356.2**	**18.71**	**27.50**	39.49	43.71	34.99	46.17	38.51	50.54
Rougher Conc 3	**43.7**	2.20	**0.06**	**0.08**	**4.86**	**364.6**	**4.92**	**5.76**	4.31	3.27	4.29	6.10	4.98	5.20
Rougher Tails	**1793.0**	90.32	**0.01**	**0.00**	**0.93**	**14.6**	**0.20**	**0.20**	14.01	1.62	33.44	10.02	8.30	7.41
Calculated Head	**1985.2**	**100.00**	**0.03**	**0.06**	**2.50**	**131.6**	**2.18**	**2.44**	**100.00**	**100.00**	**100.00**	**100.00**	**100.00**	**100.00**
ERD Head	2000.0	100.00	0.04	0.06	2.28	126.7	2.24	2.42	-	-	-	-	-	-
Call Factor	99.3	-	74.4	88.2	109.4	103.9	97.3	100.8	-	-	-	-	-	-

Combined Products:

Rougher Conc 1	**59.5**	2.99	0.45	0.96	22.77	1658	35.05	30.00	42.19	51.40	27.29	37.71	48.21	36.85
Rougher Conc 1-2	**148.4**	7.48	0.35	0.71	20.82	1477	25.26	28.50	81.68	95.11	62.28	83.88	86.72	87.39
Rougher Conc 1-3	**192.1**	9.68	0.29	0.57	17.19	1224	20.63	23.33	85.99	98.38	66.56	89.98	91.70	92.59

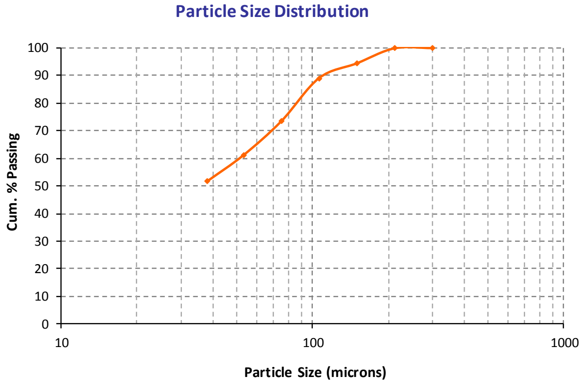
Size Distribution Determination Worksheet

Sample Tracking	
Sample ID:	HG-F2 Ro Tails
Project No.:	PJ124
Project Name:	Almaden
Date:	November 20th, 2012
Technician:	LH
Objective:	Confirm Grind at 25min

Screen Size (µm)	Sample Dry Wt (g)	Weight (%)	Cum. Weight (%)	Cum. Weight (%) Passing
300	**0.1**	0.03	0.03	99.97
212	**0.2**	0.06	0.09	99.91
150	**13.2**	5.44	5.54	94.46
106	**13.0**	5.37	10.91	89.09
75	**37.8**	15.63	26.54	73.46
53	**29.8**	12.32	38.86	61.14
38	**22.9**	9.46	48.32	51.68
-38 pan	**3.8**	1.56		
-38 Total	124.8	51.68	100.00	0.00
Total	241.5	100.00		

Mass Accountability	
Start Mass	**230.7**
+38µm wet screen	**109.7**
-38µm wet screen	**121.0**
Mass Rec. (%)	104.69

p 80	88 µm



Test Description:

Test #:	High Grade F-3
Project #:	PJ124 - Almaden Ixtaca
Operator:	Marjorie Colebrook
Date:	November 15th, 2012
Purpose:	Baseline Bulk Flotation - coarser grind
Procedure:	Natural pH, 300g/t CuSO4 and SIPX
Feed:	2kg of minus 1.7 mm High Grade Met Sample
Grind:	p80 = 313 microns. 2000g @ 60% solids in lab rod mill
Comments:	

Flotation Schedule:

	Reagents (g/tonne)				Reagents (ml or g)				Time, minutes				
Stage	$CuSO_4$	SIPX	3418A	F-140	CuSO4	SIPX	3418A	F-140	Grind	Cond.	Froth	pH	Ep
Primary Grind	300				30				10.00				
Rougher 1		100	15	11.5		20	15	0.023		1	2	7.7	-55.9
Rougher 2		50	15	11.5		10	15	0.023		1	4	8.0	-68.5
Rougher 3		50	15	11.5		10	15	0.023		1	5	8.0	-73.6
Total	300	150	45	34.5	30	40	45	0.069	10.00	3	11		

Stage	Rougher
Flotation Cell	4 litre cell
Speed: rpm	1400

Observations:	viscous - increased speed to 1400 rpm

Product	Weight g
Conc 1	31.4
Conc 2	43.5
Con 3	46.4
Rougher Tail	1850.5

Reagents:

CuSO4	2.0	%
SIPX	1	%
3418A	0.2	%
F-140	100	%
Charge	2000	g

Prep/Assay Instructions:	**Assay for Pb, Zn, Fe, Ag by AA at BCA. Assay Au by FA at BCA. Assay S by Leco at SGS** **Do not pulverise** **PSD on Tails**

Mass Balance:

Product	Weight		Assays, %, g/t						% Distribution					
	g	%	Pb	Zn	Fe	Ag	Au	S	Pb	Zn	Fe	Ag	Au	S
Rougher Conc 1	31.4	1.59	0.75	1.42	30.99	3389.4	51.18	39.50	40.27	43.76	20.92	41.14	34.94	26.64
Rougher Conc 2	43.5	2.21	0.42	0.71	23.30	1571.0	26.73	29.50	30.85	30.08	21.81	26.44	25.30	27.58
Rougher Conc 3	46.4	2.35	0.13	0.18	11.93	458.2	6.30	16.80	9.91	8.05	11.91	8.23	6.36	16.76
Rougher Tails	1850.5	93.85	0.01	0.01	1.14	33.8	0.83	0.73	18.97	18.12	45.36	24.19	33.40	29.02
Calculated Head	1971.8	100.00	0.03	0.05	2.36	131.1	2.33	2.36	100.00	100.00	100.00	100.00	100.00	100.00
ERD Head	2000.0	100.00	0.04	0.06	2.28	126.7	2.24	2.42	-	-	-	-	-	-
Call Factor	98.6	-	68.5	81.8	103.3	103.5	104.3	97.5	-	-	-	-	-	-

Combined Products:

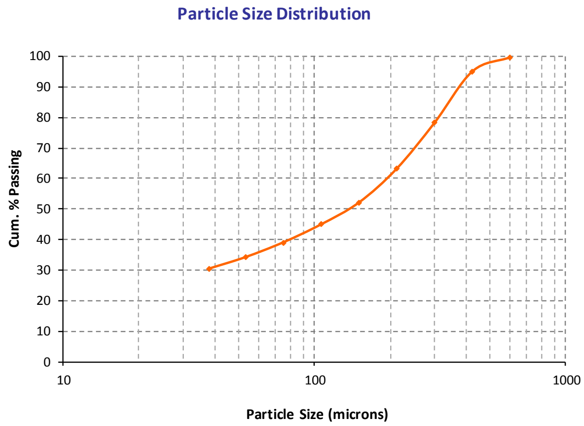
Rougher Conc 1	31.4	1.59	0.75	1.42	30.99	3389	51.18	39.50	40.27	43.76	20.92	41.14	34.94	26.64
Rougher Conc 1-2	74.9	3.80	0.56	1.01	26.52	2333	36.98	33.69	71.12	73.84	42.72	67.58	60.24	54.22
Rougher Conc 1-3	121.3	6.15	0.39	0.69	20.94	1616	25.24	27.23	81.03	81.88	54.64	75.81	66.60	70.98

Size Distribution Determination Worksheet

Sample Tracking	
Sample ID:	HG-F3 Ro Tails
Project No.:	PJ124
Project Name:	Almaden
Date:	November 20th, 2012
Technician:	LH
Objective:	Confirm Grind at 10min

p 80	313 µm

Screen Size (µm)	Sample Dry Wt (g)	Weight (%)	Cum. Weight (%)	Cum. Weight (%) Passing
600	**0.8**	0.30	0.30	99.70
425	**12.3**	4.58	4.88	95.12
300	**45.2**	16.87	21.76	78.24
212	**40.3**	15.04	36.80	63.20
150	**29.6**	11.06	47.86	52.14
106	**19.2**	7.15	55.01	44.99
75	**15.7**	5.87	60.88	39.12
53	**13.0**	4.84	65.73	34.27
38	**10.0**	3.75	69.47	30.53
-38 pan	**1.7**	0.63		
-38 Total	81.7	30.53	100.00	0.00
Total	**267.7**	**100.00**		



Mass Accountability	
Start Mass	**268.0**
+38µm wet screen	**187.9**
-38µm wet screen	**80.0**
Mass Rec. (%)	99.91

Test Description:

Test #:	High Grade F-4
Project #:	PJ124 - Almaden Ixtaca
Operator:	Marjorie Colebrook
Date:	26-Oct-12
Purpose:	Generate Rougher Conc For Cyanidation Test
Procedure:	Natural pH, 300g/t CuSO4 and SIPX
Feed:	2kg of minus 1.7 mm High Grade Met Sample
Grind:	p80 = 116 microns. 2000g @ 60% solids in lab rod mill
Comments:	Subsample Conc for Assay - agitate and siphon method

Flotation Schedule:

	Reagents (g/tonne)				Reagents (ml or g)				Time, minutes				
Stage	$CuSO_4$	SIPX	3418A	F-140	CuSO4	SIPX	3418A	F-140	Grind	Cond.	Froth	pH	Ep
Primary Grind	300				60				16.00				
Rougher 1		100	15	23.0		20	15	0.046		1	2	7.9	-58.6
Rougher 2		50	15	11.5		10	15	0.023		1	4	8.0	-65.6
Rougher 3		50	15			10	15			1	5	8.1	-70.2
Total	300	200	45	34.5	60	40	45	0.069	16.00	3	11		

Stage	Rougher
Flotation Cell	4 litre cell
Speed: rpm	1400

Observations:	288 wet weight w/ paper

Product	Weight g
Combined Conc	**213.1**
Conc Subsample	**10.0**
Rougher Tail	**1735.5**

Reagents:

CuSO4	1.0	%
SIPX	1	%
3418A	0.2	%
F-140	100	%
Charge	2000	g

Prep/Assay Instructions:	**Leach Entire Rougher Conc as per separate CN worksheet** **Submit Pulversied Rougher Tails and Conc Subsample for Au (FA) and Ag (AA) assay at BCR**

Mass Balance:

Product	Weight		Assays, %, g/t		% Distribution	
	g	%	Ag	Au	Ag	Au
Rougher Conc	**223.1**	11.39	**920.2**	**18.90**	88.75	92.04
Rougher Tails	**1735.5**	88.61	**15.0**	**0.21**	11.25	7.96
Calculated Head	1958.6	100.00	118.1	2.34	100.00	100.00
ERD Head	2000.0	100.00	126.7	2.24	-	-
Call Factor	97.9	-	93.3	104.6	-	-

Test Description:

Test #:	Dyke F-1
Project #:	PJ124 - Almaden Ixtaca
Operator:	Marjorie Colebrook
Date:	December 13th, 2012
Purpose:	Baseline Bulk Flotation
Procedure:	Natural pH, 300g/t CuSO4 and SIPX
Feed:	2kg of minus 1.7 mm Dyke
Grind:	p80 = 154 microns. 2000g @ 60% solids in lab rod mill
Comments:	

Flotation Schedule:

Stage	Reagents (g/tonne) $CuSO_4$	SIPX	3418A	F-140	Reagents (ml or g) CuSO4	SIPX	3418A	F-140	Time, minutes Grind	Cond.	Froth	pH	Ep
Primary Grind	300				30				16.00				
Rougher 1		100	15	11.5		20	3	0.023		1	2	7.7	-49.5
Rougher 2		50	15			10	3			1	4	7.8	-52.4
Rougher 3		50	15			10	3			1	5	7.9	-59
Total	300	200	45	11.5		40	9.0	0.0	16.00	3	11		

Stage	Rougher
Flotation Cell	4 litre cell
Speed: rpm	1400

Observations:

Product	Weight g
Conc 1	128.8
Conc 2	87.1
Con 3	45.5
Rougher Tail	1723.2

Reagents:

CuSO4	2.0	%
SIPX	1	%
3418A	1	%
F-140	100	%
Charge	2000	g

Prep/Assay Instructions: Assay for Pb, Zn, Fe, Ag by AA at BCA. Assay Au by FA at BCA. Assay S by Leco at SGS
Do not pulverise
PSD on Tails

Mass Balance:

Product	Weight g	%	Assays, %, g/t Pb	Zn	Fe	Ag	Au	S	% Distribution Pb	Zn	Fe	Ag	Au	S
Rougher Conc 1	128.8	6.49	0.14	0.29	31.83	428.9	7.56	37.40	56.59	58.18	49.21	60.74	71.66	72.32
Rougher Conc 2	87.1	4.39	0.03	0.08	10.38	247.8	1.92	11.50	9.16	10.17	10.85	23.73	12.33	15.04
Rougher Conc 3	45.5	2.29	0.02	0.03	5.13	75.6	1.60	3.35	2.25	2.13	2.80	3.78	5.35	2.29
Rougher Tails	1723.2	86.83	0.01	0.01	1.80	6.2	0.08	0.40	31.99	29.52	37.13	11.75	10.66	10.35
Calculated Head	1984.6	100.00	0.02	0.03	4.20	45.8	0.68	3.36	100.00	100.00	100.00	100.00	100.00	100.00
ERD Head	2000.0	100.00	0.02	0.03	3.60	37.5	0.68	3.22	-	-	-	-	-	-
Call Factor	99.2	-	74.0	107.8	116.5	122.2	101.4	104.2	-	-	-	-	-	-

Combined Products:

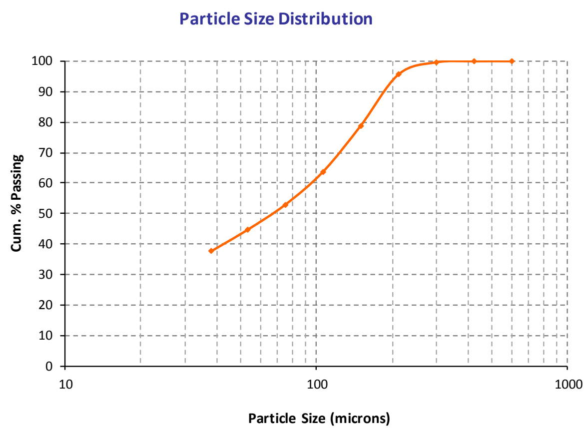
Rougher Conc 1	128.8	6.49	0.14	0.29	31.83	429	7.56	37.40	56.59	58.18	49.21	60.74	71.66	72.32
Rougher Conc 1-2	215.9	10.88	0.10	0.20	23.18	356	5.28	26.95	65.76	68.35	60.07	84.47	83.99	87.36
Rougher Conc 1-3	261.4	13.17	0.08	0.17	20.03	307	4.64	22.84	68.01	70.48	62.87	88.25	89.34	89.65

Size Distribution Determination Worksheet

Sample Tracking	
Sample ID:	Dyke F-1 Rotail
Project No.:	PJ124
Project Name:	Almaden
Date:	21-Dec-12
Technician:	Leena Heikkila
Objective:	Confirm of grind at 16 minutes

p 80 154 µm

Screen Size (µm)	Sample Dry Wt (g)	Weight (%)	Cum. Weight (%)	Cum. Weight (%) Passing
600	**0.0**	0.00	0.00	100.00
425	**0.0**	0.00	0.00	100.00
300	**0.9**	0.35	0.35	99.65
212	**10.3**	3.96	4.31	95.69
150	**43.7**	16.80	21.11	78.89
106	**39.5**	15.19	36.29	63.71
75	**28.1**	10.80	47.10	52.90
53	**21.5**	8.27	55.36	44.64
38	**18.3**	7.04	62.40	37.60
-38 pan	**1.6**	0.62		
-38 Total	97.8	37.60	100.00	0.00
Total	**260.1**	**100.00**		



Mass Accountability	
Start Mass	**260.8**
+38µm wet screen	**164.9**
-38µm wet screen	**96.2**
Mass Rec. (%)	99.73

Test Description:

Test #:	Dyke F-2
Project #:	PJ124 - Almaden Ixtaca
Operator:	Marjorie Colebrook
Date:	
Purpose:	Baseline Bulk Flotation
Procedure:	Natural pH, 300g/t CuSO4 and SIPX
Feed:	2kg of minus 1.7 mm Dyke
Grind:	p80 = 106 microns. 2000g @ 60% solids in lab rod mill
Comments:	0

Flotation Schedule:

Stage	Reagents (g/tonne)				Reagents (ml or g)				Time, minutes				
	$CuSO_4$	SIPX	3418A	F-140	CuSO4	SIPX	3418A	F-140	Grind	Cond.	Froth	pH	Ep
Primary Grind	300				30				21.00				
Rougher 1		100	15	11.5		20	3	0.023		1	2	7.5	-32.8
Rougher 2		50	15			10	3			1	4	7.6	-38.9
Rougher 3		50	15			10	3			1	5	7.8	-46.8
Total	300	200	45	11.5	30	40	9.0	0.0	21.00	3	11		

Stage	Rougher
Flotation Cell	4 litre cell
Speed: rpm	1400

Observations:	

Product	Weight g
Conc 1	148.9
Conc 2	119.0
Con 3	81.7
Rougher Tails Cut	89.4
Rougher Tail Cake	1543.6

Reagents:

CuSO4	2.0	%
SIPX	1	%
3418A	1	%
F-140	100	%
Charge	2000	g

Prep/Assay Instructions:	**Assay for Pb, Zn, Fe, Ag by AA at BCA. Assay Au by FA at BCA. Assay S by Leco at SGS** **Do not pulverise** **PSD on Tails**

Mass Balance:

Product	Weight		Assays, %, g/t						% Distribution					
	g	%	Pb	Zn	Fe	Ag	Au	S	Pb	Zn	Fe	Ag	Au	S
Rougher Conc 1	148.9	7.51	0.14	0.25	28.80	392.8	8.22	35.00	56.75	71.16	52.57	64.73	78.55	80.20
Rougher Conc 2	119.0	6.00	0.04	0.05	6.20	130.0	1.74	6.63	13.48	10.68	9.04	17.12	13.28	12.14
Rougher Conc 3	81.7	4.12	0.01	0.014	3.47	56.8	0.50	2.29	2.71	2.23	3.47	5.13	2.61	2.88
Rougher Tails	1633.0	82.37	0.01	0.005	1.74	7.2	0.05	0.19	27.06	15.93	34.92	13.01	5.56	4.78
Calculated Head	1982.5	100.00	0.02	0.026	4.11	45.6	0.79	3.28	100.00	100.00	100.00	100.00	100.00	100.00
ERD Head	2000.0	100.00	0.02	0.030	3.60	37.5	0.68	3.22	-	-	-	-	-	-
Call Factor	99.1	-	83.0	86.2	114.2	121.5	116.4	101.8	-	-	-	-	-	-

Combined Products:

Rougher Conc 1	148.9	7.51	0.14	0.25	28.80	393	8.22	35.00	56.75	71.16	52.57	64.73	78.55	80.20
Rougher Conc 1-2	267.9	13.51	0.09	0.16	18.76	276	5.34	22.40	70.23	81.84	61.61	81.85	91.84	92.35
Rougher Conc 1-3	349.5	17.63	0.08	0.12	15.19	225	4.21	17.70	72.94	84.07	65.08	86.99	94.44	95.22

Test Description:

Test #:	Limestone F-1
Project #:	PJ124 - Almaden Ixtaca
Operator:	Marjorie Colebrook
Date:	December 13th, 2012
Purpose:	Baseline Bulk Flotation
Procedure:	Natural pH, 300g/t CuSO4 and SIPX
Feed:	2kg of minus 1.7 mm Limestone
Grind:	p80 = 156 microns. 2000g @ 60% solids in lab rod mill
Comments:	

Flotation Schedule:

Stage	Reagents (g/tonne)				Reagents (ml or g)				Time, minutes				
	$CuSO_4$	SIPX	3418A	F-140	CuSO4	SIPX	3418A	F-140	Grind	Cond.	Froth	pH	Ep
Primary Grind	300				30				16.00				
Rougher 1		100	15	11.5		20	3	0.023		1	2	8.0	-66.4
Rougher 2		50	15	11.5		10	3	0.023		1	4	8.1	-68.6
Rougher 3		50	15	11.5		10	3	0.023		1	5	8.1	-69.3
Total	**300**	**150**	**45**	**34.5**	**30**	**30**	**9.0**	**0.1**	**16.00**	**3**	**11**		

Stage	Rougher
Flotation Cell	4 litre cell
Speed: rpm	1400

Observations:	No visible mineralization in froth

Product	Weight g
Conc 1	**11.6**
Conc 2	**30.9**
Con 3	**32.4**
Rougher Tail	**1910.9**

Reagents:

CuSO4	2.0	%
SIPX	1	%
3418A	1	%
F-140	100	%
Charge	2000	g

Prep/Assay Instructions:	**Assay for Pb, Zn, Fe, Ag by AA at BCA. Assay Au by FA at BCA. Assay S by Leco at SGS** **Do not pulverise** **PSD on Tails**

Mass Balance:

Product	Weight		Assays, %, g/t						% Distribution					
	g	%	Pb	Zn	Fe	Ag	Au	S	Pb	Zn	Fe	Ag	Au	S
Rougher Conc 1	**11.6**	0.58	**0.11**	**0.39**	**4.64**	**1508.0**	**15.93**	**5.06**	6.55	13.47	2.51	17.87	10.22	4.19
Rougher Conc 2	**30.9**	1.56	**0.12**	**0.20**	**10.03**	**1082.6**	**18.94**	**10.80**	18.69	18.72	14.46	34.18	32.36	23.83
Rougher Conc 3	**32.4**	1.63	**0.04**	**0.06**	**6.31**	**328.0**	**6.57**	**8.70**	6.75	5.39	9.54	10.86	11.77	20.13
Rougher Tails	**1910.9**	96.23	**0.01**	**0.01**	**0.82**	**19.0**	**0.43**	**0.38**	68.01	62.42	73.49	37.09	45.65	51.85
Calculated Head	**1985.8**	**100.00**	**0.01**	**0.017**	**1.08**	**49.3**	**0.91**	**0.71**	**100.00**	**100.00**	**100.00**	**100.00**	**100.00**	**100.00**
ERD Head	2000.0	100.00	0.01	0.013	0.84	43.8	0.67	0.69	-	-	-	-	-	-
Call Factor	99.3	-	198.1	130.4	128.5	112.5	136.7	102.2	-	-	-	-	-	-

Combined Products:

Rougher Conc 1	**11.6**	0.58	0.11	0.39	4.64	1508	15.93	5.06	6.55	13.47	2.51	17.87	10.22	4.19
Rougher Conc 1-2	**42.5**	2.14	0.12	0.26	8.56	1199	18.12	9.23	25.24	32.19	16.98	52.05	42.58	28.02
Rougher Conc 1-3	**74.9**	3.77	0.08	0.17	7.58	822	13.12	9.00	31.99	37.58	26.51	62.91	54.35	48.15

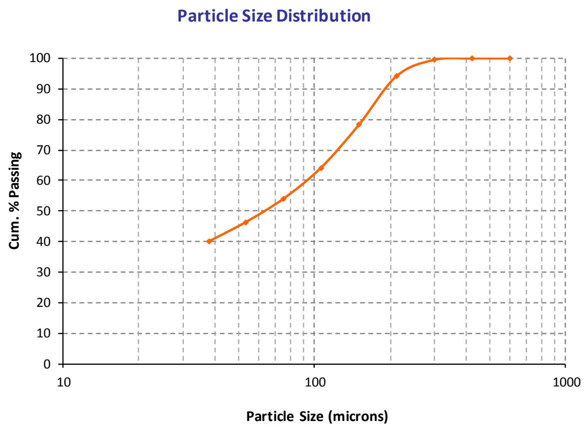
Size Distribution Determination Worksheet

Sample Tracking	
Sample ID:	Limestone F-1 Rotail
Project No.:	PJ124
Project Name:	Almaden
Date:	14-Dec-12
Technician:	Leena Heikkila
Objective:	Confirm of grind at 16 minutes

Screen Size (µm)	Sample Dry Wt (g)	Weight (%)	Cum. Weight (%)	Cum. Weight (%) Passing
600	**0.0**	0.00	0.00	100.00
425	**0.0**	0.00	0.00	100.00
300	**1.3**	0.48	0.48	99.52
212	**14.3**	5.31	5.79	94.21
150	**43.0**	15.97	21.77	78.23
106	**37.8**	14.04	35.81	64.19
75	**27.4**	10.18	45.99	54.01
53	**20.7**	7.69	53.68	46.32
38	**16.7**	6.20	59.88	40.12
-38 pan	**1.5**	0.56		
-38 Total	108.0	40.12	100.00	0.00
Total	269.2	100.00		

Mass Accountability	
Start Mass	**271.1**
+38µm wet screen	**163.4**
-38µm wet screen	**106.5**
Mass Rec. (%)	99.30

p 80 156 µm



Test Description:

Test #:	Limestone F-2
Project #:	PJ124 - Almaden Ixtaca
Operator:	Marjorie Colebrook
Date:	
Purpose:	Baseline Bulk Flotation
Procedure:	Natural pH, 300g/t CuSO4 and SIPX
Feed:	2kg of minus 1.7 mm Limestone
Grind:	p80 = 105 microns. 2000g @ 60% solids in lab rod mill
Comments:	

Flotation Schedule:

	Reagents (g/tonne)				Reagents (ml or g)				Time, minutes				
Stage	**$CuSO_4$**	**SIPX**	**3418A**	**F-140**	**CuSO4**	**SIPX**	**3418A**	**F-140**	**Grind**	**Cond.**	**Froth**	**pH**	**Ep**
Primary Grind	300				30				**21.00**				
Rougher 1		100	15	23.0		20	3	0.046		1	2	8.0	-60.8
Rougher 2		50	15	11.5		10	3	0.023		1	4	8.1	-65.6
Rougher 3		50	15	11.5		10	3	0.023		1	5	8.1	-66.6
Total	**300**	**150**	**45**	**46.0**	**30**	**30**	**9.0**	**0.1**	**21.00**	**3**	**11**		

Stage	**Rougher**
Flotation Cell	4 litre cell
Speed: rpm	1400

Observations:	No visible mineralization in froth

Product	**Weight g**
Conc 1	**27.7**
Conc 2	**40.1**
Con 3	**51.7**
Rougher Tail Cut	**110.4**
Rougher Tail Cake	**1760.6**

Reagents:

CuSO4	2.0	%
SIPX	1	%
3418A	1	%
F-140	100	%
Charge	2000	g

Prep/Assay Instructions:	**Assay for Pb, Zn, Fe, Ag by AA at BCA. Assay Au by FA at BCA. Assay S by Leco at SGS** **Do not pulverise** **PSD on Tails**

Mass Balance:

Product	**Weight**		**Assays, %, g/t**						**% Distribution**					
	g	**%**	**Pb**	**Zn**	**Fe**	**Ag**	**Au**	**S**	**Pb**	**Zn**	**Fe**	**Ag**	**Au**	**S**
Rougher Conc 1	**27.7**	1.39	**0.208**	**0.350**	**12.74**	**1852.5**	**26.93**	**18.00**	24.68	61.49	17.81	47.54	54.79	32.95
Rougher Conc 2	**40.1**	2.01	**0.068**	**0.088**	**5.30**	**501.8**	**5.99**	**9.12**	11.68	22.38	10.73	18.64	17.64	24.17
Rougher Conc 3	**51.7**	2.60	**0.034**	**0.013**	**3.21**	**134.4**	**1.14**	**3.14**	7.53	4.26	8.36	6.44	4.34	10.73
Rougher Tails	**1871.0**	94.00	**0.007**	**0.001**	**0.67**	**15.8**	**0.17**	**0.26**	56.11	11.87	63.09	27.39	23.23	32.15
Calculated Head	**1990.5**	**100.00**	**0.012**	**0.008**	**1.00**	**49.3**	**0.68**	**0.76**	**43.89**	**88.13**	**36.91**	**72.61**	**76.77**	**67.85**
ERD Head	2000.0	100.00	0.005	0.013	0.84	43.8	0.67	0.69	-	-	-	-	-	-
Call Factor	99.5	-	234.5	60.9	118.5	112.6	102.7	110.2	-	-	-	-	-	-

Combined Products:

Rougher Conc 1	**27.7**	1.39	0.21	0.35	12.74	1852	26.93	18.00	24.68	61.49	17.81	47.54	54.79	32.95
Rougher Conc 1-2	**67.8**	3.41	0.13	0.20	8.34	1054	14.54	12.75	36.36	83.87	28.54	66.18	72.43	57.12
Rougher Conc 1-3	**119.5**	6.00	0.09	0.12	6.12	656	8.75	8.59	43.89	88.13	36.91	72.61	76.77	67.85

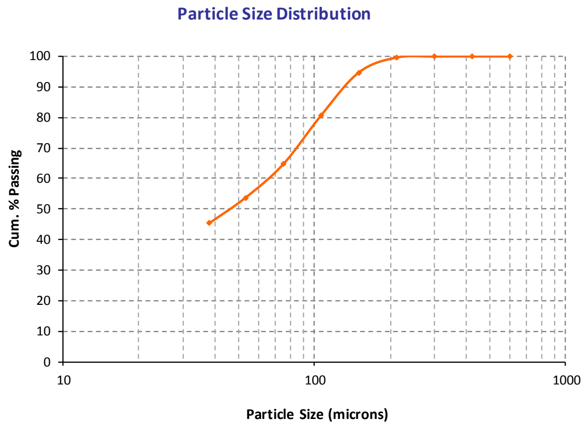
Size Distribution Determination Worksheet

Sample Tracking	
Sample ID:	Limestone F-2 Rotail
Project No.:	PJ124
Project Name:	Almaden
Date:	Jan 14th, 2013
Technician:	CB
Objective:	Confirm of grind at 21 minutes

Screen Size (µm)	Sample Dry Wt (g)	Weight (%)	Cum. Weight (%)	Cum. Weight (%) Passing
600	**0.0**	0.00	0.00	100.00
425	**0.0**	0.00	0.00	100.00
300	**0.1**	0.04	0.04	99.96
212	**0.9**	0.34	0.37	99.63
150	**13.1**	4.88	5.26	94.74
106	**37.9**	14.13	19.38	80.62
75	**42.8**	15.95	35.33	64.67
53	**29.5**	11.00	46.33	53.67
38	**22.4**	8.35	54.68	45.32
-38 pan	**5.0**	1.86		
-38 Total	121.6	45.32	100.00	0.00
Total	268.3	100.00		

Mass Accountability	
Start Mass	**270.2**
+38µm wet screen	**151.4**
-38µm wet screen	**116.6**
Mass Rec. (%)	99.30

p 80	105 µm



Test Description:

Test #:	TUFF F-1
Project #:	PJ124 - Almaden Ixtaca
Operator:	Marjorie Colebrook
Date:	December 13th, 2012
Purpose:	Baseline Bulk Flotation
Procedure:	Natural pH, 300g/t CuSO4 and SIPX
Feed:	2kg of minus 1.7 mm TUFF
Grind:	p80 = 93 microns. 2000g @ 60% solids in lab rod mill
Comments:	

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Flotation Schedule:

	Reagents (g/tonne)				Reagents (ml or g)				Time, minutes				
Stage	$CuSO_4$	SIPX	3418A	F-140	CuSO4	SIPX	3418A	F-140	Grind	Cond.	Froth	pH	Ep
Primary Grind	300				60				16.00				
Rougher 1		100	15	11.5		20	3	0.023		1	2	7.5	-36.1
Rougher 2		50	15	11.5		10	3	0.023		1	4	7.7	-45.8
Rougher 3		50	15	11.5		10	3	0.023		1	5	7.7	-48.9
Total	**300**	**150**	**45**	**34.5**	**60**	**30**	**9.0**	**0.069**	**16.00**	**3**	**11**		

Stage	Rougher
Flotation Cell	4 litre cell
Speed: rpm	1400

Observations:	Very Viscous. Has to filter some mill discharge. Very sticky to rinse.

Product	Weight g
Conc 1	**83.4**
Conc 2	**140.0**
Con 3	**144.8**
Rougher Tail	**1560.0**

Reagents:

CuSO4	1.0	%
SIPX	1	%
3418A	1	%
F-140	100	%
Charge	2000	g

Prep/Assay Instructions:	**Assay for Pb, Zn, Fe, Ag by AA at BCA. Assay Au by FA at BCA. Assay S by Leco at SGS** **Do not pulverise** **PSD on Tails**

Mass Balance:

Product	Weight		Assays, %, g/t						% Distribution					
	g	%	Pb	Zn	Fe	Ag	Au	S	Pb	Zn	Fe	Ag	Au	S
Rougher Conc 1	**83.4**	4.33	**0.009**	**0.04**	**8.87**	**56.0**	**2.91**	**9.14**	3.75	13.78	14.12	18.96	15.97	21.03
Rougher Conc 2	**140.0**	7.26	**0.009**	**0.02**	**4.03**	**28.6**	**1.46**	**4.04**	6.29	13.88	10.78	16.26	13.46	15.61
Rougher Conc 3	**144.8**	7.51	**0.006**	**0.02**	**4.02**	**24.0**	**1.39**	**3.47**	4.34	14.35	11.11	14.11	13.20	13.86
Rougher Tails	**1560.0**	80.90	**0.011**	**0.01**	**2.15**	**8.0**	**0.56**	**1.15**	85.63	57.99	64.00	50.67	57.37	49.50
Calculated Head	**1928.2**	**100.00**	**0.01**	**0.01**	**2.72**	**12.8**	**0.79**	**1.88**	**100.00**	**100.00**	**100.00**	**100.00**	**100.00**	**100.00**
ERD Head	2000.0	100.00	0.01	0.01	2.54	11.8	0.78	1.95	-	-	-	-	-	-
Call Factor	96.4	-	79.9	125.6	107.2	108.2	100.9	96.4	-	-	-	-	-	-

Combined Products:

Rougher Conc 1	**83.4**	4.33	0.01	0.04	8.87	56	2.91	9.14	3.75	13.78	14.12	18.96	15.97	21.03
Rougher Conc 1-2	**223.4**	11.59	0.01	0.03	5.84	39	2.00	5.94	10.03	27.66	24.89	35.22	29.43	36.64
Rougher Conc 1-3	**368.2**	19.10	0.01	0.03	5.12	33	1.76	4.97	14.37	42.01	36.00	49.33	42.63	50.50

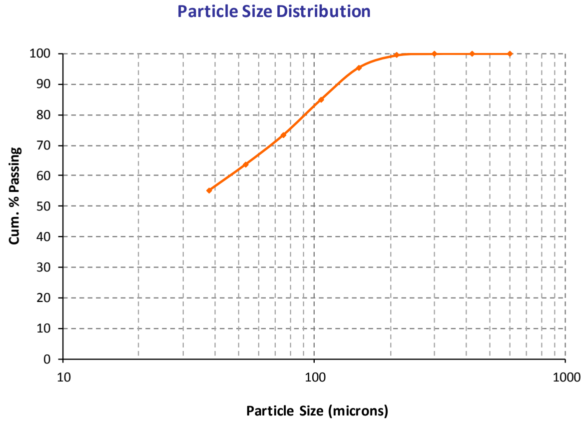
Size Distribution Determination Worksheet

Sample Tracking	
Sample ID:	TUFF F-1 Rotail
Project No.:	PJ124
Project Name:	Almaden
Date:	14-Dec-12
Technician:	Leena Heikkila
Objective:	Confirm grind at 16 minutes

Screen Size (µm)	Sample Dry Wt (g)	Weight (%)	Cum. Weight (%)	Cum. Weight (%) Passing
600	**0.0**	0.00	0.00	100.00
425	**0.0**	0.00	0.00	100.00
300	**0.1**	0.04	0.04	99.96
212	**1.3**	0.51	0.55	99.45
150	**10.2**	3.98	4.52	95.48
106	**26.9**	10.49	15.02	84.98
75	**29.8**	11.62	26.64	73.36
53	**24.8**	9.67	36.31	63.69
38	**21.7**	8.46	44.77	55.23
-38 pan	**2.3**	0.90		
-38 Total	141.6	55.23	100.00	0.00
Total	256.4	100.00		

Mass Accountability	
Start Mass	**259.1**
+38µm wet screen	**118.1**
-38µm wet screen	**139.3**
Mass Rec. (%)	98.96

p 80	93 µm



Test Description:

Test #:	TUFF F-2
Project #:	PJ124 - Almaden Ixtaca
Operator:	Marjorie Colebrook
Date:	09-Jan-13
Purpose:	Baseline Bulk Flotation - Lower % Solids
Procedure:	Natural pH, 300g/t CuSO4 and SIPX
Feed:	2kg of minus 1.7 mm TUFF
Grind:	p80 = 98 microns. 2000g @ 60% solids in lab rod mill
Comments:	

Flotation Schedule:

Stage	Reagents (g/tonne)				Reagents (ml or g)				Time, minutes				
	$CuSO_4$	SIPX	3418A	F-140	CuSO4	SIPX	3418A	F-140	Grind	Cond.	Froth	pH	Ep
Primary Grind	300				60				15.00				
Rougher 1		100	15	23.0		20	3	0.046		1	2	7.3	-13
Rougher 2		50	15			10	3			1	4	7.4	-20.9
Rougher 3		50	15	11.5		10	3	0.023		1	5	7.5	-24.5
Total	300	150	45	34.5	60	30	9.0	0.069	15.00	3	11		

Stage	Rougher
Flotation Cell	8 litre cell
Speed: rpm	1000

Observations:	

Product	Weight g
Conc 1	57.3
Conc 2	63.8
Con 3	85.3
Rougher Tail	1742.4

Reagents:

CuSO4	1.0	%
SIPX	1	%
3418A	1	%
F-140	100	%
Charge	2000	g

Prep/Assay Instructions:	**Assay for Pb, Zn, Fe, Ag by AA at BCA. Assay Au by FA at BCA. Assay S by Leco at SGS** **Do not pulverise**

Mass Balance:

Product	Weight		Assays, %, g/t						% Distribution					
	g	%	Pb	Zn	Fe	Ag	Au	S	Pb	Zn	Fe	Ag	Au	S
Rougher Conc 1	57.3	2.94	0.02	0.080	22.88	178.4	7.69	26.80	3.31	51.02	28.64	39.97	29.88	41.54
Rougher Conc 2	63.8	3.27	0.01	0.027	8.61	59.8	2.98	10.40	1.93	19.16	11.99	14.91	12.89	17.94
Rougher Conc 3	85.3	4.38	0.02	0.011	4.89	25.0	1.65	4.90	3.75	10.44	9.10	8.34	9.53	11.31
Rougher Tails	1742.4	89.41	0.02	0.001	1.32	5.4	0.40	0.62	91.01	19.38	50.26	36.78	47.70	29.21
Calculated Head	1948.8	100.00	0.02	0.005	2.35	13.1	0.76	1.90	100.00	100.00	100.00	100.00	100.00	100.00
ERD Head	2000.0	100.00	0.01	0.010	2.54	11.8	0.78	1.95	-	-	-	-	-	-
Call Factor	97.4	-	143.6	46.1	92.7	111.2	97.0	97.3	-	-	-	-	-	-

Combined Products:

Rougher Conc 1	57.3	2.94	0.02	0.08	22.88	178	7.69	26.80	3.31	51.02	28.64	39.97	29.88	41.54
Rougher Conc 1-2	121.1	6.21	0.02	0.05	15.36	116	5.21	18.16	5.24	70.17	40.63	54.88	42.77	59.48
Rougher Conc 1-3	206.4	10.59	0.02	0.04	11.03	78	3.74	12.68	8.99	80.62	49.74	63.22	52.30	70.79

Size Distribution Determination Worksheet

Sample Tracking	
Sample ID:	TUFF F-2 Rotail
Project No.:	PJ124
Project Name:	Almaden
Date:	Jan 25/13
Technician:	CM
Objective:	Confirm grind at 21 minutes

Screen Size (µm)	Sample Dry Wt (g)	Weight (%)	Cum. Weight (%)	Cum. Weight (%) Passing
600	**0.0**	0.00	0.00	100.00
425	**0.0**	0.00	0.00	100.00
300	**0.5**	0.23	0.23	99.77
212	**1.6**	0.81	1.03	98.97
150	**10.1**	5.08	6.11	93.89
106	**21.4**	10.79	16.90	83.10
75	**23.9**	12.03	28.93	71.07
53	**17.0**	8.54	37.47	62.53
38	**15.3**	7.70	45.17	54.83
-38 pan	**4.0**	2.03		
-38 Total	109.0	54.83	100.00	0.00
Total	198.7	100.00		

Mass Accountability	
Start Mass	**198.8**
+38µm wet screen	**93.9**
-38µm wet screen	**104.9**
Mass Rec. (%)	99.94

p 80 98 µm

Particle Size Distribution