NI 43-101
TECHNICAL REPORT

PRELIMINARY ECONOMIC ASSESSMENT
OF THE
IXTACA PROJECT

Puebla State, Mexico
618,800E and 2,176,100N
(NAD83 Zone 14)

Submitted to:
Almaden Minerals Ltd.

Effective Date: 09 October 2014
Updated: 06 November 2015

Qualified Persons	**Company**
Jesse Aarsen, P.Eng.	Moose Mountain Technical Services
Kristopher Raffle, P.Geo.	Apex Geoscience Ltd
G.H. Giroux, P.Eng.,	Giroux Consultants Ltd
Tracey Meintjes, P.Eng.	Moose Mountain Technical Services
Ken Embree, P.Eng.	Knight Piésold Ltd

DATE & SIGNATURE PAGE

I, **Jesse J. Aarsen, B.Sc. Mining Engineering, P.Eng.**, of Penticton B.C. certify that I have overseen the assembly of this Technical Report titled "***Preliminary Economic Assessment of the Ixtaca Project***" dated 09 October 2014, and updated on 06 November 2015. I have relied on the expert opinions of the Qualified Persons listed in the report for areas outside of my relevant experience. This report fairly and accurately represents the information that has been made available to myself as of the date of the report and complies with the National Instrument 43-101 standards.

"ORIGINAL SIGNED AND SEALED"

Jesse J. Aarsen
P.Eng.

Dated the 6th day of November, 2015.

CERTIFICATE & DATE PAGES

I, Jesse J. Aarsen, B.Sc. Mining Engineering, P.Eng., of Penticton B.C. do hereby certify that:

1. I am an Associate (Mining Engineer) with Moose Mountain Technical Services with a business address of 1975-1st Avenue South, Cranbrook BC, V1C 6Y3.
2. This certificate applies to the technical report entitled "Preliminary Economic Assessment of the Ixtaca Project" dated 09 October 2014, updated on 06 November 2015 (the "Technical Report")
3. I graduated with a Bachelor of Science degree in Mining Engineering Co-op from the University of Alberta in April 2002.
4. I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#38709) and the Association of Professional Engineers and Geoscientists of Alberta (#74969).
5. I have worked as a mining engineer for a total of 11 years since my graduation from university. I have also taken a 2 year period for personal travel throughout the world. My relevant experience for the purpose of the Technical Report includes:
   - 2002 to 2005 – employed at complex coal mine in the Elk Valley working as a short range, long range, dispatch, and pit engineer. Preparation of budget levels mine plans and cost inputs, oversaw operation of personal designs and acting in supervisory-role positions as needed.
   - Since 2007 – Consulting mining engineer specializing in mine planning and project development. Completion of mine plans for complex coal operating mines in north-eastern British Columbia and an open-pit copper/molybdenum mine in central British Columbia. Supervisory role in large multi-disciplinary studies for projects in both coal and hard-rock settings in Canada and Mongolia. Responsible for building several coal geology and block models and calculation of mineral resources under the supervision of a P.Geo.
6. I have read the definition of "qualified person" set out in National Instrument 43-101 ("the Instrument") and certify that by reason of my education, affiliation with a professional associations and past relevant work experience, I am a "Qualified Person" for the purposes of the Instrument.
7. I have visited the site on April 30-May 01, 2013 and again on August 27-28, 2014.
8. I have prepared and am responsible for the mining, infrastructure and economic components of Chapter 1; as well as Chapters 15, 16, 18, 21, 22, 25, and 26 of the Technical Report.
9. I am independent of Almaden Minerals applying the tests in Section 1.5 of the Instrument.
10. I have been involved with the Ixtaca Project during the preparation of the Preliminary Economic Assessment and the Technical Report that is based on the Preliminary Economic Assessment.
11. I have read the Instrument, and the Technical Report has been prepared in compliance with the Instrument.
12. 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.

Dated the 6th day of November 2015.

"ORIGINAL SIGNED AND SEALED"

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Signature of Qualified Person

Jesse J. Aarsen, B.Sc., P.Eng.

I, Kristopher J. Raffle, B.Sc., P.Geo., of Vancouver B.C. do hereby certify that:

1. I am a Principal (Geologist) of APEX Geoscience Ltd. with a business address 200-9797, 45 Avenue, Edmonton, Alberta, Canada T6E-5V8.
2. This certificate applies to the technical report entitled "Preliminary Economic Assessment of the Ixtaca Project" dated 09 October 2014, updated on 06 November 2015 (the "Technical Report").
3. I graduated with a Bachelor of Science degree in Geology (Honours) from the University of British Columbia in 2000.
4. I am a member of the Association of Professional Engineers and Geoscientists of British Columbia (#31400).
5. I have worked as an exploration geologist for a total of 14 years since my graduation from university. My relevant experience for the purpose of the Technical Report includes:
    - I have supervised numerous exploration programs specific to low sulphidation epithermal gold-silver deposits having similar geologic characteristics to the Tuligtic Property throughout British Columbia, Canada; and Jalisco, Nayarit and Puebla States, Mexico.
    - I have authored two previous Technical Reports with respect to the Tuligtic Property dated March 13, 2013 and February 12, 2014.
    - During 2013 and 2014, I supervised the compilation of surface geological, geochemical, and geophysical and data for the Tuligtic Property, and conducted a review and audit of Almaden's drill hole and QA/QC databases.
6. I have read the definition of "qualified person" set out in National Instrument 43-101 ("the Instrument") and certify that by reason of my education, affiliation with a professional associations and past relevant work experience, I am a "Qualified Person" for the purposes of the Instrument.
7. I have visited the site on three (3) separate occasions: October 17-20, 2011; September 23, 2012 and most recently on November 20, 2013.
8. I have prepared and am responsible for Chapters 2 through 12, 23, 24 and 27; including relevant portions of Chapters 1 and 26 of the Technical Report.
9. I am independent of Almaden Minerals applying the tests in Section 1.5 of the Instrument.
10. I have had no previous involvement with the Property that is the subject of the Technical Report than that which is stated in 5 and 7 above.
11. I have read the Instrument, and the Technical Report has been prepared in compliance with the Instrument.
12. 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.

Dated the 6th day of November 2015.

"ORIGINAL SIGNED AND SEALED"

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Signature of Qualified Person
Kristopher J. Raffle, B.Sc., P.Geo.

I, G.H. Giroux, P.Eng. MASc, of Vancouver B.C., do hereby certify that:

1. I, G.H. Giroux, of 982 Broadview Drive, North Vancouver, British Columbia, do hereby certify that:
2. I am a consulting geological engineer with an office at #1215 - 675 West Hastings Street, Vancouver, British Columbia.
3. 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.
4. I am a member in good standing of the Association of Professional Engineers and Geoscientists of the Province of British Columbia.
5. 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.
6. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, 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.
7. I am responsible for the preparation of Section 14 and relevant portions of Chapters 1 and 26 of the Technical Report titled "Preliminary Economic Assessment of the Ixtaca Project" dated 09 October, 2014, updated on 06 November 2015 (the "Technical Report").
8. I have not visited the Property.
9. I have completed a previous resource estimate on the Property that is the subject of the Technical Report in 2013.
10. 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.
11. I am independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.
12. 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 the 6th day of November 2015.

"ORIGINAL SIGNED AND SEALED"

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Signature of Qualified Person
G. H. Giroux, P.Eng., MASc.

I, Tracey Meintjes, P.Eng., of Vancouver B.C. do hereby certify that:

1. I am a Metallurgical Engineer with Moose Mountain Technical Services with a business address at 1975 1st Avenue South, Cranbrook, BC, V1C 6Y3.
2. This certificate applies to the technical report entitled "Preliminary Economic Assessment of The Ixtaca Project" dated 09 October 2014, updated on 06 November 2015 (the "Technical Report").
3. I am a graduate of the Technikon Witwatersrand, (NHD Extraction Metallurgy – 1996)
4. I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#37018).
5. My relevant experience includes process engineering, operation, and supervision, and mine engineering in South Africa and North America. I have been working in my profession continuously since 1996.
6. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
7. I visited the Property from 01 July 2014 to 02 July 2014.
8. I am responsible for Sections 13, 17, and 19; including metallurgical and processing portions of Chapters 1 and 26 of the Technical Report.
9. I am independent of Almaden Minerals as defined by Section 1.5 of the Instrument.
10. I have had no previous involvement with the Property that is the subject of the Technical Report.
11. I have read the Instrument and the Technical Report has been prepared in compliance with the Instrument.
12. 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.

Dated the 6th day of November 2015.

"ORIGINAL SIGNED AND SEALED"

Signature of Qualified Person
Tracey D. Meintjes, P.Eng.

I, Ken Embree, P.Eng., of Vancouver B.C. do hereby certify that:

1. This certificate applies to the technical report titled “Preliminary Economic Assessment of the Ixtaca Project”, with an effective date 09 October 2014, updated on 06 November 2015 prepared for Almaden Minerals Ltd.;
2. I am currently employed as Managing Principal with Knight Piésold Ltd. with an office at 1400 – 750 West Pender St, Vancouver, BC Canada;
3. I am a graduate of the University of Saskatchewan with a Degree (B.A.Sc.) in Geological Engineering, 1986. I have practiced my profession continuously since 1986;
4. I am a Professional Engineer (17,439) with the Association of Professional Engineers and Geoscientists of British Columbia;
5. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the National Instrument 43-101;
6. I visited the Ixtaca Project site on March 10 to 12, 2015;
7. I am responsible for and/or shared responsibility for Section numbers 16.7, 18.5 and 20; and including relevant portions of Chapters 1 and 26 of the Technical Report.
8. I have not had prior involvement with the Property that is the subject of the Preliminary Economic Assessment;
9. As of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading;
10. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated the 6th day of November 2015.

“ORIGINAL SIGNED AND SEALED”

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Signature of Qualified Person
Ken Embree, P.Eng.

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1.0 SUMMARY

1.1 Introduction

This Preliminary Economic Assessment ("PEA") Technical Report is written for the Ixtaca Project (the "Project") as an update to the previously issued PEA ("maiden PEA" dated 13 May 2014), and has been prepared by Moose Mountain Technical Services ("MMTS") in conjunction with APEX Geoscience Ltd., Giroux Consultants Ltd, ("GCL") and Knight Piésold Ltd. ("KP"). The Ixtaca Project encompasses the Ixtaca Zone Deposit (Ixtaca Gold-Silver Deposit) that includes the Ixtaca Main, North, and Chemalaco Zones of the Tuligtic Property.

All currency amounts are referred to in U.S. dollars (USD) unless otherwise indicated.

1.2 Property Description and Location

The Tuligtic Property (the "Property" or the "Tuligtic Property") 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 Property comprises two mineral claims totalling 14,229.55 hectares (ha) located within Puebla State, 80 kilometres (km) north of Puebla City, and 130km east of Mexico City.

1.3 Accessibility, Climate, Local Resources, Infrastructure, Physiography

The Tuligtic Property is road accessible and is located within Puebla State, 80 kilometres (km) north of Puebla City, and 130km east of Mexico City. The Ixtaca Deposit within the Tuligtic Property is located 8km northwest of the town of San Francisco Ixtacamaxtitlán, the county seat of the municipality of Ixtacamaxtitlán, Puebla State.

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

Electricity is available on the Property as the national electricity grid services nearby towns such as Santa Maria and Zacatepec.

Almaden has negotiated voluntary surface land use agreements with landowners prior to entering the exploration area and commencing work. Additional or revised landowner agreements may be required in the event advanced operations are anticipated (for example potential tailings storage areas, potential rock storage areas, and potential processing plant sites). The Federal Mining Law in Mexico provides mineral claim owners the right to obtain the temporary occupancy or creation of land easements necessary to carry out exploration and mining operations.

1.4 History

Throughout the Property there is evidence that surficial clay deposits have once been mined prior to Almaden's acquisition of the project. Almaden acquired the Cerro Grande claims of the Tuligtic Property in 2001 following the identification of surficial clay deposits that have been interpreted to represent high-level epithermal alteration. Subsequent geologic mapping, rock, stream silt, soil sampling, and induced polarization (IP) geophysical surveys identified porphyry copper and epithermal gold targets within an approximately 5 x 5km 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 (m) of 1.01g/t Au and 48g/t Ag and multiple high grade intervals including 44.35m of 2.77g/t Au and 117.7g/t Ag.

1.5 Geological Setting and Mineralization

Within the Tuligtic Property, 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 has been 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.

The epithermal vein system at the Main Ixtaca and Ixtaca North zones is associated with two sub parallel ENE (060 degrees) trending, sub-vertical to steeply north dipping dyke zones. A series of 2m to over 20m true width dykes occur within an approximately 100m 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 20m 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 synform 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 200m. 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 Main and North zones have been defined over 650m and tested over 1000m strike length with high-grade mineralization intersected to depths up to 350m vertically from surface. The strike length of the Chemalaco Zone has been extended to 450m with high-grade mineralization intersected to a vertical depth of 550m, or approximately 700m down-dip. An additional sub-parallel zone has been defined underneath the Chemalaco Zone dipping 25 to 50 degrees to the WSW, intersected to a vertical depth of 250m, approximately 400m down-dip over a 250m strike length.

The Chemalaco Zone (also known as the Northeast Extension) has a strike length of approximately 450m as defined by drilling along a series of ENE (070 degrees) oriented sections spaced at intervals of 25 to 50m, and near-surface oblique NNW-SSE oriented drillholes. The Chemalaco Zone dips moderately-steeply at approximately 55 degrees to the WSW. Chemalaco 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 of structurally thinned, brecciated, and mineralized limestone is unconformably overlain by mineralized tuff rocks. At a vertical depth of approximately 50m 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 30m true-thickness felsic porphyry dyke (Chemalaco Dyke), which is also mineralized. The Chemalaco Dyke has been interested in multiple drillholes ranging from 250 to 550m vertically below surface, and its lower contact currently marks the base of Chemalaco Zone mineralization.

1.6 Exploration

Between 2001 and 2013, Almaden's exploration at the Tuligtic Property included geologic mapping and prospecting, alteration mineralogical characterization, rock and soil geochemical sampling, ground magnetics, IP and resistivity, Controlled Source Audio-frequency Magnetotelluric (CSAMT), and Controlled Source Induced Polarization (CSIP) geophysical surveys resulting in the identification of additional anomalous zones including the Ixtaca, Ixtaca East, Caleva, Azul, and Sol zones. Since 2010, a total of 423 diamond drillholes have been drilled at the Tuligtic Gold-Silver Project, totalling 137,438m. To date all the drilling has been carried out in the Ixtaca deposit area.

1.7 Drilling

The 225 holes drilled between July, 2010 and November 13, 2012 totalled 81,971m and identified the Main Ixtaca, Ixtaca North and Chemalaco zones. Diamond drilling at 25 to 50m section spacing defined the Main Ixtaca and Ixtaca North as NE-oriented sub-vertical zones and a strike length of approximately 650m. High-grade mineralization was intersected to depths of 200 to 300m vertically from surface. The Chemalaco Zone was identified as dipping moderately-steeply over a strike length of 350m along a series of five ENE (070 degrees) oriented sections spaced at intervals of 50 to 100m. High grade mineralization having a true-width ranging from less than 30 and up to 60m was intersected beneath approximately 30m of tuff to a vertical depth of 550m, or approximately 600m down-dip.

During 2013 and subsequent to the November 13, 2012 cut-off of the maiden mineral Resource Estimate, Almaden drilled 198 holes totalling 55,467m. A total of 79 holes have been drilled at the Main Ixtaca Zone, 40 holes at the Ixtaca North Zone and 79 holes at the Chemalaco Zone. Drilling during 2013 focused on expanding the deposit and upgrading resources previously categorized as Inferred to higher confidence Measured and Indicated categories.

1.8 Sample Preparation, Analyses and Security

All strongly altered or epithermal-mineralized intervals of core have been sampled. Almaden employs a maximum sample length of 2 to 3m in unmineralized lithologies, and a maximum sample length of 1m in mineralized lithologies (50cm minimum sample length). Drill core is half-sawn using industry standard diamond core saws. After cutting, half the core is placed in a new plastic sample bag and half are placed back in the core box. Sample numbers are written on the outside of the sample bags and a numbered tag placed inside the bag. Sample bags are sealed using a plastic cable tie. Sample numbers are checked against the numbers on the core box and the sample book.

ALS sends its own trucks to the Project to take custody of the samples at the Santa Maria core facility and transports them to its sample preparation facility in Guadalajara or Zacatecas, Mexico. Prepared sample pulps are then forwarded by ALS personnel to the ALS North Vancouver, British Columbia laboratory for analysis.

Drill core samples have been subject to gold determination via a 50 gram (g) AA finish FA fusion with a lower detection limit of 0.005ppm Au (5ppb) and upper limit of 10ppm Au (ALS method Au-AA24). Over limit gold values (>10ppm Au) are subject to gravimetric analysis (ALS method Au-GRA22). Silver, base metal and pathfinder elements for drill core samples are analyzed by 33-element ICP-AES, with a 4-acid digestion, a lower detection limit of 0.5ppm Ag and upper detection limit of 100ppm Ag (ALS method ME-ICP61). Over limit silver values (>100ppm Ag) are subject to 4-acid digestion ICP-AES analysis with an upper limit of 1,500ppm Ag (ALS method ME-OG62). Ultra-high grade silver values (>1,500ppm Ag) are subject to gravimetric analysis with an upper detection limit of 10,000ppm Ag (Ag-GRA22).

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 fifteen QA/QC samples are present in every 100 samples sent to the laboratory. QA/QC sample results are reviewed 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 ten preceding and succeeding samples.

1.9 Data Verification

Mr. Kristopher J. Raffle, P.Geo., first visited the Tuligtic Property from October 17 to October 20, 2011. Additional visits to the Tuligtic Property have been carried out by Mr. Raffle on September 23, 2012 and November 20, 2013. During each of the property visits Mr. Raffle completed a traverse of the Ixtaca Zone, observed the progress of ongoing diamond drilling operations, and recorded the location of select drill collars. Almaden's complete drill core library has been made available and Mr. Raffle reviewed mineralized intercepts from a series of holes across the Ixtaca Zone. Mr. Raffle has collected quartered drill core samples as 'replicate' samples from select reported mineralized intercepts.

Based on the results of the traverses, drill core review, and 'replicate' sampling Mr. Raffle has no reason to doubt the reported exploration results. The analytical data is considered to be representative of the drill samples and suitable for inclusion in the Resource Estimate. In addition to the in-house Quality Assurance Quality Control (QAQC) measures employed by Almaden, Kris Raffle, P.Geo. of APEX Geoscience Ltd., completed an independent review of Almaden's drillhole and QAQC databases. The review included an audit of approximately 10% of drill core analyses used in the mineral resource estimate. A total of 10,885 database gold and silver analyses were verified against original analytical certificates. Similarly, 10% of the original drill collar coordinates and down hole orientation survey files were checked against those recorded in the database; and select drill sites were verified in the field by Kris Raffle, P.Geo. The QAQC audit included independent review of blank, field duplicate and certified standard analyses. All QAQC values falling outside the limits of expected variability were flagged and followed through to ensure completion of appropriate reanalyses. No discrepancies were noted within the drillhole database, and all QAQC failures were dealt with and handled with appropriate reanalyses.

1.10 Metallurgy

Exploratory metallurgical testwork was completed on each of the Ixtaca Zone geologic domains in 2012. Additional preliminary metallurgical testwork on the Ixtaca Zone geologic domains has been completed in 2013.

The preliminary metallurgical testwork results from Ixtaca's composite samples shows that the Ixtaca deposit responds well to gravity concentration followed by flotation and leaching of the gravity and flotation concentrates. Overall, gold and silver process recoveries to a silver-gold doré of 90.3% are recommended for the PEA.

1.11 Resource Estimate

The previous maiden NI 43-101 compliant mineral Resource Estimate for the Ixtaca Deposit was derived from the drilling of 225 diamond drillholes between July, 2010 and November 13, 2012. The maiden resource for the Ixtaca deposit was announced on January 31, 2013 and consisted of an indicated mineral resource of 56.99 million-tonnes, comprising 2.02 million-ounces AuEq at an average grade of 1.10g/t AuEq; and an Inferred mineral resource of 41.53 million-tonnes, comprising 1.55 million-ounces AuEq at an average grade of 1.16g/t AuEq, each using a cut-off grade of 0.5g/t AuEq. Ixtaca Deposit resource was classified as Indicated and Inferred mineral resource according to the definitions from NI 43-101 and from CIM (2005). A cut-off of 0.50g/t AuEq was highlighted as a possible cut-off for open pit mining.

Based upon the results of the diamond drilling since November 13, 2012, an update to the maiden mineral resource for the Ixtaca deposit has been prepared by Giroux Consultants Ltd. (GCL). Preliminary metallurgy has shown roughly equivalent metal recoveries for Au and Ag, therefore the mineral Resource Estimate is presented at a series of AuEq cut-offs based on a three years trailing average price of $1,540 per-ounce Au, and $30 per-ounce Ag, and assuming one can mine to the limits of the mineralized solids and no edge dilution is included. The updated Ixtaca Deposit source has been classified as a Measured, Indicated, and Inferred Mineral Resource according to the definitions from NI 43-101 and from CIM (2014). A cut-off of 0.50g/t AuEq is highlighted as a possible cut-off for open pit mining. Table 1-1 below compares the 2013 maiden mineral Resource Estimate with the updated 2014 mineral Resource Estimate.

Table 1-1 Comparison of 2014 vs. 2013 Mineral Resource Estimation (with 0.5g/t AuEq Cut-off)

Year	Classification	Measured Resource				
		Tonnes	Grade AuEq[1] (g/t)	Au (g/t)	Ag (g/t)	Contained Metal x1000 AuEq[1] (ozs)
2014	Measured Resource	30,420,000	1.38	0.61	39.44	1,350
2013		-	-			-
2014	Indicated Resource	62,250,000	1.09	0.52	28.92	2,182
2013		56,780,000	1.10	0.52	29.94	2,014
2014	Inferred Resource	22,150,000	0.99	0.50	25.14	704
2013		41,120,000	1.16	0.56	31.44	1,539

1. *AuEq = Au + (Ag * 30/1540)*

Diamond drilling by Almaden has resulted in the identification of a Measured mineral resource of 30.42 million-tonnes, comprising 1.35 million-ounces AuEq at an average grade of 0.61 g/t Au, 39.44 g/t Ag and 1.38g/t AuEq; an Indicated mineral resource of 62.25 million-tonnes, comprising 2.18 million-ounces AuEq at an average grade of 0.52 g/t Au, 28.92 g/t Ag and 1.09g/t AuEq; and an Inferred mineral resource of 22.15 million-tonnes, comprising 0.70 million-ounces AuEq at an average grade of 0.50 g/t Au, 25.14 g/t Ag and 0.99 g/t AuEq.

1.12 Proposed Development Plan

A PEA level mining design, production schedule, and cost model has been developed for the Ixtaca Zone of the Tuligtic Property. This current work is based on the results of the 2014 Technical Report; model resource update and optimization work, and includes details of a Ramp-Up production schedule designed to reduce initial capital costs. The Base Case includes an open pit mining operation with a 30,000 tonne per day process plant to produce gold and silver doré. The Ramp-Up Case includes an open pit mining operation starting with a 7,000 tonnes per day process plant to produce gold and silver doré. The Ramp-Up Case increases to 9,000 tonnes per day in Year 3 and then increases to 30,000 tonnes per day by Year 6 of production. The Ramp-up Case mines to the same ultimate limit as the Base Case. The process plant includes conventional crushing, grinding, gravity, flotation, concentrate leaching and Merrill-Crowe extraction processes. Direct mining will be done using a contractor owned and operated fleet.

A series of pit optimizations are run using the resource block model, applying a range of metal prices and recoveries, and estimated costs for mining, processing, and pit slope. The operational pits are designed based on the optimized shells, and the potentially mineable portion of the resource is estimated within those pits. The ultimate pit contains a total of 343.4Ktonnes (kt) of combined mill feed and waste material including 125.3Ktonnes (kt) of mill feed, for a strip ratio of 1.7:1. The mill feed tonnages include a mining loss factor of 3%, and operational grade dilution of 3%. Pit resources are shown in the Table below:

Table 1-2 Recovered In-pit Resources and Diluted Grade

CLASS	Mill Feed kt	NSR ($/tonne)	Au (g/t)	Ag (g/t)
Measured	36,995	33.46	0.504	32.25
Indicated	68,357	26.33	0.421	23.57
Sub-Total of Measured and Indicated	105,352	28.83	0.450	26.62
Inferred	*19,944*	*21.56*	*0.323*	*20.89*

The potentially mineable tonnages in the PEA selected ultimate pit include Inferred Resources. The reader is cautioned that Inferred Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that Inferred Resources will ever be upgraded to a higher category.

A mill production rate of 30,000 tonnes per day is assumed. The Base Case production schedule includes two years of pre-production pre-stripping, followed by twelve years of operating mine life. During the first eight years of production, mill feed is mostly direct feed from the mining operations. Low grade material encountered during the first eight years is directed to a low grade stockpile at the toe of the Rock

Storage Facility (RSF). For the remaining mine life, the low grade stockpile is reclaimed and supplies the mill. See Figure 1-1 below for the Base Case production schedule details.

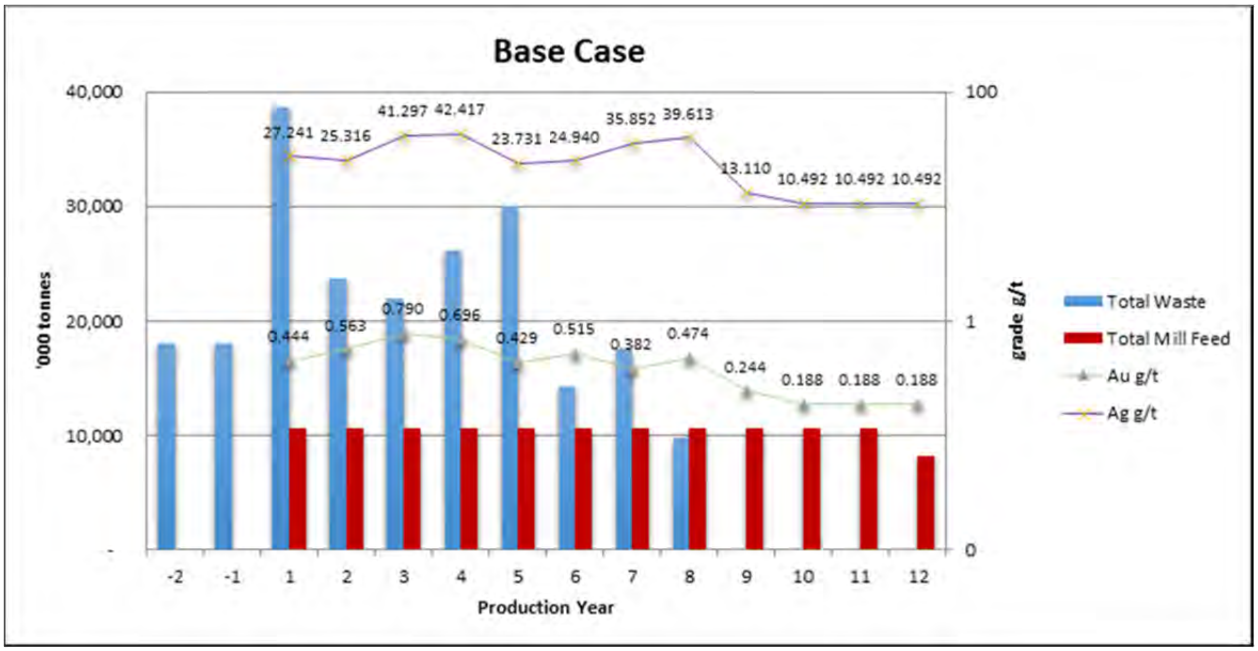


Figure 1-1 Ixtaca Base Case Production Schedule

The Ramp-Up Case production schedule includes one year of pre-production pre-stripping, followed by fifteen years of operating mine life. During the first five years of production, mill feed is almost entirely from the mining operations with low grade material directed to a low grade stockpile at the toe of the RSF. The low grade stockpile reaches maximum size at the end of Year 5. Once the process plant expands to 30,000 tonnes per day in Year 6, the mill feed is a combination of low grade stockpile reclaim blended with material from mining operations. During the last two years of production, the mill feed is comprised entirely of low grade stockpile reclaim. A total of 4.2 million tonnes of material remains in the low grade stockpile at the end of the mine life. See Figure 1-2 below for the Ramp-Up Case production schedule details.

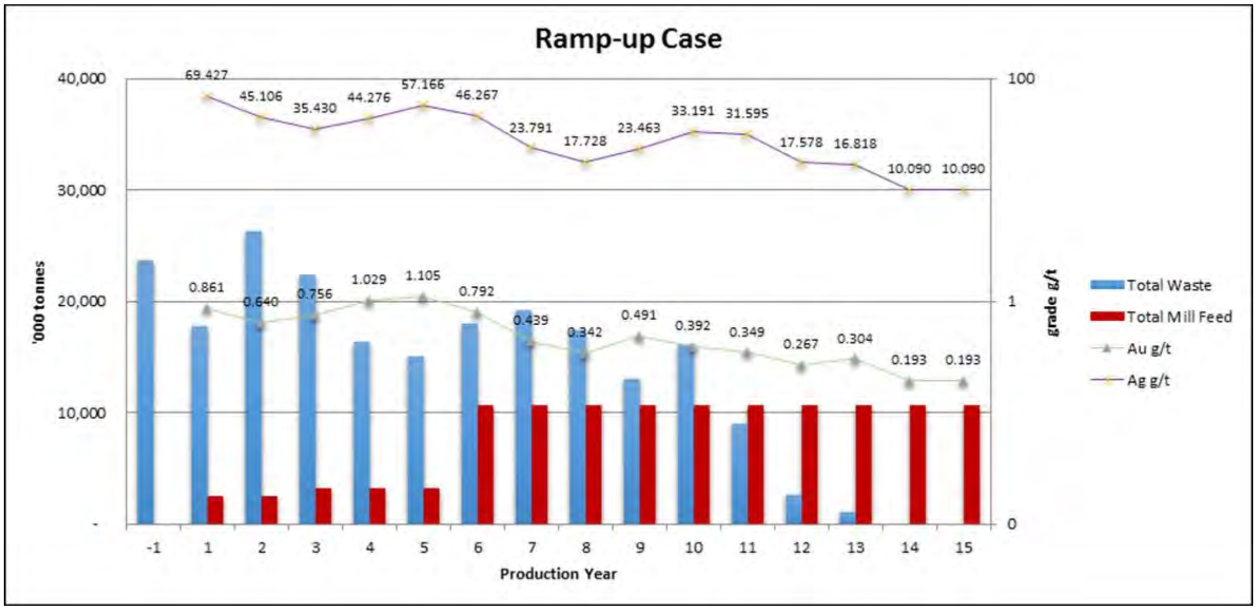


Figure 1-2 Ixtaca Ramp-Up Case Production Schedule

1.13 Production and Processing

The Base Case includes a 30,000 tonne per day process plant to produce gold and silver doré on site. The process plant includes conventional crushing, grinding, gravity, flotation, concentrate leaching, and Merrill-Crowe extraction process. Average total process recoveries for gold and silver are expected to be 90.3%. The Ramp-Up Case starts with a 7,000 tonne per day process plant using the same methods as the Base Case. Processing increases to 9,000 tonnes per day in Year 3. The Ramp-Up Case includes the purchase of a 30,000 tonne per day process plant which starts operations in Year 6 of production. In the Ramp-Up Case the 7,000 tonne per day process plant will be de-commissioned and sold once the 30,000 tonne per day mill is in operation.

1.14 Capital and Operating Costs

The Capital Cost Estimate for the Ixtaca Project is developed to a level appropriate for a PEA study in order to evaluate the overall project viability. As such, the level of accuracy is +/-35%. All Capital and Operating costs are reported in USD unless specified otherwise. The initial capital costs are summarized in Table 1-3 below:

Table 1-3 Projected Initial Capital Costs (USD million)

	Base Case	Ramp-Up Case
Site Infrastructure	$20.4	$19.4
TMF and Water Management	$44.7	$29.0
Pre-stripping activities	$64.5	$37.8
Mining Equipment	$8.0	$7.7
Process Plant	$194.5	$105.5
Indirects, EPCM, Contingencies and Owner's Costs	$67.4	$44.1
Total Start-up Capital*	$399.4	$243.5

** Numbers may not add due to rounding*

Life-of-Mine (LOM) sustaining capital includes raising the elevation of the Tailings Management Facility (TMF) through the mine schedule, Process Plant sustaining capital, and capital required to expand the fleet size. Total LOM sustaining capital costs are estimated to be $110.3 million for the Base Case and $110.8 million for the Ramp-Up Case. Net LOM expansion capital includes the capital cost to purchase a new, larger mill as well as the credit received for the sale of the small mill. Total LOM expansion capital for the Ramp-Up Case is $116.0 million. There is no expansion capital required for the Base Case.

The total LOM operating costs for the Ixtaca Project are $14.48/tonne mill feed for the Base Case and $15.85/tonne mill feed for the Ramp-Up Case. This estimate includes the contractor mining, processing, G&A, GME, re-handle, reclamation and TMF and water management operating costs during the period of operations (initial capital costs are not included in the LOM operating costs). The LOM average breakdown of these costs is shown in Table 1-4 below:

Table 1-4 Projected Average LOM Operating Costs ($/tonne mill feed)

	Base Case	Ramp-Up Case
Contractor Mining	$3.89	$4.34
Processing	$9.00	$9.60
Stockpile re-handle	$0.34	$0.52
TMF Management	$0.10	$0.09
Reclamation	$0.18	$0.19
G&A	$0.80	$0.92
GME	$0.17	$0.18
Total*	**$14.48**	**$15.85**

**Numbers may not add due to rounding*

The Base Case economic prices use a 62.9:1 silver to gold ratio. This ratio is used to calculate the cost of production in $/oz AuEq as shown in Table 1-5 below:

Table 1-5 Life of Mine Cost of Production

	Base Case		Ramp-Up Case	
	$ million	**$/oz AuEq**	**$ million**	**$/oz AuEq**
Operating Costs	$1,814	$595	$1,918	$638
Sustaining Capital	$110	$36	$111	$37
All-in Sustaining Costs	**$1,924**	**$631**	**$2,029**	**$675**
Initial Capital	$399	$131	$244	$81
Expansion Capital	$0	$0	$116	$39
All-in Costs	**$2,324***	**$762**	**$2,389**	**$795**

**Numbers may not add due to rounding*

1.15 Economic Analysis

The updated PEA project economics are based on gold price of $1320/oz and silver price of $21/oz. These prices are matched to the May 13, 2014 PEA (maiden PEA) and are a combination of spot prices to that date and current common peer usage. The project revenue is split between gold and silver with 54% coming from gold and 46% coming from silver. The after-tax economic analysis includes a corporate income tax rate of 30% (as per the Mexican Tax Reform increase effective Jan 01, 2014) as well as the two new mining duties:

a) 7.5% special mining duty and,
b) 0.5% extraordinary mining duty.

A summary of the Base Case and Ramp-Up Case economic analysis is presented in Table 1-6 below.

Table 1-6 Summary of Ixtaca Gold-Silver Economic Results

	Base Case		Ramp-Up Case	
	Pre-Tax	**After-Tax**	**Pre-Tax**	**After-Tax**
Net Cash Flow ($ million)	$1,334	$852	$1,231	$779
NPV 5% ($ million)	$842	$515	$699	$427
NPV 8% ($ million)	$640	$378	$497	$294
Internal Rate of Return	37.2%	28.3%	28.9%	23.2%
Initial Capital Payback (years)	2.3	2.5	4.2	4.5
Expansion Capital Payback (years)	N/A	N/A	0.3	0.4

The economic results are based on the potentially mineable tonnages in the selected ultimate pit. The potentially mineable tonnages in the PEA selected ultimate pit include Inferred Resources. The reader is cautioned that Inferred Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that Inferred Resources will ever be upgraded to a higher category. The reader is further cautioned that the preliminary economic assessment is preliminary in nature, and that there is no certainty that the preliminary economic assessment will be realized.

A sensitivity analysis is conducted for varying operating costs, initial capital costs and metal prices. The production schedule, stockpiling and in-pit resources are kept constant for this analysis. The pre-tax NPV 5% and pre-tax IRR sensitivities are summarized in the Tables below:

Table 1-7 Base Case Pre-Tax NPV (5%) and Pre-Tax IRR Sensitivities ($ million)

	Operating Cost Sensitivity		Initial Capital Cost Sensitivity		Metal Price Sensitivity	
Variance	**NPV (5%)**	**IRR**	**NPV (5%)**	**IRR**	**NPV (5%)**	**IRR**
-20%	$1,081	43.1%	$913	45.0%	$334*	20.6%*
-10%	$962	40.2%	$878	40.8%	$573*	29.7%*
Base	$842	37.2%	$842	37.2%	$842	37.2%
+10%	$708*	33.5%*	$807	34.0%	$1,100	43.8%
+20%	$584**	29.7%**	$771*	31.3%*	$1,357	49.8%

** The lowest grade stockpiled material processed at the end of the mine life is below break-even cut-off grade in these scenarios. When this is the case, this material is not processed and is counted as waste. This in turn shortens the mine life to 9 years*

*** Mine life is shortened to 8 years in this scenario*

Table 1-8 Ramp-Up Case Pre-Tax NPV (5%) and Pre-Tax IRR Sensitivities

	Operating Cost Sensitivity		Capital Cost Sensitivity		Metal Price Sensitivity	
Variance	**NPV (5%)**	**IRR**	**NPV (5%)**	**IRR**	**NPV (5%)**	**IRR**
-20%	$934	34.5%	$744	34.0%	$234*	15.1%*
-10%	$817	31.8%	$722	31.2%	$457*	22.4%*
Base	$699	28.9%	$699	28.9%	$699	28.9%
+10%	$572*	25.6%*	$676	26.8%	$934	34.4%
+20%	$464*	22.4%*	$654	25.0%	$1,169	39.4%

** The lowest grade stockpiled material processed at the end of the mine life is below break-even cut-off grade in these scenarios. When this is the case, this material is not processed and is counted as waste. This in turn shortens the mine life to 13 years (from 15 years)*

1.16 Environmental and Social Considerations

Environmental and Community/Social programs are in progress and will continue as the Project progresses into advanced studies. Currently there are no known issues that can materially impact the ability to extract the mineral resources at the Ixtaca Project. Previous and ongoing environmental studies include meteorology, water quantity and quality, and flora and fauna. KP has been retained by Almaden to help with long lead item studies concerning environmental monitoring, assessment and permitting matters. Almaden established the following environmental objectives for the Project:

- Protect surface and ground water quality;
- Incorporate environmental enhancement opportunities into the mine and final reclamation plans;
- Minimize the project footprint.

In order to achieve these objectives Almaden and KP have instituted the following management strategies:

Water Management – Almaden, with KP, has developed a comprehensive 2014 water management strategy including the commencement of a hydrometric and climate monitoring program, and the drilling of groundwater monitoring wells. The latest modeling, conducted using regional weather patterns, suggest that the management of rainfall and runoff from within the project area will provide sufficient water for continuous operations for the Ixtaca mine plan. Currently local communities use existing water supplies that come from natural springs located at higher elevations and upstream of the Ixtaca deposit. Stream flow upstream of the project will be either diverted around or collected, potentially creating a new fresh water supply source for local use, or used for mining and milling processes and before any would be discharged it would be treated to meet environmental guidelines.

Management of Rock – The limestone host rock, which constitutes approximately 1/3 of the total waste rock, has buffering capacity. Static geochemical testing is currently underway to characterize this further.

Environmental Monitoring – Groundwater monitoring to ensure compliance with all applicable best management practice (BMP) technologies is a fundamental component of the Project. Flora and fauna studies are also underway.

Almaden has developed good relations with the community in and around the project area, including an aggressive local hire policy for exploration activities, and involvement in health and social welfare projects.

Almaden has negotiated voluntary surface land use agreements with landowners prior to entering the exploration area. Additional or revised landowner agreements may be required in the event advanced operations are anticipated (for example potential tailings storage areas, potential rock storage areas, and potential processing plant sites). The Ixtaca deposit and any potential mining operation will be located in an area previously logged or cleared. Existing land use in the project area is minimal. The Company has employed up to 70 local people in its drilling program who live local to the Ixtaca deposit. Local employees make up virtually all the drilling staff, who have been trained on the job to operate the Company's wholly owned drills. Almaden has implemented a comprehensive community relations and education program for employees and all local stakeholders to explain the exploration program underway as well as the potential impacts and benefits of any possible future mining operation at Ixtaca. This program includes general mining education in the form of tours to mines operated by third parties elsewhere in Mexico.

Economic Impacts – The economic analysis set out in the PEA also provides some possible indications of the potential economic impact of the Ixtaca Project on the local, Puebla State and Mexico economies, should the future work and permitting support development of a mining operation. Highlights of the Base Case include:

- Direct employment of more than 400 people during the construction phase and 430 people during the subsequent approximately 12 year operating phase;
- Gross investment of approximately over US$80 million in capital equipment and equipment manufacturing during the construction phase, and,
- Approximately $483 million in direct taxes to all levels of government, including payments to the local Municipality ($60 million), Puebla State ($109 million) and Federal ($314 million) governments over the approximately 12 year operating life of the project, but excluding payroll taxes, sales taxes and income taxes paid by employees.

In due course, the permitting requirements of the Project will be fulfilled, which is a known and regulated process in Mexico.

1.17 Conclusions and Recommendations

The Ixtaca deposit is well suited for a potential mining operation. A Base Case PEA mine plan at 30,000tpd for 12 years has been developed with LOM mill feed grades of 0.430g/t gold and 25.71g/t silver. Higher grade mineralized material is mined and processed early in the mine schedule enabling quick initial capital payback. 84% of the PEA pit resource is included in the Measured and Indicated resource categories.

A Ramp-Up Case PEA mine plan is a viable alternative with significantly reduced initial capital compared to the Base Case. The Ramp-Up Case PEA mine plan starts at 7,000tpd increasing to 9,000tpd in Year 3 with an expansion to 30,000tpd in Year 6. The LOM mill feed grades are 0.438g/t gold and 26.26g/t silver. Higher grade mineralized material is mined and processed early in the mine schedule enabling quick initial capital payback. Expansion capital is delayed to allow quicker payback.

The Project exhibits strong economics at a range of metal prices.

A detailed budget and plan for a PFS is recommended with the additional work plans included for geotechnical, geomechanical, metallurgical testing, mine planning optimization, environmental characterization, and baseline studies.

2.0 INTRODUCTION

This PEA is written for the Ixtaca Gold-Silver Deposit (or "Ixtaca Project") of 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 Property comprises two mineral claims totalling 14,229.55 hectares (ha) within Puebla State, Mexico (Figure 4-1 and Figure 4-2). The purpose of this Technical Report is to present the results of the Preliminary Economic Assessment (PEA) of the Ixtaca Project. This Technical Report (PEA) supersedes the previous 2014 Technical Report entitled "NI 43-101 Technical Report Preliminary Economic Assessment of the Ixtaca Project" dated 13 May 2014.

During 2013, Almaden retained Moose Mountain Technical Services ("MMTS") to complete a mining study on the Ixtaca Project for the purpose of producing a PEA. The lead author, Jesse Aarsen, P.Eng., associate of MMTS, an independent qualified person as defined by NI 43-101, conducted a property visit on August 27-28, 2014 and on a previous occasion between April 30, 2013 and May 01, 2013. Another 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 November 20, 2013 and on previous occasions on September 23, 2012 and between October 17 and 20, 2011.

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 drillhole information. The authors, in writing this report use sources of information as listed in the references section. Government reports have been prepared by qualified persons holding post-secondary geology, or related university degree(s), and are therefore deemed to be accurate. These reports, which are used as background information, are referenced in this Report in the "Geological Setting and Mineralization" Section 7.0 below. All currency amounts are referred to in USD 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.

Several authors contributed to or supervised the completion of this Technical Report, and are all independent Qualified Persons ("QP") within the meaning of Canadian Securities Administrator's National Instrument 43-101 Standards. Each QP in this report takes responsibility for their work as outlined in their QP Certificates included in this report and found in the following chart:

Qualified Person	Company	Sections of Responsibility
Jesse Aarsen	Moose Mountain Technical Services	1,15-16, 18, 21-22, 25-26
Tracey Meintjes	Moose Mountain Technical Services	1, 13, 17, 19, 26
Gary Giroux	Giroux Consultants Ltd	1, 14, 26
Kris Raffle	APEX Geoscience	1,2-12, 23-24, 27, 26
Ken Embree	Knight Piésold	1, 20, 26

3.0 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/).

4.0 PROPERTY DESCRIPTION AND LOCATION

The Tuligtic Property consists of two mineral claims totaling 14,229.55ha (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.

Table 4-1 Tuligtic Property 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	3,028.00
		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: 618,800m east and 2,176,100m north. The Tuligtic Property is road accessible and is located within Puebla State, 80 kilometres (km) north of Puebla City, and 130km 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 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 Regulation of the Mining Law indicates the minimum exploration expenditures or the value of the mineral products to be obtained (Table 4-2).

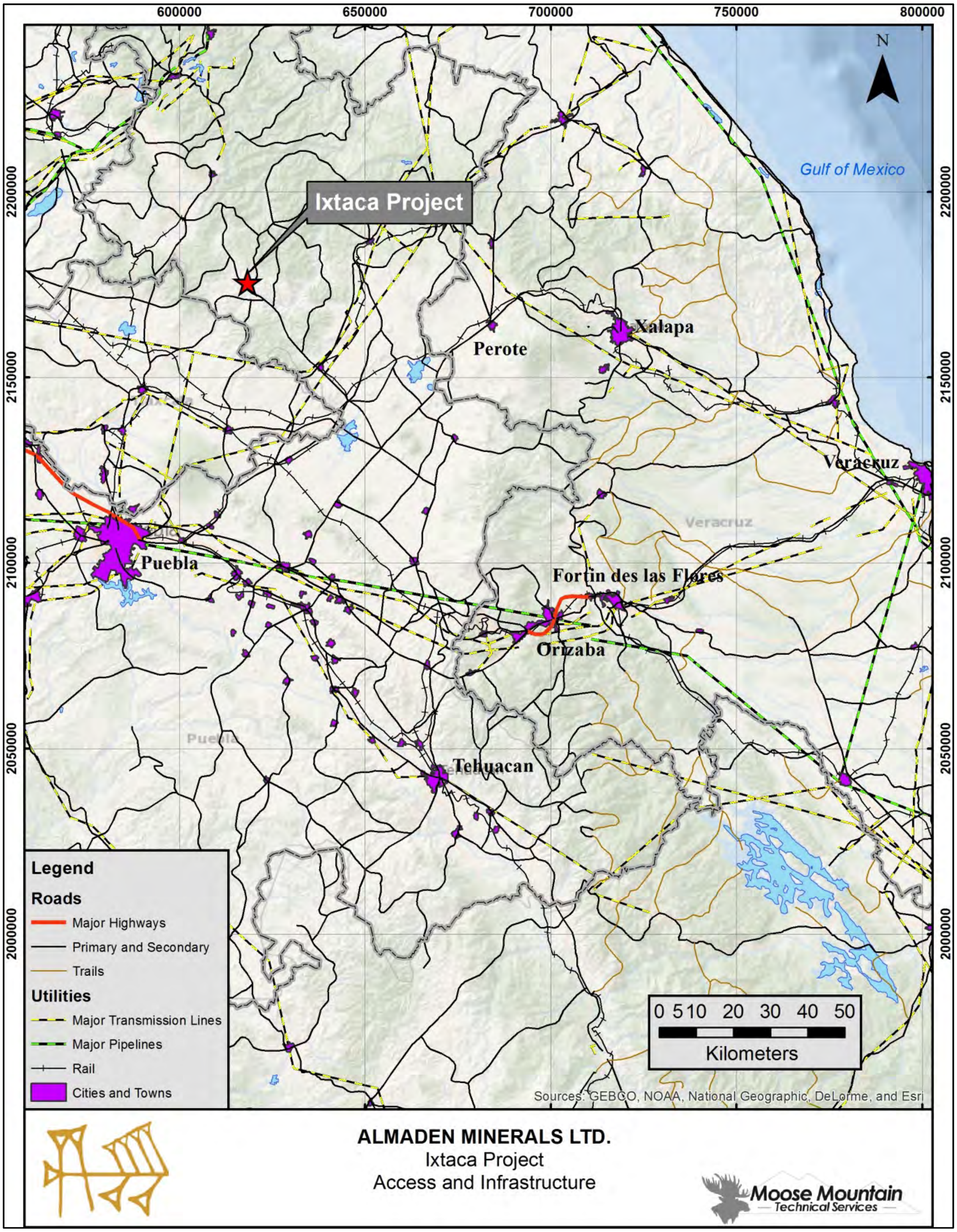


Figure 4-1 General Location

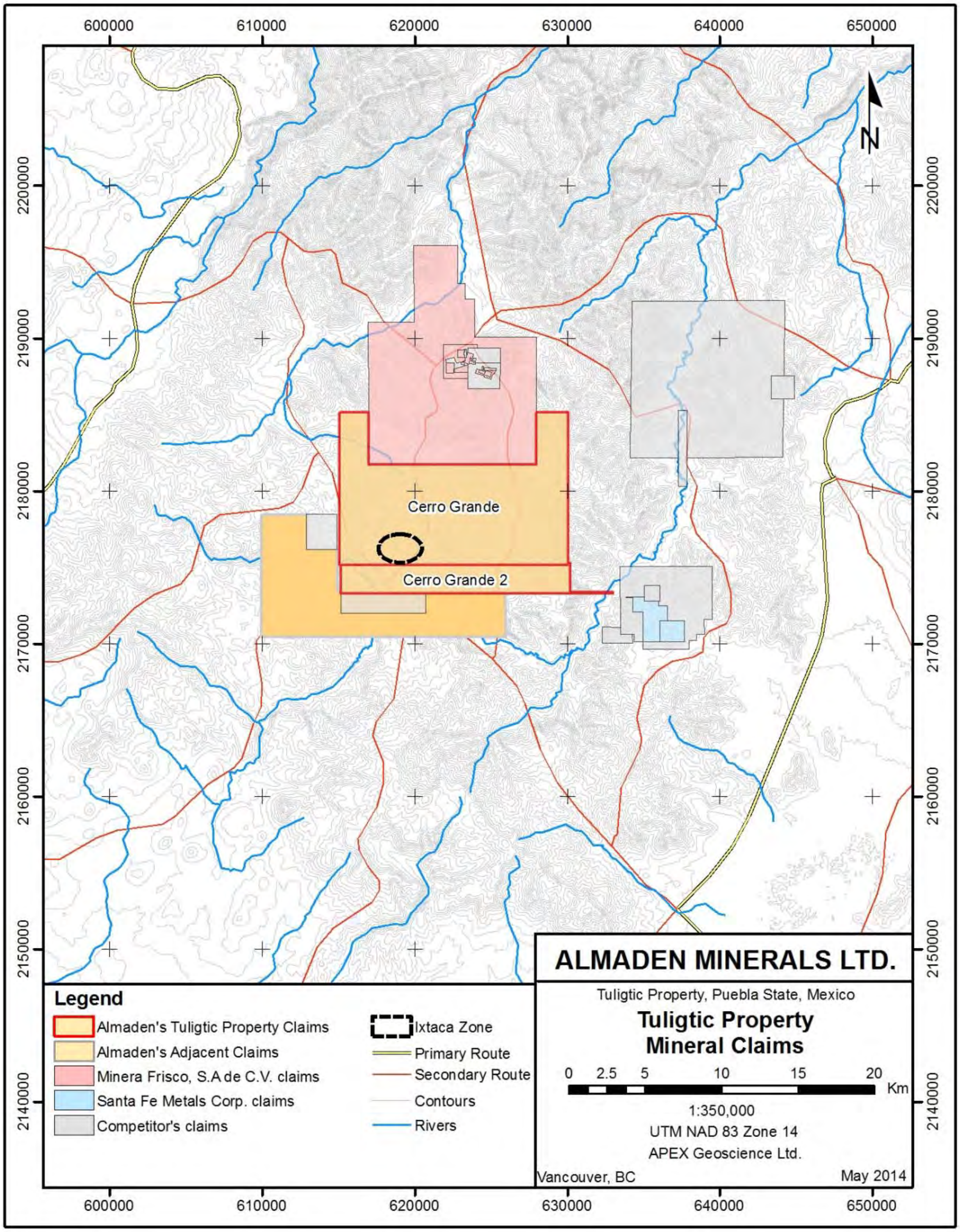


Figure 4-2 Tuligtic Property Mineral Claims

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

Area (hectares)	Fixed quota in Pesos	Additional annual quota per hectare in Pesos (USD per hectare)			
	(USD)	Year 1	Year 2-4	Year 5-6	Year 7+
<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 = 0.08USD*

The Tuligtic Property is currently subject to annual exploration/exploitation expenditure requirements of approximately $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 Property 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 voluntary surface land use agreements with surface landowners within the exploration area prior to beginning activities.

At present, the authors are 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.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

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

The Project is accessible by driving 40km east along Highway 119 from Apizaco; an industrial centre located approximately 50km north of Puebla City, and then north approximately 20km 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 Industrial complex lies 30km southwest by paved road from the Tuligtic Property, and houses agricultural, chemical, biomedical and industrial manufacturing facilities and is serviced by rail. 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 Property is generally moderate to steep hills with incised stream drainages. Elevation ranges from 2,300 metres (m) above sea level in the south to 2,800m in the north. Vegetation is dominantly cactus and pines and the general area is also somewhat cultivated with subsistence vegetables, bean and corn crops. The region has a temperate climate with average temperatures ranging from 19°C in June to 10°C in December. The area experiences up to 600mm 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.

Almaden has negotiated voluntary surface land use agreements with landowners prior to entering the exploration area and commencing work. Additional or revised landowner agreements may be required in the event advanced operations are anticipated (for example potential tailings storage areas, potential rock storage areas, and potential processing plant sites). The Federal Mining Law in Mexico provides mineral claim owners the right to obtain the temporary occupancy or creation of land easements necessary to carry out exploration and mining operations.

6.0 HISTORY

Throughout the Property there is evidence that surficial clay deposits have once been mined. This clay alteration attracted Almaden to the area and has been interpreted to represent high-level epithermal alteration. To the best of the authors' knowledge no modern exploration has been conducted on the Project prior to Almaden's acquisition of claims during 2003 and there is no record of previous mining; as such, this is a maiden discovery.

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 has been 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 5km 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 identified a greater than 2km long elevated chargeability response coincident with the exposed altered and mineralized intrusive system. Volcanic rocks exposed 1km 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 have been identified within limestone roughly 100m 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, 3km in length, spaced at intervals of 200m have been completed over mineralized intrusive rocks intermittently exposed within gullies cutting through the overlying unmineralized ash deposits (Almaden, 2006).

The Tuligtic Property has been optioned to Pinnacle Mines Ltd. in 2006 and the option agreement has been terminated in 2007 without completing significant exploration (Almaden, 2007).

The Property has been 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 (Figure 7-2, Figure 9-1). Three additional IP survey lines have been completed, and in conjunction with the previous nine (9) IP lines, a 2 x 2.5km chargeability high anomaly, open to the west and south, has been defined (Almaden, 2011). The 2009 drilling consisted of 2,973m within seven (7) holes that largely intersected skarn type mineralization.

Highlights of the drill program include:

- 38m of 0.13% Copper (Cu) from 164 to 202m and 0.11% Cu from 416 to 462m within hole DDH-01;
- 20m of 0.17% Cu from 94 to 114m and 26m of 0.14% Cu from 316 to 342m in hole DDH-02;
- 58m of 0.17% Cu from 366 to 424m in hole DDH-03 (including 14m of 0.27% Cu from 410 to 424m);
- 2m of 0.63% Cu from 18 to 20m in hole DDH-04; and
- 20m of 0.11% Cu from 276 to 296m and 8m of 0.13% Cu in hole DDH-05.

Molybdenum values are anomalous ranging up to 801 parts-per-million (ppm) (0.08%). Elevated gold values are also encountered including 2m of 1.34 grams-per-tonne (g/t) from 178 to 180m 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 is 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, has 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 44.35 metres of 2.77g/t gold and 117.7g/t silver.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Ixtaca 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 300km (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 10Ma (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 ~13.5Ma ago, followed by a nearly 10Ma hiatus. Volcanic activity in the area resumed around 3.0-1.5Ma. 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 have been controlled by NE-SW trending normal faults, associated with horst-and-graben structures, resulting from a stress field with a least principal stress ($\sigma3$) oriented to the NW.

The regional trend of the arc rocks is WNW; though more northerly trending transforms 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 is 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) show that NNW trending regional faults have been 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 development and likely to be purely extensional with possibly a component of right lateral movement, or transtensional.

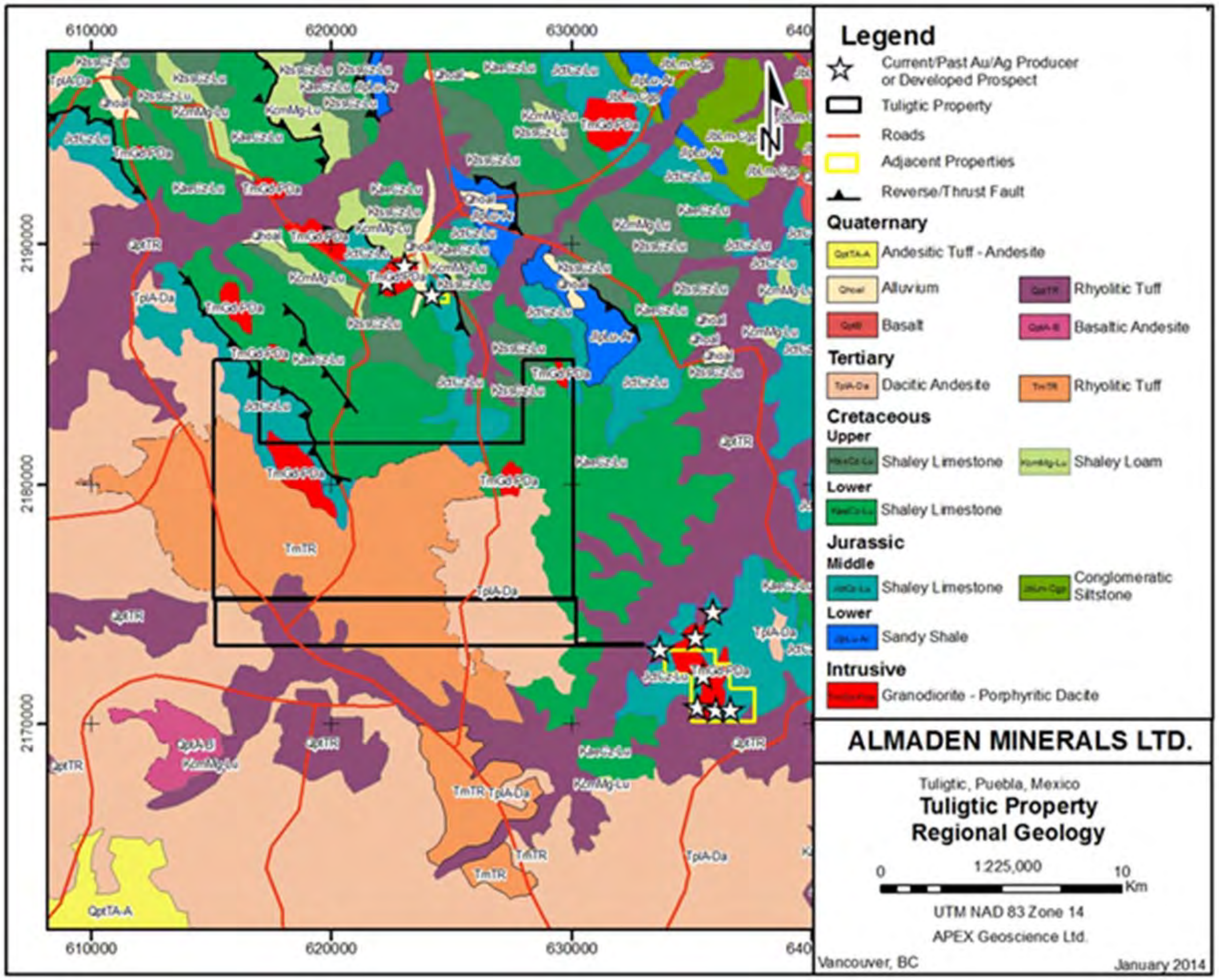


Figure 7-1 Regional Geology

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, is regionally 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 allows 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 is 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 have been 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 are 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 Property porphyry mineralization 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 mineralized especially at their contacts with limestone country rock. Petrography has shown that epithermal alteration in the dykes, marked by illite, adularia, quartz and pyrite overprints earlier calc-silicate endoskarn mineralogies (Leitch, 2011). Two main orientations are identified for dykes in the Ixtaca area; 060 degrees (parallel to the Main Ixtaca and Ixtaca North zones) and 330 degrees (parallel to the Chemalaco Zone).

An erosional unconformity surface has been formed subsequent to the intrusion of the porphyry mineralization-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 are recognized in the area of Tuligtic: the Coyoltepec Pyroclastic deposit and the Xaltipan Ignimbrite

(Carrasco-Nunez et al., 1997). Both units are covered by a thin (up to 1m) 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 is 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, 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 mineralized. Gold silver mineralization is marked by widespread disseminated pyrite and quartz-calcite veinlets. The Coyoltepec volcanics are locally oxidised and weathered near surface and along structures.

The post-mineral Xaltipan ignimbrite is not seen in the Ixtaca area and mainly found in topographic lows south of the Tuligtic Property. It consists of a very recent (0.45 ± 0.09Ma, 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).

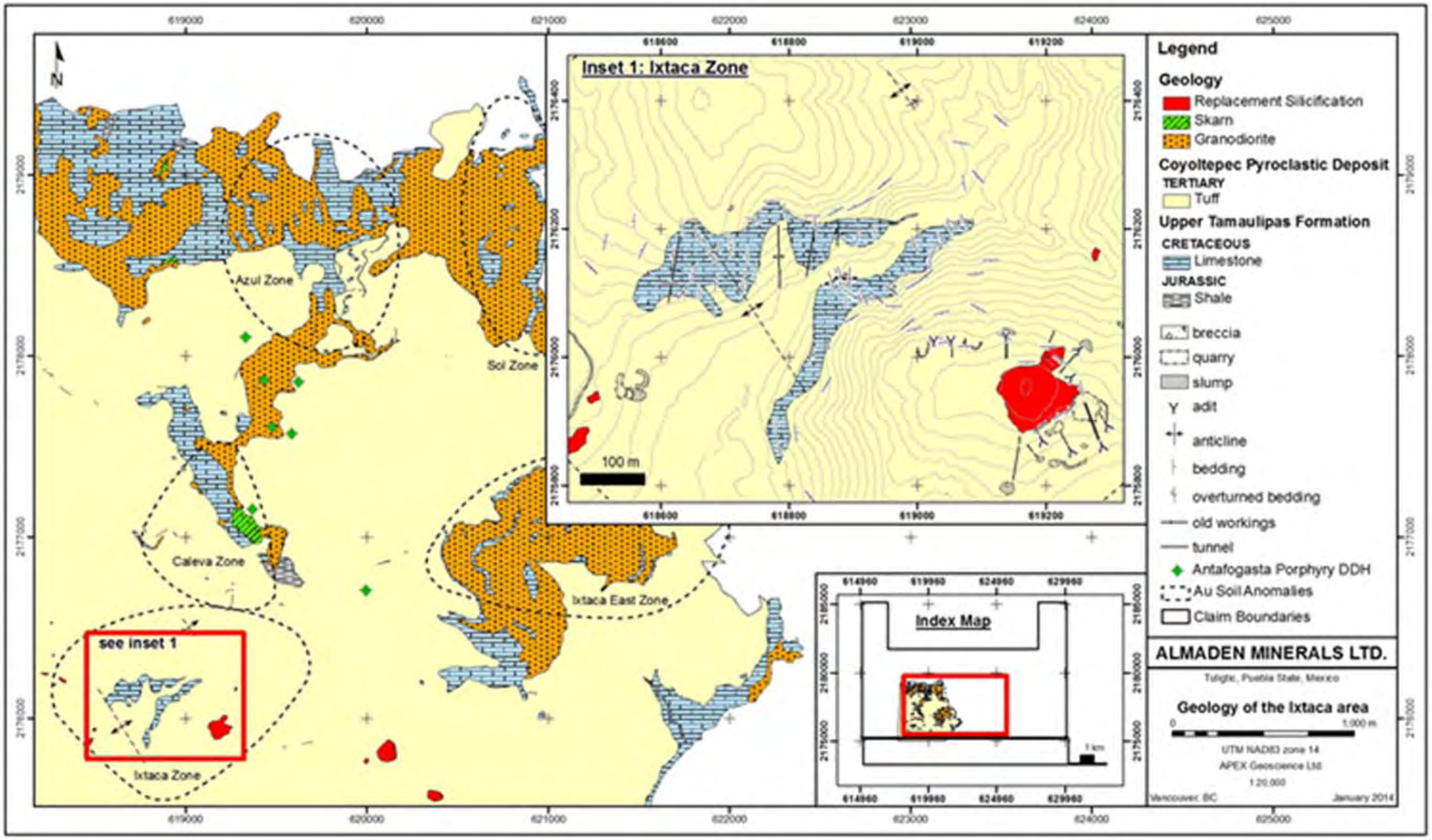


Figure 7-2 Geology of the Ixtaca Area

7.3 Mineralization

Two styles of alteration and mineralization are 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 are 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 mineralization on the Tuligtic Property is considered to have no genetic relationship to the porphyry alteration and mineralization 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 mill feed and gangue mineralogy (electrum, sphalerite, galena, adularia, and carbonates), mineralization 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 Chemalaco Zone are hosted by the shaley carbonate units and strike to the NNW, dipping to the SSW. In the footwall to Chemalaco Zone a parallel dyke has been identified which is altered and mineralized. The Chemalaco 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 shale units. The Chemalaco 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 have been carried out in order to construct a paragenetic sequence of mineral formation. This work completed by Herrington (2011) and Staffurth (2012) reveals that veining occurs 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 40-60wt (weight) % gold but a few have down to 20wt% (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 (Ag3AuS2). This mineral is associated with electrum, chalcopyrite, galena, alabandite, silver minerals, and quartz in stage three mineralization (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 mineralization at Ixtaca does not have hard geologic boundaries. The mineralized 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 5km 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 has not been significantly affected by erosion. At Ixtaca, the sinter facies and replacement silicification, where preserved, are located within the altered volcanic units.

8.0 DEPOSIT TYPES

The principal deposit-type of interest on the Tuligtic Property is low- to intermediate- sulphidation epithermal gold-silver mineralization. This style of mineralization is recognised at the Ixtaca Zone but property scale high level epithermal alteration suggests that mineralization 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 (<1,500m) 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 mill feed 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 mill feed minor stockwork mill feed common; overlying sinter common; bonanza zones common	vein arrays; open space veins dominant; disseminated and replacement mill feed minor; stockwork mill feed common; productive veins may be km-long, up to 800m in vertical extent	veins subordinate, locally dominant; disseminated and replacement mill feed common; stockwork mill feed minor.
Alteration Zoning	mill feed with quartz-illite-adularia (argillic); barren silicification and propylitic (quartz-chlorite-calcite +/- epidote) zones; vein selvedges are commonly narrow	mill feed with sericite-illite (argillic-sericitic); deep base metal-rich (Pb-Zn +/- Cu) zone common; may be spatially associated with HS and Cu porphyry deposits	mill feed 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, especially with elevated base metals	chalcedony and opal uncommon; laminated colloform-crustiform veins uncommon; breccia veins; rhodochrosite uncommon
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 fluids 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 2km 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).

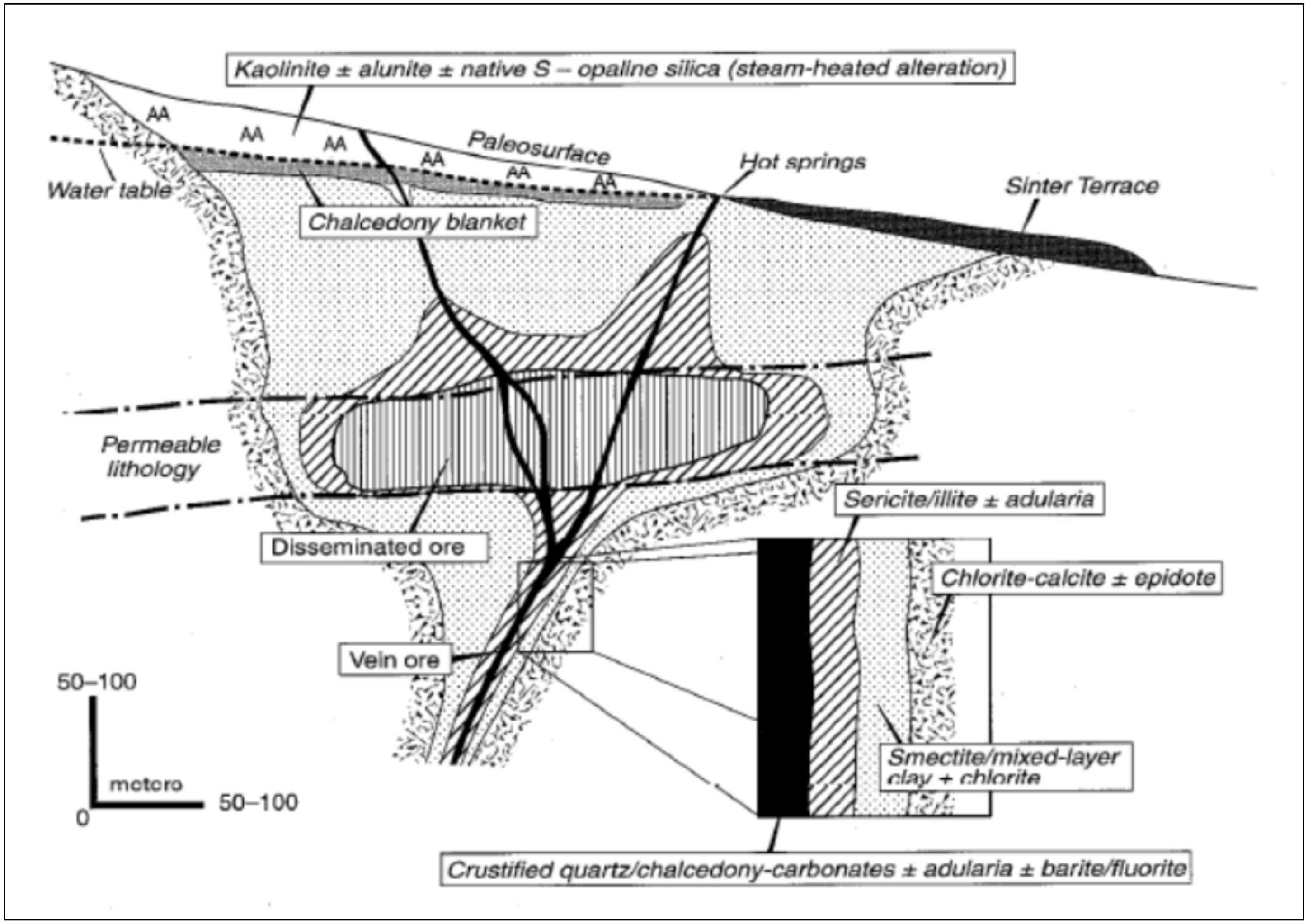
Above the water table H_2S condenses in the vadose zone to form a low pH H_2SO_4 (hydrogen-sulphate) dominant acid sulphate fluid (Hedenquist and White, 1990). These fluids can result in widespread tabular steam-heated alteration zones dominated by fine grained and friable kaolinite and alunite. Steam-heated waters collect at the water table and create aquifer-controlled strataform 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.

Mill feed zones are typically localized in structures, but may occur in permeable lithologies. Upward-flaring mill feed zones centred on structurally controlled hydrothermal conduits are typical. Large (bigger than 1m 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 mill feed shoots have relatively restricted vertical extent. High-grade ores are commonly found in dilational zones in faults at flexures, splays and in stockworks.

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 hot spring at surface to deeper, structurally and permeability focused fluid flow zones are the sites of mineralization. The mill feed 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.



**Hedenquist, 2000*

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

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. They also occur 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 oxidizing 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, mill feed 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).

9.0 EXPLORATION

Between 2004 and 2013, Almaden's exploration at the Tuligtic Property has 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. The work to date has resulted in the identification of five anomalous areas: the Ixtaca, Ixtaca East, Caleva, Azul, and Sol zones (Figure 7-2 and Figure 9-1). Detailed exploration results for the Tuligtic Property have been disclosed in a previous Technical Report for the Tuligtic Property by Raffle et al. (2013) and are summarized below.

9.1 Rock Geochemistry

Between 2004 and 2011 a total of 436 rock geochemical samples have been collected on the Property over a 6 x 6km area. Rock sampling, guided by concurrent soil geochemical surveys, has been concentrated around the Ixtaca Zone and an area extending 4km to the NNE over the copper porphyry target located between the Caleva and Azul zone soil geochemical anomalies (Figure 7-2, Figure 9-1).

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

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 10g/t silver (Ag) and up to 291g/t Ag.

Gold and silver mineralization occurs within the Ixtaca Zone, and is associated with anomalous arsenic, mercury (± antimony). To the northeast of the Ixtaca Zone zinc, copper and locally anomalous gold, silver and lead (± arsenic) values occur in association with calc-silicate skarn and altered intrusive rocks.

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 100m 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.

9.2 Soil Geochemistry

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 (Figure 7-2). During 2013, an additional 1,035 soil samples have been collected to extend soil grid lines to the west and locally infill existing grid lines, for a total of 5,795 soil samples.

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

Anomalous thresholds (greater than the 95th percentile) for gold and silver are calculated to be 17.1ppb Au and 0.59ppm Ag, respectively. A total of 288 samples contain anomalous Au, including 141 samples with coincident Ag anomalies.

The Ixtaca Zone produces the largest Au and Ag response within the Tuligtic Property (Figure 9-1). Base 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 (Figure 7-2, Figure 9-1). Soil responses are consistent with these zones being prospective for both epithermal and earlier porphyry-skarn mineralization.

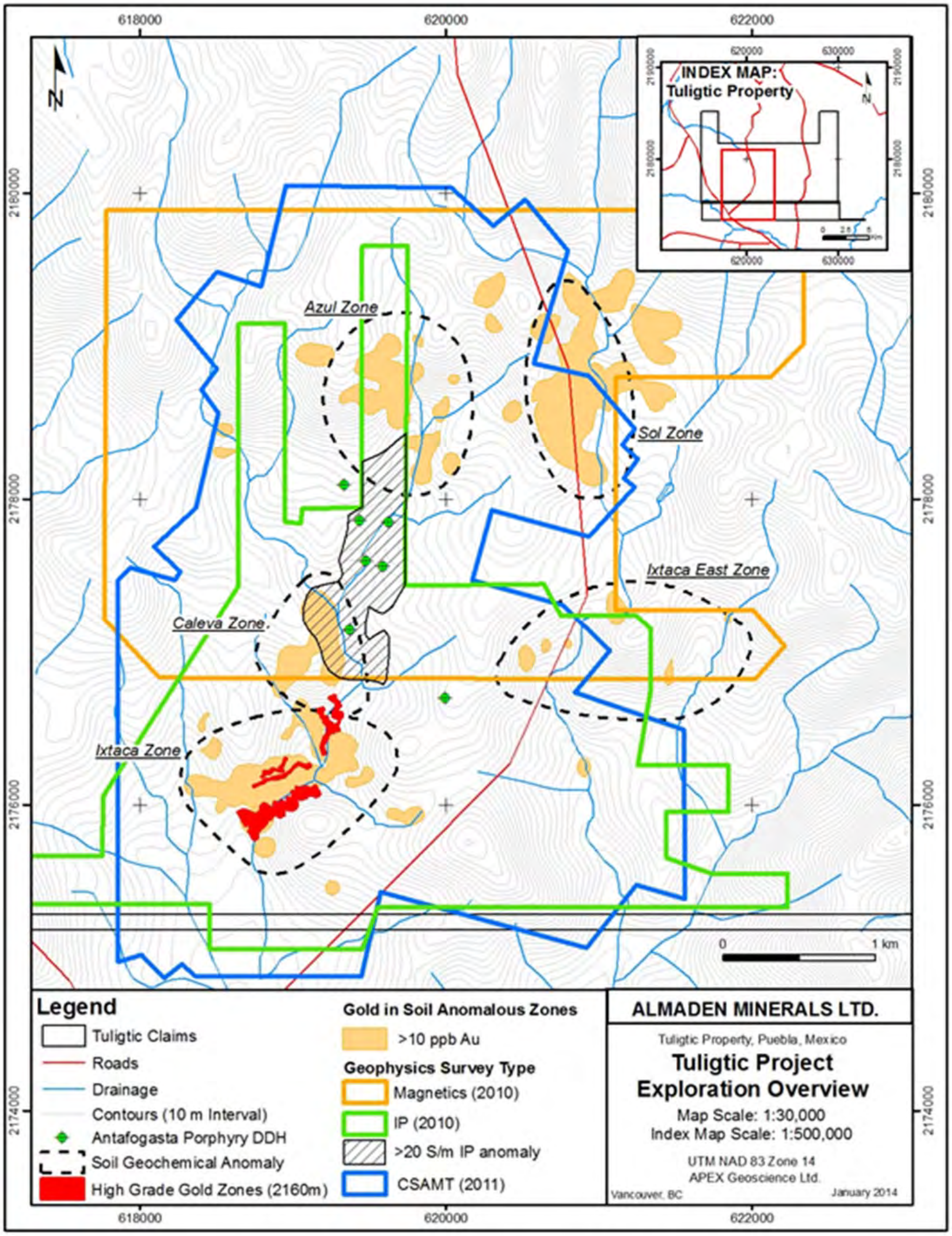


Figure 9-1 Exploration Overview Showing Gold in Soil Anomalies and Extent of Geophysical Surveys

9.3 Ground Geophysics

9.3.1 Magnetics

During 2010, Almaden completed an 84 line-km ground magnetic survey over a 4km by 4.5km area covering the copper porphyry target area north of the Ixtaca Zone (Figure 9-1). The survey comprised a series of 200m spaced east-west oriented lines with magnetic readings collected at 12.5m intervals along each line.

The survey identified a broad poorly defined, approximately 100 nano-Tesla (nT) magnetic high anomaly that corresponds in part with mapped altered quartz-monzonite porphyry rocks. Numerous, 30 to 50nT 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 suggesting structural and/or lithologic control of magnetic anomalies.

9.3.2 Induced Polarization/Resistivity

Concurrent with 2010 ground magnetic surveys, Almaden completed 108 line-km of 100m “a” spacing pole-dipole induced polarization (IP) / resistivity geophysical surveys over the Ixtaca and Cavela zones, and portions of the Azul and Ixtaca East zones (Figure 9-1). The survey employed a series of overlapping east-west and north-south oriented lines spaced at intervals of 100m.

The survey defines a 1,000 x 200m north-northwest trending 20 to 30mV/V chargeability anomaly coincident with mapped calc-silicate skarn mineralization and the Caleva Zone soil geochemical anomaly (Figure 9-1). While poorly constrained by a single north-south oriented survey line, the anomaly extends a further 1 km north over the porphyry copper anomaly area. Partial survey coverage of the Ixtaca East Zone multi-element soil geochemical anomaly defines a 700 x 500m elliptical 7 to 15mV/V chargeability anomaly along its western margin.

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.

9.3.3 CSAMT/CSIP

During 2011, Zonge International Inc. on behalf of Almaden completed a Controlled Source Audio-frequency Magnetotelluric (CSAMT) and Controlled Source Induce Polarization (CSIP) geophysical survey at the Tuligtic Property over a 6 by 4km area (Figure 9-1).

The survey totalled 48.5 line-km, including six lines oriented N-S (N16E azimuth, CSAMT and CSIP), and eight perpendicular E-W oriented lines (N104E azimuth, CSAMT only). Survey line spacing varied from 170 to 550m utilizing an array of six 25m dipoles.

2-D (N-S Line) smooth-model resistivity data defines a NW trending resistivity anomaly west of the Ixtaca Main Zone, and an E-W trending resistivity anomaly through the Ixtaca Zone. 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. 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.

2-D (E-W Line) smooth-model resistivity data shows a strong resistivity anomaly associated with the core of the Ixtaca Main Zone, and surface outcropping limestone. To the northeast, a resistivity anomaly coincident with the Chemalaco Zone may reflect complex structural geology patterns and the relatively resistive limestone and Chemalaco Dyke lithologies.

CSIP data does not appear to have identified significant anomalies.

10.0 DRILLING

10.1 Introduction

The purpose of the 2014 Technical Report is to provide a technical summary and updated mineral Resource Estimate with respect to the Ixtaca Deposit in relation to diamond drilling completed subsequent to the November 13, 2012 cut-off date of the maiden mineral Resource Estimate (Raffle et al., 2013). Since 2010, a total of 423 diamond drillholes have been drilled at the Tuligtic Property, totalling 137,438m (Figure 10-1). Drilling progress since 2010 is summarized below (Table 10-1).

The Main Ixtaca Zone of mineralization has been defined as a sub-vertical body trending northeast over a 650m strike length (Figure 10-1). The Ixtaca North Zone has been further defined over a 400m strike length as two discrete parallel sub-zones having a true-thickness of 5 to 35m, and spaced 20 to 70m apart (Figure 10-3). The Chemalaco Zone (Figure 10-1, Figure 10-4) is moderate to steeply WSW dipping that has been defined over a 450m strike length with high-grade mineralization intersected to a vertical depth of 600m or approximately 700m down-dip.

Table 10-1 Tuligtic Property Drilling Summary 2010-2013

Year	Holes Drilled (total m)	Main Ixtaca Zone	Ixtaca North Zone	Chemalaco Zone
2010	14 (6,465m)	- Discovered as sub-vertical body trending NE defined over 400m strike		
2011	85 (30,644m)	- Defined over 600m strike	- Discovered as parallel sub-vertical zone to Ixtaca Main	
2012	126 (44,862m)	- Defined over 650m strike - High-grade mineralization intersected to 300m	- Defined over 400m strike - High-grade mineralization intersected to 300m	- Discovered as a WSW moderate-steeply dipping body, defined over 350m strike, trending approximately N-S - High-grade mineralization intersected to 550m (600m down-dip)
2013	198 (55,467m)	- Tested over 1,000m strike - High-grade mineralization intersected to 300m	- Delineated as two distinct parallel zones - High-grade mineralization intersected to 32m	- Defined over 450m strike as splayed body dipping 55 degrees WSW with overall down-dip 700m - Splayed subzone dips 25-50 degrees, defined over 250m strike, 400m down-dip

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 302.42m of 1.01g/t Au and 48g/t Ag and multiple high grade intervals including 1.67m of 60.7g/t Au and 2,122g/t Ag. Almaden drilled 14 holes totalling 6,465m during 2010, defined the Main Ixtaca Zone over a 400m strike length, and initiated drilling along 50m NNW oriented sections. During 2011, Almaden drilled an additional 85 holes totalling 30,644m, which resulted in the discovery of the Ixtaca North Zone and testing of the Main Ixtaca Zone over a 600m strike length on 50m sections. Almaden discovered the Chemalaco Zone in early 2012 and continued drilling of the Ixtaca North and Main Ixtaca zones. Almaden drilled 126 holes totalling 44,862m on the Property from the beginning of 2012 until the November 13, 2012 maiden mineral Resource Estimate cut-off, for a total of 81,971m in 225 drillholes.

During 2013 and subsequent to the November 13, 2012 cut-off of the maiden mineral Resource Estimate, Almaden drilled 198 holes totalling 55,467m. A total of 79 holes have been drilled at the Main Ixtaca Zone, 40 holes at the Ixtaca North Zone and 79 holes at the Chemalaco Zone. Drilling during 2013 focused on expanding the deposit and upgrading resources previously categorized as Inferred to higher confidence Measured and Indicated categories.

Of the 423 holes to date, approximately 189 holes have been completed on the Main Ixtaca Zone, 112 at the Ixtaca North Zone, and 122 at the Chemalaco Zone (Figure 10-1). The diamond drillholes range from a minimum length of 60m to a maximum of 701m, and average 325m. All drilling completed at the Ixtaca Zone has been diamond core of NQ2 size (5.08 cm diameter). Drilling has been 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 2013 diamond drill programs have been completed under the supervision of Almaden personnel. Drillhole collars have been spotted using a handheld GPS and compass, and subsequently have been surveyed using a differentially corrected GPS. Each of the holes is marked with a small cement cairn inscribed with the drillhole number and drilling direction.

Drillholes have been surveyed down hole using Reflex EZ-Shot or EX-Trac instruments following completion of each hole. Down hole survey measurements have been spaced at 100m intervals during 2010 drilling and have been decreased to 50m intervals in 2011. During 2012 and 2013, select drillholes within all three mineralized zones have been surveyed at 15m intervals. A total of 4,672 drillhole orientation measurements (excluding 423 collar surveys) have been collected for an average down hole spacing of 27m. A total of 35 drillholes (10,354m), apart from the collar survey, have not been surveyed downhole; and a total of five drillholes (1,672m) have been surveyed at the end of hole only. Drillholes having no down hole survey have been assumed to have the orientation of the collar. Drillhole data has been plotted in the field and has been inspected. Down hole data returning unrealistic hole orientations have been flagged and removed from the database. Down hole survey summary statistics are provided in Table 10-2, below.

At the rig, drill core is placed in plastic core boxes labeled with the drillhole number, box number, and an arrow to mark the start of the tray and the down hole direction. Wooden core blocks are placed at the end of each core run (usually 3m, 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 10-2 Tuligtic Property Down Hole Survey Statistics

	Number of Drillholes	Metres
Number of Down Hole Surveys	4,672	137,438
Average Survey Spacing (not including casing)	423	27
Drillholes (No Down Hole Survey)	35 (8%)	10,354
Drillholes (End Of Hole Survey Only)	5 (1%)	1,672
Drillholes (15m Survey Spacing)	220 (52%)	67,276
Drillholes (50m Survey Spacing)	139 (33%)	49,047
Drillholes (100m Survey Spacing)	24 (6%)	9,089

Geotechnical logging is comprised of 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.
Drill core is logged based on lithology, and the presence of epithermal alteration and mineralization. All strongly altered or epithermal-mineralized intervals of core are sampled. Almaden employs a maximum sample length of 2 to 3m in unmineralized lithologies, and a maximum sample length of 1m in mineralized lithologies (50cm minimum sample length). Geological changes in the core such as major alteration or mineralization intensity (including large discrete veins), or lithology are 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 Chemalaco Zone are hosted by the shaley carbonate units and strike to the NNW, dipping to the SSW. In the footwall to Chemalaco Zone a parallel dyke has been identified which is altered and mineralized. The Chemalaco Zone and the dyke are interpreted to strike parallel to bedding and to core an antiform comprised of shale.

10.2 Main Ixtaca and Ixtaca North Zones

The Main Ixtaca and Ixtaca North zones have a strike length of approximately 650m and have been drilled at 25 and 50m section spacing. The vast majority of holes have been drilled at an azimuth of 150 or 330 degrees and at dips between 45 and 60 degrees from horizontal. Infill drilling at 25m sections has also been completed over the majority of the Ixtaca North Zone and 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 300m vertically from surface. High-grade zones occur within a broader zone of mineralization extending laterally (NNW-SSE) over 1000m and to a vertical depth of 600m below surface (Table 10-3 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 mineralization and trend 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 than 2m to over 20m true width occur within an approximately 100m wide zone (Figure 10-2, Figure 10-3). Wider dykes often correlate within individual drill sections, where they are inferred to pinch or splay. The broader dyke zone itself is relatable 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 20m 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 (Figure 10-2, Figure 10-3).

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. The synform 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.

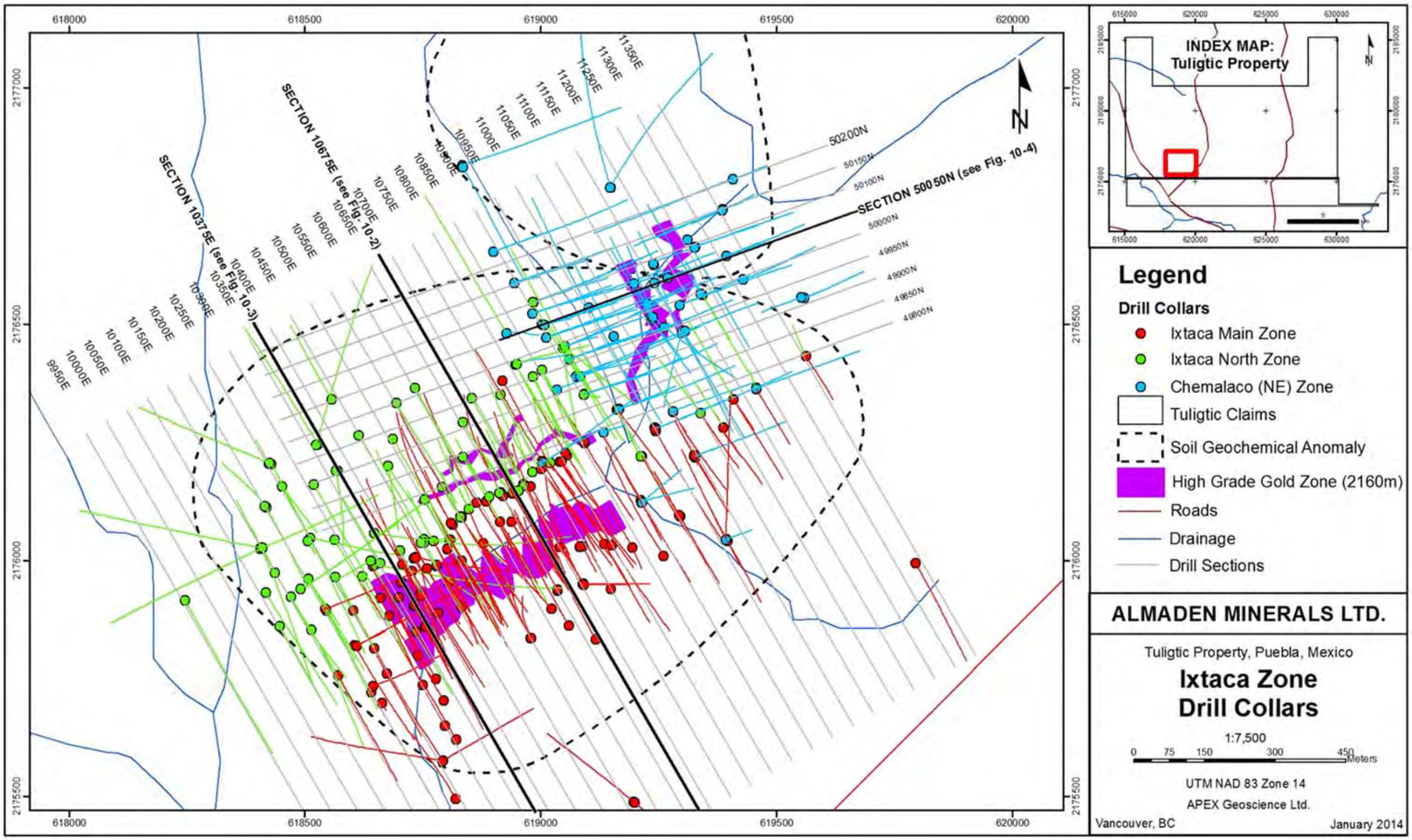


Figure 10-1 Drillhole Locations

Table 10-3 Section 10+675E Significant Drill Intercepts (Main Ixtaca and Ixtaca North Zones)

Hole ID	From (m)	To (m)	Interval (m)	Gold (g/t)	Silver (g/t)	AuEq* (g/t)
TU-12-120	260.90	290.90	30.00	0.74	96.7	**2.6**
including	260.90	266.10	5.20	2.78	437.0	**11.3**
TU-12-124	116.50	301.50	185.00	1.00	60.5	**2.2**
including	167.50	181.40	13.90	6.04	179.7	**9.5**
TU-12-127	155.95	186.00	30.05	0.70	56.7	**1.8**
including	174.00	186.00	12.00	1.05	105.7	**3.1**
TU-12-127	210.00	233.50	23.50	1.02	20.2	**1.4**
including	213.90	218.30	4.40	3.92	86.0	**5.6**
TU-12-127	243.00	285.60	42.60	0.57	10.8	**0.8**
TU-12-127	297.00	314.00	17.00	0.38	8.7	**0.5**
TU-12-132	64.50	204.20	139.70	0.22	18.0	**0.6**
including	137.00	166.60	29.60	0.35	27.8	**0.9**
including	148.25	153.30	5.05	1.16	79.0	**2.7**
including	174.40	204.20	29.80	0.33	34.1	**1.0**
TU-12-136	63.10	123.60	60.50	0.84	48.9	**1.8**
including	82.20	93.00	10.80	1.10	85.2	**2.8**
including	98.00	110.50	12.50	1.84	98.5	**3.8**
TU-12-138	43.50	87.27	43.77	0.59	4.3	**0.7**
including	61.00	71.50	10.50	0.88	4.9	**1.0**
including	84.00	87.27	3.27	2.07	10.5	**2.3**
TU-12-138	135.50	184.25	48.75	0.22	16.7	**0.5**
including	179.95	182.50	2.55	2.98	216.4	**7.2**
TU-12-138	202.00	359.50	157.50	0.36	41.4	**1.2**
including	264.30	359.50	95.20	0.54	61.1	**1.7**
including	292.50	302.00	9.50	1.27	234.3	**5.8**
including	304.00	307.00	3.00	3.87	439.9	**12.4**
TU-12-144	45.50	92.60	47.10	0.52	3.7	**0.6**
TU-12-144	210.00	258.00	48.00	0.52	32.0	**1.1**
including	227.40	235.80	8.40	1.68	59.3	**2.8**
TU-13-324	32.92	62.00	29.08	1.31	16.5	**1.6**
including	42.50	57.75	15.25	2.10	23.7	**2.6**
including	43.00	45.25	2.25	1.71	72.0	**3.1**
TU-13-324	113.50	128.00	14.50	0.25	47.0	**1.2**
including	120.00	121.00	1.00	0.59	117.5	**2.9**
including	125.00	128.00	3.00	0.79	155.0	**3.8**
TU-13-324	154.00	174.00	20.00	0.08	29.1	**0.6**
including	160.00	161.00	1.00	0.42	167.0	**3.7**
including	167.50	172.00	4.50	0.07	53.4	**1.1**
TU-13-325	128.50	136.50	8.00	0.58	132.2	**3.2**
TU-13-325	190.00	236.50	46.50	1.06	53.1	**2.1**
including	193.40	216.00	22.60	1.72	97.2	**3.6**
including	194.00	195.20	1.20	2.05	147.0	**4.9**
including	203.90	205.00	1.10	3.97	175.0	**7.4**
including	210.50	216.00	5.50	4.40	240.8	**9.1**

Hole ID	From (m)	To (m)	Interval (m)	Gold (g/t)	Silver (g/t)	AuEq* (g/t)
TU-13-388	199.00	229.50	30.50	0.67	23.9	**1.1**
TU-13-388	337.50	346.50	9.00	1.35	287.5	**6.9**
including	339.25	340.35	1.10	6.54	1982.7	**45.2**
TU-13-388	363.50	416.00	52.50	0.58	50.3	**1.6**
including	363.50	378.40	14.90	0.74	87.0	**2.4**
including	372.00	378.40	6.40	1.19	138.9	**3.9**
including	390.00	403.90	13.90	1.11	82.9	**2.7**
including	398.60	401.10	2.50	1.78	173.0	**5.1**

Table 10-4 Section 10+375E Significant Drill intercepts (Main Ixtaca Zone)

Hole ID	From (m)	To (m)	Interval (m)	Gold (g/t)	Silver (g/t)	AuEq* (g/t)
TU-11-065	26.00	126.80	100.80	0.58	46.2	**1.5**
including	26.00	74.78	48.78	0.95	77.0	**2.5**
including	43.60	68.00	24.40	1.67	134.4	**4.4**
including	49.80	59.80	10.00	3.05	198.8	**7.0**
TU-11-067	24.30	145.00	120.70	1.02	72.6	**2.5**
including	36.50	136.80	100.30	1.20	85.0	**2.9**
including	54.90	96.30	41.40	1.91	144.1	**4.8**
including	63.55	85.50	21.95	2.75	210.1	**7.0**
including	65.60	80.85	15.25	3.26	253.4	**8.3**
including	107.20	116.95	9.75	2.54	112.6	**4.8**
including	125.55	127.43	1.88	2.51	242.2	**7.3**
TU-12-202	26.50	66.50	40.00	0.35	1.4	**0.4**
including	26.50	38.00	11.50	0.78	0.5	**0.8**
TU-12-202	137.10	172.50	35.40	0.62	12.3	**0.9**
including	139.10	145.10	6.00	2.57	35.4	**3.3**
TU-12-202	249.30	260.80	11.50	0.10	16.7	**0.4**
TU-12-211	31.20	187.85	156.65	0.59	28.6	**1.2**
including	70.70	84.50	13.80	0.97	82.9	**2.6**
including	97.80	105.65	7.85	1.07	59.4	**2.3**
including	129.85	142.40	12.55	1.38	53.3	**2.4**
including	172.85	183.85	11.00	0.91	56.7	**2.0**
TU-13-389	21.34	95.50	74.16	1.02	50.9	**2.0**
including	47.00	71.00	24.00	1.52	60.6	**2.7**
including	51.50	69.00	17.50	1.92	64.4	**3.2**
including	88.60	95.50	6.90	2.54	139.9	**5.3**
TU-13-389	104.00	106.80	2.80	2.86	169.3	**6.2**
TU-13-391	16.00	126.00	110.00	0.62	42.0	**1.5**
including	48.16	89.50	41.34	1.16	76.2	**2.7**
including	48.16	59.30	11.14	1.79	110.9	**4.0**
including	71.80	84.50	12.70	1.40	106.4	**3.5**
including	71.80	74.50	2.70	3.06	230.3	**7.7**
TU-13-393	27.43	141.80	114.37	0.92	53.7	**2.0**
including	54.50	81.50	27.00	1.03	76.0	**2.6**
including	56.00	62.20	6.20	2.21	150.5	**5.2**
including	89.95	124.70	34.75	1.67	70.4	**3.1**
including	100.30	104.00	3.70	2.08	89.0	**3.9**
including	110.40	118.30	7.90	4.42	158.7	**7.6**

**Gold Equivalent based on a three-year trailing average price of $1,540/ounce gold and $30/ounce silver*

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 200m (Table 10-4 and Figure 10-3).

Mineralized limestone, shale and the cross-cutting dykes are unconformably overlain by bedded crystal tuff, which is also mineralized. Mineralization within 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.

10.3 Chemalaco Zone

The Chemalaco Zone (also known as the Northeast Extension) of the Ixtaca deposit has an approximate strike length of 450m oriented roughly north-south (340 azimuth) and has been drilled via a series of ENE (070 degrees) oriented sections spaced at intervals of 25 to 50m, and near-surface oblique NNW-SSE oriented drillholes (Figure 10-1). The Chemalaco Zone dips moderately-steeply at 55 degrees WSW. High grade mineralization having a true-width ranging from less than 30 and up to 60m has been intersected beneath approximately 30m of tuff to a vertical depth of 550m, or approximately 700m down-dip. An additional sub-parallel zone has been defined underneath the Chemalaco having a true-width ranging from 5 to 40m and dipping 25 to 50 degrees to the WSW, resulting in a splayed zone extending from near-surface to a vertical depth of 250m. The sub-parallel zone has an approximate down-dip length up to 400m over a 250m strike length (Table 10-5, Figure 10-4).

The Chemalaco Zone vein lies northeast of the Main Ixtaca Zone and occurs within the hinge zone of a shale cored antiform. Near surface, along the axis of the antiform, a zone of structurally thinned, brecciated, and mineralized limestone is unconformably overlain by mineralized tuff rocks (Figure 10-4). At a vertical depth of 80m below surface, high-grade shale-hosted mineralization dips moderately-steeply at 25 to 55 degrees 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 30m true-thickness felsic porphyry dyke (Chemalaco Dyke), which is also mineralized. The Chemalaco Dyke has been interested in multiple drillholes ranging from 250 to 550m vertically below surface, and its lower contact currently marks the base of Chemalaco Zone mineralization.

Table 10-5 Section 50+050N Significant Drill intercepts (Chemalaco Zone)

Hole ID	From (m)	To (m)	Interval (m)	Gold (g/t)	Silver (g/t)	AuEq* (g/t)
TU-12-190	85.00	89.00	4.00	0.25	0.5	0.3
TU-12-190	100.00	112.00	12.00	0.17	1.9	0.2
TU-12-190	259.00	272.90	13.90	0.17	12.3	0.4
TU-12-190	278.85	321.00	42.15	1.06	47.4	2.0
including	293.50	300.50	7.00	1.34	72.0	2.7
including	306.00	317.80	11.80	1.67	71.7	3.1
including	310.00	314.00	4.00	2.45	116.4	4.7
TU-12-190	377.90	386.00	8.10	0.24	2.8	0.3
TU-12-194	83.50	87.50	4.00	0.46	2.8	0.5
TU-12-194	112.60	124.00	11.40	0.22	4.4	0.3
TU-12-194	272.50	279.50	7.00	0.15	40.9	0.9
TU-12-194	294.50	300.00	5.50	0.14	81.1	1.7
TU-12-194	313.00	371.80	58.80	1.04	19.4	1.4
including	317.60	347.00	29.40	1.63	23.9	2.1
TU-12-199	66.00	70.00	4.00	0.26	2.4	0.3
TU-12-199	91.00	93.80	2.80	0.19	3.0	0.2
TU-12-199	344.20	424.00	79.80	0.84	20.6	1.2
including	365.70	385.70	20.00	1.19	25.6	1.7
including	396.50	402.50	6.00	1.43	16.0	1.7
including	408.30	423.40	15.10	1.48	37.6	2.2
including	414.30	416.10	1.80	4.90	175.5	8.3
TU-12-205	81.00	132.00	51.00	0.51	6.0	0.6
including	101.50	106.00	4.50	3.41	6.1	3.5
TU-12-205	254.50	293.50	39.00	0.61	88.8	2.3
including	255.50	281.20	25.70	0.86	127.8	3.3
including	256.00	272.40	16.40	1.08	164.8	4.3
including	256.00	265.00	9.00	1.57	244.5	6.3
TU-12-205	312.00	319.00	7.00	0.19	207.2	4.2
TU-13-265	488.40	531.80	43.40	0.50	9.2	0.7
including	500.60	507.20	6.60	2.15	11.6	2.4
including	504.20	507.20	3.00	3.36	17.1	3.7
TU-13-265	539.00	545.00	6.00	0.07	22.2	0.5
TU-13-265	550.30	558.00	7.70	0.07	28.1	0.6
TU-13-268	41.30	56.25	14.95	0.05	11.5	0.3
TU-13-268	61.25	120.50	59.25	0.11	41.1	0.9
including	74.90	79.75	4.85	0.25	126.9	2.7
including	103.00	106.00	3.00	0.23	81.2	1.8
TU-13-268	133.00	138.00	5.00	0.03	22.3	0.5
TU-13-268	151.50	208.00	56.50	0.36	42.0	1.2
including	166.00	178.50	12.50	0.56	91.4	2.3
including	166.00	167.50	1.50	0.74	223.7	5.1
including	192.00	199.50	7.50	0.75	51.6	1.8
TU-13-268	222.75	239.00	16.25	0.08	14.6	0.4
TU-13-272	48.00	138.50	90.50	0.20	31.4	0.8
including	66.05	70.20	4.15	0.44	49.5	1.4

Hole ID	From (m)	To (m)	Interval (m)	Gold (g/t)	Silver (g/t)	AuEq* (g/t)
including	77.50	84.80	7.30	0.29	71.1	**1.7**
including	112.75	119.75	7.00	0.43	40.1	**1.2**
including	129.00	138.50	9.50	0.41	114.0	**2.6**
TU-13-272	146.00	161.00	15.00	0.22	47.1	**1.1**
including	147.00	148.50	1.50	0.65	252.7	**5.6**
TU-13-272	187.00	193.50	6.50	0.11	11.5	**0.3**
TU-13-272	220.00	231.00	11.00	0.14	9.5	**0.3**
TU-13-275	68.50	84.00	15.50	0.15	10.6	**0.4**
TU-13-275	105.00	112.00	7.00	0.11	15.8	**0.4**
TU-13-275	120.00	134.50	14.50	0.18	6.2	**0.3**
TU-13-275	149.00	227.00	78.00	0.39	23.8	**0.9**
including	164.50	193.50	29.00	0.43	43.3	**1.3**
TU-13-275	254.00	258.00	4.00	0.01	13.5	**0.3**
TU-13-287	106.00	131.00	25.00	0.11	15.2	**0.4**
including	122.00	125.00	3.00	0.30	50.3	**1.3**
TU-13-287	156.50	182.00	25.50	0.66	102.3	**2.7**
including	168.00	170.08	2.08	4.35	975.0	**23.3**
TU-13-289	134.00	153.00	19.00	0.22	48.4	**1.2**
including	144.50	151.80	7.30	0.40	82.8	**2.0**
TU-13-289	160.00	188.00	28.00	0.21	10.8	**0.4**

**Gold Equivalent based on a three-year trailing average price of $1,540/ounce gold and $30/ounce silver*

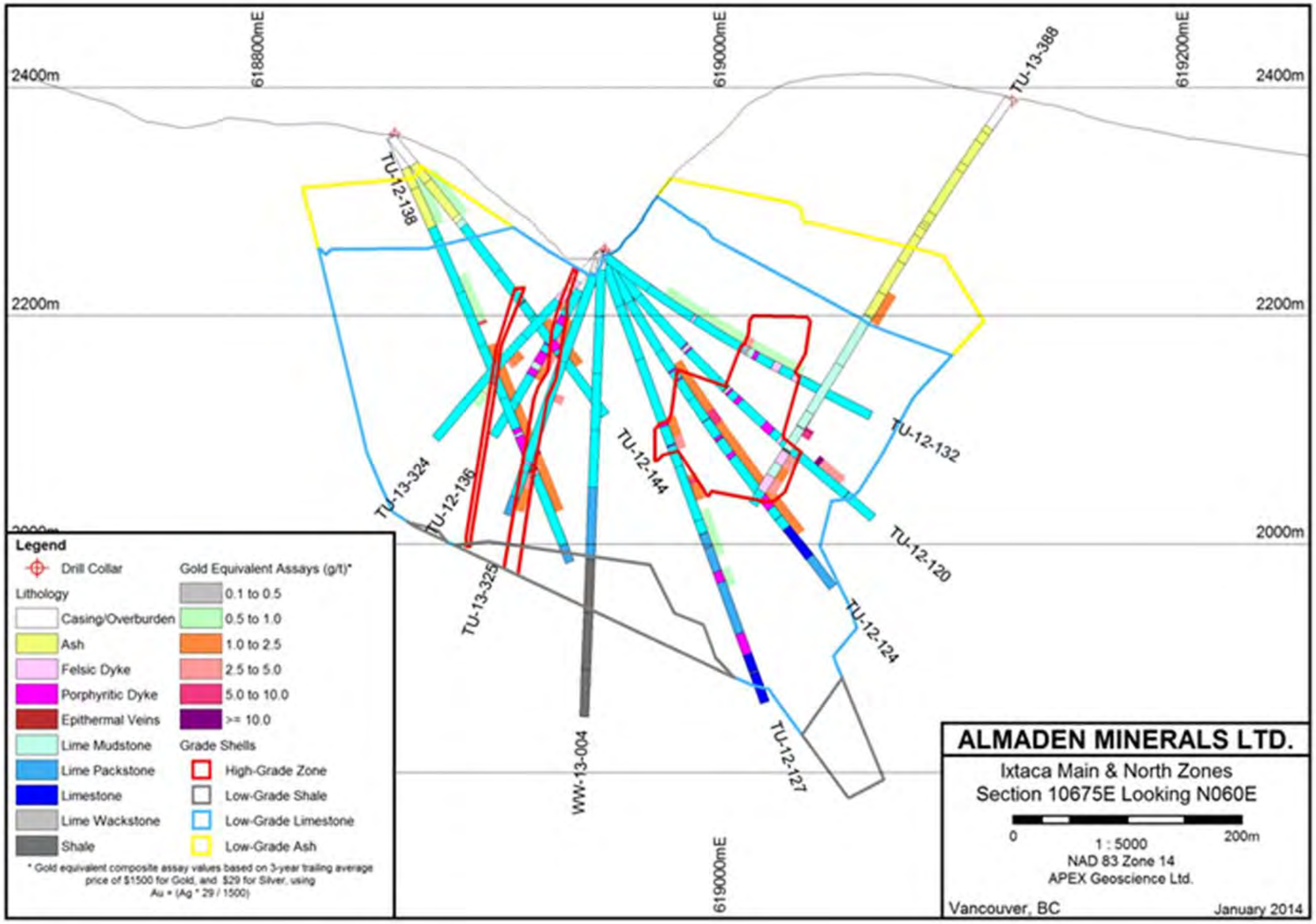


Figure 10-2 Section 10+675E through the Ixtaca Main and North Zones

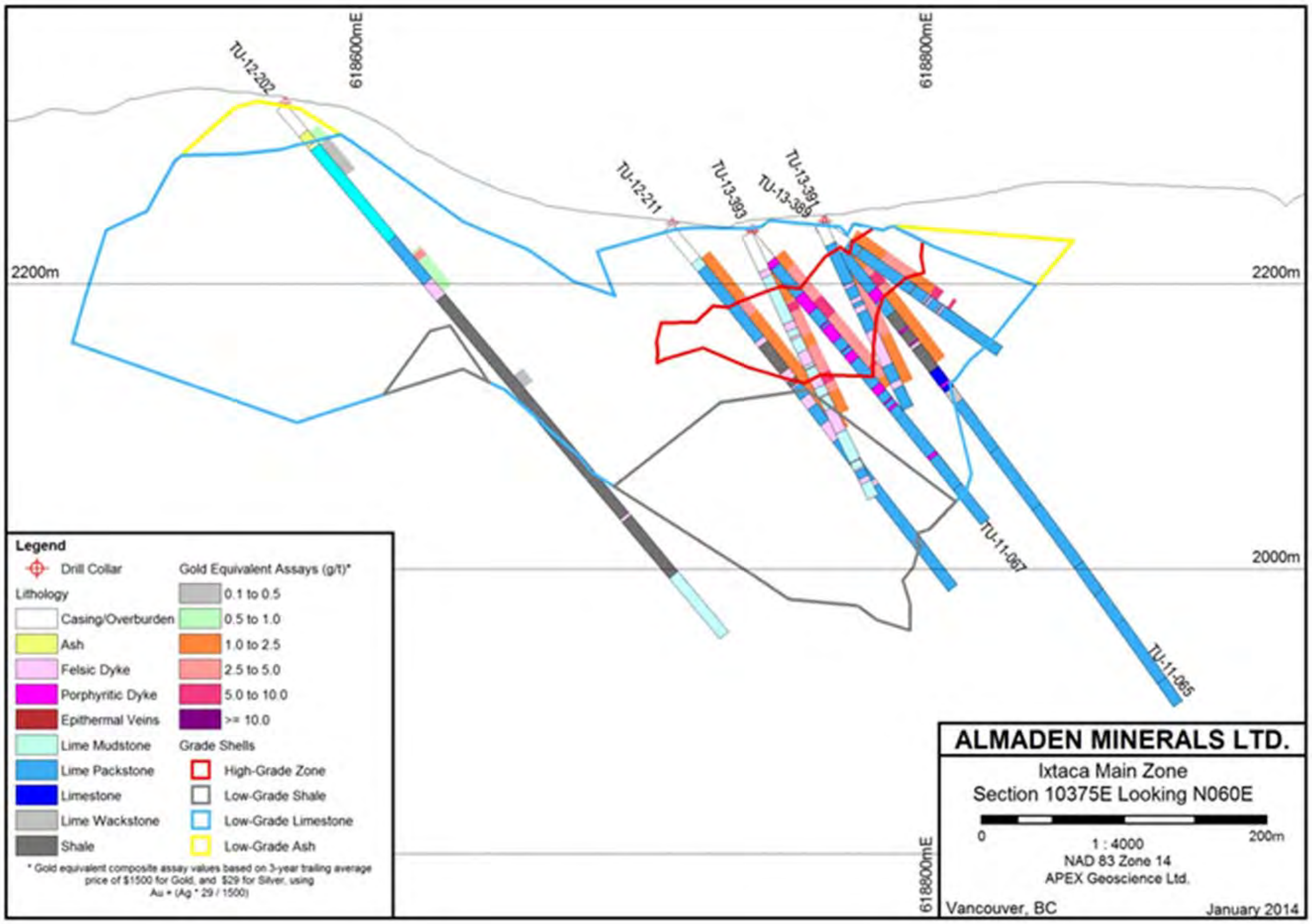


Figure 10-3 Section 10+375E through the Ixtaca Main Zone

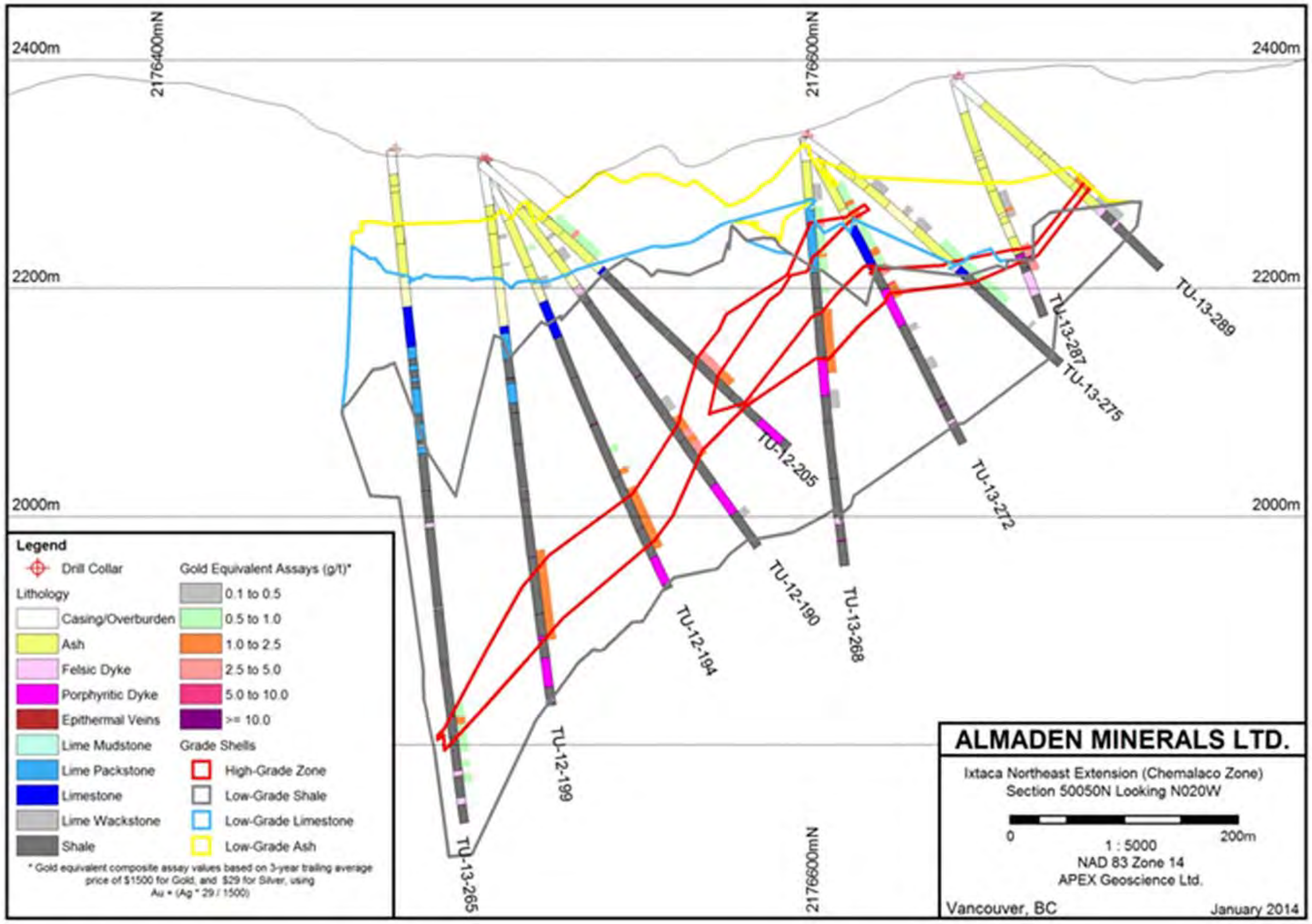


Figure 10-4 Section 50+050N through the Chemalaco Zone

11.0 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 have been transported by Almaden field personnel to the Santa Maria core facility where they are 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 Maria 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 are 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 and Mr. Kristopher J. Raffle, P.Geo., has no reason to believe that the security of the samples has been compromised.

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

Rock grab samples are 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 0.005ppm Au (5 ppb) and upper limit of 10ppm 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 6mg of gold-free silver and then cupelled to yield a precious metal bead. The bead is digested in 0.5ml dilute nitric acid and 0.5ml concentrated hydrochloric acid. The digested solution is cooled, diluted to a total volume of 4ml with de-mineralized water, and analyzed by atomic absorption spectroscopy against matrix-matched standards.

Soil samples are subject to gold determination via is digestion of a 50g 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 are analyzed by 33-element inductively coupled plasma atomic emission spectroscopy (ICP-AES), with a 4-acid digestion (ALS method ME-ICP61). A 0.25g 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 have been sampled. Almaden employs a maximum sample length of 2 to 3m in unmineralized lithologies, and a maximum sample length of 1m in mineralized lithologies (50cm minimum sample length). Sampling always begins at least five samples above the start of mineralization. Geological changes in the core such as major alteration or mineralization intensity (including large discrete veins), or lithology are used as sample breaks.

Drill core is half-sawn using industry standard gasoline engine-powered diamond core saws, with 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 <1cm) are sampled using spoons. In all cases, the right hand side of the core (looking down the hole) is sampled. After cutting, half the core is placed in a new plastic sample bag and half is placed back in the core box. Between each sample, the core saw and sampling areas are washed to ensure no contamination between samples. Field duplicate, blank and analytical standards are added into the sample sequence as they are being cut.

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

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

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

Over limit gold values (>10ppm Au) are subject to gravimetric analysis, whereby a 50g 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 have been analyzed by 33- element ICP-AES, with a 4-acid digestion, a lower detection limit of 0.5ppm Ag and upper detection limit of 100ppm Ag (ALS method ME-ICP61). A 0.25g 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 (>100ppm Ag) have been subject to 4-acid digestion ICP-AES analysis with an upper limit of 1,500ppm 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 100ml 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 (>1,500ppm Ag) are subject to gravimetric analysis with an upper detection limit of 10,000ppm Ag (Ag-GRA22).

11.1.3 Author's Drill Core

The collected drill core samples have been placed into sealed plastic bags and transported by Mr. Kristopher J. Raffle, P.Geo., (considered "the author" in this Section of the report) 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; however the author has no reason to believe that the security of the samples has been compromised.

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

Drill core samples collected by Kristopher J. Raffle, P.Geo., have been subject to gold determination via a 50 gram (g) AA finish FA fusion with a lower detection limit of 0.005ppm Au (5ppb) and upper limit of 10ppm Au (ALS method Au-AA24). A 50g prepared sample is fused with a flux mixture, inquarted with 6mg of gold-free silver and then cupelled to yield a precious metal bead. The bead is digested in 0.5mL dilute nitric acid and 0.5mL concentrated hydrochloric acid. The digested solution is cooled, diluted to a total volume of 4mL 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 are analyzed by 33-element inductively coupled plasma atomic emission spectroscopy (ICP-AES), with a 4-acid digestion. A 0.25g 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 (>100ppm Ag) are subject to 4-acid digestion, ICP-AES analysis with an upper limit of 1,500ppm 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 100ml 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 Tuligtic rock grab sample and soil geochemical programs, Almaden relies 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 reviewed 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 five samples preceding and five samples succeeding the reviewable QA/QC sample have been 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 Mr. Raffle's opinion, Almaden's QA/QC procedures are reasonable for this type of deposit and the current level of exploration. A total of 12,873 QA/QC analytical standard and blank samples have been submitted for analysis. 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 mineral Resource Estimate.

11.2.1 Analytical Standards

A total of 17 different analytical standards have been used on the Project. Since November 13, 2012 and drillhole TU-12-221 (the end of the Maiden Resource Estimate cut-off), 7 different analytical standards have been used and are the basis for the section herein. Please refer to the 2013 Almaden NI 43-101 (Raffle et al. 2013) report for a detailed discussion of the previously used standards.

Each standard has an accepted gold and silver concentration as well as known "between laboratory" standard deviations, or expected variability, associated with each standard. The standards include seven multi-element gold-silver standards with accepted values ranging from 0.564 to 3.88g/t Au, and 14.4 to 103.0g/t Ag. One analytical standard for every 20 samples (5%) is 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 11-1 below.

Between 2010 and 2013 Almaden employed two separate criteria by which standards have been assigned "pass" or "reviewable" status.

Up to drillhole TU-12-130 a reviewable standard had been 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. Beginning with drillhole TU-12-154, standards returning *less than* three (3) standard deviations (3SD) below the accepted value for gold or silver have also been flagged for review. Failed standards occurring outside of reported mineralized intercepts have not been flagged as reviewable. Beginning with drillhole TU-12-131, a reviewable standard is now defined as any standard occurring anywhere in a drillhole returning >3SD above or below the accepted value for gold or silver. In addition, two standards analyzed consecutively returning values >2SD above or below the accepted value for the same element (gold or silver) are classified as reviewable.

All standard samples returning gold or silver values outside the established criteria are reviewed. A decision to conduct reanalysis of samples surrounding the reviewable standard is based on whether the standard returned a value above or below the accepted value (low, or slightly high >3SD values are allowed after data review) or if it occurred within a reported interval (>3SD values are allowed outside of reported intervals) Prior to August 7, 2012, when a reviewable standard has been recognized the five preceding and five succeeding samples, in addition to the standard have been subject to review and possibly re-analysis. After August 7, 2012 when a reviewable standard is recognized, the ten preceding and ten succeeding samples, in addition to the standard is subject to review and possibly re-analysis. The results of re-analysis are 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 is considered resolved and the re-analyzed standard value and surrounding reanalyzed samples are added to the drillhole database.

A total of 6,438 analytical standards have been inserted into the sample stream of 109,570 assays for gold and silver for the 423 drillholes. Of the 6,438 standards, a total of 2,356 have been subject to review criteria in place up to drillhole TU-12-130. Of the remaining 4,082 samples subject to the current review criteria (TU-12-131 and later), 1,708 samples have been included in the maiden mineral Resource Estimate up to hole TU-12-221 (Raffle et al., 2013). QA/QC results with respect to the remaining 2,374 standards are reported herein (TU-12-222 and later).

Of the 2,374 QA/QC samples inserted into the sample stream since November 13, 2012, a total of 162 (6.8%) have been initially 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. These standards have been re-analysed and all but four passed the repeat analysis (Figure 11-1). Of the four (4) re-analysis failures, one (1) has been outside the reported mineralized interval, and the remaining three (3) assayed within the accepted value for Au but failed the accepted value for Ag. Of these three re-analysis failures, two have been consecutive standards that returned >2SD from the accepted Ag value and one re-assayed >3SD for the accepted Ag value.

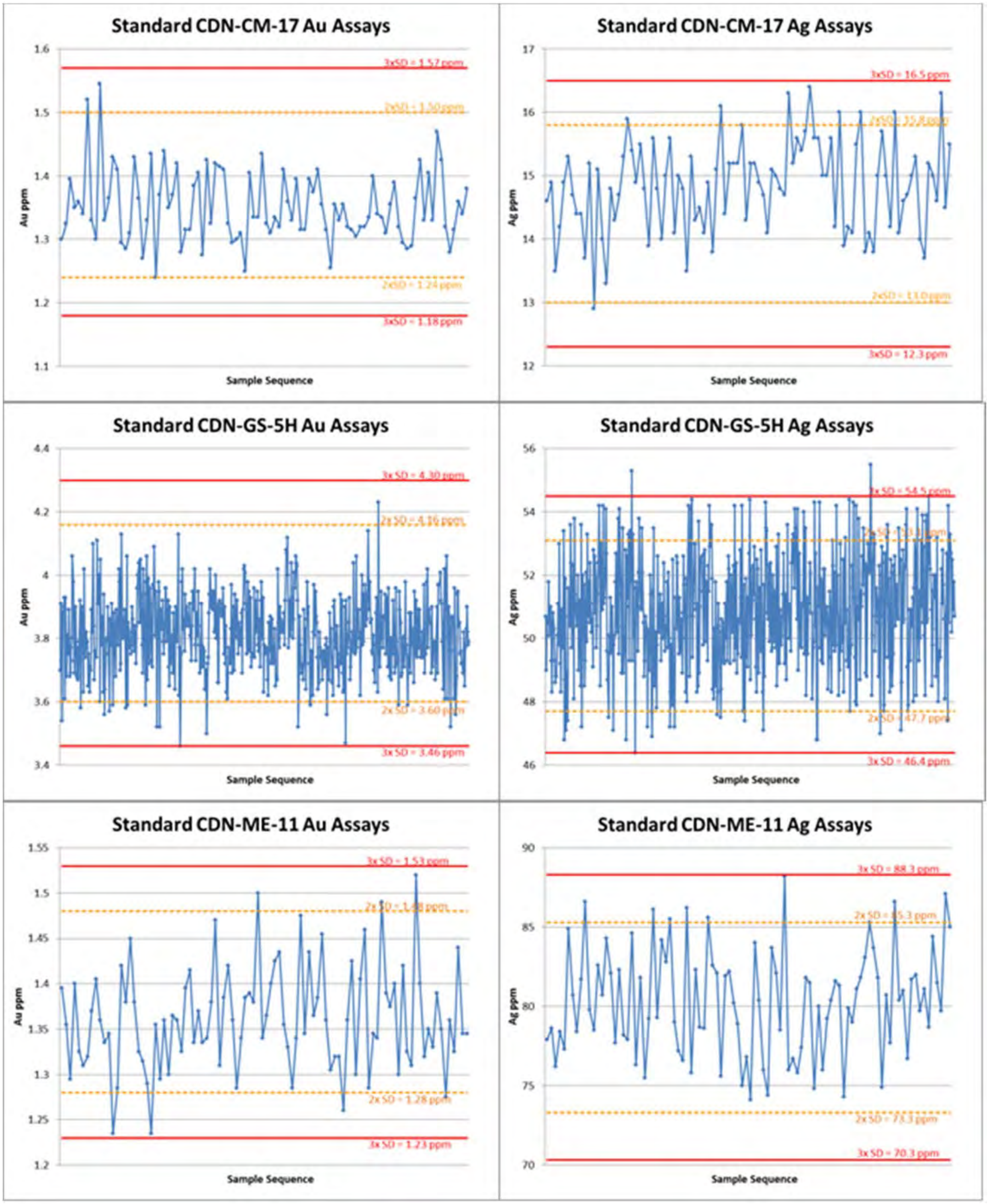


Figure 11-1 QA/QC Analytical Standards

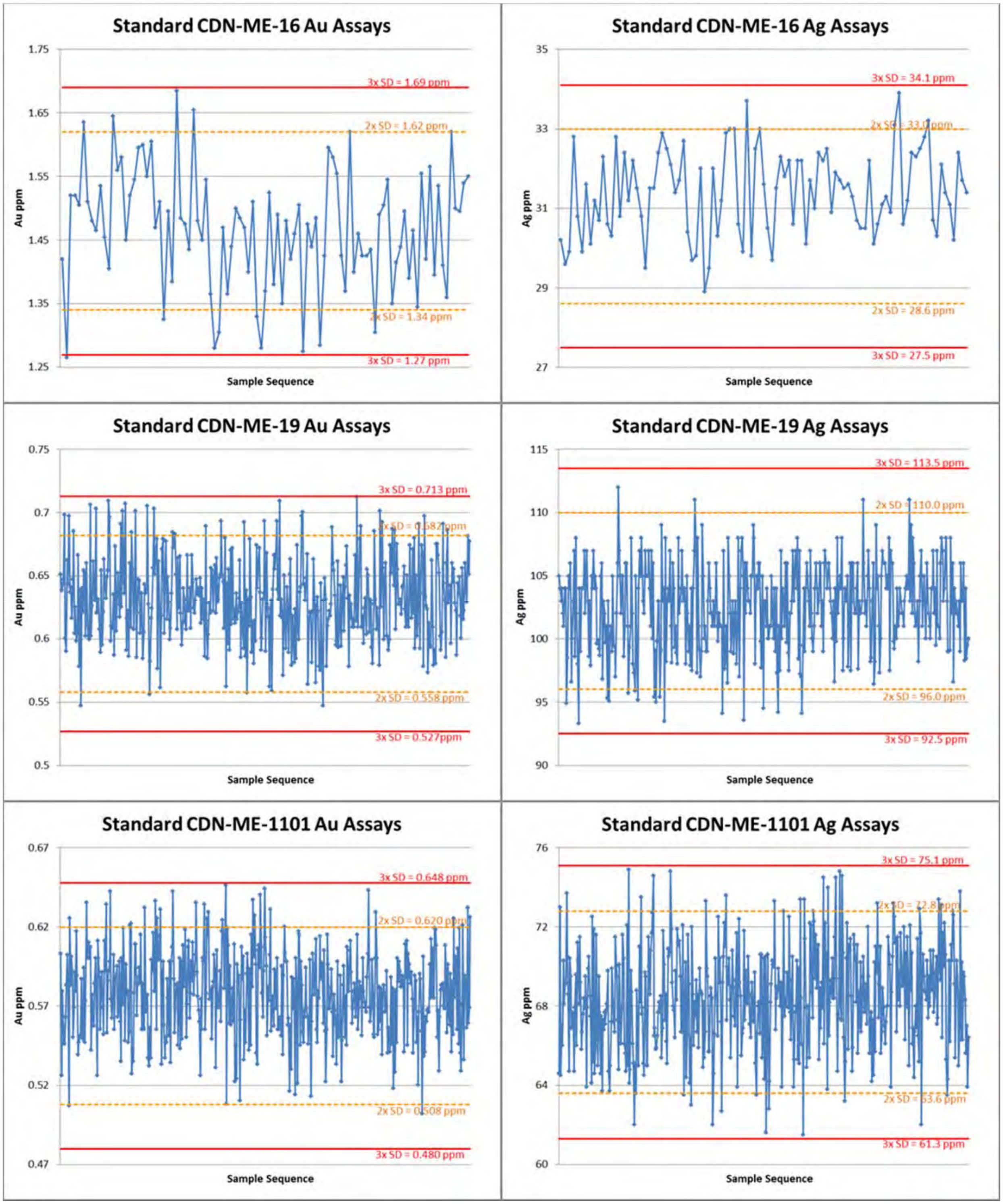


Figure 11-1 QA/QC Analytical Standards cont…

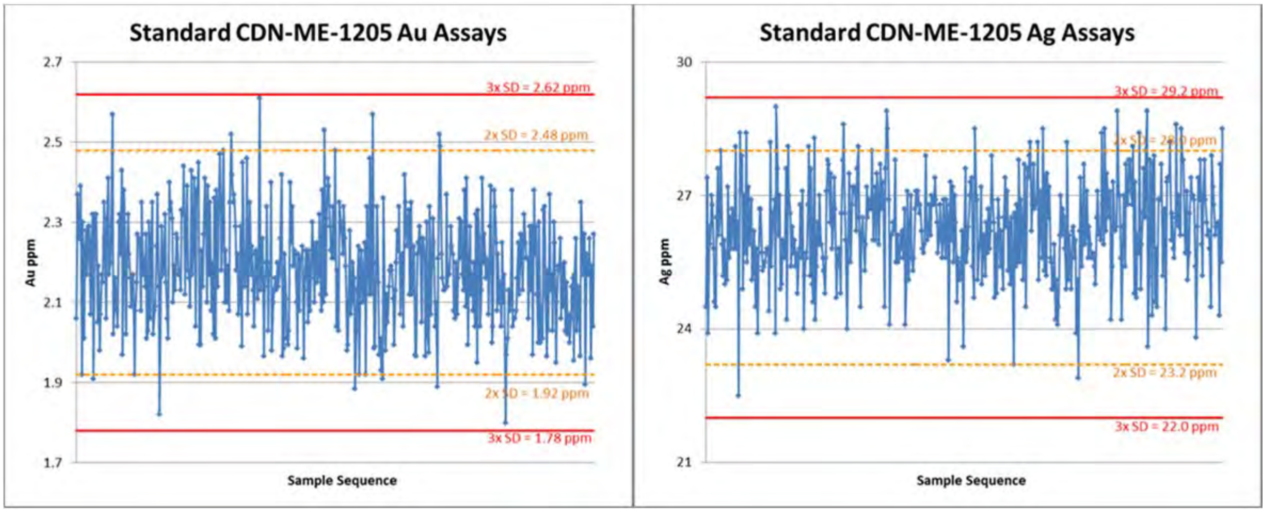


Figure 11-1 QA/QC Analytical Standards cont…

11.2.2 Blanks

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

Prior to August 7, 2012, reviewable blank samples occurring outside a reported mineralized intercept have not been 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 five samples on either side have been re-analyzed. To provide additional confidence, on August 7, 2012, Almaden increased the number of samples re-analyzed to ten samples on either side of the blank in question. The results of re-analysis are 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 does not return values above the accepted limits; the QA/QC concern is considered resolved and the re-analyzed blank value and surrounding reanalyzed samples are added to the drillhole database.

Of the 2,374 blank samples analyzed since November 13, 2012, a total of nine blanks have returned assays greater than the accepted values of 50ppb Au and 5ppm Ag. Of these, eight blanks have returned greater than 50ppb Au, and seven blanks returned greater than 5ppm Ag. These blanks occurred within mineralized intervals, and as such have been re-assayed. When re-assayed, all blanks except one sample returned values below the accepted values for Au and Ag (Figure 11-2). The single remaining failed blank sample immediately follows a high grade sample that returned an assay of 5,310ppm Ag and in this case it is reasonable that a certain amount of carryover occurred.

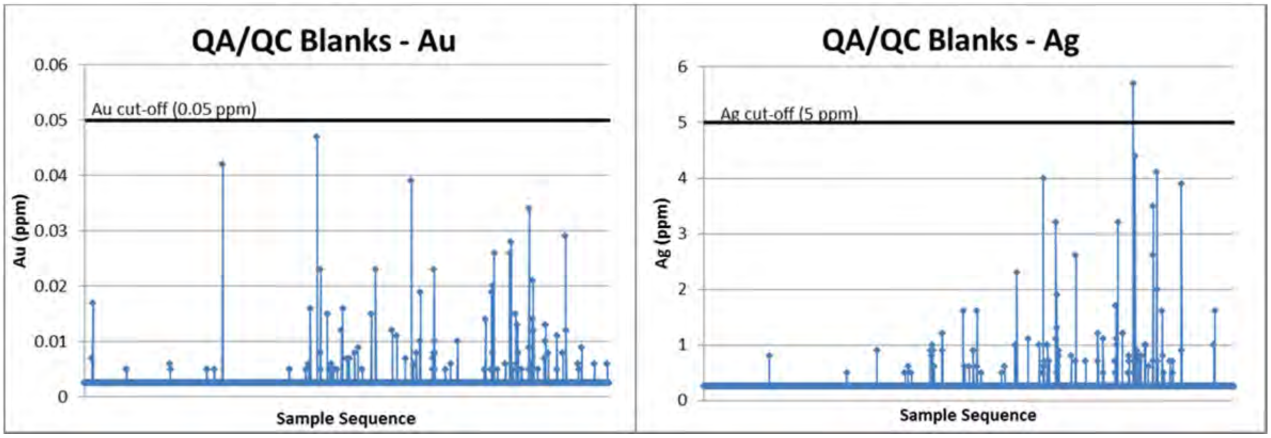


Figure 11-2 QA/QC Blanks

11.2.3 Duplicates

Quartered-core duplicate samples are collected to assess the overall repeatability of individual analytical values. One core duplicate for every 20 samples (5%) is inserted into the sample stream at the '15', '35', '55', '75', and '95' positions. A total of 2,369 quarter-core duplicates have been inserted into the sample stream beginning with drillhole TU-12-222.

As part of their internal QA/QC program, ALS completes routine re-analysis of prep (coarse reject) and pulp duplicates to monitor precision. ALS analyzed a total of 772 prep duplicates for gold, and 798 for silver. A total of 1,836 pulp duplicates have been analyzed for gold and 1,459 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). The data 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.

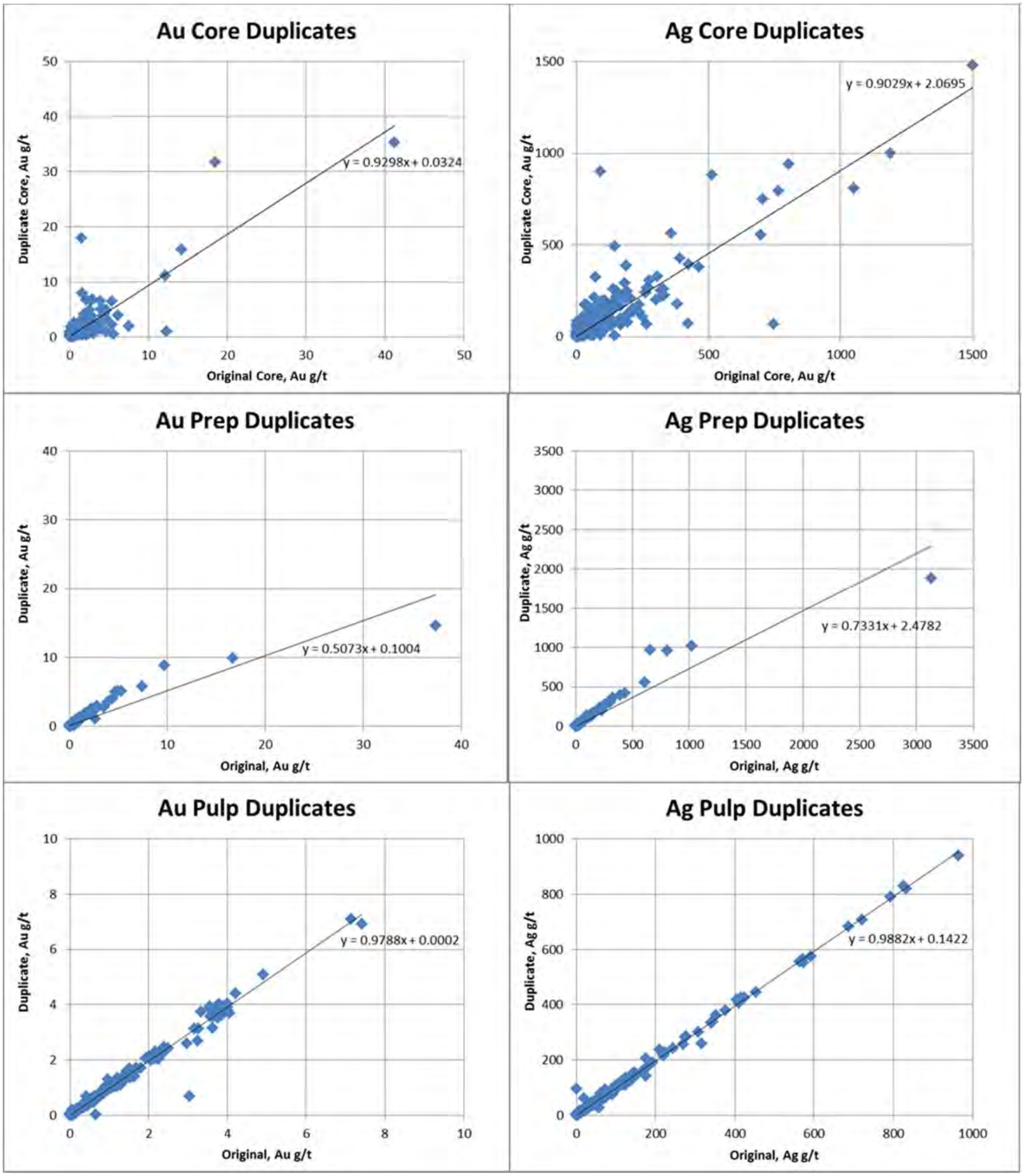


Figure 11-3 QA/QC Duplicates

11.3 Independent Audit of Almaden Drillhole Database

Between August 23 and September 26, 2012 and subsequently January 2 and January 21, 2014 APEX personnel, under the direct supervision of Kristopher J. Raffle, P.Geo., conducted an independent audit of Almaden's drillhole 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 the 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 22 diamond drillhole collar locations have been confirmed by Kristopher J. Raffle, P.Geo., following site visits to the Tuligtic Property on October 18, 2011, September 23, 2012 and November 20, 2013. The drill locations have been compared with the Almaden database used in the mineral Resource Estimate and are deemed to be accurate. In addition, Almaden has provided APEX with copies of all original down hole survey field records. Original field records for a total of 42 drillholes have been checked against database values used for the mineral Resource Estimate. No discrepancies have been found.

11.3.2 Drill Core Assay Database

A total of 109,570 drill core samples exist within the drill database (423 drillholes in total). The database audit consisted of checking 10,885 database gold and silver values against the original ALS analytical certificates. The audit specifically focused on assays within reported mineralized intercepts. No discrepancies have been identified between the original ALS analytical certificates and Almaden's drillhole database values.

12.0 DATA VERIFICATION

Kristopher J. Raffle, P.Geo., (considered "the author" in this Section of the report) 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 has been 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.

Additional visits to the Tuligtic Property were carried out by the author on September 23, 2012 and November 20, 2013 to observe current operations, review additional mineralized intercepts in drill core, and collect quarter drill core samples from the recently completed drillholes. A comparison of the results of the author's '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	Drillhole	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	91.70	92.20	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
13KRP201	126912	TU-13-238	216.00	216.50	0.50	3.78	92	2.69	63.4
13KRP202	142029	TU-13-287	166.98	168.00	1.02	0.668	48	0.775	87.7
13KRP203	141281	TU-13-308	375.50	376.00	0.50	2.36	19	2.41	33.2
13KRP204	143281	TU-13-309	195.00	195.50	0.50	11.35	756	14.4	1000

Based on the results of the traverses, drill core review, and 'replicate' sampling Mr. Raffle has no reason to doubt the reported exploration results. Slight variation in assays is expected due to variable distribution of mill feed 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.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

In 2012, Almaden drilled and shipped samples for metallurgical testing to Blue Coast Research's laboratory located in Parksville, British Columbia. Almaden completed a total of ten scoping level tests on four different composite samples.

In 2013, MMTS was requested to design and supervise an additional scoping level testing program to support the PEA using the original four Ixtaca composite samples. This metallurgical testing program was completed in September 2013 with a total of 74 tests that included gravity concentration and flotation tests.

13.2 Composite Sample Location

Figure 13-1 shows location of the diamond drillhole collars (and its trace projection) used to create each of the composite samples as well as the whole 2012 drilling campaign. Metallurgical drillholes are within Ixtaca Deposit's boundary and are adequately distributed throughout the Ixtaca Deposit.

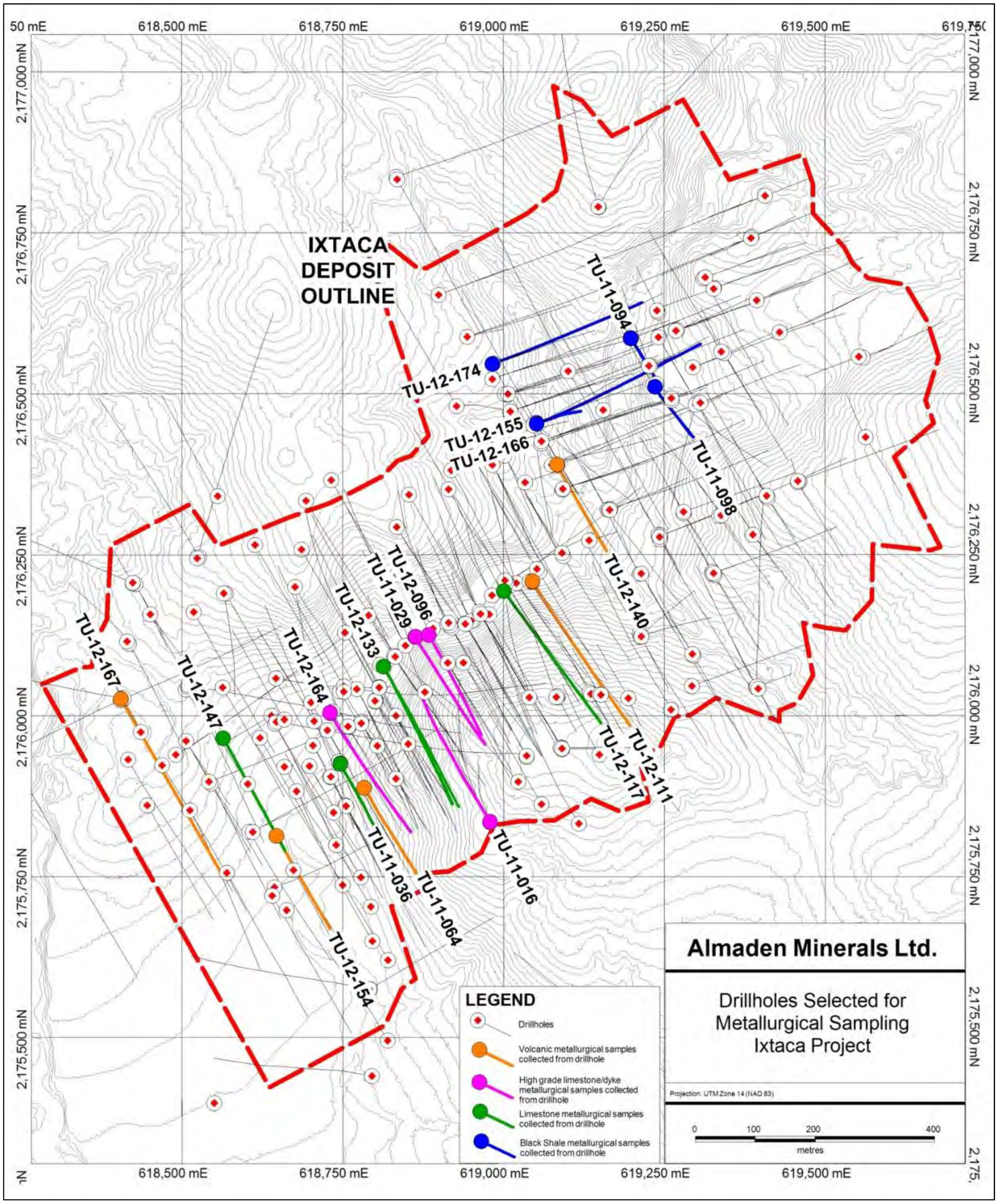


Figure 13-1 Composite Sample Location

13.3 Composite Sample's Characteristics

The four composite's head assays are shown in Table 13-1 and the expected life of mine average head grades are shown in Table 13-2. Figure 13-2 to Figure 13-4 compare the expected gold and silver assays in the PEA mine plan with the composite head grades.

Figure 13-4 shows that composite samples fall generally in the 60th to 80th percentile of the grade tonnage distribution curves. Composite sample head grades used in the PEA metallurgical testwork are higher than their corresponding mill feed type's LOM average.

Table 13-1 Metallurgical Composite Sample's Head Assays

	Pb (%)	Zn (%)	Fe (%)	Ag (g/t)	Au (g/t)	S (%)
Limestone	0.008	0.015	1.11	50.6	0.66	0.68
Volcanic	0.006	0.011	2.45	12.6	0.76	1.88
High Grade Limestone-Dyke	0.036	0.061	2.59	132.9	2.11	2.37
Black Shale	0.404	0.608	4.67	45.4	1.14	3.71

Table 13-2 Ixtaca Minable Deposit's Average Head Grade

Composite	kt	Average Head Assay					
		Pb (%)	Zn (%)	Fe (%)	Ag (g/t)	Au (g/t)	S (%)
Limestone	53,724	0.010	0.022	n/a	20.921	0.323	0.951
Volcanic	32,934	0.004	0.018	n/a	11.034	0.423	1.407
High Grade Limestone-Dyke	19,291	0.023	0.043	n/a	55.973	0.838	1.090
Black Shale	22,733	0.041	0.086	n/a	30.130	0.303	2.189
Total	**128,682**	**0.016**	**0.035**	**0.000**	**25.272**	**0.422**	**1.307**

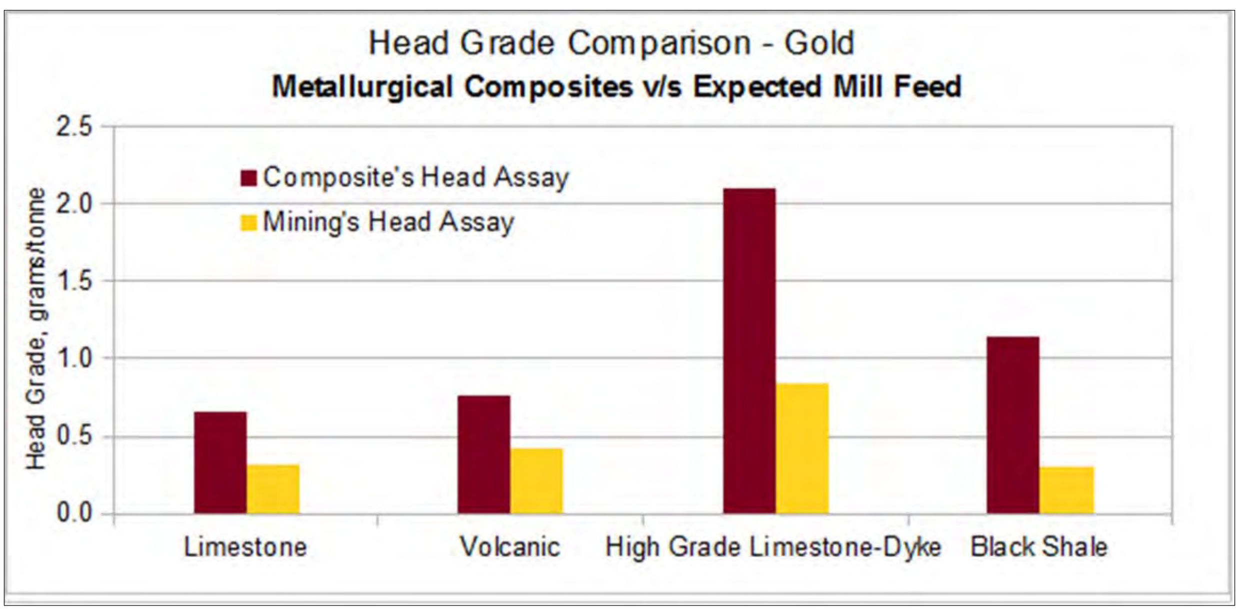


Figure 13-2 Gold's Head Grade Comparison between Metallurgical Composite and Mill Feed

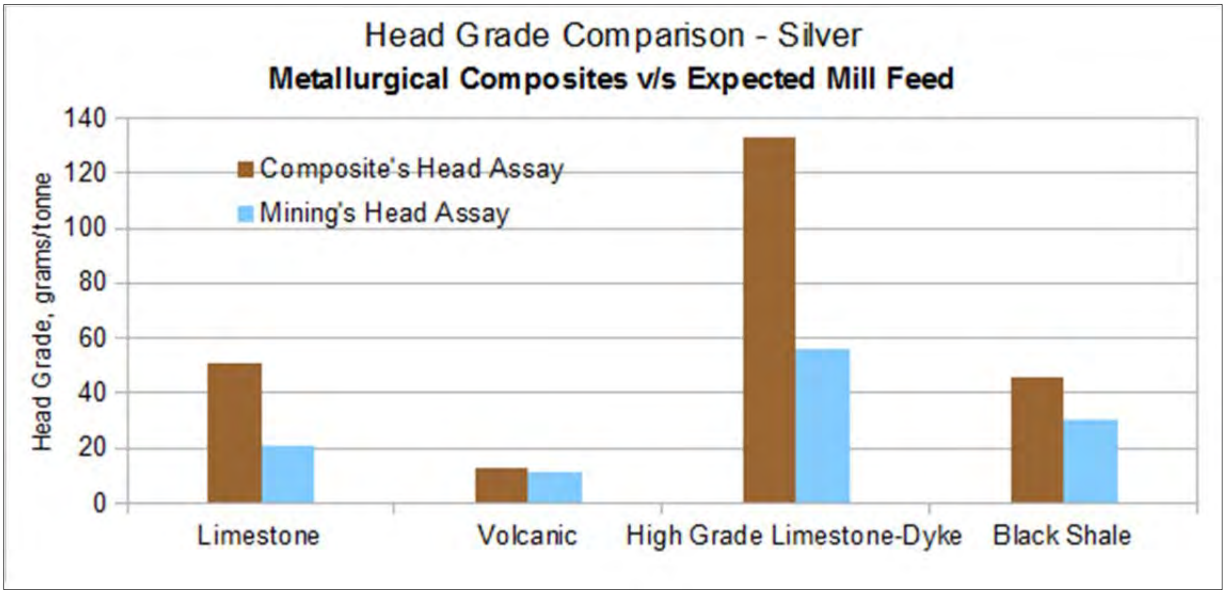


Figure 13-3 Silver's Head Grade Comparison between Metallurgical Composite and Mill Feed

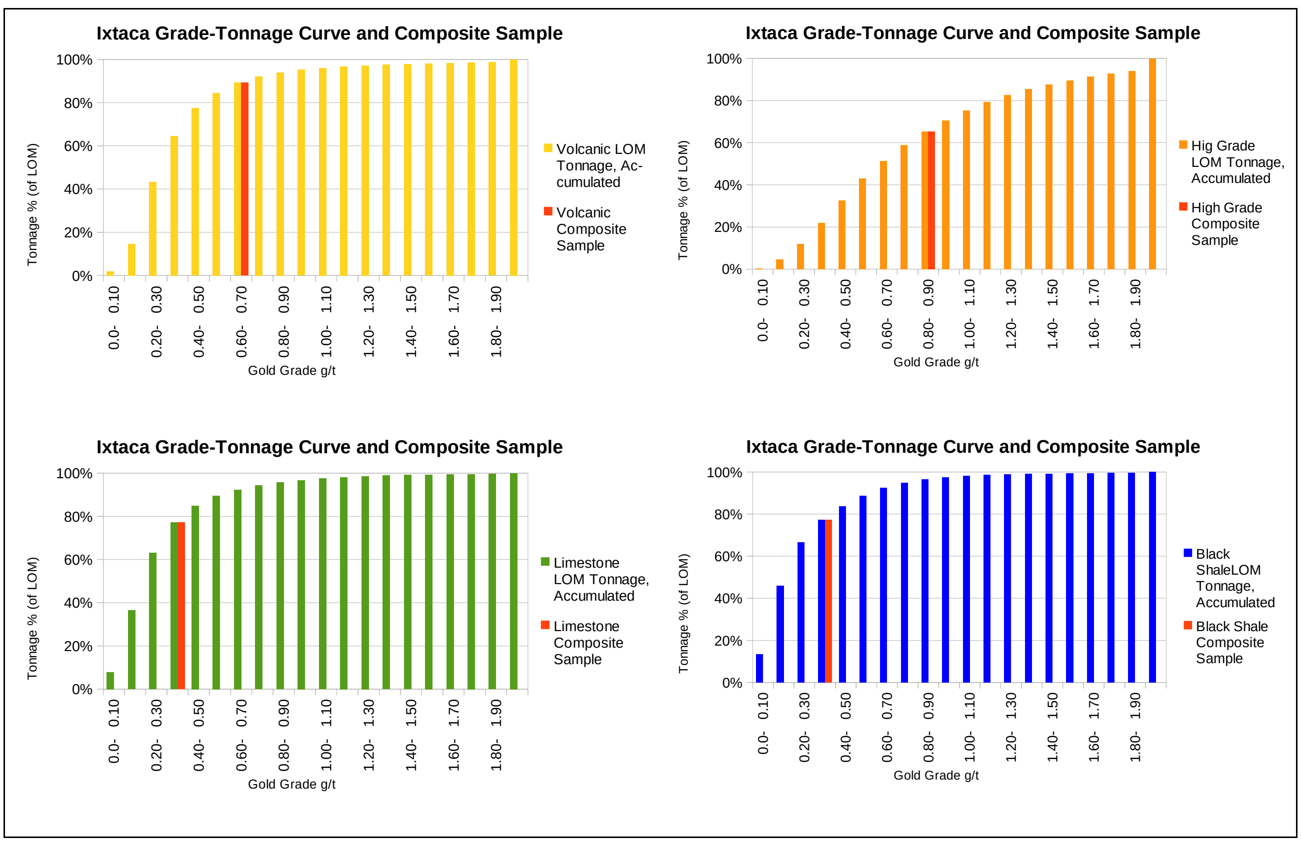


Figure 13-4 Composite Gold Grade Relative to LOM

13.4 Metallurgical Testwork

The preliminary metallurgical testwork program has included 34 gravity concentration tests and 40 kinetic rougher flotation tests, as shown in Table 13-3. Composite samples have been subject to gravity concentration using a Knelson laboratory machine followed by rougher flotation of gravity concentrator's tails. The gravity concentrate from the Falcon laboratory machine is further upgraded using a shaking table (mozzley table). The mozzley table concentrate became the final gravity concentrate, and mozzley table tails have been blended with the Knelson concentrator's tails to become the rougher flotation feed.

Table 13-3 Total Metallurgical Tests

Composite	Scoping Rougher Flotation	Preliminary Tests		Total tests
		Gravity Recovery	Rougher Kinetics	
Limestone	2	15	15	32
Volcanic	2	14	20	36
High Grade Limestone-Dyke	5	3	3	11
Black Shale	1	2	2	5
Totals	**10**	**34**	**40**	**84**

13.5 Bond Ball Work Index (BWi)

Results from standard Bond Work Index on each composite sample are shown in Table 13-4. The Dyke Limestone and Black Shale samples have a medium hardness. The volcanic sample is significantly softer than the mill feed types at 10.5kWh/tonne, which has benefits in terms of reduced energy required for comminution, but as discussed later in the document, the volcanic ores tend to produce ultra-fine particles which negatively impacts recovery and general process performance.

Table 13-4 Bond Work Index Results

Mill feed Type	Bond Work Index, kWh/tonne
Dyke	14.6
Limestone	13.2
Black Shale	18.6
Volcanic	10.5

13.6 Gravity Recoverable Gold

Samples have been tested for gravity recoverable metals using a laboratory scale Knelson concentrator. There is a distinctive performance difference between gold and silver recovery by gravity concentration, as shown in Table 13-5. Black Shale, High Grade Limestone Dyke, and Limestone samples produced gravity recoverable gold ranging between 30% and 40%. Corresponding silver recoveries are between 5% and 10%. Volcanic mill feed type shows poor gravity recoveries for both metals.

Table 13-5 Gravity Recoverable Gold

Mill feed Type	Recovery Au	Recovery Ag
Black Shale	30.47%	5.5%
HG LD	38.60%	6.0%
Limestone	30.24%	9.3%
Volcanic	7.99%	3.9%

13.7 Flotation Reagents

Flotation testwork uses a gold industry reagent suite typical for the Ixtaca type of mineralization (see Table 13-6).

Table 13-6 Flotation Reagents

Reagent	Type	Dosage Tested, g/tonne
$CuSO_4$	Surface activator	0 – 300
SIPX	Collector	25 – 450
3477	Collector	0 – 225
3418A	Collector	0 – 45
Na_2SiO_3	Dispersant	0 – 500
F-140	Frother	0 – 140

13.8 Effect of Grind Size on the Recovery of Gold and Silver

Gold recovery responded well to a finer grind size, see Figure 13-5. The test varied the P_{80} from approximately 120µm to 40µm. The optimum P_{80} is expected to be between 45µm to 65µm. Volcanic composite, because it's evident "softness", easily generates ultra-fine particles that negatively affect metallurgical recovery and overall metallurgical performance (excepting energy consumption in grinding). Volcanic samples will need further evaluation to define optimal process conditions that can minimize, or compensate the presence of ultra-fine particles.

Silver recovery variability at different grind sizes show a similar response to gold, but silver recovery is generally higher than gold for Volcanic and Limestone samples, and lower than gold for High Grade Limestone Dyke and Black Shale samples.

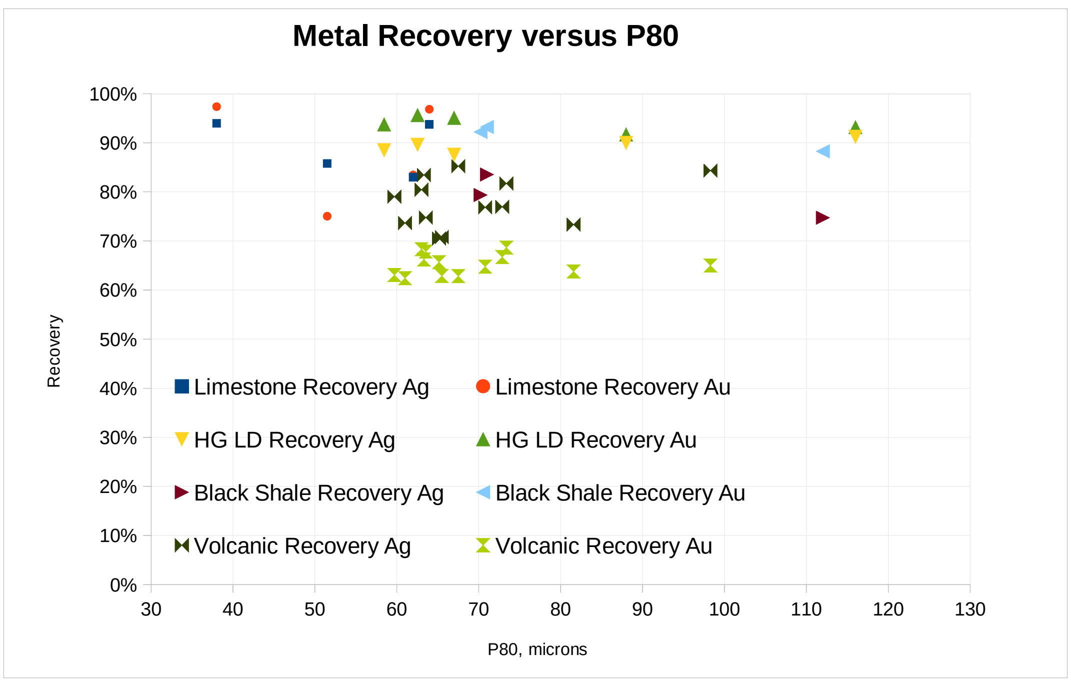


Figure 13-5 Effect of Grind Size on Flotation Metal Recovery

13.9 Effect of %Solids on the Recovery of Gold and Silver

Limestone and Volcanic samples have responded well to lower solids density in flotation. The effect of solids concentration on the metal recovery has been evaluated by floating a 2-kilogram charge on 4-liter flotation cell and in 8-liter flotation cell, which is equivalent to slurry having a solids concentration of approximately 38%w/w and 22%w/w respectively. Figure 13-6 shows that average increase in recovery is approximately 3% (from +76% to 80%) for gold and 8% (from 73% to 81%) for silver.

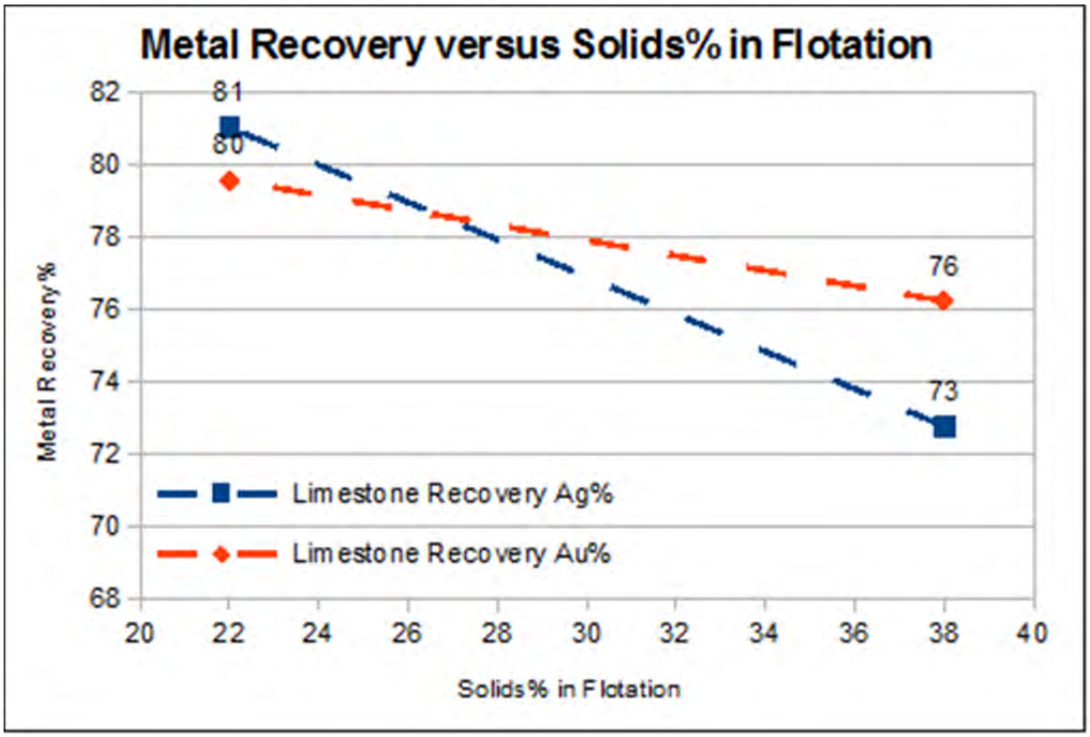


Figure 13-6 **Effect of Solids Concentration of Metal Recovery**

13.10 Effect of Flotation pH on the Recovery of Gold and Silver

The Volcanic samples show slight recovery improvement when floating at lower than natural pH in the range of pH=5.0 as shown in Figure 13-7. A possible explanation for this response would be the acid neutralization effect of clay components within the volcanic sample; this subject will need further investigation. All other samples have shown no positive response to variations in natural pH.

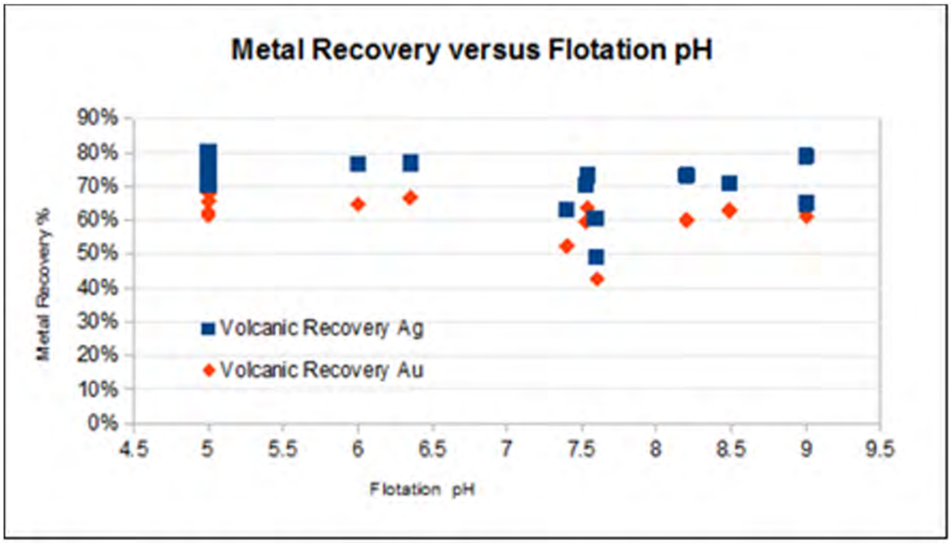


Figure 13-7 Effect of PH on Metal Recovery for Volcanics

13.11 Preliminary Leach Testwork

Preliminary bottle roll cyanide leach tests on Ixtaca High Grade Limestone Dyke concentrate resulted in 93% recovery of silver and 88.1% recovery of gold as shown in Figure 13-8. These recoveries were achieved after 48 hours of leaching with a coarse grind size of 92 microns.

The tests indicate that Ixtaca mill feed is amenable to leaching. Leach optimization tests have not been carried out on Ixtaca concentrates at this preliminary stage.

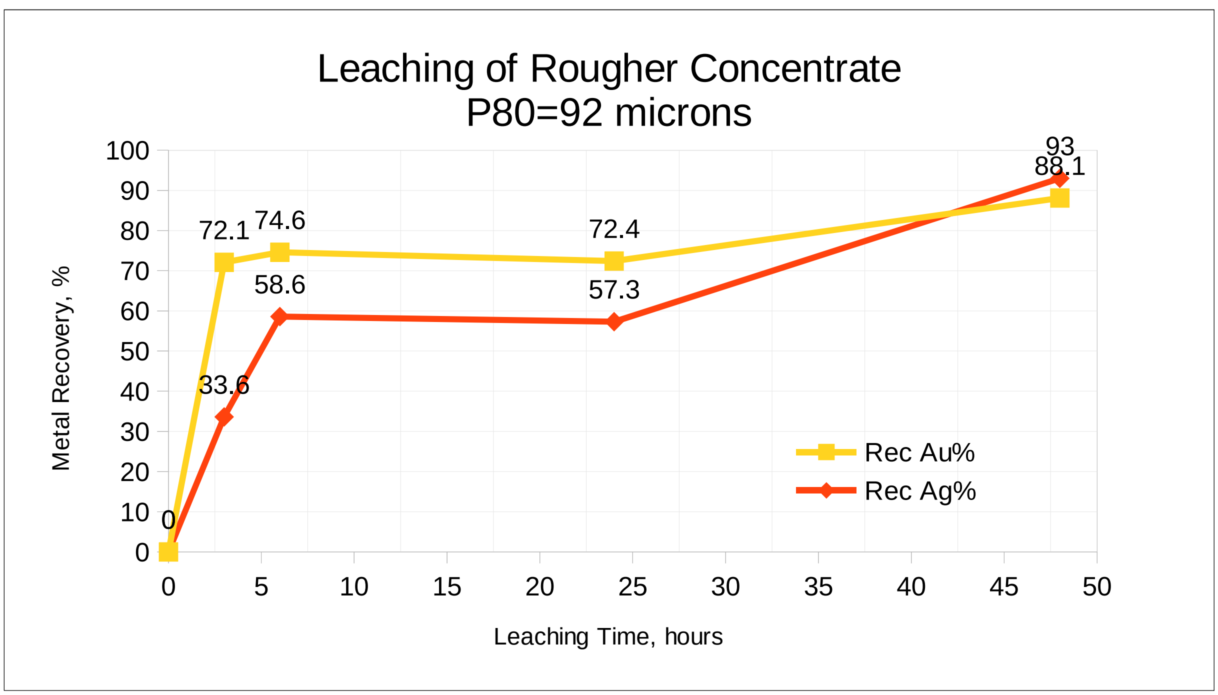


Figure 13-8 Leaching of Rougher Concentrate

13.12 Deleterious Elements

No deleterious elements have been identified within Ixtaca mineralization at this stage of study.

13.13 Process Criteria

The preliminary metallurgical testwork tested four composite samples representing the dominant lithologies identified in Ixtaca's geology.

Gravity concentration produced good gold recovery averaging 30% to +40% in three out of four samples, with corresponding silver recoveries ranging from 5% to 10% approximately. The Volcanic composite sample (the fourth sample in the set) achieved consistently lower gravity concentration recovery at approximately 8% Au and 4% silver. Future Volcanic sample testwork should use a higher G-Force gravity concentration machine.

This preliminary metallurgical testwork is focused on maximizing the grade of a saleable silver-gold gravity and flotation concentrate. A decision has subsequently been made to base the PEA on a process that produces a silver-gold bar from the leaching of the gravity and flotation concentrates.

The rougher flotation testwork, when focused on achieving maximum metal recovery contained in the minimum possible concentrate mass (lowest possible mass pull) has achieved the following recoveries:

- 95% Au recovery, 88% Ag recovery for High Grade Limestone Dyke; 7% mass pull,
- 93% Au recovery, 84% Ag recovery for Black Shale; 10% mass pull,
- 84% Au recovery, 77% Ag recovery for Limestone; 5% mass pull,
- 69% Au recovery, 82% Ag recovery for Volcanic; 22% mass pull.

These recoveries of gold, silver, and mass pull values are for a preliminary flotation program aiming to produce a concentrate as final product, and have not been optimized for a process that includes leaching of concentrate to enable production of a silver-gold doré.

With the focus of producing a silver-gold bar by means of leaching of the combined gravity and flotation concentrates, it is estimated that rougher concentrate mass pull could increase in average to 12% in order to achieve a combined gravity and flotation recovery of 95% average for gold and silver. This assumption needs confirmation in the next phase of metallurgical testwork. Leaching recovery of the combined gravity and rougher concentrate has been estimated at 95%.

The preliminary metallurgical testwork results from Ixtaca's composite samples shows that the Ixtaca deposit responds well to gravity concentration followed by flotation and leaching of the flotation concentrate. Overall, gold and silver process recoveries to a silver-gold doré of 90.3% are recommended for the PEA.

14.0 MINERAL RESOURCE ESTIMATES

At the request of Morgan Poliquin, President of Almaden, Giroux Consultants Ltd. (GCL) was retained to produce an updated Resource Estimate on the Ixtaca Main Zone of the Tuligtic Property located in Puebla State, Mexico. There have been 198 additional diamond drillhole completed on the Tuligtic Property by Almaden since the last 43-101 Resource Estimate (K. Raffle, et.al. March 4, 2013) bringing the total number of drillhole on the Property to 423. The effective date for this estimate is January 8, 2014, the date the data was received.

Gary 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 has supplied a total of 423 drillhole with 5,095 down hole surveys and 109,570 assays for gold and silver. Of these drillholes, 400 totalling 129,734m outline the Ixtaca Main zone and NE Extension which are estimated in this resource. All drillholes are included in Appendix A with the holes used in this resource highlighted. A total of 705 gaps have been found in the from – to record and in these gaps values of 0.001g/t Au and 0.01g/t Ag are inserted. Included in these gaps are 422 intervals at the start or end of holes that are not sampled due to broken rock which is cased or ends of holes that are not considered mineralized. Two gold and silver assays reported as blank are set to 0.001g/t and 0.01g/t respectively.

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

Code	**Description**
ASH	A clay altered tuff overlying 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 of Shale surrounding the NEHG zone

From this list, 3 dimensional solids for each domain have been created in Gemcom software by Almaden geologists, to constrain the estimation. Figure 14-1 shows the various mineralized domains.

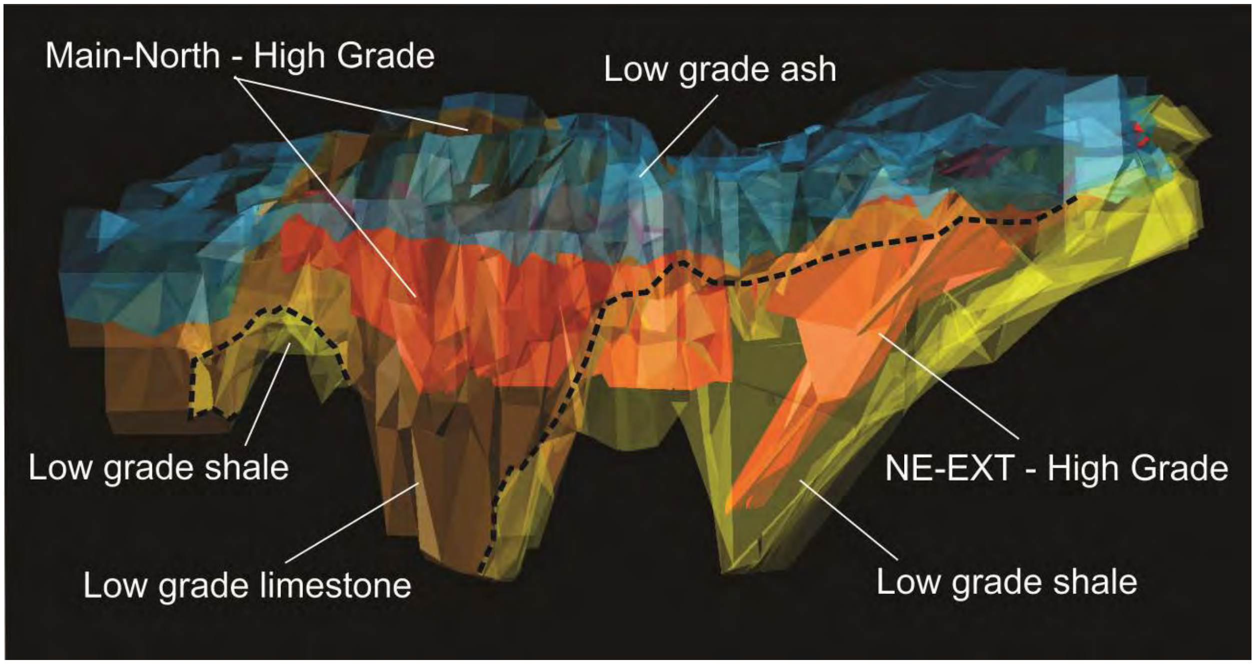


Figure 14-1 Isometric View Looking N Showing the Geologic Solids

Drillholes have then been compared to the solids and each assay has been tagged with a code. The statistics for gold and silver are tabulated in Table 14-1 below sorted by mineralized zone. Assays outside the mineralized solids are 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
ASH	Au (g/t)	13,778	0.422	4.647	0.001	470.00	11.00
	Ag (g/t)	13,778	8.68	58.80	0.01	4340.00	6.78
MHG	Au (g/t)	11,643	1.247	5.080	0.001	336.00	4.08
	Ag (g/t)	11,643	82.51	236.21	0.01	9660.00	2.86
LGLM	Au (g/t)	38,382	0.261	1.784	0.001	167.00	6.83
	Ag (g/t)	38,382	17.40	95.07	0.01	5310.00	5.46
LGSH	Au (g/t)	3,376	0.186	0.992	0.001	38.00	5.34
	Ag (g/t)	3,376	11.58	60.44	0.01	2370.00	5.22
NELGSH	Au (g/t)	20,705	0.118	1.059	0.001	94.00	8.98
	Ag (g/t)	20,705	9.49	41.80	0.01	1490.00	4.41
NEHG	Au (g/t)	3,858	0.791	2.584	0.003	96.40	3.27
	Ag (g/t)	3,858	50.15	118.69	0.25	3140.00	2.37
WASTE	Au (g/t)	18,532	0.012	0.070	0.001	5.44	5.61
	Ag (g/t)	18,532	0.72	6.62	0.01	646.00	9.23

To determine if each of these geologic domains is unique the lognormal cumulative frequency plots for gold and silver are examined. The two high grade units are significantly different from the low grade units so 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 also very similar but occur on different ends of the deposit.

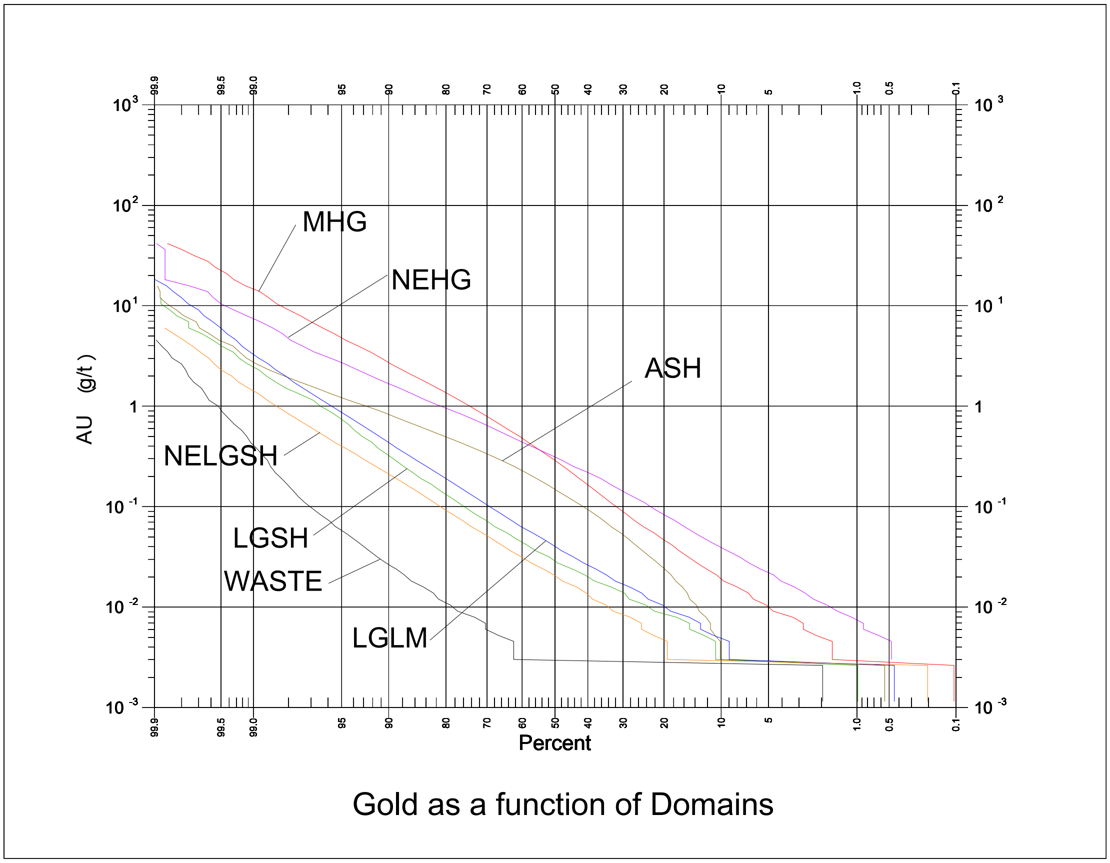


Figure 14-2 Lognormal Cumulative Frequency Plot for Au as a Function of Domain

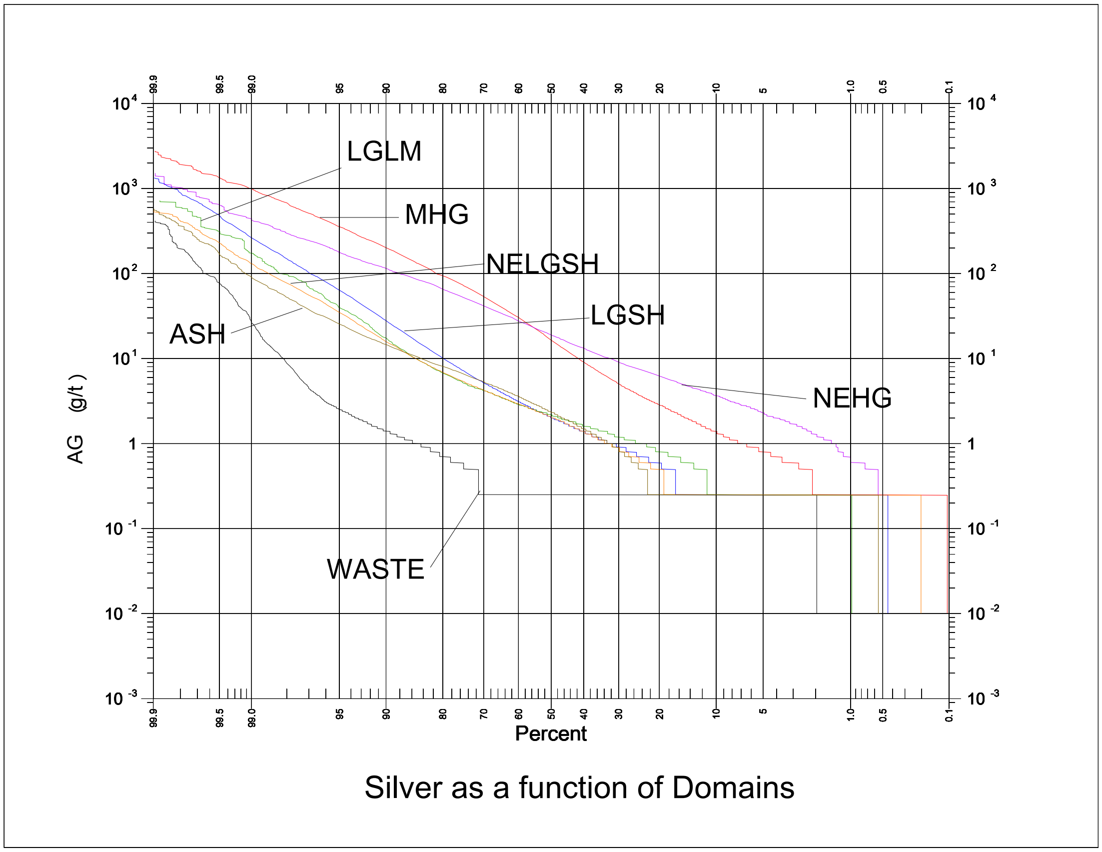


Figure 14-3 Lognormal Cumulative Frequency Plot for Ag as a Function of Domain

The grade distributions for gold and silver, within each mineralized domain, have been examined to determine if capping is required and if so, at what levels. Both elements show skewed distributions in all domains and have been converted to lognormal cumulative frequency plots. Each variable has been examined within each domain with thresholds selected for capping if required. (Table 14-2)

Table 14-2 Capped Levels for Gold and Silver

Domain	Variable	Cap Level (g/t)	Number of Assays capped
MHG	Au	56.0 g/t	6
	Ag	2150.0 g/t	18
ASH	Au	20.0 g/t	10
	Ag	500.0 g/t	16
LGLM	Au	41.0 g/t	11
	Ag	2200 g/t	10
LGSH	Au	6.0 g/t	8
	Ag	360.0 g/t	11
NELGSH	Au	13.0 g/t	5
	Ag	1100.0 g/t	4
NEHG	Au	17.0 g/t	5
	Ag	960.0 g/t	9
WASTE	Au	0.5 g/t	32
	Ag	50.0 g/t	14

The effects of capping are shown in the following Table 14-3 with minor reductions in mean grade but significant reductions in standard deviations and coefficients of variation.

Table 14-3 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
ASH	Au (g/t)	13,778	0.358	0.887	0.001	20.00	2.48
	Ag (g/t)	13,778	7.82	27.85	0.01	500.00	3.56
MHG	Au (g/t)	11,643	1.195	3.298	0.001	56.00	2.76
	Ag (g/t)	11,643	80.10	190.56	0.01	2150.00	2.38
LGLM	Au (g/t)	38,382	0.251	1.315	0.001	40.00	5.23
	Ag (g/t)	38,382	17.09	84.75	0.01	2200.00	4.96
LGSH	Au (g/t)	3,376	0.163	0.513	0.001	6.00	3.16
	Ag (g/t)	3,376	10.15	34.15	0.01	360.00	3.37
NELGSH	Au (g/t)	20,705	0.108	0.455	0.001	13.00	4.22
	Ag (g/t)	20,705	9.44	40.35	0.01	1100.00	4.27
NEHG	Au (g/t)	3,858	0.736	1.470	0.003	17.00	2.00
	Ag (g/t)	3,858	48.37	89.16	0.25	960.00	1.84
WASTE	Au (g/t)	18,532	0.011	0.034	0.001	0.50	2.98
	Ag (g/t)	18,532	0.64	2.12	0.01	50.00	3.32

14.2 Composites

Of the 110,274 assays, within the seven domains, 109,003 or 99% are less than or equal to 3m in length. As a result, a 3m composite length is selected. Down hole composites 3m in length are formed to honour the domain boundaries. Composite intervals at the domain boundaries that are less than 1.5m in length are combined with adjoining samples while those greater than or equal to 1.5m are left alone. As a result, the composites form a uniform support of 3±1.5m. Material outside the six mineralized solids is considered waste. (See Table 14-4)

Table 14-4 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
ASH	Au (g/t)	6,699	0.270	0.519	0.001	12.20	1.92
	Ag (g/t)	6,699	5.77	15.34	0.01	355.15	2.66
MHG	Au (g/t)	2,824	0.880	1.454	0.001	20.67	1.65
	Ag (g/t)	2,824	58.85	86.99	0.01	1287.43	1.48
LGLM	Au (g/t)	13,568	0.158	0.478	0.001	11.72	3.03
	Ag (g/t)	13,568	9.94	31.45	0.01	1050.01	3.16
LGSH	Au (g/t)	1,153	0.114	0.263	0.001	3.06	2.31
	Ag (g/t)	1,153	7.12	18.50	0.01	223.96	2.60
NELGSH	Au (g/t)	7,253	0.073	0.230	0.001	8.33	3.13
	Ag (g/t)	7,253	6.40	19.15	0.01	660.60	2.99
NEHG	Au (g/t)	910	0.626	0.835	0.003	7.36	1.34
	Ag (g/t)	910	42.74	53.21	0.25	487.60	1.24
WASTE	Au (g/t)	11,061	0.008	0.021	0.001	0.46	2.70
	Ag (g/t)	11,061	0.40	1.07	0.01	56.70	2.68

To determine if hard or soft boundaries are required between the geologic domains, a series of Contact Plots have been 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 can be set up with samples allowed to influence block grades from both sides of a contact. The results are shown in Appendix B. The grades for Au across the contacts are sufficiently different for the LGLM-ASH, LGLM-NELGSH, ASH-NELGSH, MHG-LGLM and NEHG-NELGSH boundaries to make these all Hard Boundaries.

In the case of the LGLM-LGSH contact, the grades are sufficiently similar for Au across the contact, to make this a Soft Boundary. The grades for Ag across the contacts are sufficiently different for the ASH-NELGSH, MHG-LGLM and NEHG-NELGSH contacts to make these all Hard Boundaries.

For silver along the LGLM-ASH, LGLM-LGSH and LGLM-NELGSH contacts, the grades are sufficiently similar to make these Soft Boundaries.

14.3 Variography

Pairwise relative semivariograms have been produced for gold and silver within the each of the geologic domains. In all cases except for waste, a geometric anisotropy has been observed and nested spherical models are fit to the three principal directions. Due to the high correlation between Au and Ag in each of the domains, gold and silver show similar directions of anisotropy. (Table 14-5)

Table 14-5 Pearson Correlation Coefficients for Au – Ag Geologic Domains

Au:Ag Correlation Coef.	ASH	MHG	LGLS	LGSH	NEHG	NELGSH	WASTE
	0.7740	0.8781	0.8330	0.8336	0.5684	0.8013	0.7743

Within the Ash zone both gold and silver have been modelled with anisotropic models with longest range along azimuth 155^{o} dip 0^{o} and down dip along azimuth 245^{o} dip -45^{o}.

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

For the north east extension mineralization, the longest horizontal ranges in both the high grade core and low grade shale that surrounds it are found along azimuth 347^{o}.

For all of these models nested anisotropic spherical models are applied.

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

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

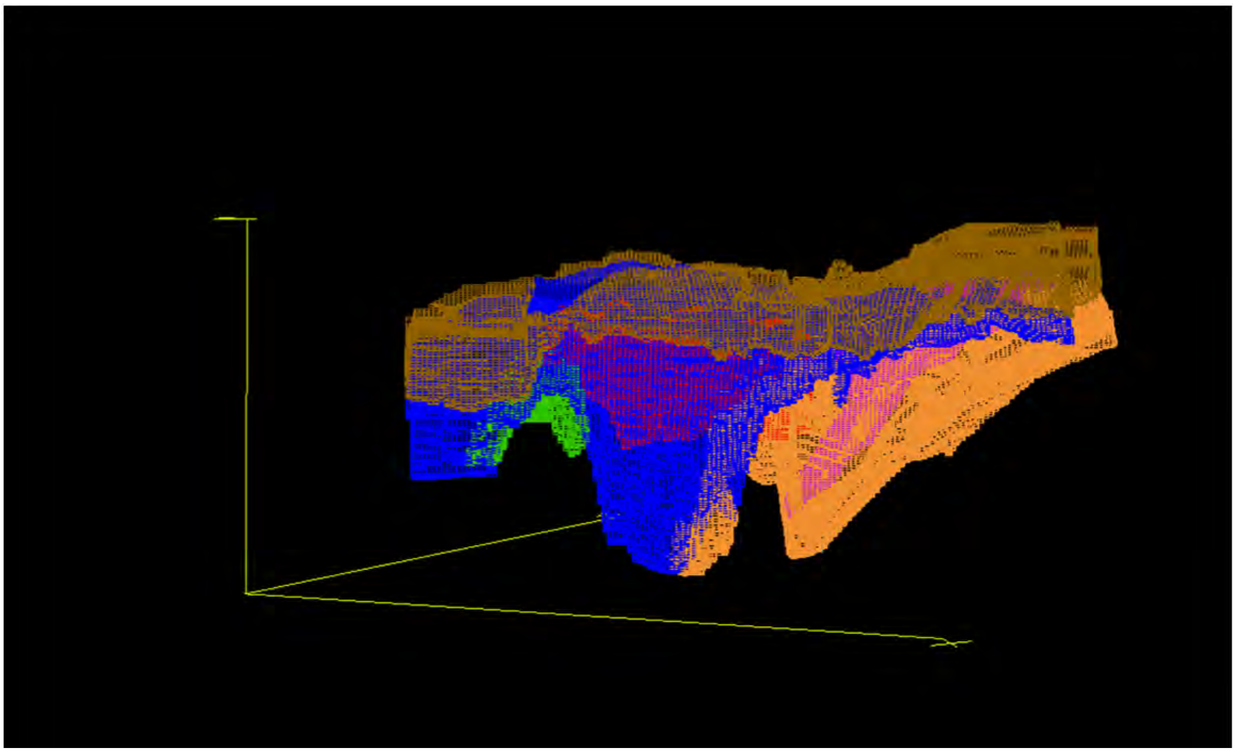
Table 14-6 Semivariogram Parameters for Gold and Silver

Domain	Variable	Az/Dip	C_0	C_1	C_2	Short Range (m)	Long Range (m)
MHG	**Au**	60° / 0°	0.35	0.38	0.20	10.0	120.0
		330° / -55°				12.0	100.0
		150° / -35°				12.0	120.0
	Ag	60° / 0°	0.45	0.30	0.18	12.0	140.0
		330° / -55°				15.0	70.0
		150° / -35°				15.0	100.0
ASH	**Au**	155° / 0°	0.20	0.18	0.70	10.0	120.0
		65° / -45°				15.0	50.0
		245° / -45°				20.0	90.0
	Ag	155° / 0°	0.20	0.20	0.60	10.0	120.0
		65° / -45°				15.0	50.0
		245° / -45°				15.0	90.0
LGLM	**Au**	60° / 0°	0.30	0.35	0.27	12.0	120.0
		330° / -55°				18.0	80.0
		150° / -35°				18.0	100.0
	Ag	60° / 0°	0.35	0.42	0.20	12.0	120.0
		330° / -55°				24.0	100.0
		150° / -35°				25.0	100.0
LGSH	**Au**	60° / 0°	0.20	0.30	0.37	15.0	30.0
		330° / -55°				10.0	50.0
		150° / -35°				15.0	60.0
	Ag	60° / 0°	0.20	0.40	0.27	20.0	50.0
		330° / -55°				10.0	50.0
		150° / -35°				30.0	80.0
NELGSH	**Au**	347° / 0°	0.20	0.25	0.35	40.0	140.0
		257° / -55°				12.0	210.0
		77° / -35°				15.0	100.0
	Ag	347° / 0°	0.20	0.35	0.15	28.0	90.0
		257° / -55°				15.0	210.0
		77° / -35°				20.0	60.0
NEHG	**Au**	347° / 0°	0.30	0.10	0.40	12.0	120.0
		257° / -55°				10.0	36.0
		77° / -35°				10.0	40.0
	Ag	347° / 0°	0.30	0.15	0.33	12.0	80.0
		257° / -55°				10.0	18.0
		77° / -35°				15.0	48.0
WASTE	**Au**	Omni Directional	0.08	0.30	0.06	36.0	110.0
	Ag	Omni Directional	0.05	0.45	0.12	36.0	110.0

14.4 Block Model

A rotated block model with blocks 10m NE-SW, 10m NW-SE and 5m high has superimposed over the mineralized solids. The model is rotated 30^o 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 are recorded. These percentages are checked to assure there is no overlap. The block model origin shown in Figure 14-4 is as follows:

Lower Left Corner		
618578 E	Column size = 10m	167 columns
2175235 N	Row size = 10m	128 rows Top of Model
2490 Elevation	Level size = 5m	180 levels Rotation 30^o counter clockwise



Note: ASH in brown, MHG in red, LGLM in blue, LGSH in green, NEHG in purple and NELGSH in orange

Figure 14-4 Isometric View Looking NW Showing Blocks

14.5 Bulk Density

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

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

The measurements have been made on drill core samples using the Archimedes (weight in air-weight in water) method. The relative number of analysis is shown in the Table below:

Table 14-7 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 is also sorted by lithology.

Table 14-8 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 14-8 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 have then averaged within the various geologic domains to produce the following specific gravities for converting volumes to tonnes:

- The ash domain has an average specific gravity of 1.67
- The low grade limestone (LGLM) domain has an average specific gravity of 2.66
- The main high grade (MHG) domain has an average specific gravity of 2.63 (this unit contains about 20% Felsic Dyke)
- The main high grade zone (MHGN) North limb has an average specific gravity of 2.60 (this north limb contains about 40% Felsic Dyke and 40% Mafic Dyke)
- The low grade shale (LGSH) domain has an average specific gravity of 2.61
- The North East extension high grade (NEHG) domain has an average specific gravity of 2.65

14.6 Grade Interpolation

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

Each kriging run has been 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 uses search dimensions equal to ¼ the semivariogram range in the three principal directions. A minimum of four composites are required to estimate a block with a maximum of three from any given drillhole. In this manner, all blocks are estimated with a minimum of two drillhole. For blocks not estimated in pass 1, a second pass using ½ the semivariogram range has been completed. A third pass using the full range and a fourth pass using twice the range has followed. Finally because there were many blocks containing multiple domains, a fifth pass has often been required to ensure all domains were estimated. In all passes the maximum number of composites used is twelve and if more were found in any search the closest twelve are used.

Once all domains are completed, estimated blocks containing some percentage outside the mineralized domains are 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 is produced. The search parameters for gold within each domain and the number of blocks estimated in each pass are tabulated in the following Table.

Table 14-9 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	14,220	60 / 0	30.0	330 / -55	25.0	150 / -35	30.0
	2	8,773	60 / 0	60.0	330 / -55	50.0	150 / -35	60.0
	3	792	60 / 0	120.0	330 / -55	100.0	150 / -35	120.0
NEHG	1	508	347 / 0	30.0	257 / -55	9.0	77 / -35	10.0
	2	4,916	347 / 0	60.0	257 / -55	18.0	77 / -35	20.0
	3	7,714	347 / 0	120.0	257 / -55	36.0	77 / -35	40.0
	4	1,578	347 / 0	240.0	257 / -55	72.0	77 / -35	80.0
LGLM	1	47,121	60 / 0	30.0	330 / -55	20.0	150 / -35	25.0
	2	106,984	60 / 0	60.0	330 / -55	40.0	150 / -35	50.0
	3	58,743	60 / 0	120.0	330 / -55	80.0	150 / -35	100.0
	4	11,282	60 / 0	240.0	330 / -55	160.0	150 / -35	200.0
NELGSH	1	65,307	347 / 0	35.0	257 / -55	52.5	77 / -35	25.0
	2	82,293	347 / 0	70.0	257 / -55	105.0	77 / -35	50.0
	3	27,998	347 / 0	140.0	257 / -55	210.0	77 / -35	100.0
	4	472	347 / 0	280.0	257 / -55	420.0	77 / -35	200.0
ASH	1	13,923	155 / 0	30.0	65 / -45	12.5	245 / -45	22.5
	2	51,013	155 / 0	60.0	65 / -45	25.0	245 / -45	45.0
	3	50,819	155 / 0	120.0	65 / -45	50.0	245 / -45	90.0
	4	12,622	155 / 0	240.0	65 / -45	100.0	245 / -45	180.0
LGSH	1	198	60 / 0	7.5	330 / -55	12.5	150 / -35	15.0
	2	2,402	60 / 0	15.0	330 / -55	25.0	150 / -35	30.0
	3	8,287	60 / 0	30.0	330 / -55	50.0	150 / -35	60.0
	4	7,123	60 / 0	60.0	330 / -55	100.0	150 / -35	120.0
WASTE	1	7,138	Omni Directional			27.5		
	2	28,078	Omni Directional			55.0		
	3	49,245	Omni Directional			110.0		
	4	19,292	Omni Directional			220.0		

14.7 Classification

Based on the study herein reported, delineated mineralisation of Ixtaca is classified as a resource according to the following definitions from National Instrument 43-101 and from CIM (2014):

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

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

> "A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling."

> "The term Mineral Resource covers mineralisation and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of Modifying Factors. The phrase 'reasonable prospects for economic extraction' implies a judgement by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. The Qualified Person should consider and clearly state the basis for determining that the material has reasonable prospects for eventual economic

extraction. Assumptions should include estimates of cut-off grade and geological continuity at the selected cut-off, metallurgical recovery, smelter payments, commodity price or product value, mining and processing method and mining, processing and general and administrative costs. The Qualified Person should state if the assessment is based on any direct evidence and testing. Interpretation of the word 'eventual' in this context may vary depending on the commodity or mineral involved. For example, for some coal, iron, potash deposits and other bulk minerals or commodities, it may be reasonable to envisage 'eventual economic extraction' as covering time periods in excess of 50 years. However, for many gold deposits, application of the concept would normally be restricted to perhaps 10 to 15 years, and frequently to much shorter periods of time."

Inferred Mineral Resource

"An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration."

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

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

Indicated Mineral Resource

"An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation. An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve."

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

> **Measured Mineral Resource**
> "A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation. A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve."
> "Mineralisation or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such that the tonnage and grade or quality of the mineralisation can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability of the deposit. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit."
> **Modifying Factors**
> "Modifying Factors are considerations used to convert Mineral Resources to Mineral Reserves. These include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors."

At Ixtaca, the geologic continuity has been established through surface mapping and drillhole 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 have been 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.

A range of cut-offs are presented to demonstrate the sensitivity of the deposit to grade variations.

The Resource Tables are shown below using gold equivalent cut-offs where:

Gold – 3 yr. trailing average price of $1540
Silver – 3 yr. trailing average price of $30

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

$$AuEq = Au + (Ag * 30 / 1540)$$

At the time the resource was estimated no economic studies had been completed at Ixtaca. In the author's judgement and experience the resource stated has reasonable prospects of economic extraction. A cut-off of 0.50g/t AuEq has been highlighted as a possible cut-off for open pit mining. An analogous deposit to Ixtaca might be the Dolores Mine in Chihuahua, Mexico. A reported mining cut-off for this deposit is 0.3 g/t AuEq (Chlumsky, et al, 2011). The deposits are comparable in both gold and silver grades but Dolores uses heap leach recovery while Ixtaca is contemplating milling. As a result a 0.5 g/t AuEq cut-off was considered reasonable. The work described further in this PEA study establishes an NSR cut-off and tabulates the resource present within an optimized pit shell.

Table 14-10 Measured Resource 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	76,600,000	0.31	19.72	0.70	768	48,570	1,714
0.20	56,340,000	0.40	25.36	0.89	725	45,940	1,619
0.25	49,730,000	0.44	27.92	0.98	704	44,640	1,573
0.30	44,550,000	0.48	30.28	1.07	682	43,370	1,527
0.40	36,460,000	0.55	34.89	1.23	640	40,900	1,437
0.50	**30,420,000**	**0.61**	**39.44**	**1.38**	**599**	**38,570**	**1,350**
0.60	25,860,000	0.67	43.82	1.53	560	36,430	1,270
0.70	22,300,000	0.73	48.02	1.67	526	34,430	1,196
0.80	19,420,000	0.79	52.08	1.80	493	32,520	1,126
1.00	15,620,000	0.88	58.66	2.03	444	29,460	1,017
2.00	6,000,000	1.33	86.54	3.01	256	16,690	581

Table 14-11 Indicated Resource 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	208,220,000	0.24	13.06	0.50	1,627	87,430	3,334
0.20	145,640,000	0.32	17.08	0.65	1,484	79,980	3,044
0.25	125,610,000	0.35	18.90	0.72	1,413	76,330	2,900
0.30	108,520,000	0.38	20.78	0.79	1,336	72,500	2,749
0.40	81,460,000	0.45	24.78	0.93	1,184	64,900	2,446
0.50	**62,250,000**	**0.52**	**28.92**	**1.09**	**1,043**	**57,880**	**2,172**
0.60	48,710,000	0.59	33.15	1.23	921	51,920	1,933
0.70	39,350,000	0.65	37.12	1.37	824	46,960	1,738
0.80	32,810,000	0.71	40.64	1.50	747	42,870	1,581
1.00	23,750,000	0.81	47.12	1.73	621	35,980	1,322
2.00	5,880,000	1.39	72.89	2.81	263	13,780	532

Table 14-12 Inferred Resource 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	101,730,000	0.21	9.63	0.40	677	31,500	1,292
0.20	62,210,000	0.29	13.71	0.56	576	27,420	1,110
0.25	50,850,000	0.32	15.68	0.63	530	25,640	1,030
0.30	42,490,000	0.36	17.58	0.70	488	24,020	956
0.40	30,250,000	0.43	21.35	0.84	415	20,760	820
0.50	**22,150,000**	**0.50**	**25.14**	**0.99**	**355**	**17,900**	**704**
0.60	16,940,000	0.57	28.55	1.12	309	15,550	612
0.70	13,400,000	0.63	31.66	1.25	273	13,640	539
0.80	10,810,000	0.70	34.57	1.37	242	12,010	476
1.00	7,620,000	0.80	39.85	1.57	195	9,760	385
2.00	1,200,000	1.18	73.69	2.61	45	2,840	101

Table 14-13 Measured + Indicated Resource 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	284,800,000	0.26	14.85	0.55	2,399	135,980	5,045
0.20	201,980,000	0.34	19.39	0.72	2,208	125,920	4,663
0.25	175,330,000	0.38	21.46	0.79	2,114	120,970	4,470
0.30	153,070,000	0.41	23.55	0.87	2,018	115,900	4,277
0.40	117,910,000	0.48	27.90	1.02	1,823	105,770	3,882
0.50	**92,680,000**	**0.55**	**32.38**	**1.18**	**1,642**	**96,490**	**3,522**
0.60	74,570,000	0.62	36.85	1.34	1,482	88,350	3,203
0.70	61,640,000	0.68	41.06	1.48	1,350	81,370	2,933
0.80	52,230,000	0.74	44.89	1.61	1,239	75,380	2,709
1.00	39,370,000	0.84	51.70	1.85	1,065	65,440	2,339
2.00	11,880,000	1.36	79.78	2.91	519	30,470	1,113

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 have been produced on 50m intervals through the deposit. Estimated block grades have been 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 Figure 14-5 to Figure 14-9 for bench levels 2250 down to 2050.

Another check on the results has been completed by comparing the average composite grade for each domain with the average kriged grades for that domain. (Table 14-14)

Table 14-14 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
ASH	Au (g/t)	6,699	0.27	128,377	0.23
	Ag (g/t)	6,699	5.77	128,377	5.75
MHG	Au (g/t)	2,824	0.88	23,785	0.88
	Ag (g/t)	2,824	58.85	23,785	59.94
LGLM	Au (g/t)	13,568	0.16	224,130	0.14
	Ag (g/t)	13,568	9.94	224,130	7.92
LGSH	Au (g/t)	1,153	0.11	18,010	0.15
	Ag (g/t)	1,153	7.12	18,010	7.54
NEHG	Au (g/t)	910	0.63	14,716	0.70
	Ag (g/t)	910	42.74	14,716	44.65
NELGSH	Au (g/t)	7,253	0.07	176,070	0.09
	Ag (g/t)	7,253	6.40	176,070	6.33
WASTE	Au (g/t)	11,061	0.008	103,753	0.016
	Ag (g/t)	11,061	0.40	103,753	0.84

The following legend can be used to show the levels of gold found in the Figures below:

- Au >= 0.0 < 0.2g/t is shown in black
- Au >= 0.2 < 0.4g/t is shown in blue
- Au >= 0.4 < 0.6g/t is shown in green
- Au >= 0.6 < 0.8g/t is shown in orange
- Au >= 0.8 < 1.0g/t is shown in red
- Au >= 1.0g/t is shown in pink
- Composite show 10m above and below bench.

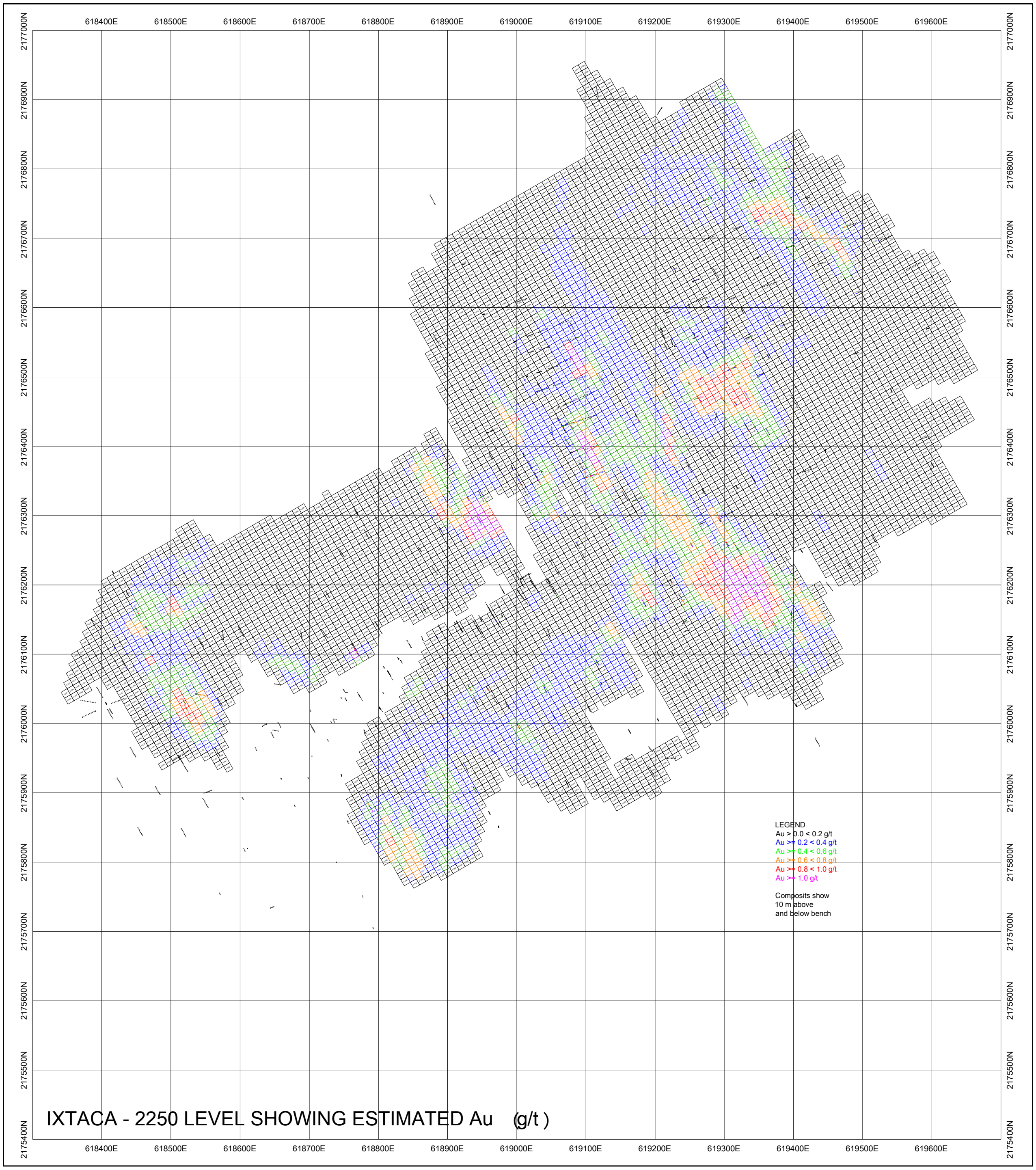


Figure 14-5 Ixtaca 2250 Level Plan Showing Estimated Gold in Blocks

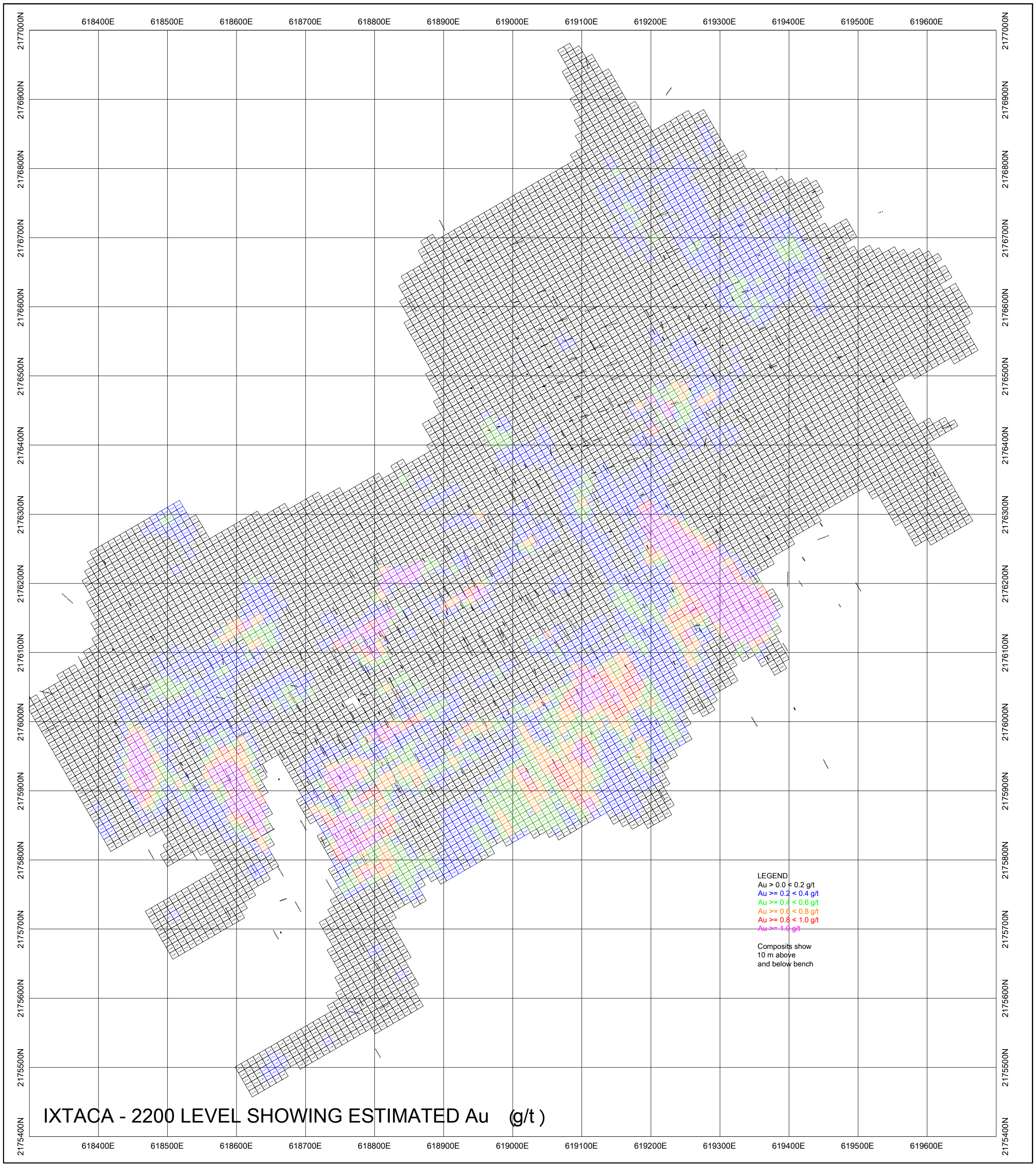


Figure 14-6 Ixtaca 2200 Level Plan Showing Estimated Gold in Blocks

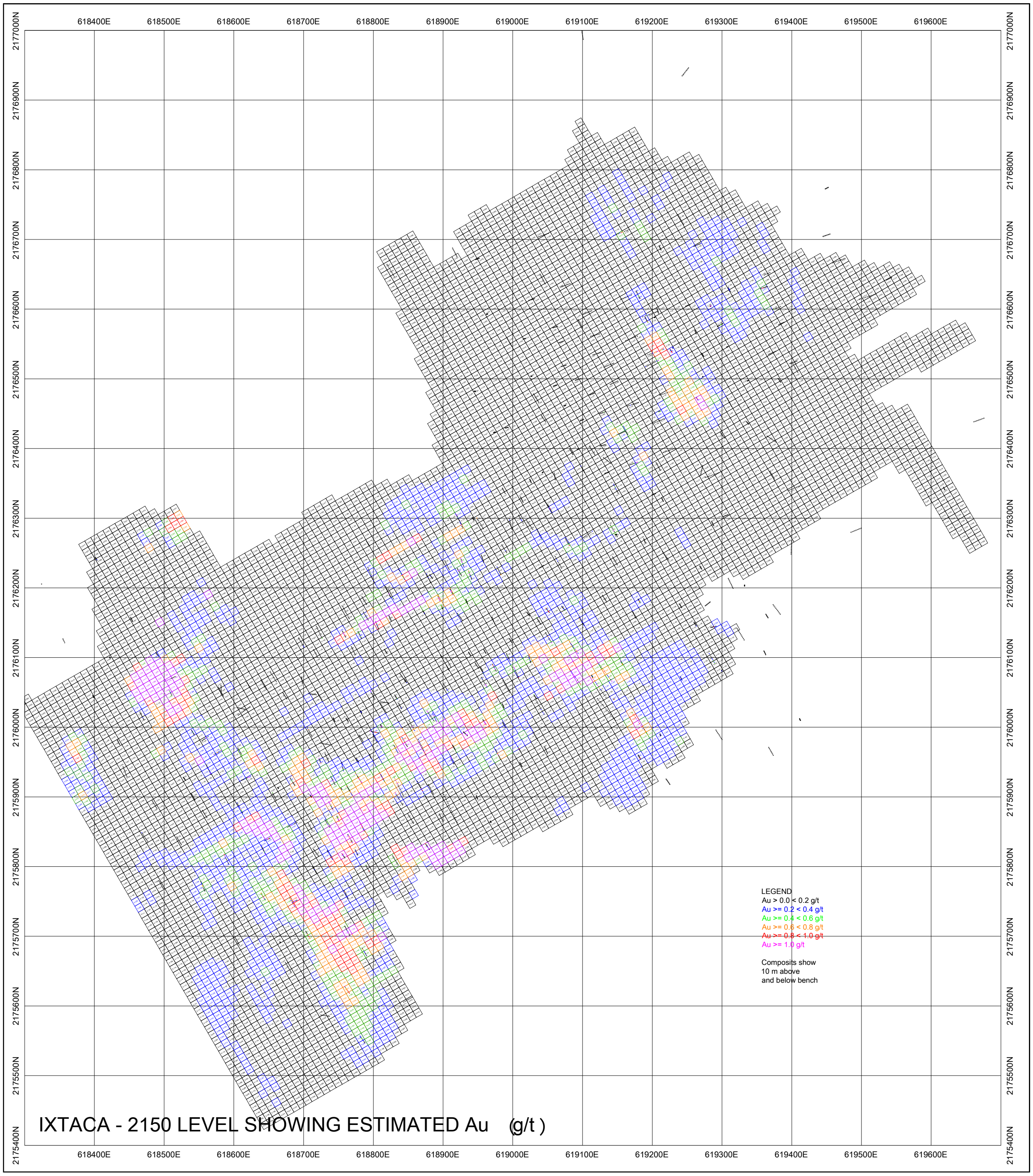


Figure 14-7 Ixtaca 2150 Level Plan Showing Estimated Gold in Blocks

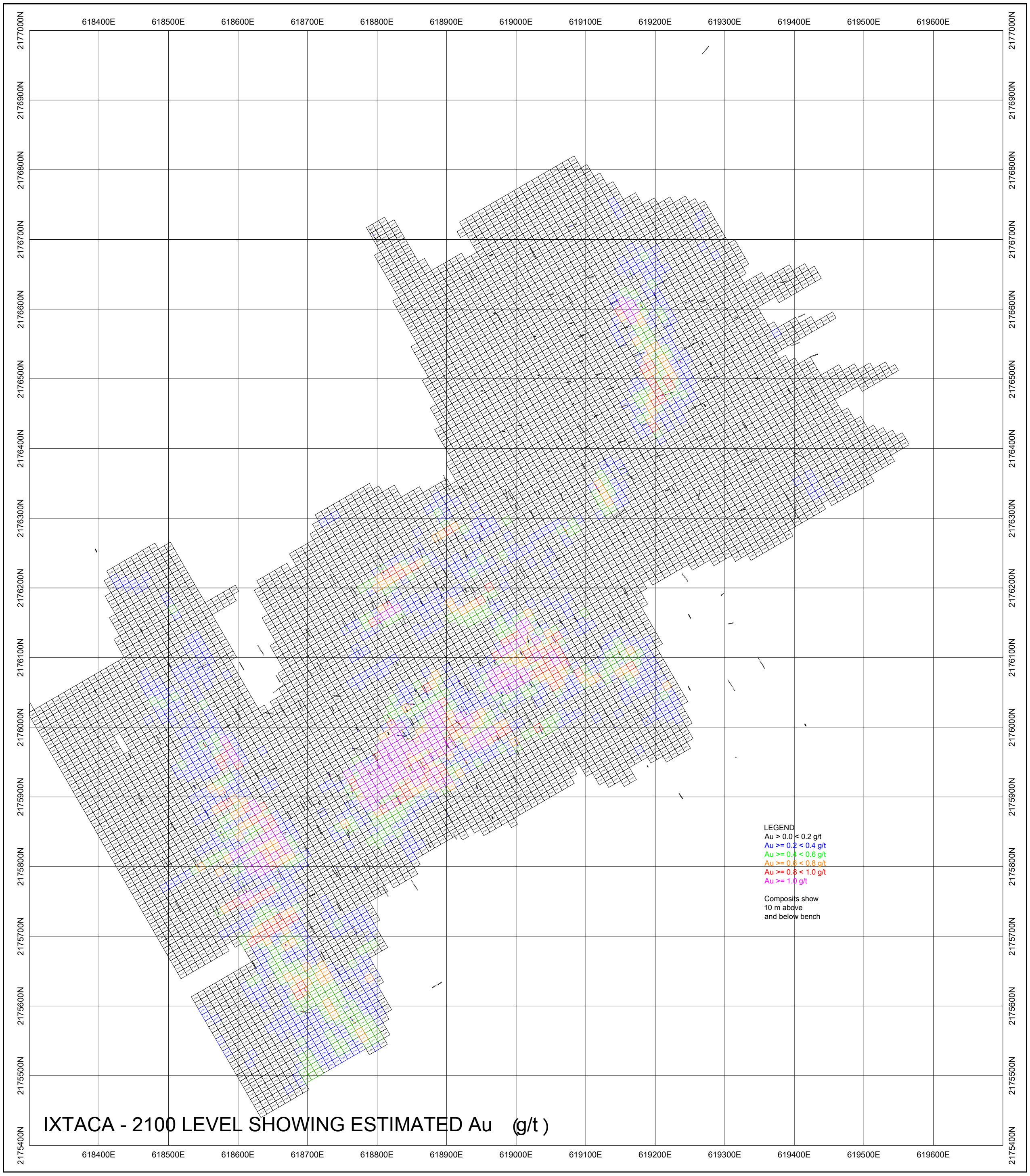


Figure 14-8 Ixtaca 2100 Level Plan Showing Estimated Gold in Blocks

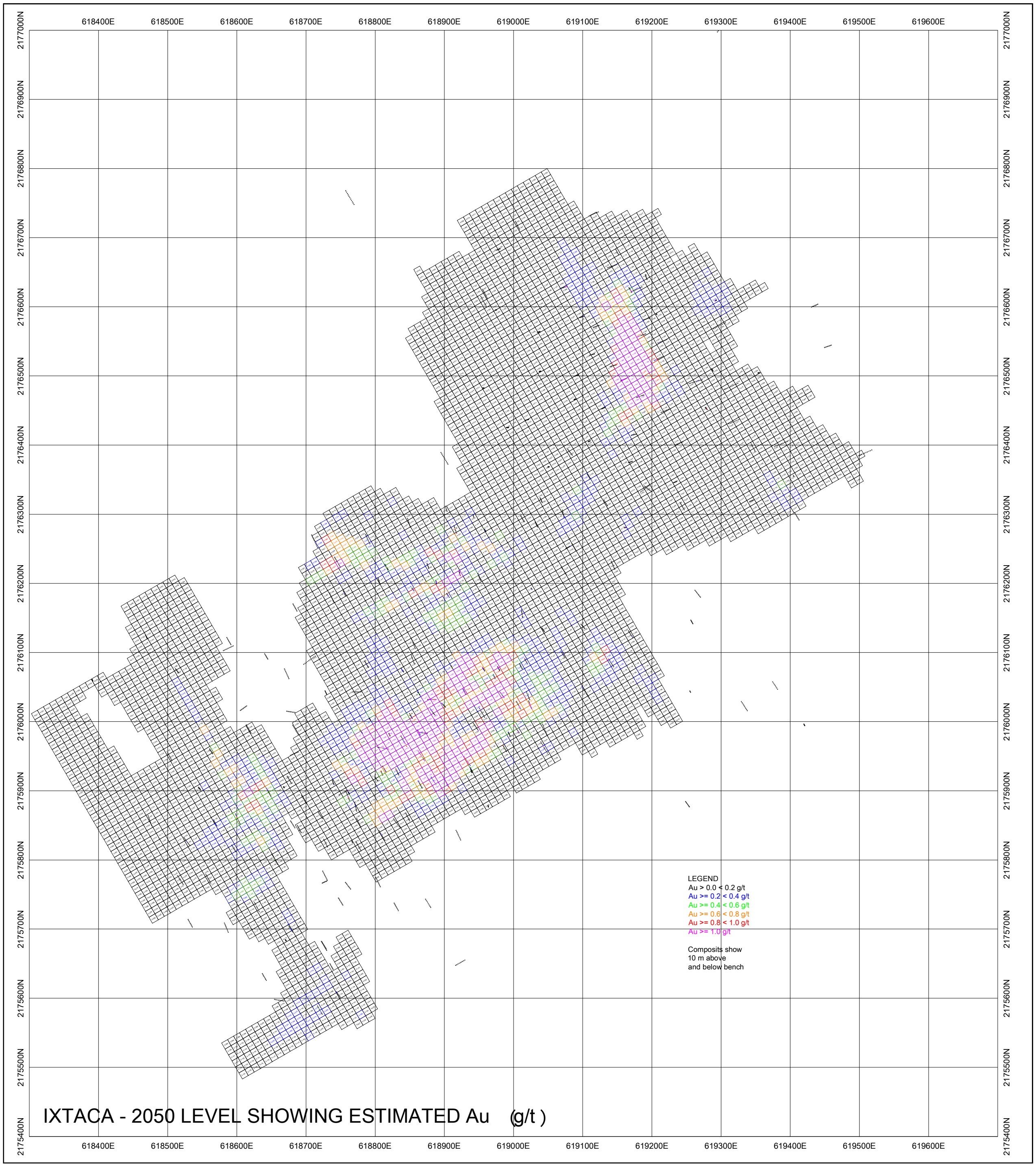


Figure 14-9 Ixtaca 2050 Level Plan Showing Estimated Gold in Blocks

15.0 MINERAL RESERVE ESTIMATES

No mineral reserve estimate has been completed on the Ixtaca deposit. This PEA study is conceptual in nature and includes Inferred Resources.

The CIM definition of a Mineral Reserve is that it is "the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study." A PFS has not been completed on the Ixtaca deposit and therefore it is not possible to declare a Mineral Reserve of any kind.

16.0 MINING METHOD

A Scoping level mine design, production schedule, and associated cost models have been developed for the Ixtaca Gold-Silver Deposit of the Tuligtic Property (hereafter referred to as “Ixtaca” or the “Project”). Two cases have been considered for this updated Preliminary Economic Assessment (PEA), as follows:

- Base Case - 30,000 tpd production schedule
- Ramp-Up Case - 7,000 tpd, ramping up to 9,000 tpd in Year 3 of production and to a maximum throughput of 30,000 tpd in Year 6 of production

This current work is based on the February 12, 2014 model resource update “Technical Report on the Tuligtic Project, Pueblo State, Mexico Pit phases are designed using the results of an economic pit limit analysis. Waste/mill feed break even cut-off grade (COG) is based on a net smelter return (NSR) of $9.0/tonne, assuming doré as the final product. Pit delineated resources calculated from the block model including mining loss and dilution are provided in Table 16-2.

Grade items use the 3D block model (3DBM) provided by GCL in January 2014. The grade items used are for each lithologic unit using gold (Au) and silver (Ag) grades and are reported in grams per tonne.

For this section the unit “t” refers to metric tonnes and “kt” refers to thousand metric tonnes. All currency amounts are in US dollars (USD) unless specified otherwise.

The following Table shows the breakdown of pit resources by category.

Table 16-1 Summarized In-Pit Resources

CLASS	Mill Feed kt	NSR ($/t)	Au (g/t)	Ag (g/t)
Measured	36,995	33.46	0.504	32.25
Indicated	68,357	26.33	0.421	23.57
Sub-Total of Measured and Indicated	105,352	28.83	0.450	26.62
Inferred	*19,944*	*21.56*	*0.323*	*20.89*

The potentially mineable tonnages in this updated PEA selected ultimate pit include Inferred Resources. The reader is cautioned that Inferred Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral reserves, and there is no certainty that Inferred Resources will ever be upgraded to a higher category.

All classes of the resources are considered as mill feed in the mine planning production schedule. To even out the waste stripping requirements in the mining schedule and to help ensure that higher grade areas are targeted sooner in the mining schedule, the pit is divided into smaller phases. The initial phase targets the higher revenue near surface, higher grade areas of the ultimate pit. The breakdown of resources (Measured, Indicated, and Inferred) by pit phase is shown in the Table below.

Table 16-2 Summarized In-Pit Resources by Pit Phase

Recovered In-pit Resources and Diluted Grade NSR ≥ 9.00						
PIT (i=incremental)	**Mill Feed kt**	**NSR ($/t)**	**Au g/t**	**Ag g/t**	**Waste kt**	**Strip Ratio t:t**
Phase1 - 631	15,401	33.45	0.472	34.64	21,702	1.4
Phase2 - 632i	53,014	29.43	0.497	24.38	104,341	2.0
Phase3 - 623i	56,881	24.47	0.356	24.53	92,015	1.6
TOTAL	**125,296**	**27.67**	**0.430**	**25.71**	**218,058**	**1.7**

16.1 Introduction

The objective of this updated PEA is to define and develop an executable mine configuration with a strong economic basis.

The main updates to this PEA from the maiden PEA include the following:

- selective mining to target higher grade areas within each lithology unit
- addition of a low capital ramp-up mining plan going to the same ultimate pit limit as the Base Case
- incorporation of the volcanics SG (average of 1.67 from the resource model) into the waste tonnages thereby reducing the capitalized pre-stripping tonnages (and costs)
- lower mining costs for volcanics due to reduced drilling and blasting in this material (some drilling and blasting is still required however)
- addition of a conveyor from the pit rim to the mill

The mine planning work for this study is based on the 3D block model (3DBM) received from GCL on January 15, 2014. For this updated PEA an updated waste model is used, incorporating significantly lower specific gravity for the volcanic rock unit. All mine-planning work is done with MineSight® (MineSight) including pit optimization, detailed designs and production scheduling with MineSight Strategic Planner (MSSP). Two production schedules; a Base Case and a low capital Ramp-Up Case are included and reported in this PEA.

In addition to the information contained in the block model, other data used for the mine planning includes the base economic parameters, mining cost data estimated by MMTS based on other similar projects, recommended preliminary pit slope angles (PSAs) and tailings design by Knight Piésold, estimated metallurgical recoveries, plant costs, and throughput rate targets.

16.2 Mining Datum

The project design work is based on NAD83 ZONE 14 N (meters). Topography (1m contours) provided by Almaden is based on a survey done by PhotoSat using WorldView2 satellite (50cm resolution, stereo). The 3D block model was provided by GCL on January 15, 2014 which was imported into MineSight.

16.3 Production Rate Consideration

A number of factors are considered when establishing an appropriate mining and processing rate. For Ixtaca, key factors include:

- Resource Size: A typical mine life is set at 12 to 20 years; less than this typically requires higher initial capital which needs time for amortization, and beyond this, time-value discounting shows an insignificant contribution to the NPV of the Project.
- Capital Payback: Capital investment typically is targeted at projects with a payback period less than 5 years.
- Operational Constraints: Power, water, or supplies and services capacities for support of operations can limit production.
- Site Delivery Constraints: Physical size and weight of equipment and shipping limitations can determine the maximum size of units that can be delivered to site.

Generally, economies of scale can be realized at higher production rates and lead to reduced unit operating costs. These are tempered to the above-mentioned physical and operational constraints and flexibility issues. Also higher tonnage throughputs generally require more capital and the size of the Project is reflected in the initial investment. Economies of scale can still apply where some access and construction issues have a high fixed component regardless of the size of the Project. Higher production rates generally pay back fixed capital earlier, and provide a higher rate of return on capital, which improves project NPV.

The Base Case mining schedule using a stockpile strategy for this Ixtaca study has a mill throughput of 30,000 tonnes per day; resulting in a project mine life of 12 years of production.

An alternate Ramp-Up Case mining schedule is analyzed to test the impact of reduced initial capital. The Ramp-Up Case uses a stockpile strategy and starts with a mill throughput of 7,000 tonnes per day for Years 1 and 2 increasing to 9,000 tonnes per day for Years 3 to 5 and then full capacity of 30,000 tonnes per day for Years 6 and onwards. The resulting project mine life is 15 years of production.

The results of the Base Case and Ramp-Up Case are presented in this report.

16.4 Mine Planning 3d Block Model and MineSight Project

A 3D Block Model received from GCL on January 15, 2014 is used for this study. An updated waste model is used in this updated PEA and incorporates significantly lower specific gravity (1.67 t/m3) as defined by the resource model for the waste rock volcanic unit, as opposed to a previously used average waste rock density of 2.55 t/m3.

The resource model is a rotated 3D block model (3DBM) with whole block and mineralized Au (g/t), Ag (g/t) grades, ORE%, specific gravity and Class. Mineralized grades are reported by lithology zone of which there are seven (shown in Table 16-6), therefore there are seven zones per block. Whole block Pb (%) and Zn (%) and Acid Rock Drainage (ARD) data grades are also included in the block model. ORE% (provided by GCL) represents the percentage of the block that is inside each lithology zone (seven zones per block). MMTS added a topography (TOPO) item representing the percentage of a block below the topography surface.

The model dimensions are shown in Table 16-3. Mine planning pit designs and scheduling utilize 10m benches (1 bench = 2 block height). This 10m bench height represents a suitable bench height for the size of equipment chosen. The 10m x 10m horizontal dimensions provide a suitable resolution for long range planning.

The block model is horizontally rotated 30^{o} counter-clockwise to line up with drill sections and with the orientation of the main mineralized structures.

Table 16-3 Ixtaca 3D Block Model Limits

	Min	Max	Block Size	# of Blocks
x	618418	620838	10	242
y	2174535	2176695	10	216
z	1590	2600	5	202

Table 16-4 Block Model Rotation Parameters

Origin		Angles	
Easting	618418	Horizontal	-30
Northing	2174535	Dip	0
Elevation	0	Plunge	0

A list of mine planning 3DBM items is given in Table 16-5 and a list of the lithological zones is given in Table 16-6. The total model area is illustrated in Figure 16-1

Table 16-5 Ixtaca 3DBM Items

Model Item	Source	Units	Description
TOPO	TOPO FULL (130315).msr		"TOPO FULL (130315).msr"
ORE1	Giroux Jan15 2014	%	% of block within Zone 1
ORE2	Giroux Jan15 2014	%	% of block within Zone 2
ORE3	Giroux Jan15 2014	%	% of block within Zone 3
ORE4	Giroux Jan15 2014	%	% of block within Zone 4
ORE5	Giroux Jan15 2014	%	% of block within Zone 5
ORE6	Giroux Jan15 2014	%	% of block within Zone 6
ORE7	Giroux Jan15 2014	%	% of block outside of all mineralized domains
AU1	Giroux Jan15 2014	g/t	gold grade of Zone 1
AU2	Giroux Jan15 2014	g/t	gold grade of Zone 2
AU3	Giroux Jan15 2014	g/t	gold grade of Zone 3
AU4	Giroux Jan15 2014	g/t	gold grade of Zone 4
AU5	Giroux Jan15 2014	g/t	gold grade of Zone 5
AU6	Giroux Jan15 2014	g/t	gold grade of Zone 6
AU7	Giroux Jan15 2014	g/t	Estimated gold grade for waste portion of block
AG1	Giroux Jan15 2014	g/t	silver grade of Zone 1
AG2	Giroux Jan15 2014	g/t	silver grade of Zone 2
AG3	Giroux Jan15 2014	g/t	silver grade of Zone 3
AG4	Giroux Jan15 2014	g/t	silver grade of Zone 4
AG5	Giroux Jan15 2014	g/t	silver grade of Zone 5
AG6	Giroux Jan15 2014	g/t	silver grade of Zone 6
AG7	Giroux Jan15 2014	g/t	estimated silver grade for waste portion of block
PCMIN	Giroux Jan15 2014	%	% of block within all mineralize domains
AUMIN	Giroux Jan15 2014	g/t	weighted average gold grade for mineralized portions of block

Model Item	Source	Units	Description
AGMIN	Giroux Jan15 2014	g/t	weighted average silver grade for mineralized portions of block
AUTOT	Giroux Jan15 2014	g/t	weight average gold grade for total block
AGTOT	Giroux Jan15 2014	g/t	weight average silver grade for total block
SG	Giroux Jan15 2014		weight average specific gravity for block (NOTE: some editing by MMTS for volcanic SG)
CLASS	Giroux Jan15 2014		block classification: 1 = Measured, 2 = Indicated, 3 = Inferred
PBTOT	Giroux Jan15 2014	%	Estimate for Pb in Total Block
ZNTOT	Giroux Jan15 2014	%	Estimate for Zn in Total Block
ORE%	MMTS	%	the sum of ore% of the mineralized zones (ORE1-6), does not include the waste portion of the mineralized blocks
ZN1	MMTS		set to 1 if the majority of the block is in Zone 1
ZN2	MMTS		set to 2 if the majority of the block is in Zone 2, otherwise undefined
ZN3	MMTS		set to 3 if the majority of the block is in Zone 3, otherwise undefined
ZN4	MMTS		set to 4 if the majority of the block is in Zone 4, otherwise undefined
ZN5	MMTS		set to 5 if the majority of the block is in Zone 5, otherwise undefined
ZN6	MMTS		set to 6 if the majority of the block is in Zone 6, otherwise undefined
ZN7	MMTS		set to 7 if the majority of the block is in Zone 7, otherwise undefined
NSR1	MMTS	$/t	NSR of the block using au1 and ag1
NSR2	MMTS	$/t	NSR of the block using au2 and ag2
NSR3	MMTS	$/t	NSR of the block using au3 and ag3
NSR4	MMTS	$/t	NSR of the block using au4 and ag4
NSR5	MMTS	$/t	NSR of the block using au5 and ag5
NSR6	MMTS	$/t	NSR of the block using au6 and ag6
NSR7	MMTS	$/t	NSR of the block using au7 and ag7
NSRd	MMTS	$/t	NSR of the block using whole block grades for a Dore product
LITHM	MMTS		The whole block code of the majority lithological zone
WST	MMTS		used to identify volcanics material for waste reporting: 1= Volcanics, 2 = everything else

Table 16-6 Lithological Zones

Zone	Lithology
1	Volcanics (ash, tuff)
2	Main high-grade limestone
3	Low-grade Limestone
4	Low-grade Shale
5	NE extension high-grade limestone
6	NE extension low-grade shale
7	Waste

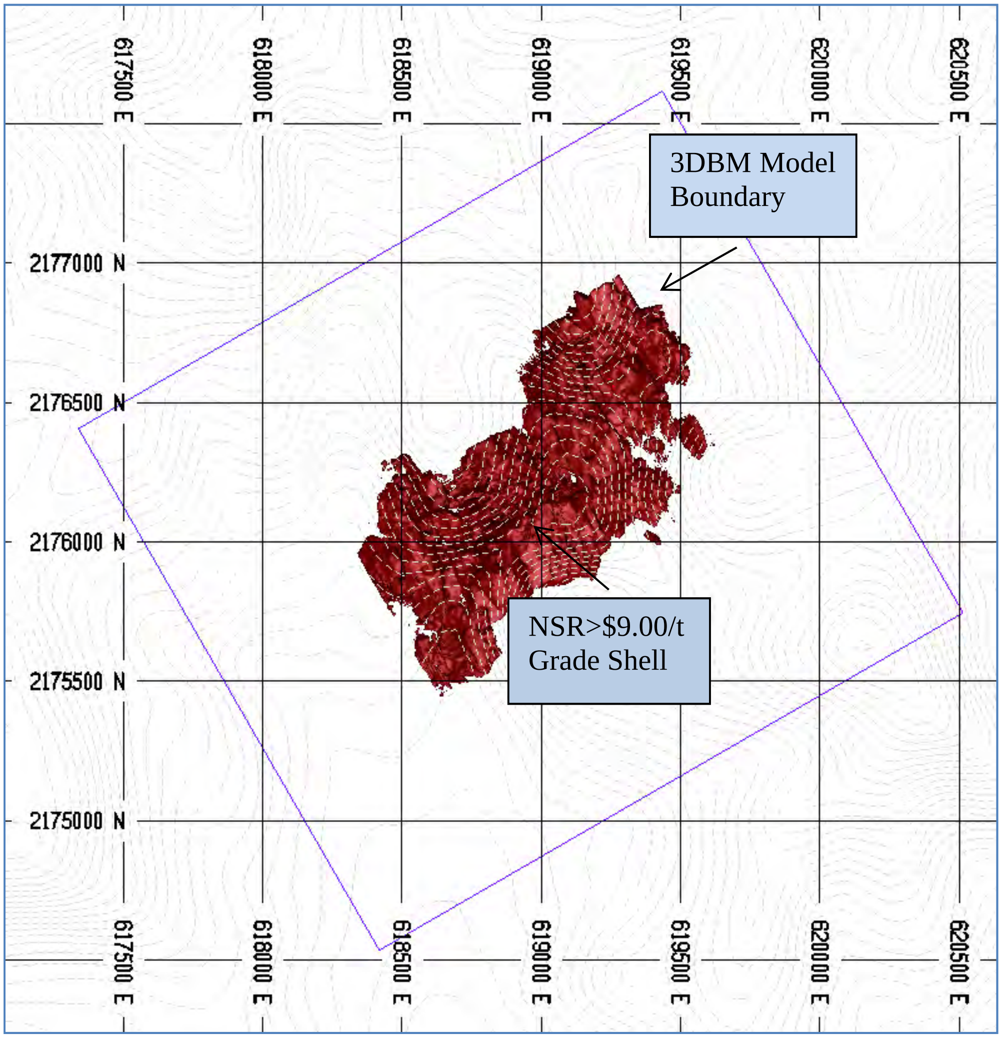


Figure 16-1 Ixtaca Resource Area with 10m Contours

16.5 Net Smelter Return (NSR)

NSR values ($/t) are calculated for each mineralized block in the resource model using Base Case Net Smelter Prices (NSP), process recoveries and the grade of each metal for each zone unit in the block. The NSR value is based on a multi-zoned estimation using zonal ORE% and grades where a single mineralized block may include as many as seven different zones of any of the seven lithology units listed in Table 16-6 above. The NSR is estimated, using the calculated value of metal for each zone in each block and added together to create a total NSR value for each block. The abbreviation NSP is used in this report to differentiate between Market prices and Net prices (net of smelting, refining, and offsite charges) where market price is the basis of scoping studies and is often used to compare to other studies. NSP is based on the market price but applies the stated smelter and refining terms (detailed in Appendix E) to arrive at an internal price value used through the analysis to generate value based resource tonnages and revenues in cash flow analysis. The NSP can be considered the price available at the mine gate to generate revenues to be applied against mining, processing, and G&A costs for the operation.

16.6 Mining Loss and Dilution

The pit-delineated resources used for scheduling are calculated from in-situ multi-zone grades in the 3DBM using detailed pit designs with the appropriate mining recoveries and dilutions applied. The recoveries and dilutions convert the in-situ mill feed tonnages and grades into a Run-of-Mine (ROM) delivered tonnage and grades to the mill. The ROM delivered tonnage (i.e. what the mill will actually "see") is used to determine the appropriate production schedule. The following describes the method used to estimate mill feed loss and dilution resulting from the mining process.

There are three main parts to mining loss and dilution:

- Dilution of waste into mill feed where the bulk mining operations don't allow selectively mining isolated waste zones from larger mill feed zones;
- Loss of mill feed into waste where the bulk mining operations don't allow selectively mining isolated zones from larger waste zones;
- General mining losses and dilution due to handling (haul back in truck boxes, stockpile floor losses, etc.)

The Selective Mining Unit (SMU) plays a large part in determining the effect of the issues listed above and are related to the size of the digging unit and the mining rate. At a PEA level of study the block size is considered to be the SMU.

The Ixtaca zone is an epithermal gold-silver deposit that can be mined with large scale open pit mining methods. Lerchs-Grossman (LG) shells are produced for the deposit and an optimal potential open pit shape is selected. The mill feed from the ultimate pit is enough to feed the mill for 12 years at a rate of 30,000tonnes/day. The block model provided by GCL has 10m x 10m x 5m block sizes. Each block in the model has a volume of $500m^3$ and weighs approximately 1,275 tonnes. The plant feed will require approximately 23.5 blocks per day, which confirms the block size as an appropriate SMU for the size of shovel utilized in the mine plan.

The incremental break-even NSR cut-off used to determine mill feed for the production schedule is NSR$\geq$9. The estimated mill feed grade includes minimal internal dilution, and an additional external mining dilution of 3% is applied to in-pit resource estimates based on typical open pit mining practices.

Table 16-7 Ixtaca Mining Loss and Dilution

Mining Loss	Contact Dilution	Dilution Grade		
		NSR	AU	AG
%	%	($/t)	g/t	g/t
3	3	8	0.11	7.7

Overall mining loss is assumed to be 3%. This is an allowance to cover mill feed material that gets misdirected in material handling. Details on how dilution and dilution grades are determined are shown in Appendix F.

16.7 Economic Pit Limits and Pit Designs

Economic pit limits for the Ixtaca deposit have been determined for this study with a Lerchs-Grossman (LG) pit optimization using MineSight Economic Planner (MS-EP). Sensitivity to metal price, mine costs, processing cost and slope angles, are included in this analysis.

16.7.1 Pit Optimization Method

The pit optimization is based on the updated resource 3DBM as described above. For this PEA study, Measured, Indicated, and Inferred (MI & I) class resources are included in the LG economics.

The LG assessment is carried out by generating sets of pit shells of varying revenue assumptions and pit slopes to test the deposits geometric/topographic and pit slope sensitivity.

The ultimate economic pit limit is often determined by time value discounting deeper or longer term material. However, the larger size of the undiscounted economic pit limit and the targeted mill throughput means discounting would apply to material in excess of 20 years of production. This does not reflect the undiscounted potential of the resource. To determine if this deposit is “time sensitive” additional analysis (using a 10% discount rate applied to the mill feed material revenue) has been done based on approximate schedules of pit phases. This analysis determines the impact of the discounted revenue from the deeper mill feed and later phases that require up-front pre-stripping costs. The “deeper” mill feed material in the north-east area of the optimized pit shapes is impacted the most with discounting. For this Project it has been decided that proper pit phase selection decreases the impact of discounted mill feed, therefore an undiscounted method is used to select the ultimate pit.

Subsequent to determining the ultimate economic pit limit, the project team selected a smaller, higher margin pit limit for the basis of this study.

16.7.2 Economic Pit Limit Assessment

This section describes basis for mining and processing costs used in the LG Analysis.

16.7.2.1 LG Pit – Unit Mining Cost

Mining unit costs to generate the LG pit shells are based on the MMTS database for actual operating costs from similar operations in Mexico.

The unit mining costs per tonne mined used in the LG pit are shown in the following Table 16-8.

Table 16-8 Unit Mining Cost per tonne

Drilling	5%	0.07	$/tonne
Blasting	15%	0.21	$/tonne
Loading	15%	0.21	$/tonne
Hauling	55%	0.77	$/tonne
Pit Maintenance and G&A	10%	0.14	$/tonne
Total	**100%**	**1.40**	**$/tonne**

Mining costs include fuel, tires, maintenance parts and labour and operator labour costs.

16.7.2.2 LG Pit – Processing Operating Cost

The processing operating cost is estimated from benchmarks of similar processing operating plants, including operations located in Mexico, and adjusted to reflect local electrical energy cost, and labor cost.

The average unit processing costs per tonne milled used for the pit optimization are shown in Table 16-9.

Table 16-9 Ixtaca Unit Process Costs

($/t)	Unit Cost
Processing	9.50
General & Administration	1.74
Total Process Cost ($/t)	**11.24**

16.7.3 Pit Slope Angles

Maximum Pit Slope Angles (PSAs) are based on recommendations from Knight Piésold.

Knight Piésold has recommended varying the maximum inter-ramp-angles (IRA) by specific material zones delineated in the pit area. The resultant overall pit slope angle considers the maximum IRA for each material zone and includes flatter upper slopes, haul ramps and/or wider benches. Table 16-10 summarizes pit slope assumptions.

Table 16-10 Ixtaca Pit Slope Assumptions

Geological Unit	Vertical Distance between Berms (m)	BFA (degrees)	Berm Width (m)	IRA (degrees)	Max Inter-Ramp Slope Height (m)	Overall Slope Angle (degrees)
Volcanic Tuff	20	65	10	46	200	43 to 45
Sedimentary Package	20	70	10	49	200	

16.7.4 Process Recoveries

The LG runs use whole block grades AuTOT & AgTOT directly from the 3DBM. The final product is assumed to be doré. The appropriate process recoveries are applied, using input from a MMTS metallurgical specialist based on completed metallurgical testing at the Blue Coast Labs. Revenues from secondary metals Pb and Zn are not considered in this PEA study and are not included in the LG runs. See Table 16-11 below for process recoveries used in the LG pit economics.

16.7.5 LG Economic Pit Limits

The ultimate economic pit limit is estimated as the pit size where an incremental increase in pit size does not significantly increase the pit-delineated resource; in other words, where the expansion of the pit shell has limited potential for a positive economic margin. The selected (undiscounted) ultimate pit limit is chosen where the incrementally larger pits produce marginal or negative economic returns.

A series of LG Pit surfaces are created to assess the incremental economics of increasing the mining limits. By varying the metal prices from low to high values, the geometry of the mineralized deposit is tested, where low metal prices require high grades and/or low strip ratios to generate an economic pit shape, and high metal prices can generate incremental revenues to mine deeper, higher strip ratio material or lower grade zones. The larger pit shapes create larger mineable resources capable of supporting larger capital expenditure, but the extra material has lower economic margins (revenues minus cost) where-as the smaller pit shapes have higher margins but create smaller projects and can be more capital sensitive. Note: This is a not price sensitivity study since all in-pit resource tonnages are calculated at the Base Case metal price and corresponding NSR cut-off grade.

The optimization parameters are shown in Table 16-11 below.

Table 16-11 Pit Optimization Parameters

Gold Price	$/oz	$1300
Silver Price	$/oz	$22
Gold Overall Process Recovery	All metallurgical zones	90.3%
Silver Overall Process Recovery	All metallurgical zones	90.3%
Process Cost + G&A	$/tonne milled	$11.24
Mining Cost	$/tonne	$1.40
Overall Pit Slope	degrees	43

The Base Case LG pit series are shown graphically in Figure 16-2.

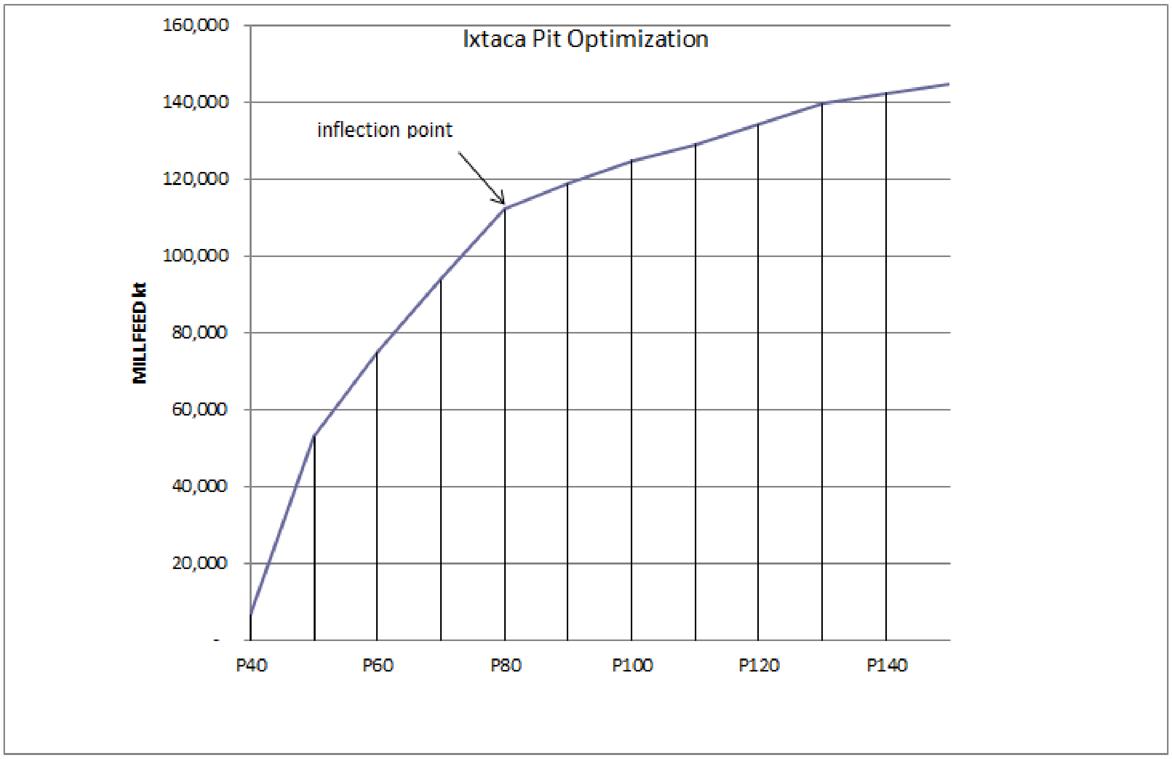


Figure 16-2 LG Sensitivity Summary

The incremental economics of the pit shell expansions to the right of the inflection point indicate only marginal economic return. The graph inflection point is at the 80% case where an incremental increase in the revenue driver (NSP) does not significantly increase the pit resource size. The biggest inflection is at 80%; however the pit profit continues to moderately increase to 100% where there is another more modest inflection point. Tables for this graph may be found in the Appendix D.

See Figure 16-3 for a representative sectional view.

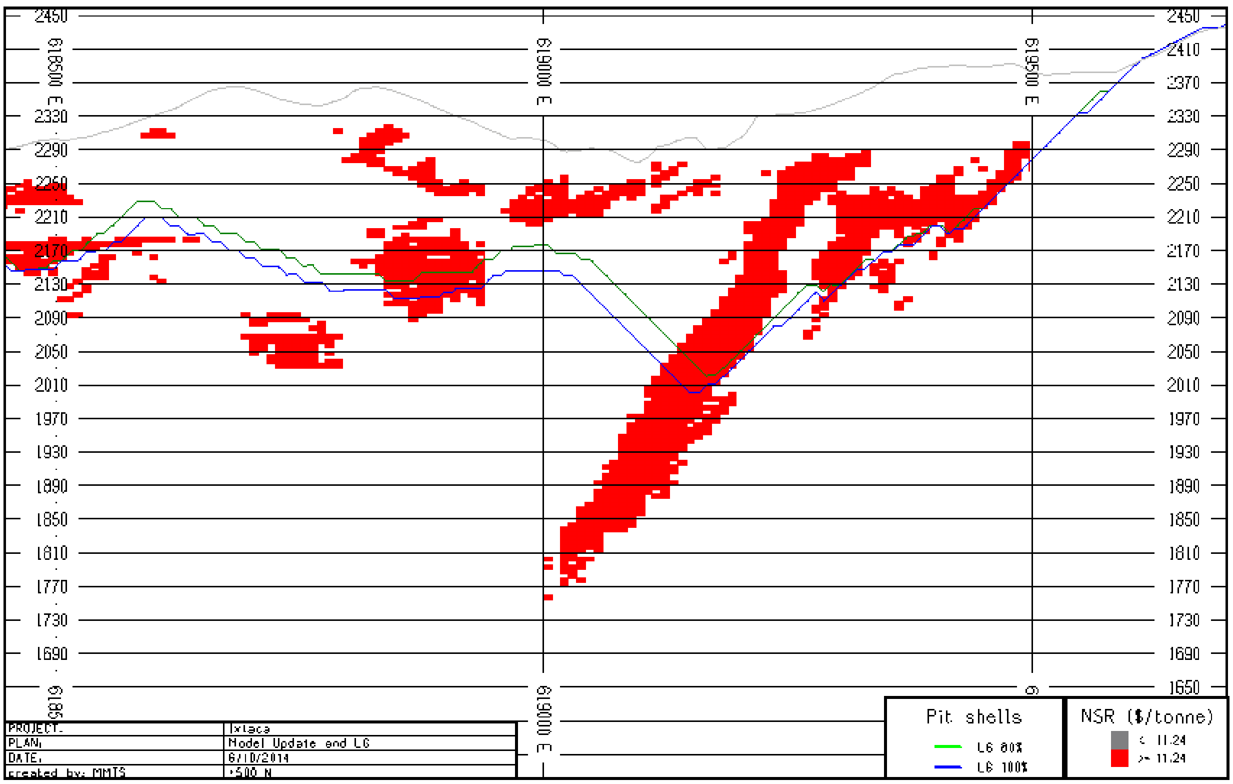


Figure 16-3 Section +500N

The P80 optimized pit shape (LG 80%) is selected to be used as a guide for the ultimate pit design. Choosing the P80 pit shell for the ultimate pit design allows for a buffer against a drop in metal prices from the base case prices used or increases in operating costs in later years of the mine life.

The ultimate pit in-situ resource with whole block grades is listed in Table 16-12 below.

Table 16-12 Ultimate LG Pit Resource

PIT	INSITU Mill Feed kt NSRd≥11.24				WASTE	TOTAL		Total Contained Metal	
	kt	NSR	Au (g/t)	Ag (g/t)	kt	kt	S/R	Au ('000 oz)	Ag ('000 oz)
P80	112,035	30.96	0.483	28.586	280,415	392,450	2.5	1,739	102,968

Cost and slope sensitivity are analysed to determine how these factors may affect the ultimate economic pit limit. Series of LG runs analyzed varying the mill feed mining cost, waste mining cost, processing cost, slope variance and recovery variances. Figure 16-4 summarizes the mill tonnes sensitivity series.

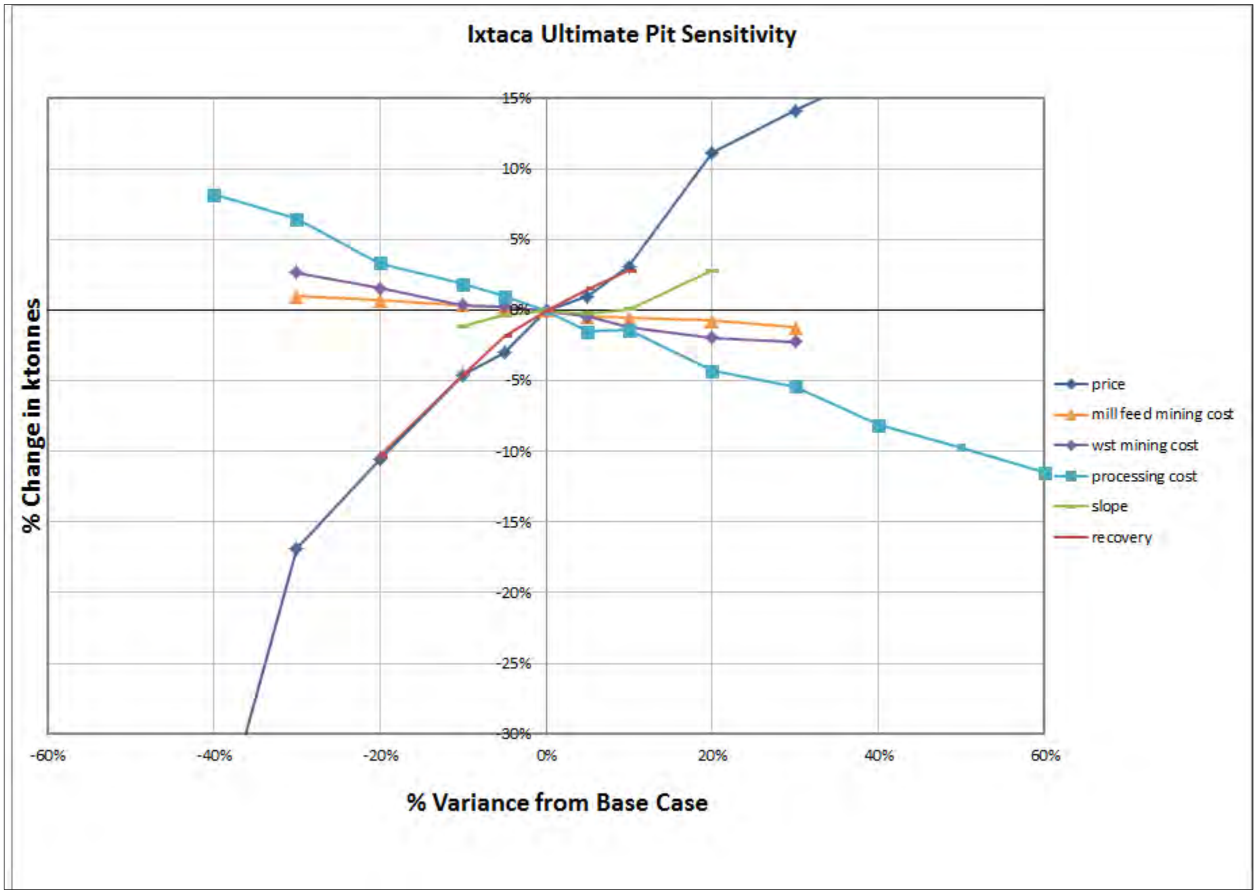


Figure 16-4 **Ultimate Pit Sensitivity**

From this sensitivity analysis, it may be noted that price, recovery and process cost have the steepest curves which indicates the highest sensitivity to resultant ultimate economic pit size. As well, pit limit is least sensitive to changes in operating cost or small changes in pit slopes.

A plan view of the LG pit limits is shown in Figure 16-5.

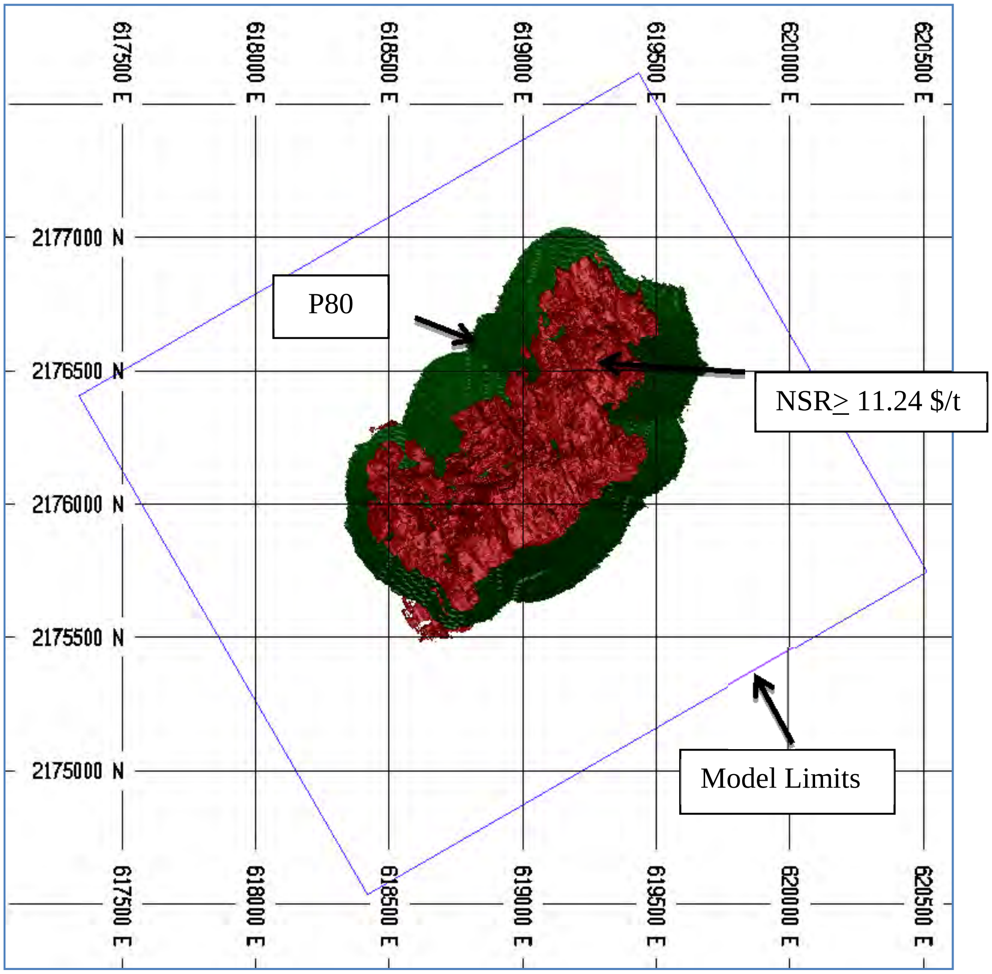


Figure 16-5 Plan view of Ixtaca P80 pit shells and NSR

The Figure below shows a cross-section of the P80 pit shell with NSR block grade item.

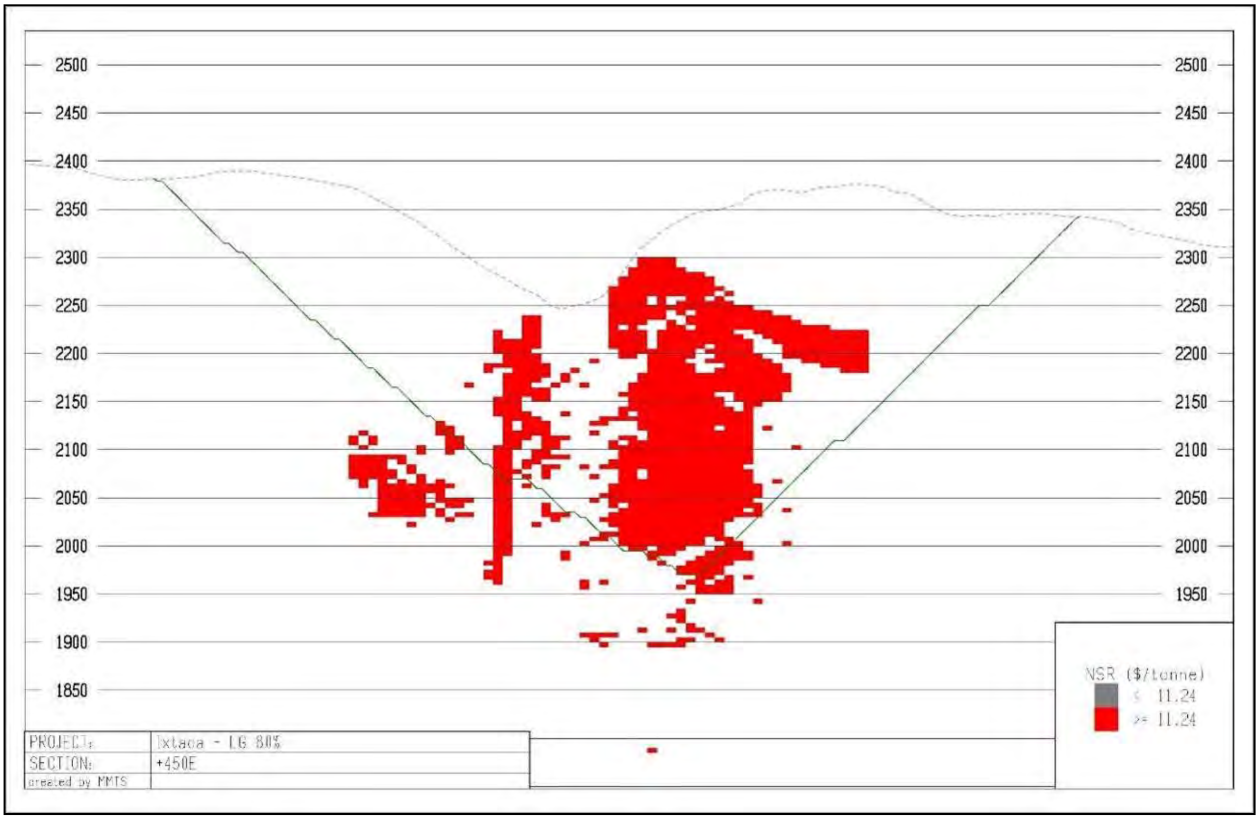


Figure 16-6 Sample Cross-section (+450E looking North-East)

16.7.6 Detailed Pit Design

MMTS has completed PEA level pit designs that demonstrate the viability of accessing and mining the potentially economic resources at the Ixtaca site. The designs are developed using MineSight software, estimated geotechnical parameters, suitable road widths for the chosen equipment size, and minimum mining widths based on efficient operation of the mining equipment chosen for the Project.

16.7.6.1 Haul Road Width

Haul road widths are designed to provide safe, efficient haulage, with the following minimum width specifications:

- For dual lane traffic, a travel width of not less than three times the width of the widest haulage vehicle used on the road is required.
- Where single lane traffic exists, a travel width of not less than two times the width of the widest haulage vehicle used on the road is required.
- Shoulder barriers should be at least three-quarters of the height of the largest tire on any vehicle hauling on the road along the edge of the haulage road wherever a drop-off greater than 3.2m exists. The shoulder barriers are designed at 34° face slope, which is slightly less than the angle of repose. The width of the barrier must be added to the travel width to get the total road width.

Ditches are included within the travel width allowance. For crowned haul roads, the width of this ditch allowance is 5m. Ditches are not added to the in-pit high wall roads; there is adequate water drainage at the edge of the road between the crowned surface and lateral embankments, such as high walls or lateral impact berms. During run-off, when water is flowing, this ditch allowance is still part of the available running surface if required, and can be used as lateral clearance for haul trucks. It can also be driven on, if required, to avoid obstructions. In practice, specifically-designed excavated ditches in haul roads tend to be filled in by road grading and, when maintained as open ditches, can create a hazard if the wheel of a haul truck or light vehicle should happen to get caught in them. Avoiding the addition of ditch width to the three-truck travel width on the in-pit high wall roads can significantly reduce the pit waste stripping.

Based on a 91-t truck, the haul road design basis is as follows:

- Largest vehicle overall width: 6.1m
- Double lane high wall haul road allowance: 23.7m
- Single lane high wall haul road allowance: 17.6m
- Minimum shoulder berm width: 5.4m

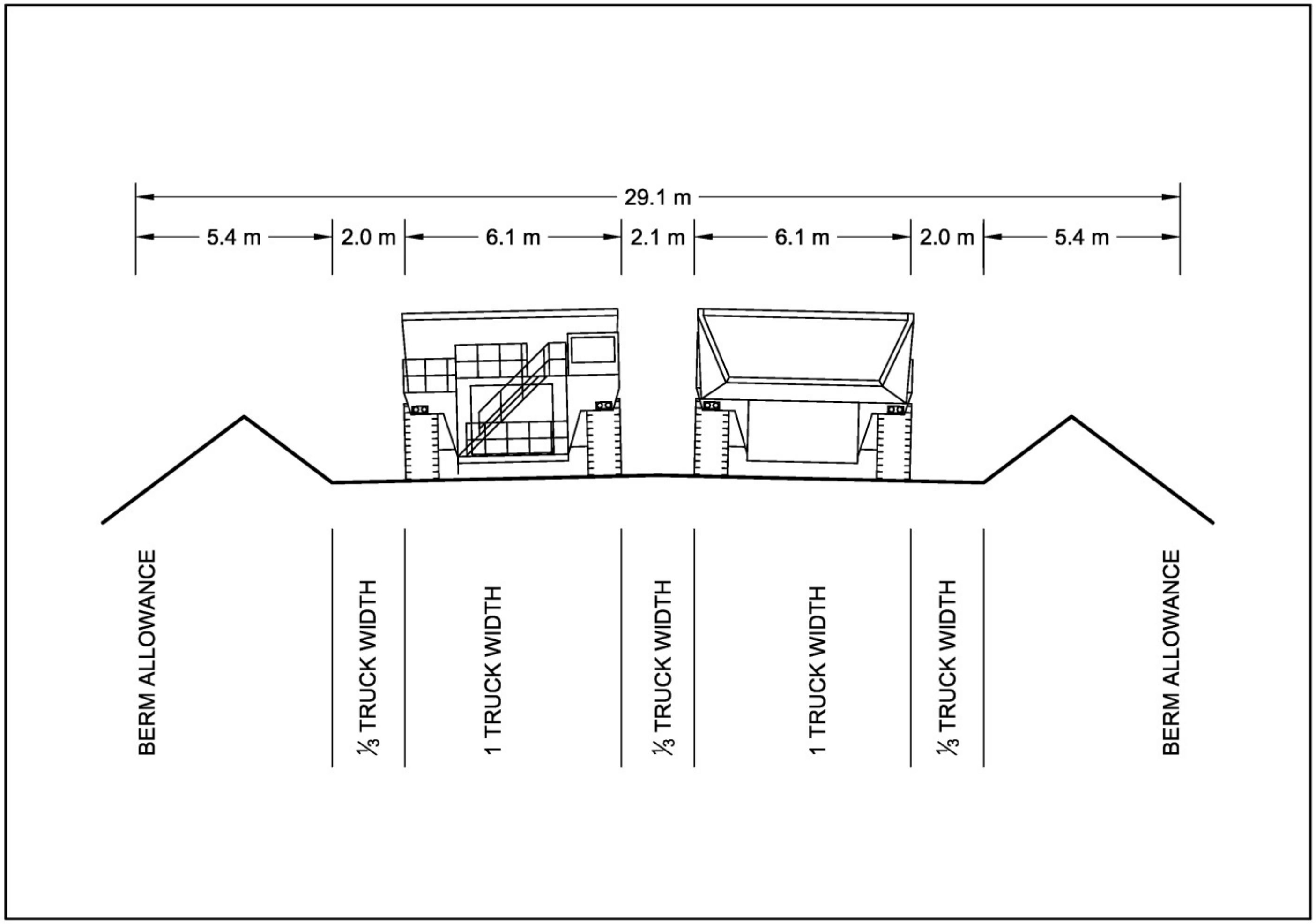


Figure 16-7 Dual Lane External Haul Road Cross Section

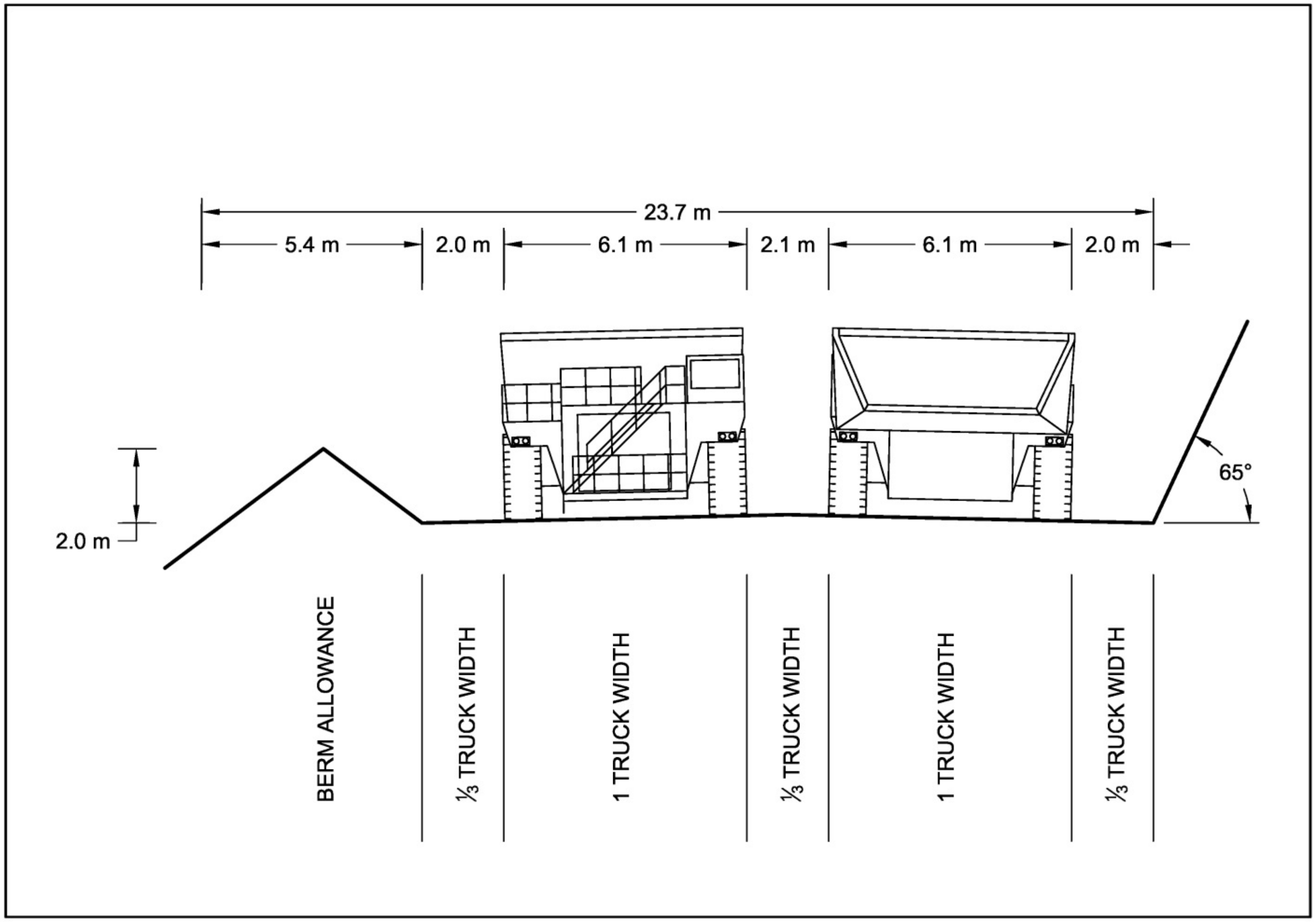


Figure 16-8 Dual Lane High Wall Road Cross Section

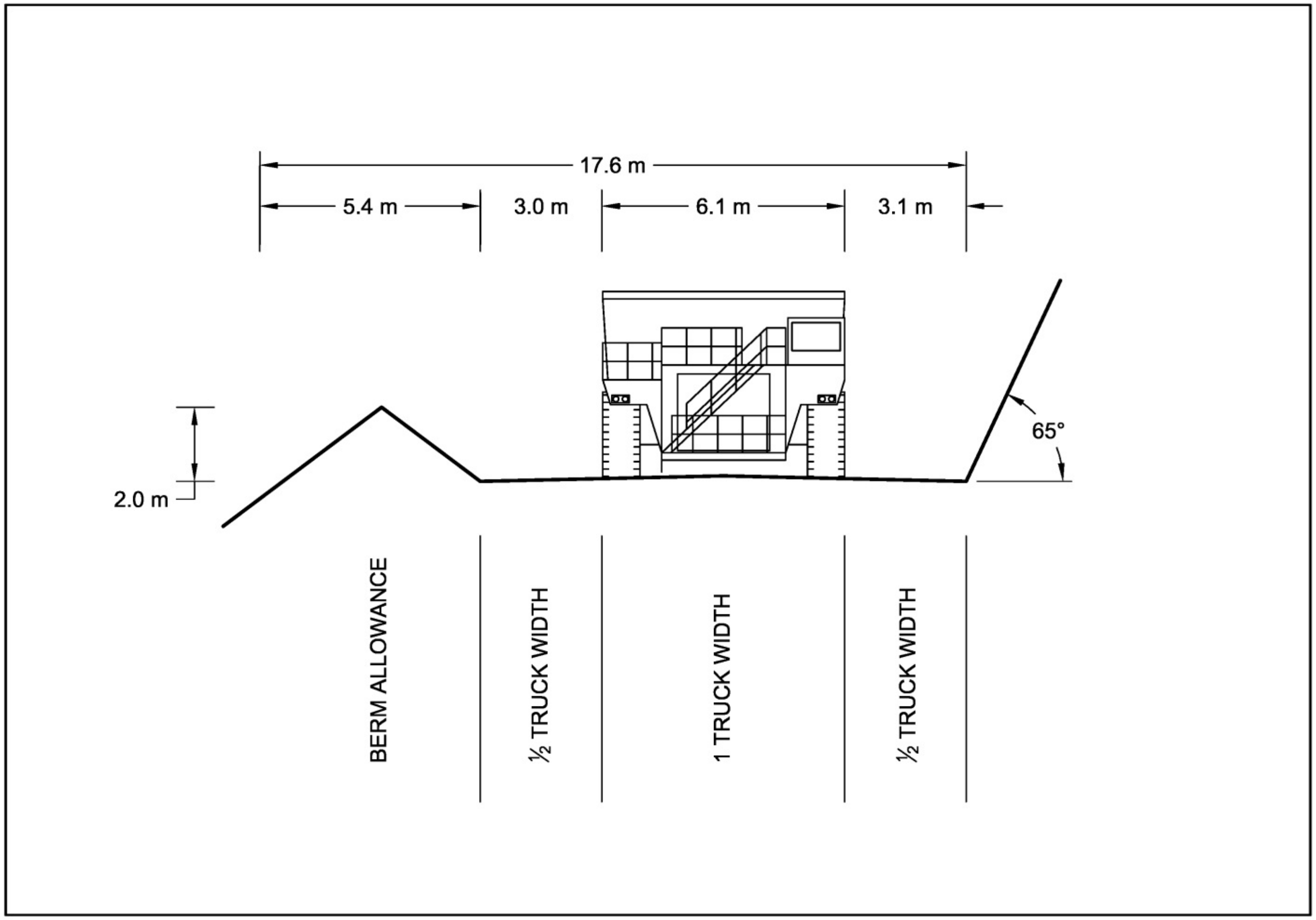


Figure 16-9 Single Lane High Wall Haul Road Cross Section

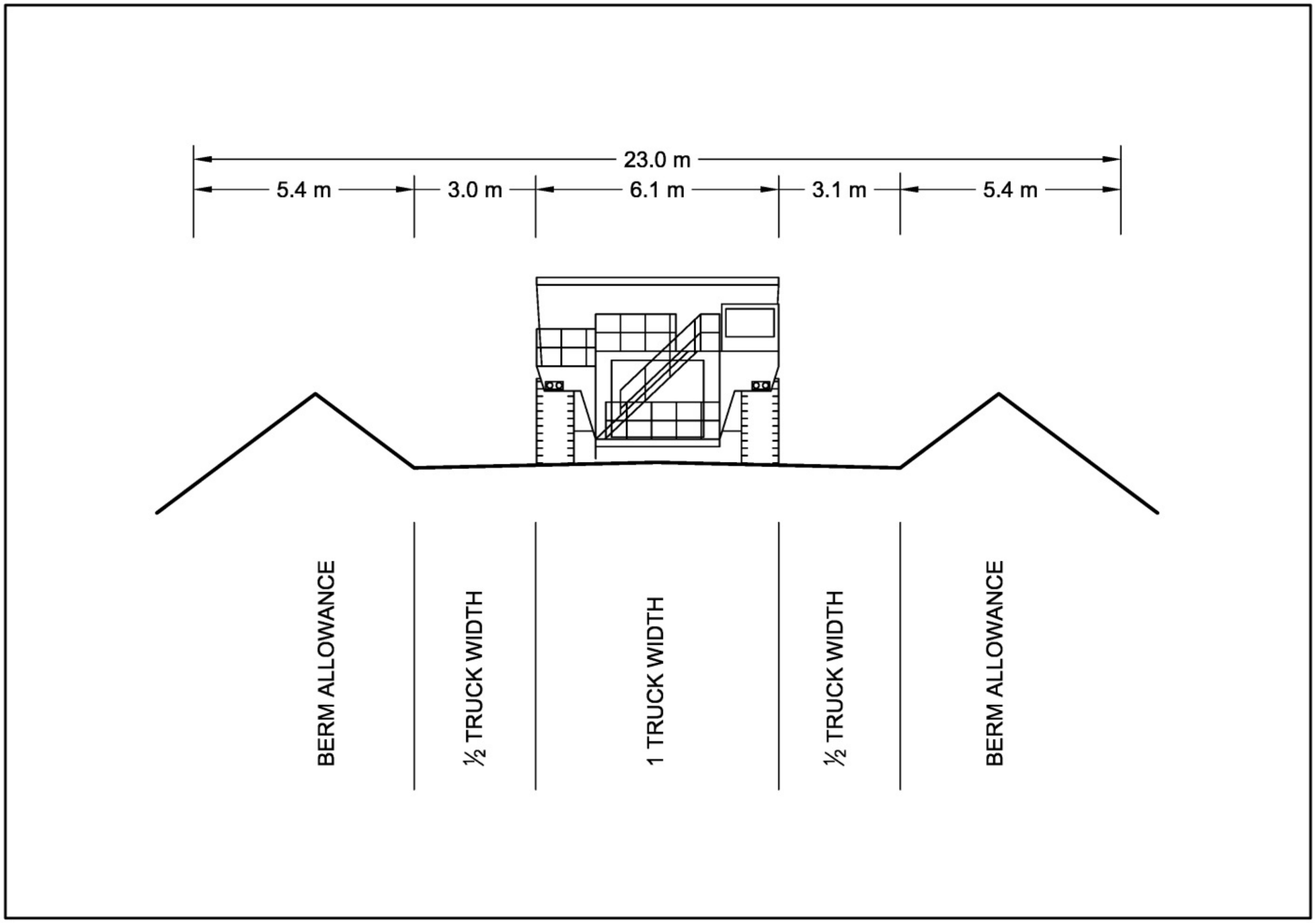


Figure 16-10 Single Lane External Haul Road Cross Section

The haul road widths and cut slope geometries outlined above may or may not be suitable for external haul roads placed on steep side slopes where significant rock cuts and fills are required. These need to be evaluated on a site-by-site basis to accommodate the geology and geometry of the road alignment. The design parameters outlined above (road width, cut slope angles, heights between benches, etc.) are suitable for this level study and will need to be assessed in higher level studies, with regards to the quality of the rock and the geologic structure in the area where the road exits the pit. This can vary depending on the location of the haul roads with respect to the pit design sectors, and may be influenced by the bench geometries in that area.

16.7.6.2 Pit Design Basis

The design standards applied in the current pit designs are summarized in the following two tables.

Note: the initial pit design basis assumed that 91tonne truck equivalent sized haul trucks would be the standard equipment, however after scoping-level production schedules were completed, a larger 177tonne equivalent sized haul truck is considered a more economic and feasible haul truck option. The pit designs have not been updated for the larger sized trucks but will be on the next study. Modifications to the ramp width to accommodate larger truck sizes would require an extra 5m width added to high wall ramps and may reduce the grade on the ramps to 8% (from 10% current design basis) but for a scoping level study the difference is not significant.

Table 16-13 Design Basis for Pit Design

Geological Unit	Vertical Distance between Berms (m)	Bench Face Angle (degrees)	Berm Width (m)	Inter-Ramp Angle (degrees)	Maximum Inter-Ramp Slope Height (m)	Overall Slope Angle (degrees)
Volcanic Tuff	20	65	10	46	200	43 to 45
Sedimentary Package	20	70	10	49	200	

**Note: The maximum height of these inter-ramp slopes should not exceed 200m. A 25m wide berm is included for geotechnical stability, where the vertical distance between high-wall ramps is greater than 200m.*

Table 16-14 Equipment Guidelines

Equipment Fleet **Major Mining Fleet**		
Shovels	$15m^3$ shovel	
Trucks	91tonne truck	
Max pit ramp slope		10%
Waste Dump Angle of Repose		37°
Largest Vehicle Overall Width		6.1m
Maximum Tire Height (27.00-R49)		2.7m
Minimum Haul road outside berm height		2.0m
Minimum Shoulder / Berm Width		5.4m
External road ditch		0m
Double lane high-wall haul road allowance		23.7m
Single lane high-wall haul road allowance		17.6m

16.7.6.3 Minimum Mining Width

A minimum mining width for each pit phase is specified to maintain a suitable mining platform for efficient mining operations. This width is established based on equipment size and operating characteristics. For this study, the minimum mining width generally conforms to 35m, which provides sufficient room for 2-sided truck loading.

16.7.6.4 Access Considerations

Access considerations maintain that haul roads must have a travel surface at least twice the width of the widest haul vehicle (one-way roads) and at least three times the width of the widest vehicle (two-way roads). One-way roads are not normally employed for main long term haul routes because they limit the safe by-passing of trucks and consequently lead to reduced productivity. One-way roads are, however, an appropriate option for low volume traffic flow or shorter-term operations. For this report, the use of one-way haul roads is limited to the bottom two or three benches of some pit phases. An access ramp is not designed for the last two benches of each pit bottom, assuming that temporary ramps will be used to access the pit bottom and will be removed upon retreat.

External pit road grades are designed at a maximum grade of 8%. Internal pit haul ramps use a maximum 10% and switchbacks are designed flat, with ramps entering and exiting at design grade. In practice however, grades will be transitioned so that visibility and haul speeds are optimized going around the switchback.

16.7.6.5 Variable Berm Width

Pit designs for Ixtaca are designed honouring overall IRAs, a nominal bench face angle of 70° (except in volcanics where it is 65°) and a minimum safety berm width of 8m. Where the vertical distance between high-wall ramps is greater than 200m, a 25m wide berm is added for geotechnical stability (based on the guidelines provided by Knight Piésold).

Where haul roads intersect designed safety benches, the haul road width is counted towards the safety berm width.

16.7.6.6 Bench Height

Ixtaca pit designs are based on the digging reach of the shovels (10m operating bench) with double benching between high wall berms; therefore, the berms are separated vertically by 20m. Single benching will be employed, if required, to ensure mill feed and maintain the safety berm sequence as warranted.

16.7.6.7 Pit Phase Selection

The ultimate pit limit is selected as the P80 pit shell, based on pit optimization economics. Phase 1 is derived from the P40 pit shell, which includes the higher grade economic material at a low strip ratio. The Phase 2 mines out to the western and southern limits of the P80 pit shell, while Phase 3 includes the remainder of the P80 pit shell material to the northeast.

For most hard rock/metal deposits the optimal sequence of pit phases arises from mining the areas within the ultimate pit that have higher economic margins due to higher grades, lower strip ratio, or both. Phase 1 has the lowest strip ratio and highest economic return based on higher grades.

Backfilling waste into mined out sections of the pit can reduce mine waste haulage costs as well as reduce the area of disturbance compared to disposing waste rock in external dumps. Generally with sequential pit phases, the third phase may partially backfill the mined out first phase, the fourth phase the mined out second phase and so on. For the Base Case mine plan, three pit phases are designed.
Backfilling is only minimally available since most of Phase 3 waste is mined out by the time Phase 1 is available for backfill.

The north-east portion of the Ixtaca ultimate pit limit is determined to be the most "time sensitive" based on results from a discounting sensitivity analysis. Because of this, the north-east portion is designed as the last pit phase which helps to reduce exposure by delaying the pre-stripping of a potentially un-economic phase. This contributes to the "robustness" of the phase designs.

The description of the detailed pit phase designs (or pushbacks) in this section uses the following naming conventions:

- The first digit signifies the type of geometry object (6 is used for pit geometry).
- The middle digit signifies the design series.
- The final digit signifies the pit phase number. This project contains three phases.

The suffix 'i' indicates that the resource tonnage for the phase is incremental from the previous phase. If there is no 'i' specified, it is cumulative up to the phase indicated.

16.7.6.7.1 PHASE1:

Phase 1 is designed to provide efficient access to a supply of mill feed large enough to sustain processing operations for one and a half years, while waste stripping progresses in Phase 2.

Phase 1, therefore, is cone shaped and has its eastern limit set to by its proximity to Phase 2. The minimum mining pushback between Phase 1 and Phase 2 is 50m, which is greater than the minimum mining width. In most areas the distance is greater. This ensures efficient mining of Phase 2.

The edges of Phase 1 are positioned to efficiently mine mill-feed material on levels 2100 through 2310. Figure 16-11 shows a plan view of Phase 1. No part of Phase 1 mines to the ultimate pit limit.

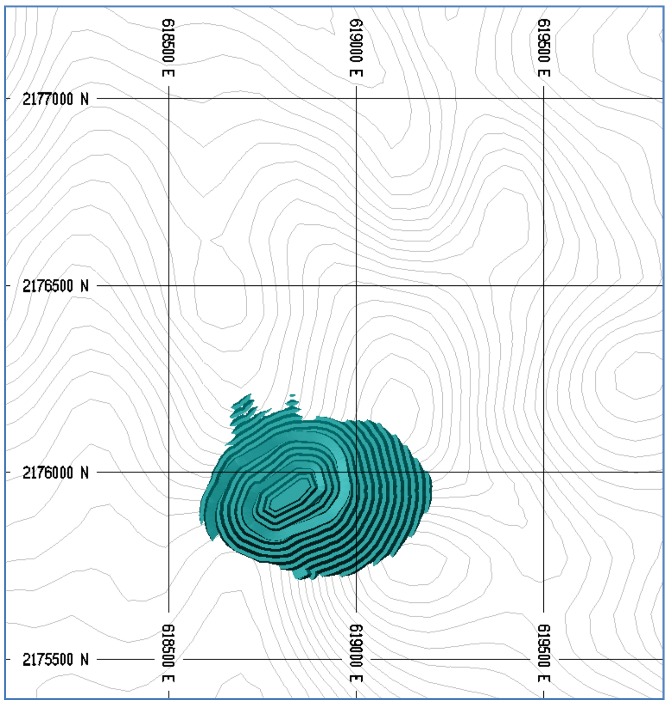


Figure 16-11 Phase 01- Design 631

16.7.6.7.2 PHASE2:

The optimized ultimate pit shell, developed by Lerch-Grossman analyses, has 3 distinct pit-bottoms, and there is also a large waste area between the central and the eastern pit-bottoms.

Phase 2 is designed to mine the western-most of these pit-bottoms', and to provide a head-start to the pre-stripping required to reach the central pit-bottom. Targeting the central pit-bottom directly results in a pit phase with very high pre-stripping requirements. Phase 2 is designed to spread this waste stripping between Phase 2 and Phase 3. The north-west, south-west, and south-east areas of Phase 2 mine to the ultimate pit limits.

Figure 16-12 shows a plan view of Phase 2.

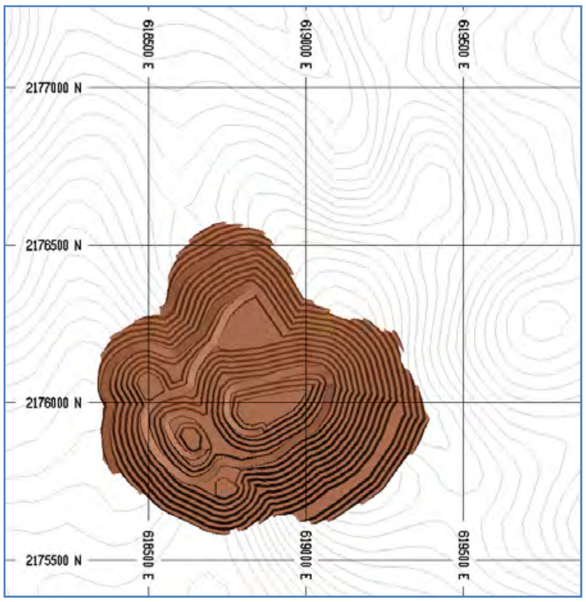


Figure 16-12 Phase 02- Design 632

16.7.6.7.3 PHASE3 (Ultimate):

Phase 3 targets the 'P80' pit optimized ultimate pit shell's central pit-bottom. Mining pushback is in the north-east area after Phase 2.

Figure 16-13 shows a plan view of Phase 3. Figure 16-14 shows all phases.

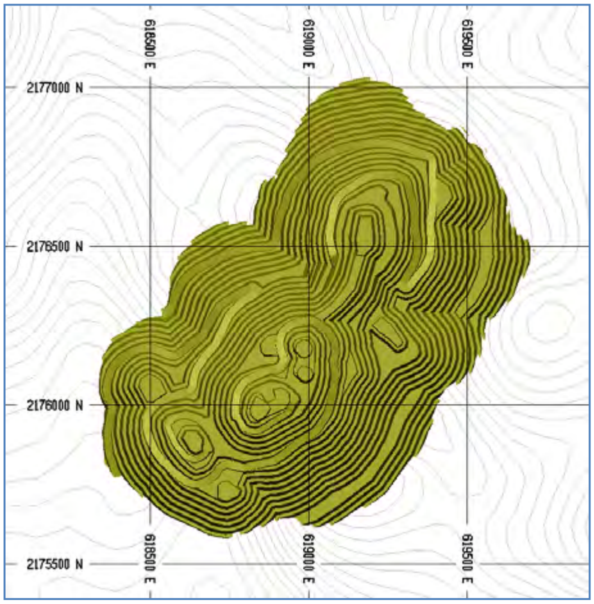


Figure 16-13 Phase 03- Design 623

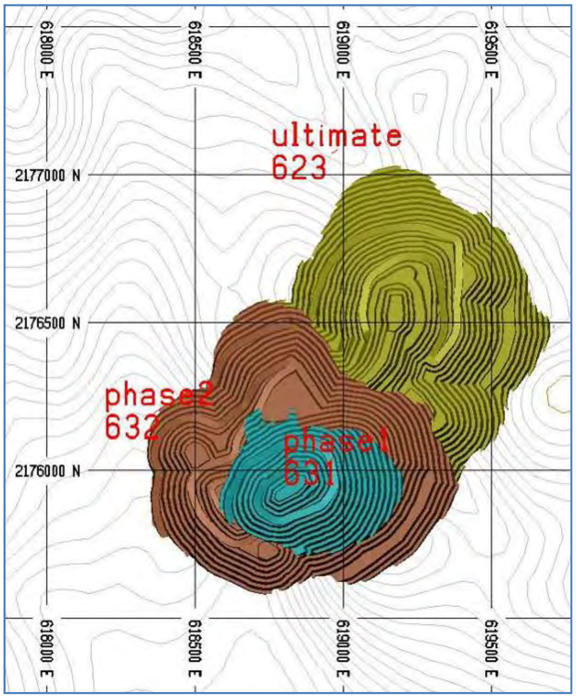


Figure 16-14 Plan View of all Ixtaca Pit Phases

16.8 Pit Resource

The resource model is a rotated 3D block model (3DBM) with whole block and mineralized Au (g/t), Ag (g/t) grades, ORE%, specific gravity and Class. Mineralized grades are reported by lithology zone of which there are seven (shown in Table 16-6), therefore there are seven zones per block. Whole block Pb (%) and Zn (%) and Acid Rock Drainage (ARD) data grades are also included in the block model. ORE% (provided by GCL) represents the percentage of the block that is inside each lithology zone (seven zones per block). MMTS added a topography (TOPO) item representing the percentage of a block below the topography surface.

16.8.1 Pit Resource Calculation

Ixtaca pit delineated resources (shown in Table 16-17) are calculated using an NSR cut-off base on mineralized gold (Au) and silver (Ag) grades by lithology unit within every block along with the lithology block percentages from the resource model and with mining dilution (3%) and loss (3%) applied.

By using the applicable NSP with the metal grades and recoveries in each zone in each block, an NSR value is calculated giving a value in $/t for each zone in each block in the resource model. A weighted average NSR is determined for the block based on a multi-zoned estimation and the ore percentages of each zone are combined to give a total Ore% for each block. Mill feed and waste cut-offs based on the NSR values are used for break-even material selection and for the grade bins for cash flow optimization in the production schedule. The metal prices and resultant NSPs used are based on January 09 2014 spot prices, are shown in Table 16-15.

Table 16-15 Metal Prices and NSP used in NSR Calculation

	Metal Price ($/oz)	NSP ($/oz)	NSP ($/gm)
Au	$1,250	$1,236	$39.8
Ag	$20	$16.4	$0.53

The flotation concentrate recovery for each metallurgical domain is 95% (See Section 13.0). The concentrate recovery to doré also has a 95% recovery, resulting in the average process recoveries for gold and silver to doré to be 90.3% based on test work carried out at Blue Coast Research Ltd. The overall recovery of 90.3% is applied to the NSR estimation for each block for every zone.

Table 16-16 Recovery Results

Flotation Concentrate Recovery	95%
Concentrate Recovery to Doré	95%
Overall Recovery	90.3%

The NSR formula is:
For every mineralized zone (1-6) within a block, NSR =(NSPAu * Au * rec1) + (NSPAg * Ag * rec2).

Where:

- NSPAu = net smelter price for gold ($/gm)
- NSPAg = net smelter price for silver ($/gm)
- Au = gold grade (g/t) of Zone 1-6
- Ag = silver grade (g/t) of Zone 1-6
- **Rec1 =** recovery Au (in %)
- **Rec2 =** recovery Ag (in %)

Including the doré recovery, the NSR $_{(1\text{-}6)}$ = (39.8*AU$_{(1\text{-}6)}$*90.3/100) + (0.53*AG$_{(1\text{-}6)}$*90.3/100)

Since this is a poly-metallic deposit, the combined value of Au & Ag for the differing lithology units is therefore represented by the block NSR, which can be treated as a COG item. For breakeven economics it is assumed that low grade material will be mined to expose higher grade material. However, if this low grade material can produce more net revenue than the cost of processing then it incrementally adds to the net revenue if milled. This low grade material is stockpiled and may be run at the end of the mine life when other operating costs have been reduced. The Ixtaca incremental breakeven NSR COG is $9.00/tonne which covers the process operating costs.

Table 16-17 Summarized Pit Delineated Resource

Recovered In-pit Resources and Diluted Grade NSR ≥ 9.00						
PIT (i=incremental)	**Mill Feed**	**NSR**	**Au**	**Ag**	**Waste**	**Strip Ratio**
	kt	**($/t)**	**g/t**	**g/t**	**kt**	**t:t**
Phase1 - 631	15,401	33.45	0.472	34.64	21,702	1.4
Phase2 - 632i	53,014	29.43	0.497	24.38	104,341	2.0
Phase3 - 623i	56,881	24.47	0.356	24.53	92,015	1.6
TOTAL	**125,296**	**27.67**	**0.430**	**25.71**	**218,058**	**1.7**

16.9 Mine Plan

16.9.1 Mine Production Schedule

The mine production schedule is developed with MineSight Strategic Planner (MSSP), a comprehensive long range schedule optimization tool for open pit mines. Annual production requirements, mine operating considerations, product prices, recoveries, destination capacities, equipment performance and operating costs are used to determine the optimal production schedule. Scheduling results are presented by period, as well as cumulatively. The production schedule includes:

- Tonnes and grade mined by period, broken down by mill feed and waste material type, bench, and mining phase;
- Truck and shovel requirements by period in number of units and operating hours;
- Tonnes transported by period to different destinations (mill, stockpiles, and rock storage facilities).

"Time 0" in the mine schedule refers to the mill start date; first full year of operation is defined as Year 1. The total mine schedule is done in annual time periods.

In the production scheduling, mining precedence is required to specify the mining order of the pit phases based on relative location of the phases. The primary program objective in each period is to maximize the Net Present Value (NPV). The MSSP NPV calculation is guided by an estimation of operating and capital costs, process recoveries, and revenues based on metal grades, prices, and recoveries. The pit phases have been designed with higher grades and lower strip ratios in the earlier phases and when this sequence is used by MSSP the project NPV is improved.

In addition to phase precedence, MSSP tracks the haul cycle time and resultant total haul truck hours from each pit and bench to the primary crusher, stockpiles, or designated rock storage facilities to determine the best sequence for each period. The optimal mining sequence is one that pursues the combined objectives of higher revenues earlier in the mine life with lower required equipment hours

16.9.1.1 Mine Load and Haul Fleet Selection

Initially, the mine load and haul fleet selected prior to production scheduling included the $15m^3$ class shovel matched with the 91 tonne truck as per pit design criteria for ramp widths. However, after reviewing the number of haul trucks required with the 91 tonne payload in the production schedule, it is considered logistically inefficient to have so many haul trucks operating inside the chosen pit phase areas. Therefore, the plan has been revised to use larger 177 tonne trucks matched with $27m^3$ class shovels in the production schedule and haul profiles.

Productivities of the selected equipment include shovel loading times and truck haul cycle estimates for multiple pit-to-destination combinations.

16.9.1.2 Schedule Criteria

The production schedule setup includes a large number of scheduling parameters, and can be modified to a high level of detail. As such, key scheduling parameters used in MSSP are defined here.

Truck and shovel criteria are a key component in the calculation of equipment hours in MSSP. The Table 16-18 below lists the truck and shovel scheduling design parameters used in MSSP.

Table 16-18 Equipment Design Criteria

	Shovel	**EX5500**
	Truck	**Cat 789**
Shovel	Bucket size (m3)	27
	fill factor	95%
	payload (BCM)	19.0
	payload (tonnes)	50.4
Truck	size (m3)	95.5
	capacity (tonnes)	177
	fill factor	90%
	payload (BCM)	63.7
	# passes (BCM)	3.4
	# passes (tonnes)	3.5
	rounded	4
	Pass time (sec)	38
	Operator Eff (%)	80%
	spot & wait (sec)	30
	load time (min)	3.67
	Prod (tonnes/op hr)	**2896**
	Mt/year	17.8

Truck and shovel availability assumptions for MSSP are shown in Table 16-19.

Table 16-19 Shovel and Truck Availabilities Used in MSSP

177tonne Haul Trucks										
Lifetime Operating Hours (‘000 hours)	**0-10**	**10-20**	**20-30**	**30-40**	**40-50**	**50-60**	**60-70**	**70-80**	**80-90**	**100+**
Availability	89%	88%	87%	86%	85%	84%	83%	82%	81%	80%
27m^3 Shovel Diesel Hydraulic Shovel										
Lifetime Operating Hours (‘000 hours)	**0-10**	**10-20**	**20-30**	**30-40**	**40-50**	**50-60**	**60-70**	**70-80**	**80-90**	**100+**
Availability	89%	88%	87%	86%	85%	84%	83%	82%	81%	80%

Equipment availability is reduced as the equipment gets older to reflect the impact of age on performance. Loading time is used with other design parameters to calculate the required equipment hours. Haulage parameters are summarized in Table 16-20.

Table 16-20 Haulage Parameters

Criteria	Value
Loading Time	3.67 min./load
Cycle Time per Pass	38 sec.
# of Passes	4
Spot & Wait Time per Load	30 sec
Job Efficiency Factor	84%
Spot & Dump Time per Load	1.0 min
Delay Time	1.0 min
Max Speed on Haul Roads	45 km/h
Max Speed on Active Bench	20 km/h
Max Speed w/in 200m of Active Dump Face	20 km/h
Rolling Resistance In-pit	5%
Rolling Resistance On Ramp or Haul Road	3%
Rolling Resistance on Dump	8%

The MSSP schedule assumes 360 mine operating days scheduled per year and a 21.5 hour operating day. This 21.5 hour day includes a total daily delay time of 2.5 hours, or a 90% factor.

Final haulage cycle times for each destination is calculated from centroid of the pit benches to each centroid of the various destinations. Exterior haul roads are designed over topography to each destination with a maximum 8% slope. Flat hauls along the Rock Storage Facility (RSF) and tailings embankment are included for each lift.

Haulage destinations include the following;

- Crusher– a location 0.5 km south-west of the pit rim is used for the crusher.
- Rock Storage Facility (RSF) – a centroid 2,490m elevation is used. This is roughly 1/3 of the way up the dump and this starting location allows for convenient haul road location. It is assumed all levels below this are flat at 2,490m, above this an 8% haul road is taken into account and material is dumped at the centroid of every lift.
- Tailings Embankments – Lift elevations used are taken from KP Memorandum VA14-01393 for the Base Case option (Figure 18-5). Construction material for the main Tailings Management Facility (TMF) embankment will be hauled, placed, spread and compacted by selective routing of mine haul trucks for the starter (Stage 1) embankment up to an elevation of 2,445m. The final (Stage 5) embankment will be at an elevation of 2,525 m. The Concentrate Tailings Management Facility (CTMF), situated within but higher in the main TMF, has a starter (Stage 1) crest elevation of 2,490 m, and a final (Stage 5) elevation of 2,525 m. Construction of these facilities will require that material be hauled to 2,550 m prior to accessing the embankment construction areas.
- Lift elevations for the Ramp-Up Case option are taken from KP Letter VA14-01394 (Figure 18-6). Construction material for the main TMF embankment will be hauled, placed, spread and compacted for the starter (Stage 1) embankment up to an elevation of 2,423 m. The final (Stage 10) embankment will be at an elevation of 2,519 m. The CTMF has a starter (Stage 1) crest elevation of 2,479 m, and a final (Stage 10) elevation of 2,519 m. As per the Base Case option, construction of these facilities will require the hauling of material to 2,550 m prior to accessing the embankment construction areas.
- Stockpile – a centroid 2,390 m elevation is used. This temporary stockpile area is south of the RSF.

16.9.1.3 Cut-off Grade Strategy

Various scoping scheduling scenarios have been investigated; the Base Case mining schedule uses a COG of NSR=9 at a full production rate of 30,000 tpd, while the Ramp-Up Case mining schedule starts at 7,000 tpd and ramps up to 9,000 tpd in Year 3, and ramps up to the full production rate of 30,000 tpd by Year 6. The COG during the lower throughput years (Years 1-5) is determined to be NSR=14. MSSP optimizer can develop a COG strategy to increase the Project NPV by stockpiling lower grade marginal material for processing later in the LOM schedule. A COG strategy increases mill head grades and revenues early in the production schedule which has a positive impact on the NPV of the Project. For the bulk mining method implemented for this deposit, it is assumed that blast-hole assaying will be used for grade control to define mill feed COG limits in the pits. Table 16-21 explains each scenario tested for this project with the MSSP optimized scheduler.

16.9.1.4 Schedule Results

The primary scheduling objective is to maximize the NPV which is accomplished by meeting the production requirements in each period (usually mill feed tonnage) while seeking the highest revenue, and lowest cost. The MSSP NPV calculation is guided by an estimation of operating and capital costs, and metal prices, grades, and recoveries. To optimize the project economics the Schedule optimizer targets lower costs and higher grades in the earlier production periods. This has to also be done within the physical constraints of the pit phase, and, equipment performance parameters, and practical operating criteria.

Table 16-21 MSSP Schedule Scenarios Stockpile Strategy

MSSP Schedule	Stockpile Strategy and NSR Annual Cut-off														
	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year10	Year11	Year12	Year13	Year14	Year15
Base Case	20	15	25	15	15	20	15	15	9	9	9	9			
Low Capital Ramp-Up	30	30	30	40	50	25	15	15	15	15	15	9	9	9	9

The summarized results for the annual Base Case and Ramp-Up production schedules are shown in the following Tables and Figures. Note that the grade scale on the right hand side of the graphs is logarithmic.

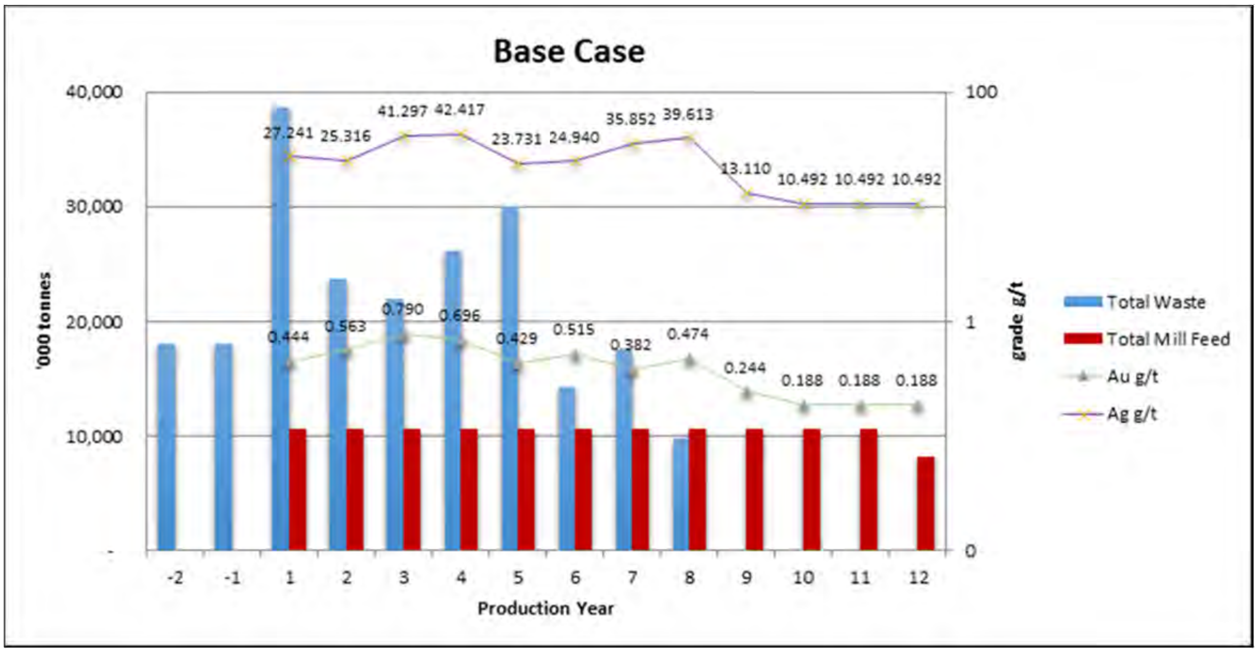


Figure 16-15 Summarized Production Schedule –Base Case

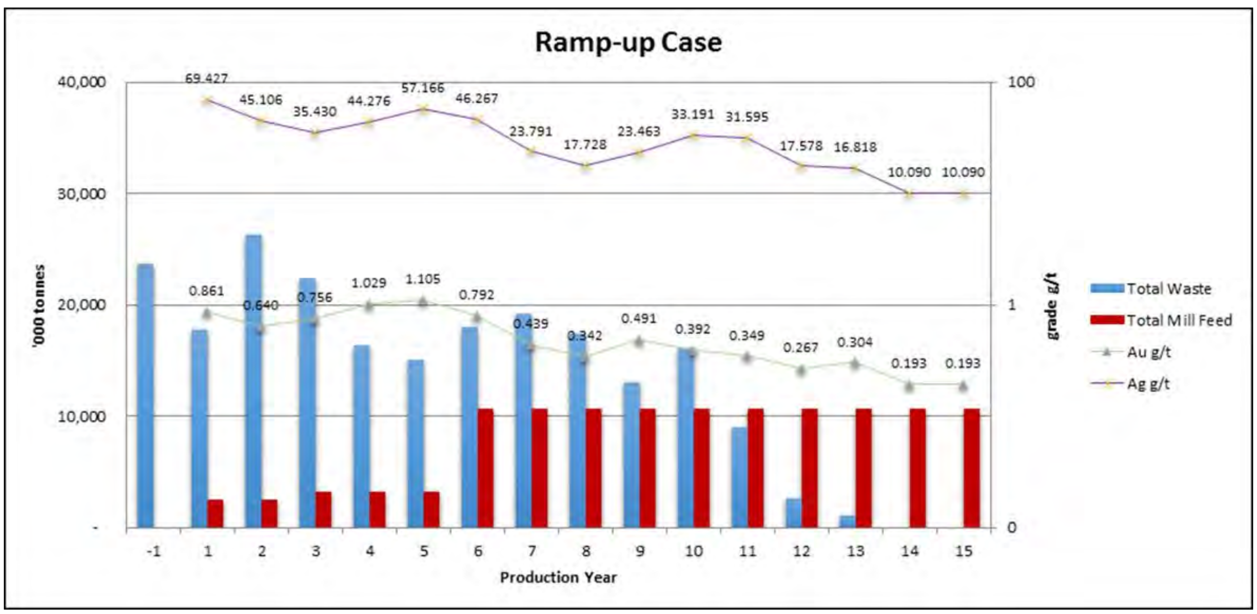


Figure 16-16 Summarized Production Schedule –Ramp-Up Case

The Table 16-22 and Table 16-23 below show the LOM mill-feed production schedule for the Base Case and Ramp-up Case. All in-pit material is mined towards the end of Year 8 and Year 13 respectively, and from that point on the mill feed is comprised of stockpiled low grade material.

Table 16-22 LOM Production Schedule - Base Case

Ixtaca Base Case 30ktpd		-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	Totals
		Pre-stripping														
Days		365	365	365	365	365	365	365	365	365	365	365	365	365	275	
Waste	kt	17,976	17,976	38,714	23,728	21,988	26,130	30,031	14,235	17,466	9,810	6.3	0	0	0	218,059
Direct Mill Feed	kt			9,326	10,640	10,660	9,085	10,640	10,660	10,640	10,640	75.5	0	0	0	82,366
from stkpl to mill	kt			1315.25			1554.8					10,564	10,640	10,640	8,215	42,929
stkpl	kt		4,047	2,998	4,260	12,849	2,500	1,434	8,197	4,222	2,419	2.1				42,929
Total Mill Feed	kt		-	10,641	10,640	10,660	10,640	10,640	10,660	10,640	10,640	10,640	10,640	10,640	8,215	125,295
Au	g/t			0.444	0.563	0.790	0.696	0.429	0.515	0.382	0.474	0.244	0.188	0.188	0.188	0.430
Ag	g/t			27.24	25.32	41.30	42.42	23.73	24.94	35.85	39.61	13.11	10.49	10.49	10.49	25.71
Strip Ratio (Waste/Mill Feed Mined)					3.6	2.2	2.1	2.5	2.8	1.3	1.6	0.9	0.0	0.0	0.0	1.7
Total Material Mined	kt	17,976	22,023	51,037	38,628	45,498	37,716	42,105	33,092	32,328	22,869	84	0	0	0	343,354
Cumulative Material Mined	kt	17,976	39,999	91,036	129,664	175,161	212,877	254,982	288,074	320,401	343,270	343,354	343,354	343,354	343,354	

Table 16-23 LOM Production Schedule - Ramp-Up Case

Ixtaca		Production Schedule																
Base Case 30ktpd		**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**Totals**
		Pre-stripping																
Days		365	365	365	365	365	365	365	365	365	365	365	365	365	365	365	365	
Waste	kt	23,728	17,762	26,312	22,422	16,332	15,009	18,063	19,215	17,429	13,019	16,088	8,995	2,599	1,084			218,059
Direct Mill Feed	kt		2,487	1,337	2,324	3,196	3,196	6,549	1,997	3,795	10,217	9,640	7,701	3,479	2,454	0	0	58,372
from stkpl to mill	kt			1,149	872			4,111	8,663	6,865	443	1,000	2,939	7,161	8,186	10,640	10,640	62,669
stkpl	kt	271	9,876	3,228	6,970	10,472	12,930	4,028	1,754	4,416	6,764	4,272	1,945					66,924
Total Mill Feed	kt	-	2,487	2,486	3,196	3,196	3,196	10,660	10,660	10,660	10,660	10,640	10,640	10,640	10,640	10,640	10,640	121,040
Au	g/t		0.861	0.640	0.756	1.029	1.105	0.792	0.439	0.342	0.491	0.392	0.349	0.267	0.304	0.193	0.193	0.430
Ag	g/t		69.43	45.11	35.43	44.28	57.17	46.27	23.79	17.73	23.46	33.19	31.59	17.58	16.82	10.09	10.09	25.71
Strip Ratio (Waste/Mill Feed Mined)				7.1	10.6	7.0	5.1	4.7	1.7	1.8	1.6	1.2	1.5	0.8	0.2	0.1	0.0	0.0
Total Material Mined	kt	23,999	30,125	30,878	31,716	30,000	31,134	28,640	22,966	25,640	30,000	30,000	18,640	6,078	3,538	0	0	343,355
Cumulative Material Mined	kt	23,999	54,124	85,002	116,717	146,717	177,852	206,492	229,458	255,098	285,098	315,098	333,738	339,816	343,354	343,354	343,354	

16.9.2 Rock Storage Facilities (RSF)

The specific gravity for waste rock averages 2.02 t/m^3 based on the ratio of volcanic material. The specific gravity for volcanic material is approximately 1.67 t/m^3 while the rest of the material is approximately 2.65 t/m^3. The Ixtaca mine plan requires total rock storage of 218 million tonnes or 142.76 million loose cubic meters of waste rock (a 30-35% swell factor is applied to in-situ material resulting in an average specific gravity of placed material of 2.02 t/m^3); several RSF designs have been reviewed and analysed. Table 16-24 summarizes the total volumes and tonnages for the waste rock destinations.

Table 16-24 Waste Rock Destinations and Volumes

Destination	Volume – *M LCMs*	Insitu Tonnage – *M tonnes*
Tailings Management Facility-TMF	79.6	123
Concentrate Tailings Management Facility-CTMF	11.7	18.1
Waste Rock Storage Facility-RSF	51.46	77
TOTAL	142.76	**218.1**

Scoping studies performed by KP (memo found in Appendix G) considered four waste rock storage locations that surrounded the proposed open pit, and it has been determined that the preferred location is immediately downstream of the Tailings Management Facility ("TMF"), where the material can be used to construct the embankments. After consideration of potential for expansion, current land use, land tenure potential and total operating cost, the RSF downstream of the Option 5B TMF has been considered for all waste haul cycles and equipment requirements. The various TMF options initially considered are shown in Figure 16-17 below.

A preliminary evaluation of the mine waste rock has been completed to assess the presence of Potentially-Acid-Generating (PAG) material, which requires specific handling and storage. Based on the preliminary assessment, a large proportion (approximately 85%) of the waste rock is expected to be not Potentially-Acid-Generating (non-PAG), while the remainder is expected to be Potentially-Acid-Generating (PAG). The non-PAG waste rock will be used as construction material for TMF and Concentrate Tailings Management Facility (CTMF) embankments. The PAG material will require special handling, such as encapsulation within non-PAG materials.

The placement of PAG and non-PAG is not currently incorporated into the RSF design, however it can be reasonably assumed to be included in future designs.

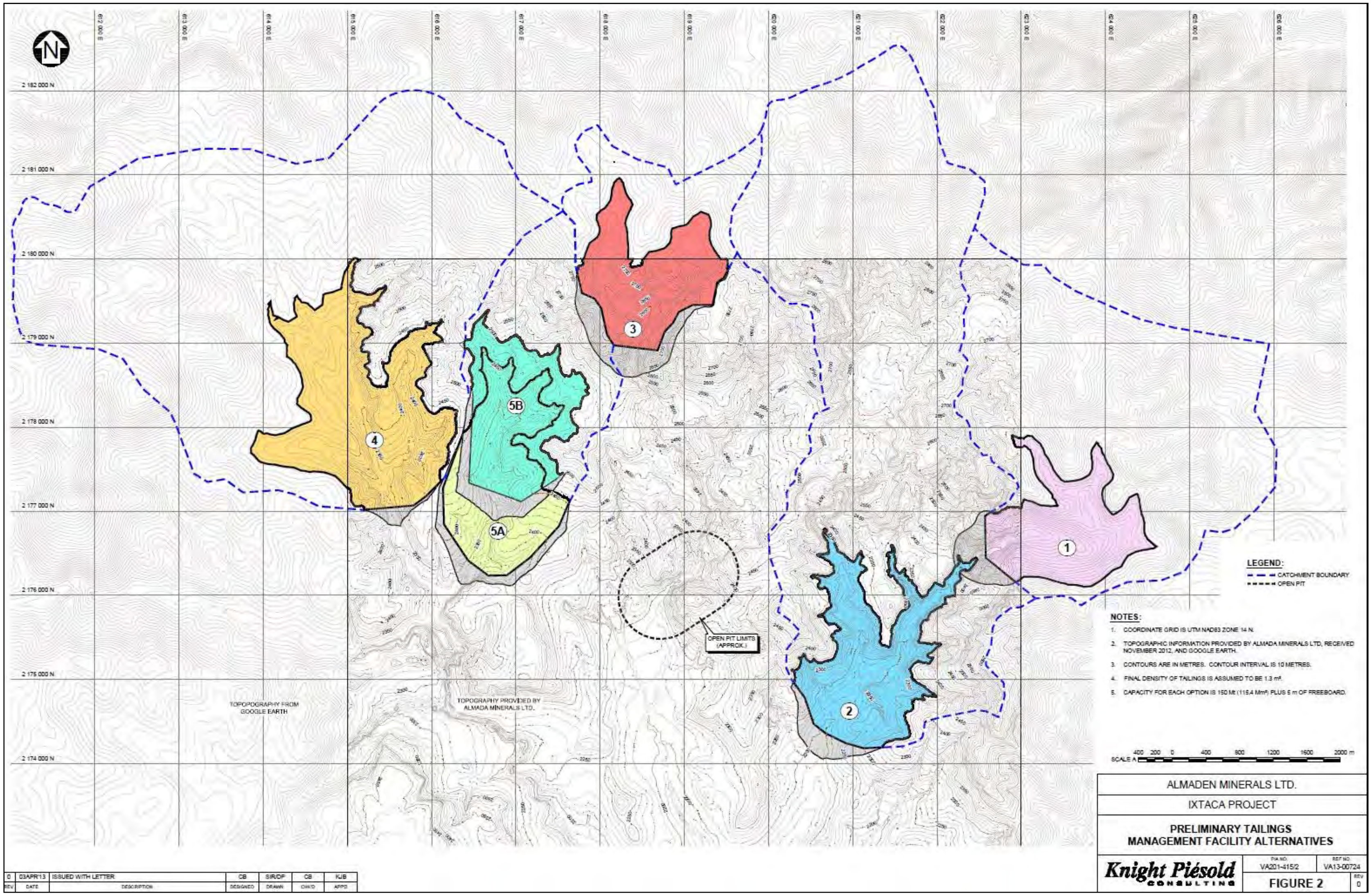


Figure 16-17 Preliminary Tailings Management Facility Locations considered by KP

16.9.2.1 **Design Parameters**

The design of the RSF is mostly dictated by the location and design of the TMF, which includes embankments for the main TMF and the CTMF. This report uses the TMF 5B-130Mt design provided by Knight Piésold as the Base Case. The TMF 3B location, discussed further in Section 18.6, can be used as an alternative should it be required. Construction of the TMF and CTMF starter dams and subsequent lifts of both the RSF and tailings embankments are derived from recommendations provided by Knight Piésold (see Section 18.8 for details). The waste rock that is dumped at the downstream toe of the TMF acts as an additional buttress to assist with maintaining and stabilizing the tailings embankment. Construction of the RSF downstream of the tailings embankment will be done in 20m lifts.

During the pre-production period, pre-stripping is done in phases 1 and 2. The quantity of pre stripping is determined by rock requirements to build the starter dams for the TMF, as well as ensuring a steady mill feed production from the pit once the mill starts up in Year 1.

Table 16-25 and Table 16-26 below detail the waste rock schedule in cubic meters for the Base Case Life of Mine, and the Ramp-Up Case Life of Mine, respectively

Table 16-25 Base Case LOM Waste Rock Schedule (Volumes in ‘000 m^3)

Base Case : 30Ktpd			Destination		Destination	
Schedule Year	Scheduled Waste Rock		TMF (5b-130Mt) and RSF		CTMF (10%)	
	‘000 LCM	CUM. NET	‘000 LCM	CUM. NET	‘000 LCM	CUM. NET
Pre-Prod	23,431	23,431	15,931	15,931	7,500	7,500
1	25,232	48,663	24,232	40,163	1,000	8,500
2	15,465	64,127	14,465	54,628	1,000	9,500
3	14,330	78,458	13,330	67,958	1,000	10,500
4	17,160	95,618	16,550	84,508	610	11,110
5	19,722	115,340	19,112	103,620	610	11,720
6	9,348	124,688	9,348	112,968	-	11,720
7	11,557	136,245	11,557	124,525	-	11,720
8	6,495	142,740	6,495	131,020	-	11,720
9	-	142,740	-	131,020	-	11,720
10	-	142,740	-	131,020	-	11,720
11	-	142,740	-	131,020	-	11,720
12	-	142,740	-	131,020	-	11,720
13	-	142,740	-	131,020	-	11,720
Total	**142,740**		**131,020**		**11,720**	

Table 16-26 Ramp-Up Case LOM Waste Rock Schedule (Volumes in ‘000 m^3)

Ramp-Up : 7 up to 30Ktpd			Destination		Destination	
Schedule Year	**Scheduled Waste Rock**		**TMF (5b-130Mt) and RSF**		**CTMF (10%)**	
	‘000 LCM	**CUM. NET**	**‘000 LCM**	**CUM. NET**	**‘000 LCM**	**CUM. NET**
Pre-Prod	15,346	15,346	8,851	8,851	6,495	6,495
1	11,488	26,834	10,488	19,339	1,000	7,495
2	17,018	43,852	16,018	35,357	1,000	8,495
3	14,558	58,410	13,558	48,915	1,000	9,495
4	10,725	69,135	9,725	58,640	1,000	10,495
5	9,857	78,992	8,857	67,497	1,000	11,495
6	11,862	90,854	11,637	79,134	225	11,720
7	12,714	103,568	12,714	91,848	-	11,720
8	11,533	115,101	11,533	103,381	-	11,720
9	8,615	123,716	8,615	111,996	-	11,720
10	10,646	134,362	10,646	122,642	-	11,720
11	5,953	140,314	5,953	128,594	-	11,720
12	1,720	142,034	1,720	130,314	-	11,720
13	702	142,736	702	131,016	-	11,720
14	-	142,736	-	131,016	-	11,720
15	-	142,736	-	131,016	-	11,720
Total	**142,736**		**131,016**		**11,720**	

The Figure 16-18 below is a plan view showing the General Arrangement that includes destinations used for haulage cycles and the 3B-TMF alternative.

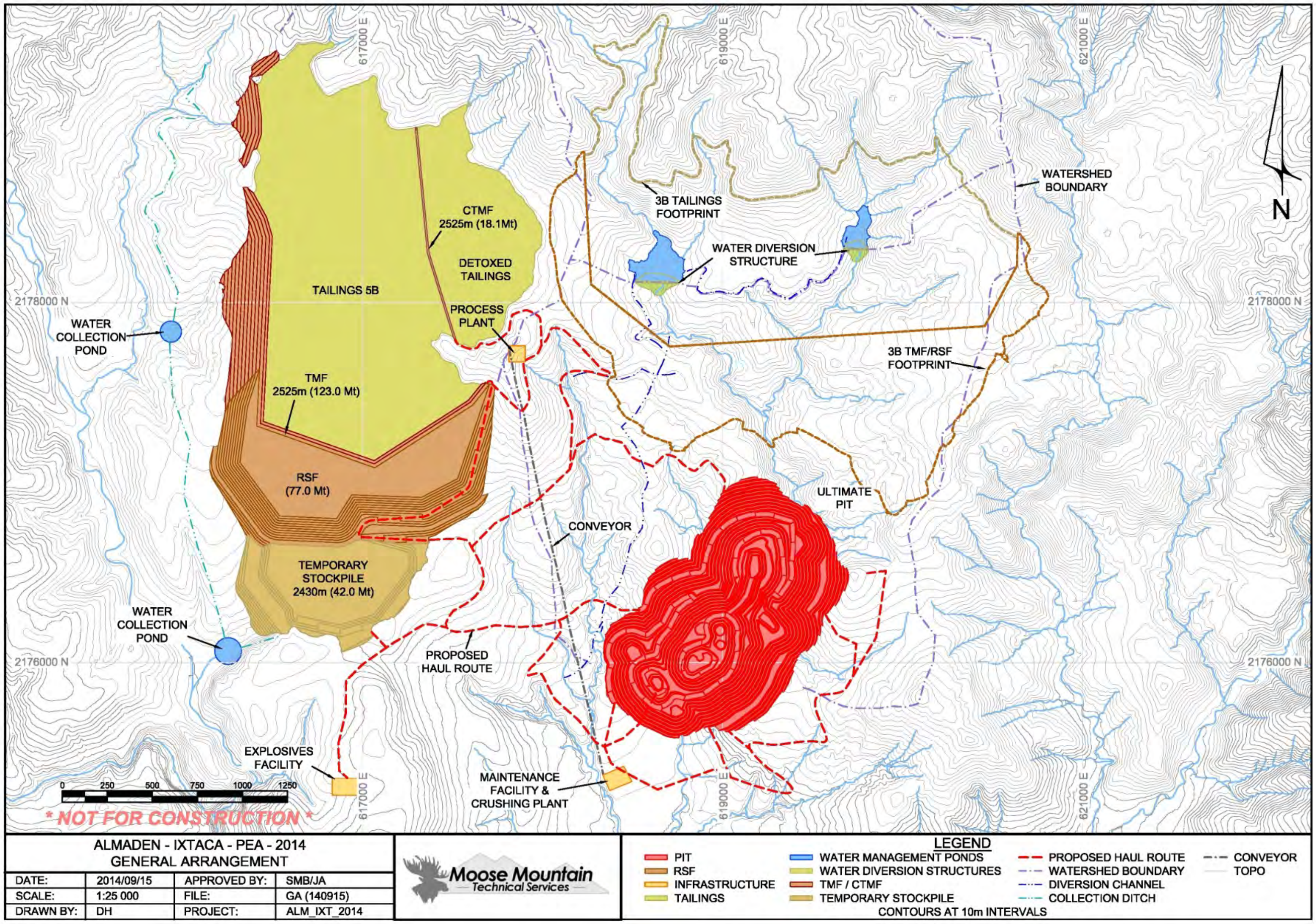


Figure 16-18 Plan View of RSF and Pit Limit General Arrangement

16.9.3 Mine Production Detail- Base Case

The Base Case (30ktpd) includes two years of pre-production mining (pre-stripping), followed by 12 years of mill feed production.

All waste rock is directed to the RSF or the tailings embankments (including the CTMF). The waste rock is comprised of approximately 55% volcanic and 45% primary rock material.

A stockpile strategy consists of three mill feed stockpiles that include low grade, midgrade and high grade material. For the first eight years, the low grade and midgrade, with some high grade is stockpiled to enable the best possible mill-feed head grade especially in the earlier years. All stockpiled mill feed material is directed to a temporary stockpile location south of the final RSF footprint.
The stockpile is fully reclaimed in year's nine to twelve.

16.9.3.1 Pre-Strip Period

Pre-stripping in two phases, Phase1 and Phase2 will commence approximately two years in advance of the mill start-up. A total of 35,950kt of waste rock is directed to the building of the tailings embankments (including the CTMF). Lower grade material (1,790kt) and mid to high grade material (1,315kt) is directed to a (temporary) stockpile. (Figure 16-19)

16.9.3.2 Year 1

The first year of production assumes a full mill start-up of 30,000 tpd. 9,330kt of mill feed is sent directly from the pit, with another 1,315kt reclaimed from stockpile, totalling 10,645kt for the year. A total of 38,714kt of waste rock (53% volcanic) is hauled from Phase 1 and 2 to both the RSF and TMF (including the CTMF). The entire mid-grade stockpile is reclaimed and sent to the mill, while the lower grade material is stockpiled to a cumulative size of 5.7 million tonnes. Pre-stripping of Phases 1-2 continues.

16.9.3.3 Year 2-5

Years two and three continue to mine Phases 1 and 2, completing Phase1 in Year three and Phase2 in Year five. Stripping of Phase 3 commences in Year four. The stockpile strategy includes building the mid to high grade stockpile back up in Year three, while all low grade material continues to go to the stockpile. The stockpile is 25,000kt by the end of Year five. 25,000kt of waste rock is removed annually and hauled to the RSF and TMF (including the CTMF). (Figure 16-20)

16.9.3.4 Years 6-8

Years six to eight continues to mine Phase3. In Year eight all of Phase 3 is completed. All low-grade material mined is sent to the stockpile during these years, while the size of the stockpile reaches a maximum 40,057kt in Year eight. Waste mining in the pit is complete by year 8 therefore the TMF dam must be built to completion at this time as well.

16.9.3.5 Years 9-12

All mill feed is from the low-grade stockpile with estimated mill feed head grade of 0.188g/t for gold (Au) and 10.49g/t for silver (Ag). The End of Period map shows final embankment crest elevations of 2,525 m for both TMF and CTMF as per scheduled material for the ultimate Base Case layout. (Figure 16-21)

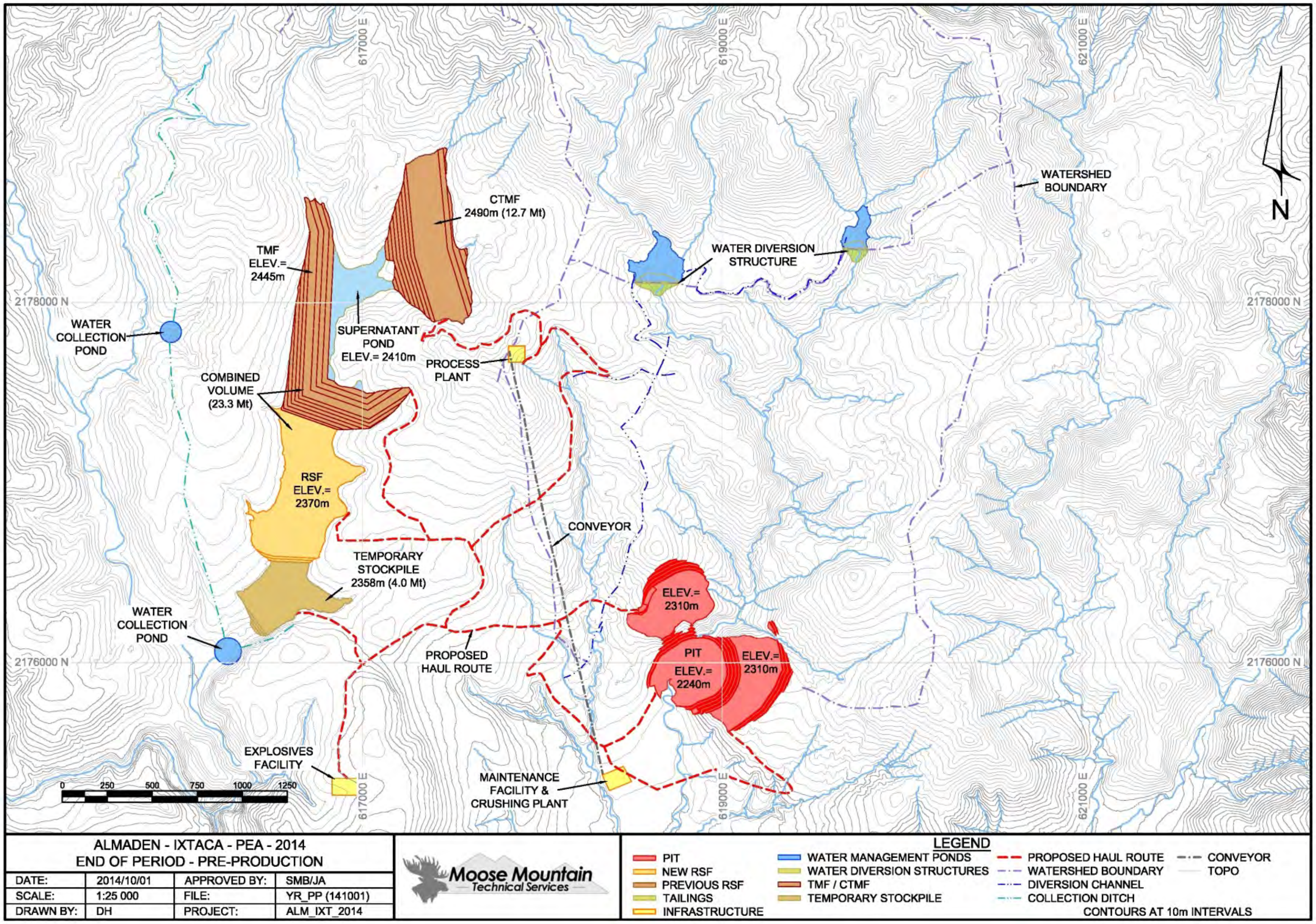


Figure 16-19 Pre-Production Base Case

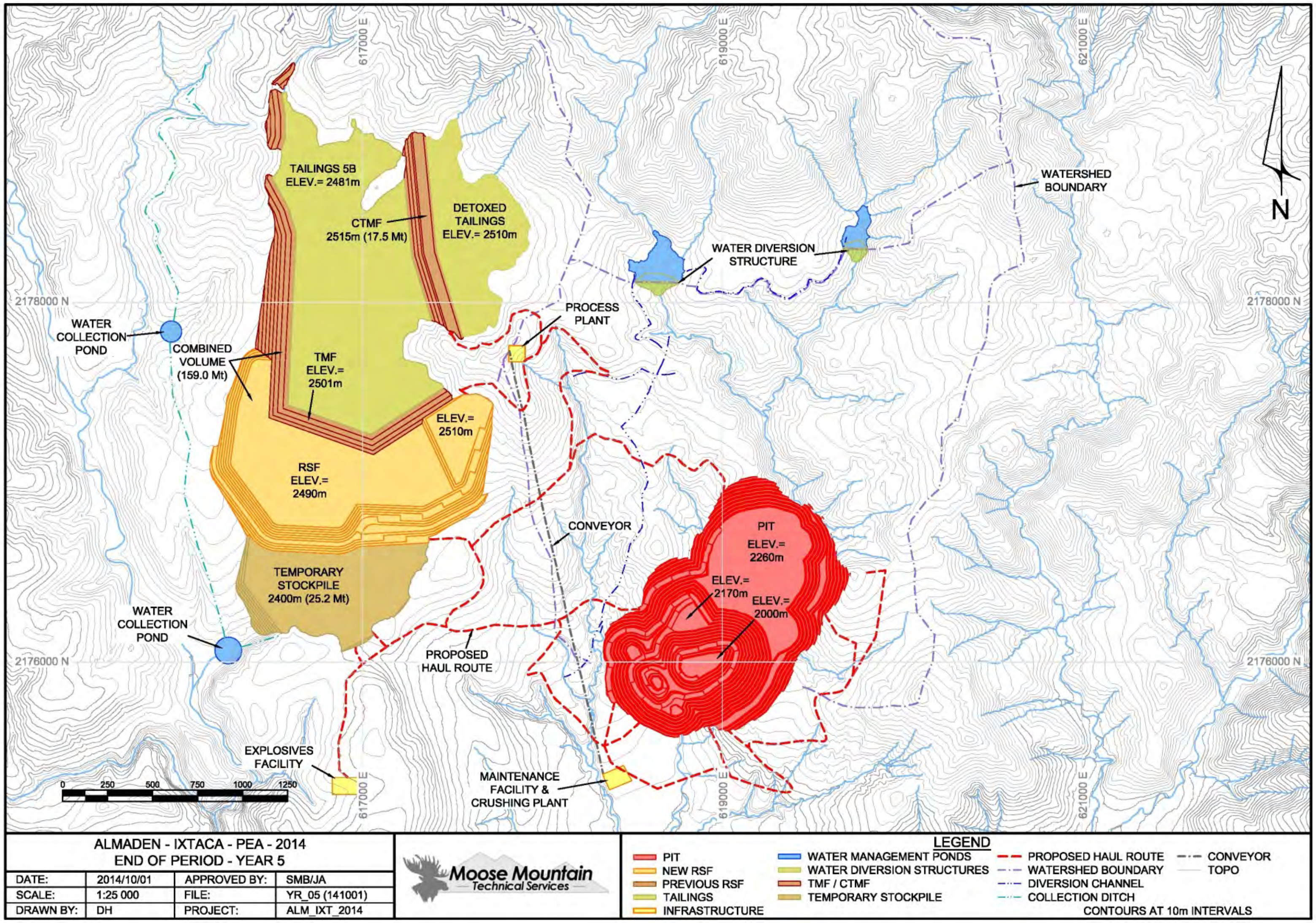


Figure 16-20 Year 5 Base Case

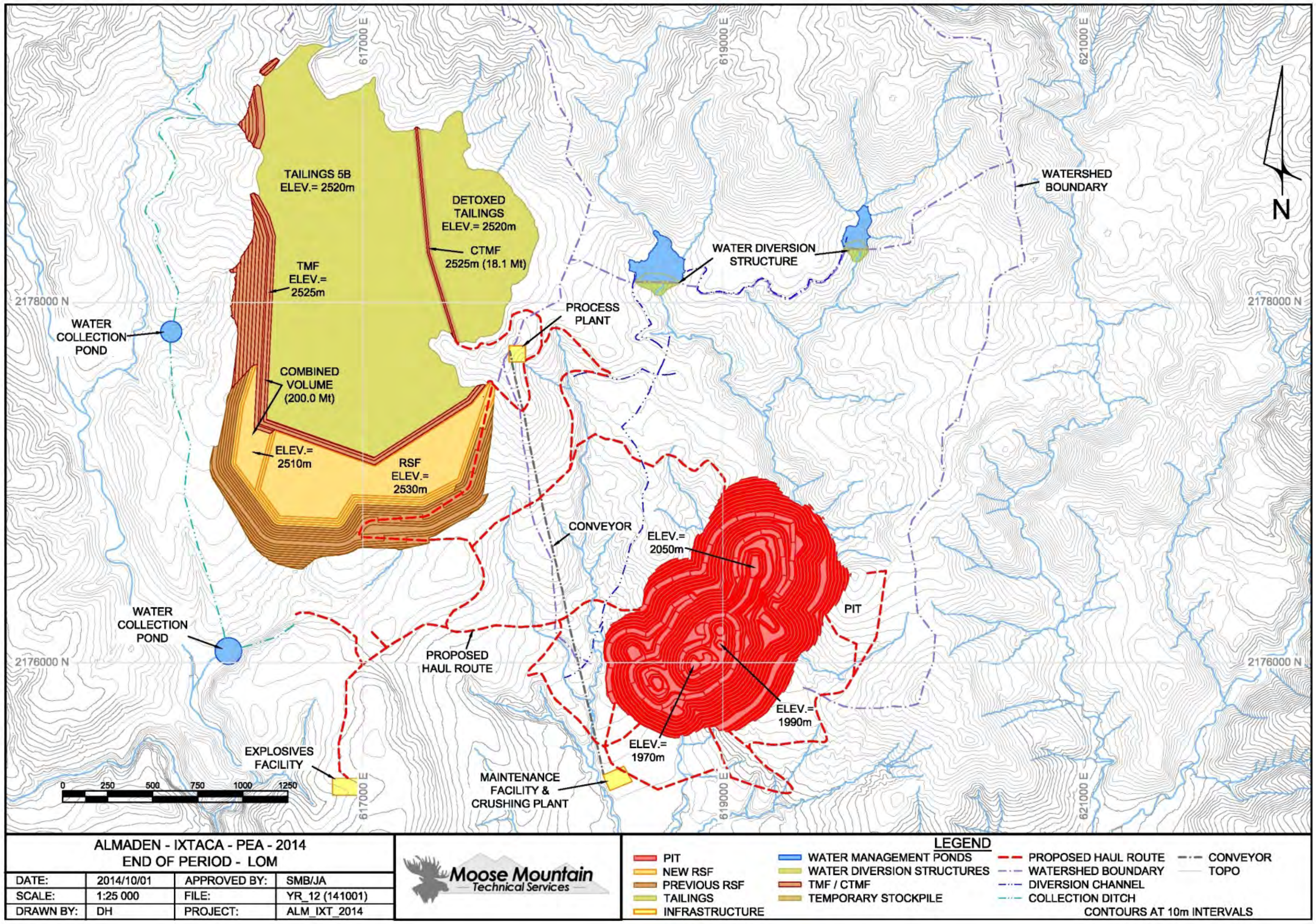


Figure 16-21 LOM Base Case

16.9.4 Mine Production Detail- Ramp-Up Case

The low capital Ramp-Up Case (7,0000 tpd in Years 1-2, 9,000 tpd in Years 3-5, 30ktpd from Year 6 to EOM) includes one year of pre-production mining (pre-stripping), followed by fifteen years of mill feed production.

All waste rock is directed to the RSF or the tailings embankments (including the CTMF). The waste rock includes 55% volcanic material and 45% primary rock material. The TMF construction schedule is included in the schedule as per the recommendations from KP in memorandum VA14-01393, with the exception of the Alternative 2 Stage 1 design as per letter VA14-01394 for Year 1 of production.

All stockpiled material is directed to the stockpile location south of the final RSF footprint.

The stockpile strategy consists of three stockpiles that include low grade, midgrade and high grade material. Mill Feed material is sent to and reclaimed from the stockpiles throughout the life of mine, to enable the best possible mill-feed head grade. In the final two years of production, all mill production consists of low grade stockpiled material. At the end of the mine life, there is 4,255kt of low grade stockpiled material that is un-reclaimed.

16.9.4.1 Pre-Strip Period

Pre-stripping in Phase1 and Phase2 will commence approximately one year in advance of the mill start-up. A total of 23,569kt of waste rock (99% volcanic material) is directed to the building of the Stage1 of the tailings embankment (including the CTMF), with remainder directed to the RSF. Lower grade material (214kt) and mid to high grade material (55kt) is directed to the stockpile. (Figure 16-22)

16.9.4.2 Year 1-2

The first two years of production assumes a low-throughput mill start-up of 7ktpd (total 2,490kt per annum). In Year one, 2,490kt of high grade mill feed is sent directly from the pit, while in Year two, in addition to 1,337kt of direct mill feed, 1,150kt is reclaimed from a high grade stockpile. 44,075kt of waste rock (60% volcanic) is hauled from Phase 1 and 2 to both the RSF and TMF (including the CTMF). Stockpiled material is 44% low-grade (5,442kt), with the remaining stockpile consisting of 6,784kt of mid to high grade. A total of 30,125kt of material is moved in Year one, and 30,878kt in Year two.

16.9.4.3 Year 3-5

The production rate in Years three to five increases to 9ktpd (total 3,196kt per annum). Phases 1 and 2 continue to be mined during these three years, with Phase 1 complete by the end of Year five. The stockpile strategy includes building the mid to high grade stockpile up to a total of 22,000kt, while all low grade material (approximately 14,000kt mined) continues to go to the stockpile. The total stockpile reaches a size of 41,724kt by the end of Year five. 18,000kt of waste rock is removed annually and hauled to the RSF and TMF (including the CTMF). (Figure 16-23)

16.9.4.4 Years 6-13

The production rate is fully ramped up to capacity by Year six to 30ktpd (10,650kt per annum). Pre-stripping of Phase3 begins in Year six, while Phase2 is completed by the end of Year seven. Mill Feed during these years optimizes between the stockpile and direct mill feed to provide an optimum mill feed head grade. All mid to high grade stockpiled material is reclaimed by the end of Year eleven, leaving a stockpile of 40,882kt of low grade material. In Year thirteen, Phase 3 is completed. The TMF (including the CTMF) is built to completion in Year thirteen as there is no more waste rock material available. The total amount of material moved for Years six to ten averages 27,500kt per annum. In years eleven to thirteen, the mining rate reduces to half and then to a quarter of the previous year's six to ten.

16.9.4.5 Years 14-15

All mill feed is from the low grade stockpile with estimated mill feed head grade of 0.193 g/t for gold (Au) and 10.09 g/t for silver (Ag). 4,225kt of low grade stockpile remains and is counted as part of the ultimate RSF. The End of Period map shows final embankment crest elevations of 2,519 m for both TMF and CTMF as per scheduled material for the ultimate Ramp-Up Case layout (Figure 16-24)

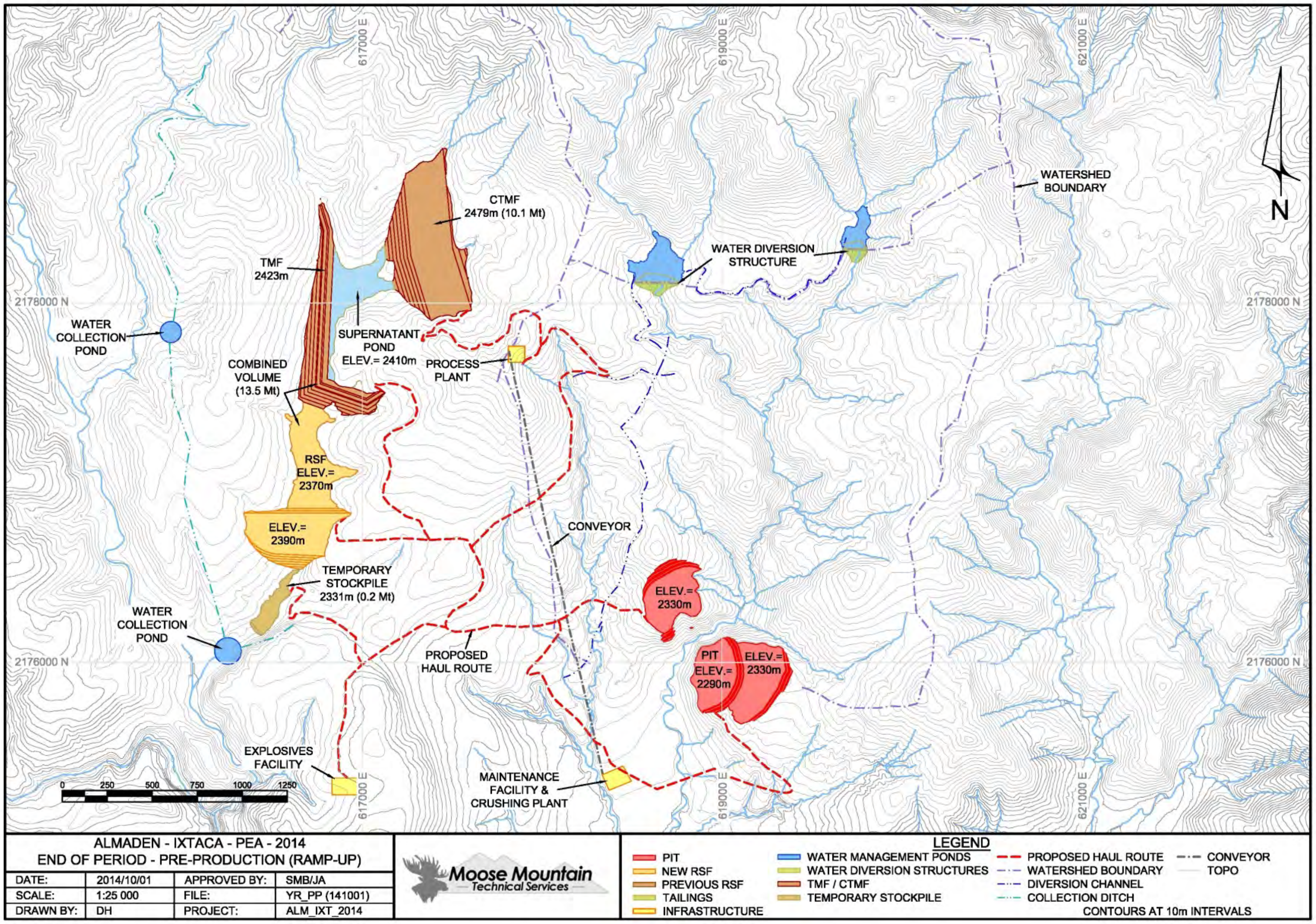


Figure 16-22 Pre-Production Ramp-Up Case

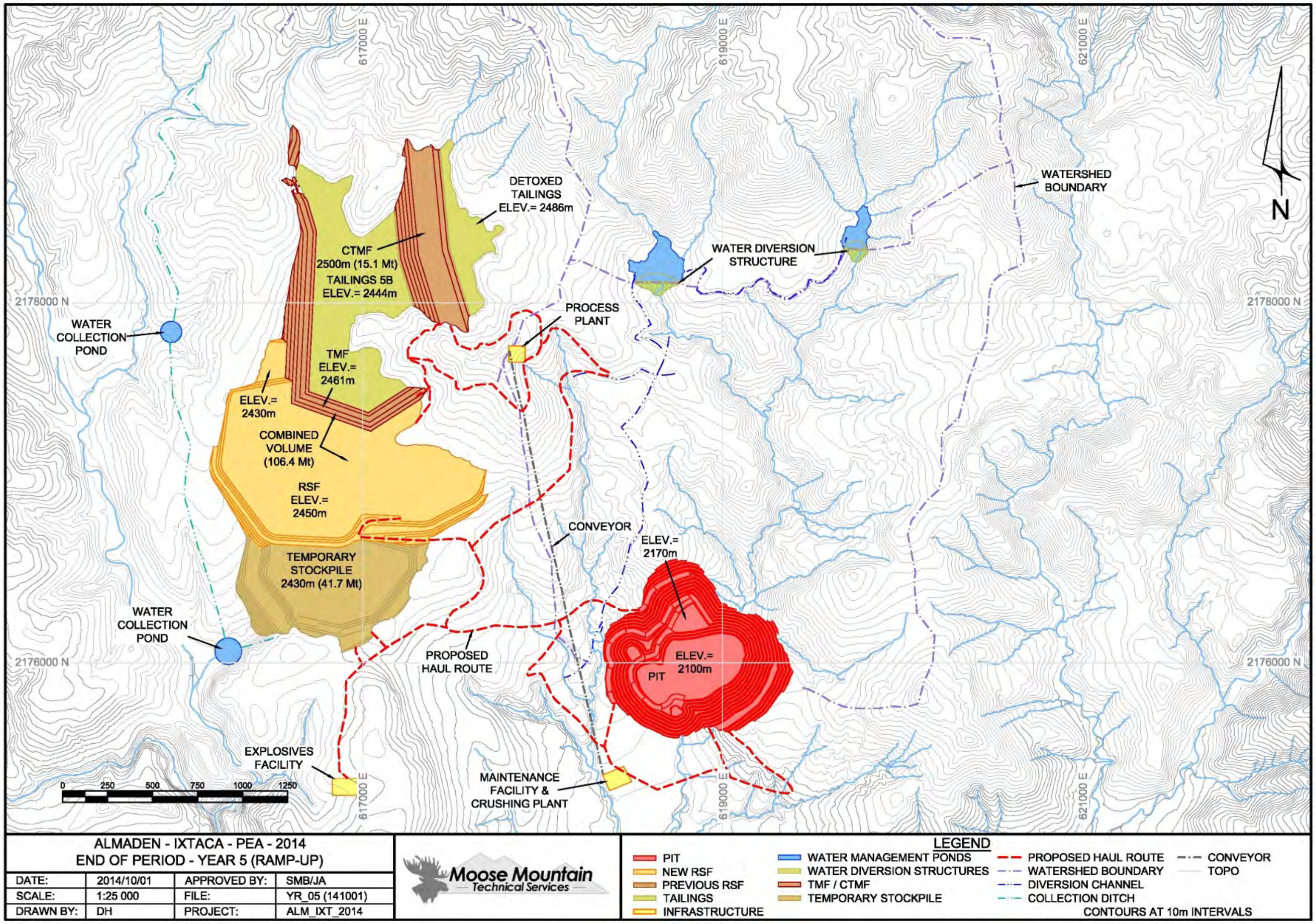


Figure 16-23 Year 5 Ramp-Up Case

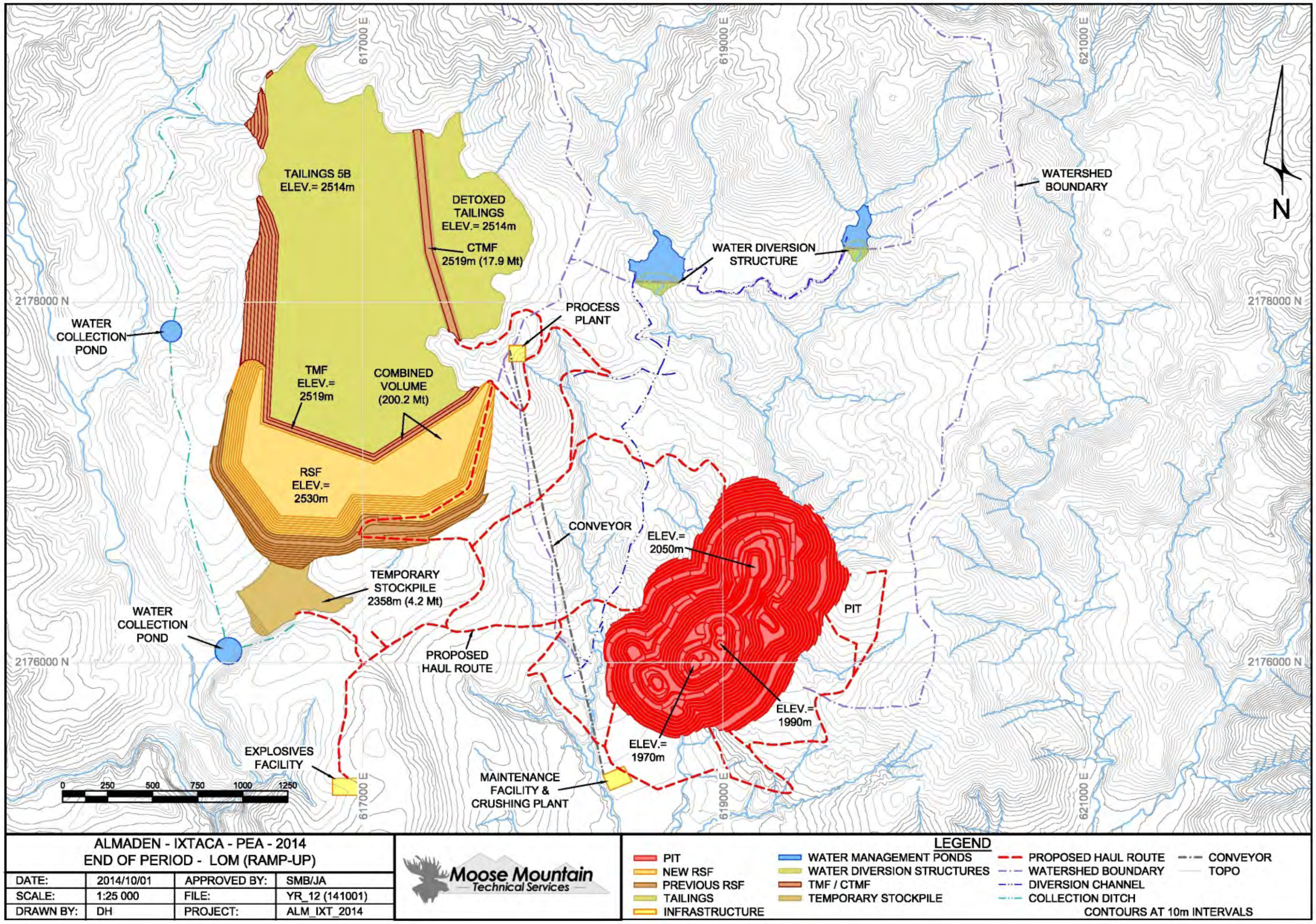


Figure 16-24 LOM Ramp-Up Case

16.9.5 Mine Operations

16.9.5.1 General Organization

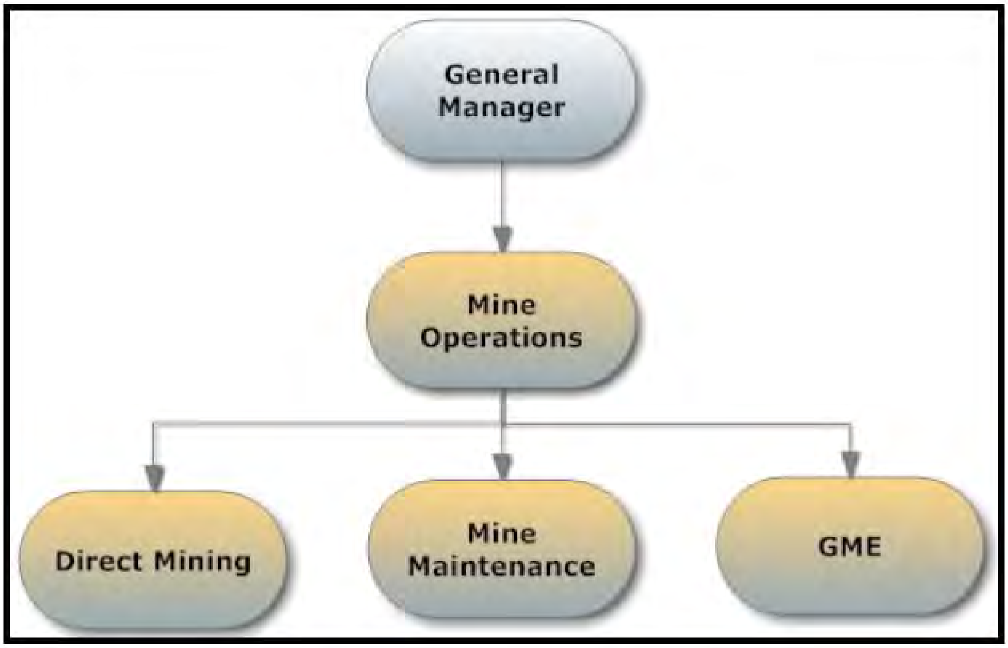


Figure 16-25 General Organization Chart

Mine operations are organized into three departments: mine maintenance, direct mining, and technical services. Other areas of the organization are dealt with elsewhere in the report. The mine department is estimated to employ 431 people initially (including temporary employees).

Each crew is close to 85 people (operations, maintenance and mill) and it is assumed that there will be 4 crews operating in a continuous shift (2 weeks on / 2 weeks off, 2 days and 2 nights on / 4 days off, etc.).

There will be a total of approximately 40 salaried personnel covering operations, administration, mill and maintenance.

It is estimated an additional 15% temporary workforce will be required for the hourly personnel in order to cover holidays and other leaves of absence.

The mine maintenance department reports to the General Manager through the Maintenance Manager. Under the supervision of the General Maintenance Foreman, mine maintenance accounts for supervision, planning, and implementation of all maintenance activities pertaining to the mine fleet or direct mining infrastructure, whether in the maintenance shops or in active mining areas.

The direct mining and technical services departments will report to the General Manager through the Mine Manager. Direct mining, under the supervision of the General Operations Foreman, accounts for supervision, training, and implementation of all drilling, blasting, loading, hauling, and pit maintenance activities in the mine. It also accounts for any other areas where mine fleet activity is present, such as the construction of haul roads. Technical services, under the supervision of the Chief Engineer, account for all technical support from mine planning, geology, surveying and mine engineering personnel.

In this study, direct mining and mine maintenance is planned as contractor owned and operated fleet/personnel. The project owner is responsible for supplying all necessary infrastructures for maintenance. All facilities required for the blasting supply contractor will be purchased by the Owner as well. The mine will employ a drilling and blasting foreman that will act as a liaison between the blasting contractor and the mine, and a drilling and blasting engineer to design and manage the blasting operations.

16.9.5.2 Drilling

Areas will be prepared on the bench floor for blast patterns in the in situ rock. The spacing and burden between blast holes will be varied as required to meet the specified powder factor for the various rock types.

If future operations decide to install automatic samplers, the drill will be responsible for bagging and tagging the drill cutting from the sampler for shipment to the assay lab. If manual sampling is done, the driller will be responsible for taking the samples from the drill cuttings, and bagging and tagging it.

Controlled blasting techniques will be used for high wall rows, pioneering drilling during pre-production, and development of initial upper benches. Where required, dozers will be used to establish initial drilling benches for the upper portions of each phase and move material until sufficient mining width is established.

16.9.5.3 Blasting

There has not been a blasting study at this level of study for the Project. Based on similar operations, an appropriate powder factor may be used to provide adequate fragmentation and digging conditions for the shovels, with a targeted top-size between 1400-1500mm for barren rock and quarried rock. The powder factors selected are dependent on rock types and will therefore vary. In typical barren rock, the powder factor may range from 0.35-0.37 kg/t. The powder factor in volcanics material is expected to be approximately 1/3 lower (between 0.23-0.25 kg/t)

16.9.5.4 Explosives

A contract explosives supplier will provide the blasting materials and technology for the mine, as well as manufacture bulk explosives on site. The nature of the business relationship between the explosives supplier and the mining operator will determine who is responsible for obtaining the various manufacture, storage and transportation permits, as well as any necessary licenses for blasting operations. This will be established during commercial negotiations. For this study, the explosives contractor delivers the prescribed explosives to the blast holes and supplies all blasting accessories. Blasting accessories will be stored in magazines.

Specifications for blasting plant and explosives storage magazines and the locations of these facilities have been designed to provide a minimum clearance distance of 750m between the explosives manufacturing facility and any inhabited buildings or work areas is designed. In future studies, compliance to Mexican regulations will need to be included into the design. The location of the blasting plant and the explosives magazines are determined by the table of distances that govern the manufacturing and storage of explosives and blasting agents. The contractor will be responsible for proper placement of magazines and facilities.

Different contractors have various explosives products and specifications. The chosen contractor will be responsible for providing all Material Safety Data Sheets (MSDS) and product fact sheets as applicable.

16.9.5.5 Explosives Loading

Loading of the explosives will be done with bulk explosives loading trucks provided by the explosives supplier. The trucks should be equipped with GPS guidance or otherwise tied into the in-pit data network, and should be able to receive automatic loading instructions for each hole from the engineering office.

Explosives loading will be carried out by the contractor's crews with their immediate supervisor; however, they must report to the owner's Drilling and Blasting Foreman who will work alongside the Drilling and Blasting Engineer to ensure that explosives loading is carried out according to the mine's specific needs.

The blast holes will be stemmed to avoid fly-rock and excessive air blasts. Crushed rock will be provided for stemming material and will be dumped adjacent to the blast pattern. A small front-end loader, owned and operated by the mine, will be used for blast hole stemming. Any crushed rock required for blast hole stemming will be provided by the onsite rock crusher specified for mine roads and quarrying operations.

16.9.5.6 Blasting Operations

The blasting crew will be provided by the contractor and will coordinate the drilling and blasting activities to ensure a minimum of two weeks of broken material inventory is maintained for each shovel. Also, the blast patterns will not be staked; therefore, the blasting activities will also need to be tied into the in-pit data network. The blasters will require hand-held electronic location devices to identify the holes for the pattern tie-in. A detonation system will be provided by the contractor and will consist of an initiation device, detonating cord, surface delay connectors, and boosters.

Blasting activities are day shift activities only and can only occur during the five days each week that the mine's Drilling and Blasting Foreman is present.

16.9.5.7 Loading

Bench widths are designed to ensure operating room is suitable for efficient double-sided loading of trucks at the shovels. This study does not reduce shovel productivities in areas where double-sided loading is not possible (such as the upper benches of the pit phases where the end of the bench meets topography), as it is assumed that ancillary equipment will be deployed in non-productive operating areas, to prepare the digging areas for higher shovel productivity. This can entail dozing small benches down slope to the next bench, trap dozing, and other dozing activities to attain minimum mining width for the shovels.

Any significant move of a primary loading unit will be carried out with the use of a rental low-boy truck and trailer in order to reduce wear and tear on the shovel, and to reduce down time.

16.9.5.8 Hauling

Haulage of mill feed and rock will be handled by off-highway 177-t haul trucks. The trucks will be outfitted with Fleet Management systems.

The study is based on a detailed haulage network database, estimated and designed using MineSight® haulage software. Haulage profiles have been created from the centroid of every bench of every phase of the open pit to each unique destination location (such as the TMF, the primary crusher, stockpile, and RSF). These haul profiles are input into the schedule optimizer, which is set to maximize project NPV by using the lowest cost haul to a feasible destination. The payload, loading time, and loaded/empty haul cycles then determine the truck productivity.

16.9.5.9 Pit Maintenance

Pit maintenance services include haul road maintenance, mine dewatering, transporting operating supplies, relocating equipment, and pit floor clean-up. Haul road maintenance is necessary for low haulage costs; dozer, grader, and water truck hours are allocated and adjusted to maintain the haul road network throughout the LOM production schedule.

Regular application and grading of crushed rock to haul road surfaces improves truck travel speeds, reduces mechanical fatigue to the haul trucks, and enhances tire life, which is a major mine operating cost. Crushed rock requirements for road maintenance are incorporated into the crushing and screening operation located at the quarry.

Additional ancillary equipment is included in this section such as maintenance vehicles, crew and supervisor pickup trucks, cranes and other support equipment.

16.9.5.10 General Mine Expense (GME)

The GME portion of direct mining accounts for the supervision, safety, environmental, and training for the direct mining activities, typically pertaining to salaried direct mining staff. Direct mining supervision will extend down to the Shift Foreman level.

The Operations General Foreman will assume responsibility for overall supervision for the mining operation and will be responsible for open pit supervision and equipment coordination. The position reports to the Mine Manager, who in turn reports to the General Manager. A mine Shift Foreman is required on each 12 hour shift, with overall responsibility for the shift. A day shift only (five days a week), Drilling and Blasting Foreman will provide supervision for drilling and blasting during day shifts. When the Drilling and Blasting foreman is not present, no blasting activities may occur; however, drilling can be supervised by the Shift Foreman.

Initial training and equipment operation will be provided by experienced operators. As performance reaches adequate levels, the number of trainers can be decreased to a sustaining level, with a single Training Foreman providing supervision for continuous day shift only shift trainers.

Direct mining GME also includes annual allowances for general equipment rental, licensing and maintenance of mine design and fleet management software.

16.9.5.11 Technical Services

Technical Services accounts for the technical support from mine engineering, planning, geology and surveying functions. The majority of Technical Services activities fall under GME.

The Chief Mine Engineer/Mining Superintendent will direct the Technical Services department and will report to the Mine Manager, who in turn will report to the General Manager. The Senior Mining Engineer will coordinate mining engineering, drilling and blasting engineering, mine planning, and surveying. The Senior Geologist will be responsible for local step out and infill drill programs for onsite exploration activities and updating the long range geological model. The geology department will also provide grade control support to mine operations, and will manage and execute the blast hole sampling and blast hole interpolation of the short range blast hole models for operations planning.

A separate Project Engineer will assume split responsibilities for all mine geotechnical issues including pit slope stability, monitoring, and hydro-geological studies, as well as TMF engineering. The Project Engineer will also have oversight for the whole property for any geotechnical monitoring and assessment programs being carried out by safety personnel or third party consultants, or any other unspecified projects on the Property.

Technical Services GME also includes engineering consulting on an ongoing basis for specialty items such as geotechnical, operations support, environmental and geo-hydrology expertise and third-party reviews. In addition, in-fill exploration drilling is included as an allowance in the GME in the year prior to phase push-backs.

16.10 Mine Closure and Reclamation

Mexico does not have detailed closure legislation, but has national environmental laws and is currently developing more specific mine closure requirements. Guidance for the construction, operation, and closure of tailings impoundments is included in a national regulation revealed in 2003 (NOM-141-SEMARNAT-2003). Post operation criteria are presented in Section 5.7 of NOM-141-SEMARNAT-2003 and include the following:

- Upon closure of the TMF, measures must be taken to ensure that:
  - Dust is not emitted into the atmosphere as a result of the loss of moisture from the surface of the TMF or embankments, among others;
  - Runoff does not affect surface water and groundwater; and
  - The TMF embankments do not fail.
- For tailings that are potentially acid generating, the following shall be implemented:
  - Cover with a mineral material in order to prevent the formation of acid drainage from the tailings;
  - When it is not appropriate to put measures in place to prevent the formation of acid drainage, measures must be put in place for its treatment to avoid harming water bodies, soils, and sediment, either because of its acidity or by pollution with toxic elements;
  - The surface shall be covered with the recovered soil, when applicable, or with materials that allow plant species to take root; and
  - The plant species that are used to cover the dump shall be native to the region, in order to guarantee their success and permanence with a minimum of conservation.

A mine reclamation bond is not required in Mexico.

16.11 Mine Equipment

The mining equipment descriptions and specifications in this section provide general information of the size, dimension, capacity, etc. of the selected equipment. These specifications are not intended to target equipment from any specific manufacturer or vendor.

The complete mining fleet for the Base Case is summarized in Table 16-27 showing the initial Year -2 requirement, and the maximum LOM requirement. The complete mining fleet for the Ramp-Up Case is summarized in Table 16-28 showing the initial Year -1 requirement and the maximum LOM requirement.

Table 16-27 Base Case LOM Fleet Requirements

Mobile Fleet			
Mine Mobile Fleet	**Task / Description**	**Initial Qty**	**LOM Max Qty**
Drilling			
Drill - Diesel Hydraulic - 270mm	Primary Drill	2	3
Blasting			
Blasthole Loader - 80kW	Blast hole stemmer	1	1
Loading			
Major:			
Diesel Hydraulic Shovel - 27m^3	Loading all material types	2	3
Support:			
Dozer – 306kW	Shovel support, pit ramps and roads	1	1
Wheel Dozer – 372kW	Pit clean up	1	2
Hauling			
Major:			
Haul Truck – 177 tonne	Hauling all material types	22	24
Support:			
Water Truck – 20,000 gallons	Road maintenance	1	1
Dozer – 306kW	Barren rock facility maintenance	1	2
Grader – 221kW	Road Grading, maintenance	1	1
PIT MAINTENANCE			
Dozer – 306kW	Pit Support	1	2
Excavator – 283kW	Utility Excavator	1	1
Light Plant – 20kW	Lighting plant	6	10
Crane – 40t	Crane	1	1
Fuel/Lube Truck - 3,500 gallons	Mobile Fuelling	1	1
Crew Cab Pickup	Crew Cabs, Supervisor trucks	12	12
Dozer – 306kW	Quarry / TSF Dozer	1	1

Table 16-28 Ramp-Up Case LOM Fleet Requirements

Mobile Fleet			
Mine Mobile Fleet	**Task / Description**	**Initial Qty**	**LOM Max Qty**
Drilling			
Drill - Diesel Hydraulic - 270mm	Primary Drill	2	2
Blasting			
Blasthole Loader - 80kW	Blast hole stemmer	1	1
Loading			
Major:			
Diesel Hydraulic Shovel - 27m^3	Loading all material types	2	2
Support:			
Dozer – 306kW	Shovel support, pit ramps and roads	1	1
Wheel Dozer – 372kW	Pit clean up	1	1
Hauling			
Major:			
Haul Truck – 177 tonne	Hauling all material types	16	17
Support:			
Water Truck – 20,000 gallons	Road maintenance	1	1
Dozer – 306kW	Barren rock facility maintenance	1	1
Grader – 221kW	Road Grading, maintenance	1	1
PIT MAINTENANCE			
Dozer – 306kW	Pit Support	1	1
Excavator – 283kW	Utility Excavator	1	1
Light Plant – 20kW	Lighting plant	6	6
Crane – 40t	Crane	1	1
Fuel/Lube Truck - 3,500 gallons	Mobile Fuelling	1	1
Crew Cab Pickup	Crew Cabs, Supervisor trucks	12	12
Dozer – 306kW	Quarry / TSF Dozer	1	1

16.11.1 Drilling

Primary production drilling at the Ixtaca Project utilizes diesel hydraulic rotary drills. These drills are outfitted with 270mm drill bits for use in all material types. Drills are outfitted with high precision drill positioning or GPS systems for efficient and accurate positioning, and superior data collection from each drill unit and drillhole.

The production drills operate in all high-wall rows, final walls, and other controlled blasting areas. Drill requirements, including the high-wall drilling, require 1 drill in preproduction and 2 drills in Year 1.

16.11.2 Blasting

Blasting at the Ixtaca will be performed by contractors at a mine owned explosives storage and handling facility. The contractor will operate a proprietary MMU (Mobile Manufacturing Unit), which will deliver blasting materials to the blast pattern and mix them into explosives at the drillhole.

A small front-end loader will be used for blast hole stemming. This loader will load drill cuttings, crushed rock, or gravel into the hole to stem it and more effectively direct the energy of the blast.
One 110 kW FEL (front end loader) is sufficient for the LOM.

16.11.3 Loading

16.11.3.1 Major Equipment

Production loading of waste rock and mill feed material will be performed by 27 m^3 diesel-hydraulic shovel units. These will be owned by the contractor mining company.

16.11.3.2 Support Equipment

A 306kW dozer will be stationed in the pit and is included for heavy ripping and in-pit ramp and road cuts.

A 372kW wheel dozer is included for cleaning up spilled rock at the shovel face. The wheel dozer is highly mobile and so is assumed to support the shovels operating in the pit. In addition, the in-pit dozer, or the pit maintenance or utility dozer may be available to support pit floor clean-up if the shovels are working far apart.

These large support pieces are fitted with vehicle health monitoring systems. High precision navigation is not specified for this equipment.

16.11.4 Hauling

16.11.4.1 Major Equipment

The hauler selected to match the 27m^3 shovels is the 177-t payload class diesel haul truck. As described in Section 16.9.5.8 Loading, the 177-t trucks have been determined from a previous truck/shovel matching study. The size of the haul fleet is determined by the production schedule material requirements and required truck operating hours to meet the scheduled tonnage over the haul road network for each operating period. The schedule optimizer evaluates the cycle time and destination requirements for each period and accumulates the required hours. These are combined with the appropriate availability and utilization factors to determine the required number of trucks. Haul trucks will be owned and operated by the contractor mining company for the life of the mine.

Where possible, optimization of the haulage fleet has been undertaken in this study to even out the truck fleet requirements. The LOM maximum haul fleet for the Base Case is 24 units and for the Ramp-Up Case is 17 units.

All haul trucks are fitted with fleet management systems. These are state-of-the-art data centers that report on all facets of machine health. These include machine operating temperatures, vibration, fuel consumption, etc. The trucks may be equipped with a dispatching or positioning system.

16.11.4.2 Support Equipment

The haul support fleet maintains roads and assists maintenance of the trucks (i.e. tire manipulator).

17.0 RECOVERY METHODS

Preliminary metallurgical testwork results discussed in Section 13.0 indicate that mill feed from the Ixtaca deposit can be processed using gravity concentration, conventional flotation, and leaching of a flotation concentrate to recover 90.3% of the total silver and gold in mill feed and produce a silver-gold doré.

A simplified process flow sheet to process 30,000 tonnes per day of mill feed is shown in Figure 17-1.

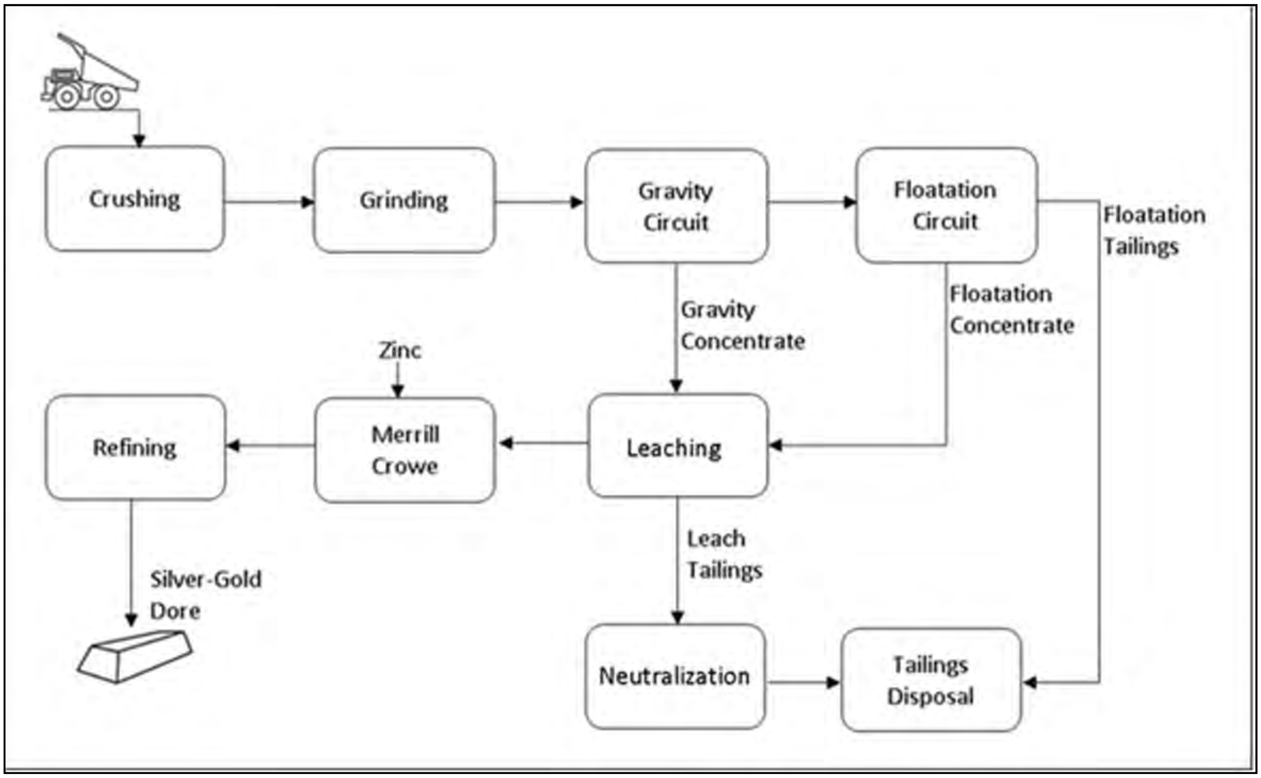


Figure 17-1 Ixtaca Simplified Flow Sheet

The processing plant includes a crushing stage, followed by grinding, gravity concentration, and flotation to produce a mineral concentrate. The gravity and flotation concentrates are leached with cyanide in agitated tanks to extract precious metals into solution. Gold and silver are precipitated from the pregnant leach solutions with zinc in a Merrill-Crowe process followed by filtration and smelting to produce a silver-gold bar with a purity of approximately 98% in precious metals.

Flotation tailings are placed in the Tailings Management Facility (TMF). Leach tailings pass through a cyanide destruction neutralization process before being placed in a separate fully lined Concentrate Tailings Management Facility (CTMF).

Process equipment planned for Ixtaca utilizes conventional technology.

18.0 PROJECT INFRASTRUCTURE

General site facilities planned for the Ixtaca Project are:

- Site access roads
- Low-grade stockpiles
- Settlement ponds for open pit drainage water
- Explosives bulk plant and magazine
- Tailings Management Facility (TMF), including the Concentrate Tailings Management Facility (CTMF)
- Rock Storage Facility (RSF)
- Power distribution
- Maintenance Facility
- Administration building
- Site wide water management facilities

Site infrastructure and ancillary buildings are presented below.

18.1 Site Access

The Ixtaca deposit, the epithermal gold-silver target within the Tuligtic Property, is located 8km 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 center located approximately 50 km north of Puebla City, and then north approximately 20 km along a gravel road to the town of Santa Maria.

Offshoots from the main access road will connect to the primary crusher, the explosive facility, and mine pit. Ancillary roads from the site process bench connect to the truck shop and fuel storage facility. Site and access roads are depicted on Figure 16-18.

Mine haul roads are designed by MMTS and are presented as part of the pit design and mining production schedule in Section 16.0.

Electric power is available on the Property as the national electricity grid services nearby towns such as Santa Maria and Zacatepec.

18.2 Process Area and Laboratory

The process area is located east and upslope of the ultimate tailings embankment crest. Process tailings are piped downslope to the tailings facility.

18.3 Maintenance Facility

The maintenance facility location is in the area of the crusher near the pit rim. Major maintenance on haul trucks will be done at the maintenance facility. Administration offices, dry, wash bays, warehouse, and fuel storage will also be located in this area.

18.4 Crushing Plant and Conveyor

The crushing plant for mill feed material is located south of the pit rim. All mill feed material that is direct fed from the pit to the mill will be crushed near the pit rim and then conveyed to the mill. Any mill feed material that is directed to the temporary low grade stockpile will be crushed near the pit rim and then truck hauled to the stockpile location.

18.5 Low Grade Stockpile

Marginal mill feed material mined in the early part of the production schedule will be placed in a stockpile location at the toe of the RSF which is downstream of the TMF. The stockpile reaches maximum size of 40.1 million tonnes in Year 8 of production in the Base Case. The entire stockpile is reclaimed for processing at the end of the mine life in the Base Case. In the Ramp-Up Case the temporary stockpile reaches a maximum size of 41.9 million tonnes in Year 10 of production. At the end of mine life in the Ramp-Up Case, there is a total of 4.3 million tonnes of material that is not reclaimed for processing and considered as part of the ultimate RSF. Reclamation activities will be done on this material along with the RSF.

18.6 Tailings and Rock Management

The proposed mining plan for the Base Case incorporates a 30,000 tpd open pit operation for a mine life of 12 years. The TMF has capacity to securely store up to 130 Mt of tailings throughout the mine life and after closure, with potential for expansion. The processing, which includes leaching the combined gravity/flotation concentrates to produce a gold and silver doré on site, will generate two separate tailings streams. Tailings management therefore includes the storage of the gravity/flotation tailings in the main (larger) part of the TMF and the concentrate tailings in a separate CTMF located within the overall TMF.

An alternative mining plan, developed for the purposes of reducing initial capital costs, is also developed. The alternative plan incorporates a Ramp-Up schedule, with an initial mill feed rate of 7,000 tpd. The mill feed rate increases to 9,000 tpd in Year 3, before increasing to the ultimate mill feed rate of 30,000 tpd in Year 6. A reduced TMF and CTMF design has been completed for this alternative plan, with a capacity to securely store a combined 121Mt of tailings and detoxed tailings over a mine life of 15 years. The TMF and CTMF designs for the Ramp-Up case have slightly lower crest heights and both can be expanded to the Base Case designs with additional building material.

A total of five potential TMF locations were originally identified based on the defined storage requirements and a site visit completed in 2012. An initial comparison of each site has been conducted, including the following items:

1. Proximity to Open Pit,
2. Elevation difference between the Open Pit and the ultimate embankment crest
3. Embankment size
4. Catchment area
5. Potential for expansion
6. Current land use
7. Land Tenure Considerations

Results of the assessment indicated that the preferred TMF location is Option 5. A viable alternative to Option 5 is Option 3, which is located above the proposed open pit and wholly contained within the open pit catchment. Option 5 is preferred mainly due to the proximity to the Open Pit, smaller catchment area and minimal land use within the catchment. Option 5 had two variations (5A and 5B) and Option 5B has subsequently been used for the PEA design. Starter and ultimate layouts of the preferred TMF Option 5B for the Base Case and Ramp-Up Case are shown in the GA drawings in the following Figures.

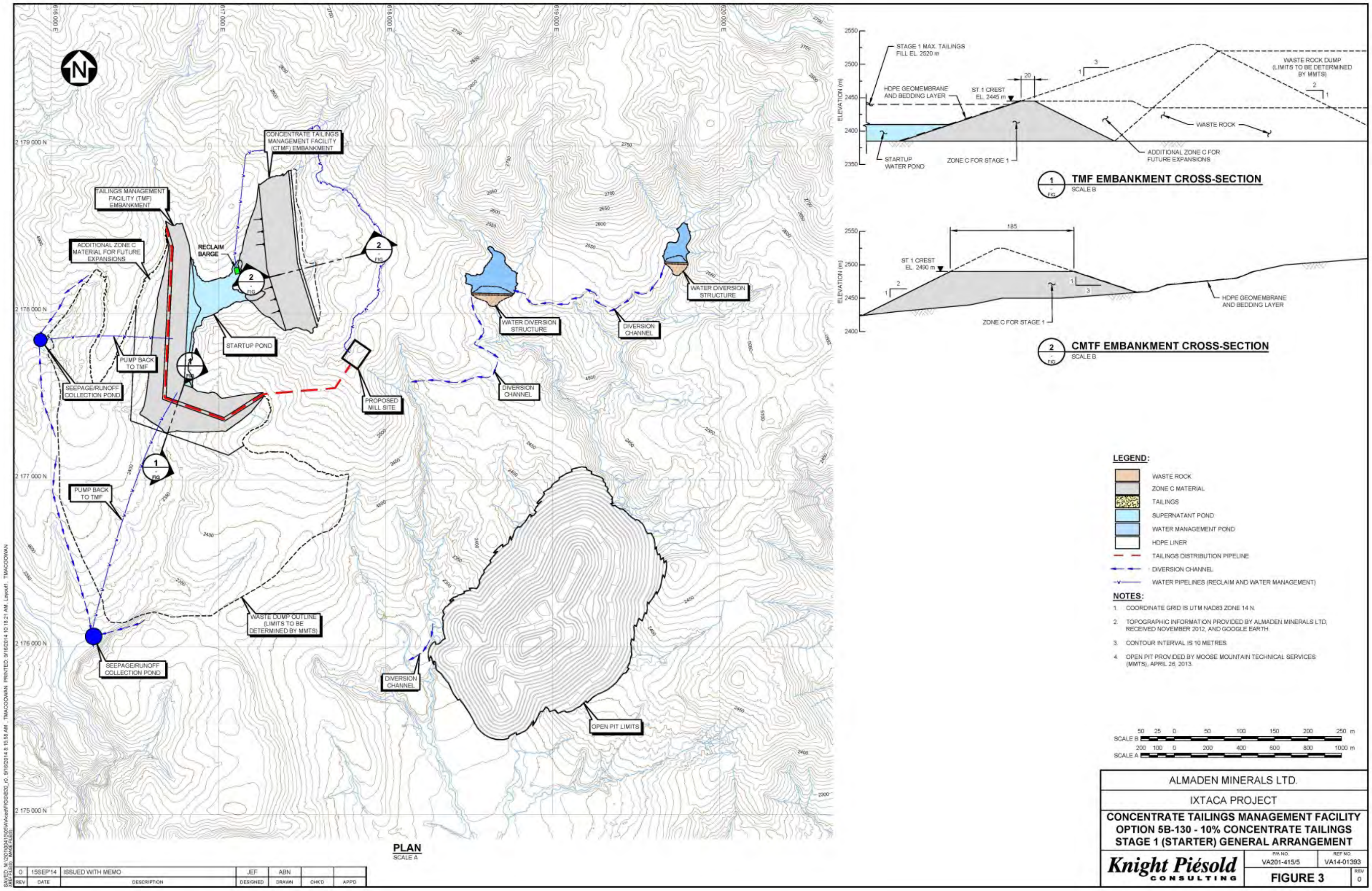


Figure 18-1 Starter Base Case Concentrate Tailings Management Facility General Arrangement

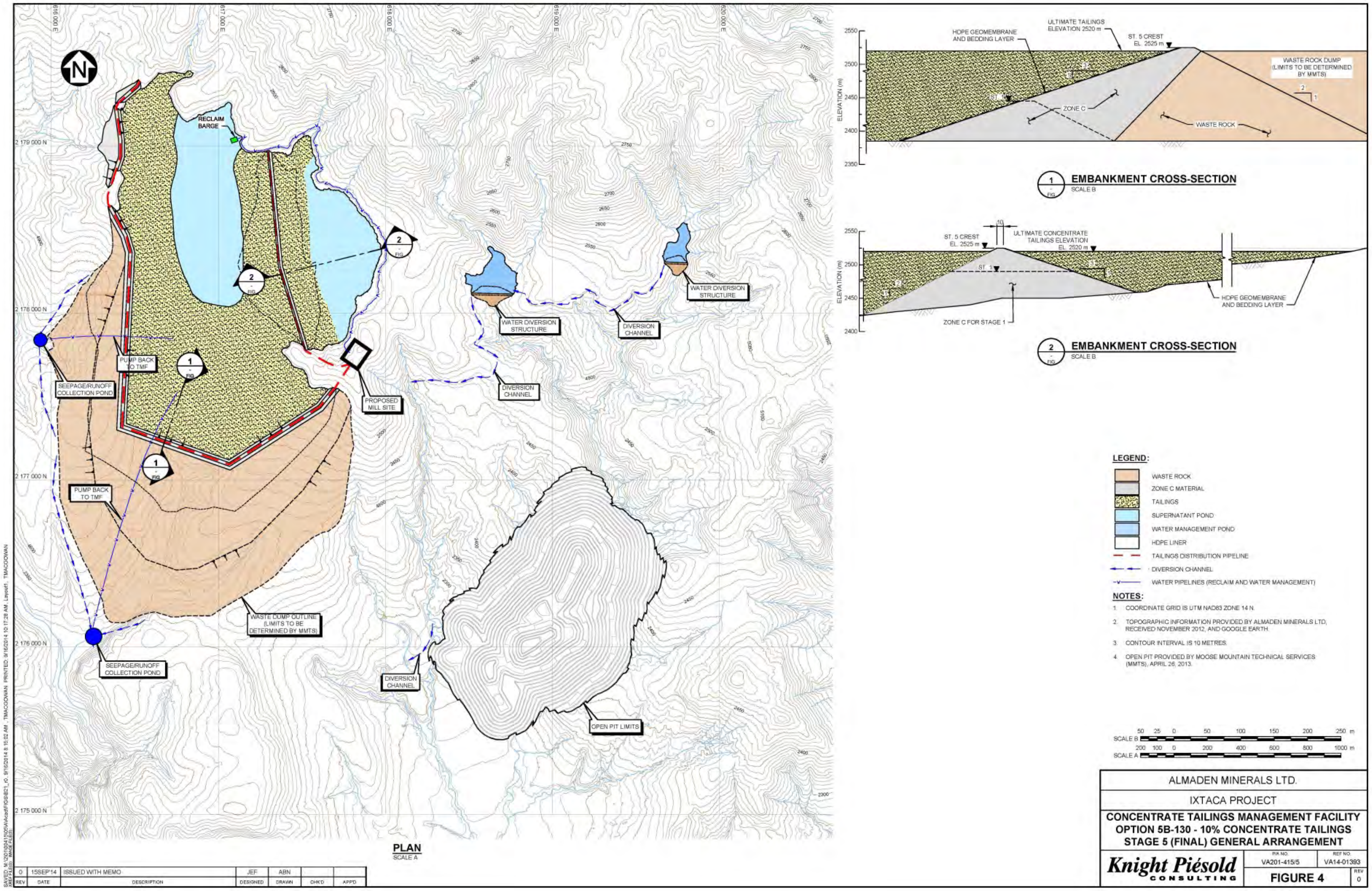


Figure 18-2 Ultimate Base Case Concentrate Tailings Management Facility General Arrangement

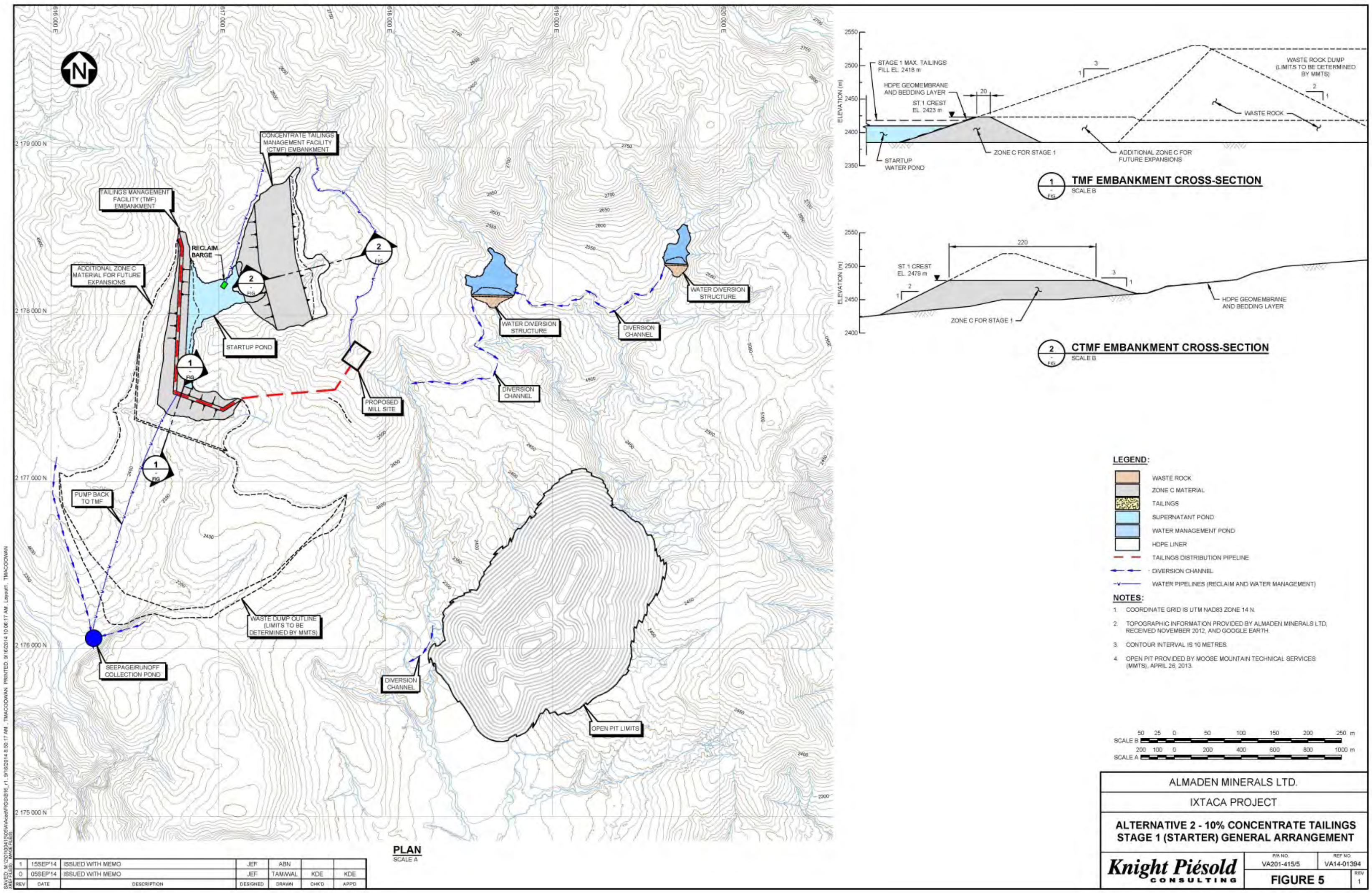


Figure 18-3 Starter Ramp-Up Case Concentrate Tailings Management Facility General Arrangement

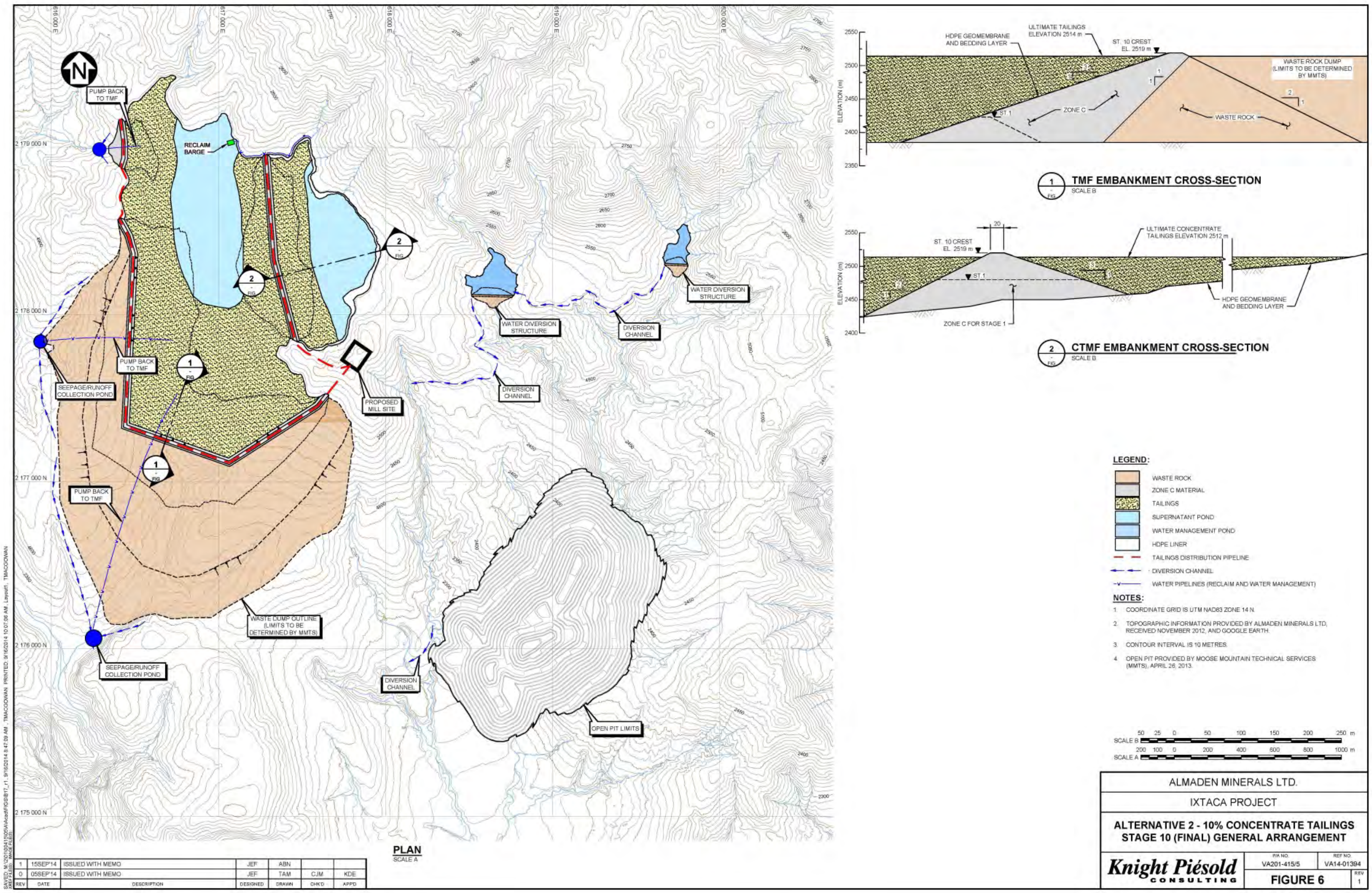


Figure 18-4 Ultimate Ramp-Up Case Concentrate Tailings Management Facility General Arrangement

Conventional slurry tailings disposal is considered as the Base Case design for the Ixtaca Project. Alternative tailings disposal methods, such as thickened or dewatered (dry stack) tailings, are not considered in this scoping level study and will be evaluated in the next phase.

The TMF is a valley-fill impoundment that incorporates a large RSF below the confining embankment, with a temporary ore stockpile downstream of the RSF. The main TMF has a lining system on the upstream face of the TMF embankment, consisting of a 100 mil HDPE geo-membrane with a non-woven geotextile underlay. The CTMF will be a separate fully lined impoundment, located within, but at the upstream end of the main TMF.

Confining embankments for both facilities will be a similar cross-section consisting of the following:

- Bedding Layer - processed material that will provide a smooth surface for installation of the geotextile and HDPE geo-membrane
- Zone C - structural embankment fill zone that will be placed and compacted in thinner lifts (approx. 600 mm)
- Waste Rock - shell zones to be placed in thicker lifts (approx. 1000 mm)
- Rock Storage Facility - located beyond the limit of the ultimate embankment, to be placed in thick lifts (approximately 5 m or as required by the mine) and compacted by the mine haul fleet

Drainage systems will be included in the TMF embankment and foundation. The drains will consist of a longitudinal underdrain system with connecting transverse drains.

The embankments will be constructed and expanded in stages to distribute costs over the life of the mine. Most of the embankment fill materials will consist of rock from the open pit. Based on the preliminary results of waste rock characterization, there is sufficient waste rock that is not Potentially Acid Generating (non-PAG) for embankment construction. Any Potentially Acid Generating (PAG) materials would be stored separately, or encapsulated within non-PAG materials within the TMF.

For preliminary design purposes, conceptual staged construction and filling schedules have been developed for both the Base Case and the Ramp-Up Case. Cross-sections through the embankments and below the RSF for Stage 1 and Stage 5 are given on Figure 18-5 and Figure 18-6, respectively for the Base Case, and similar sections for Stage 1 and Stage 10 are given on Figure 18-7 and Figure 18-8, respectively for the Ramp-Up Case.

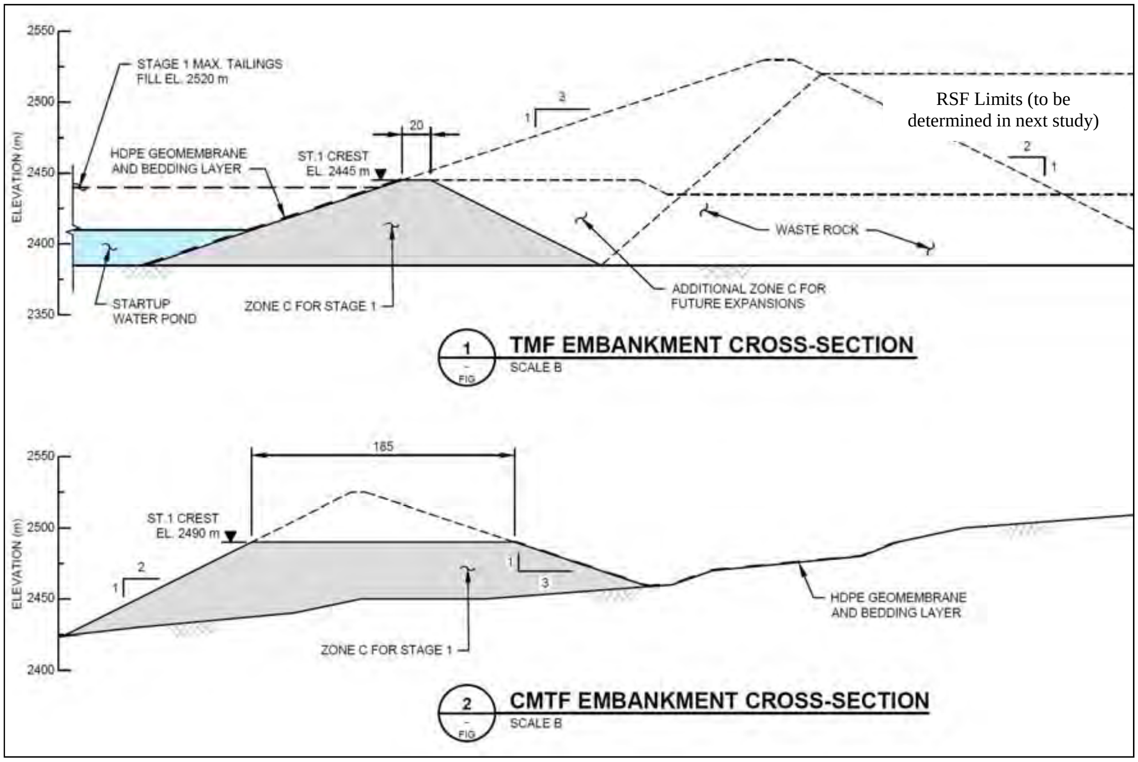


Figure 18-5 Stage 1 (Starter) Embankment Cross-section for TMF and CTMF (Base Case)

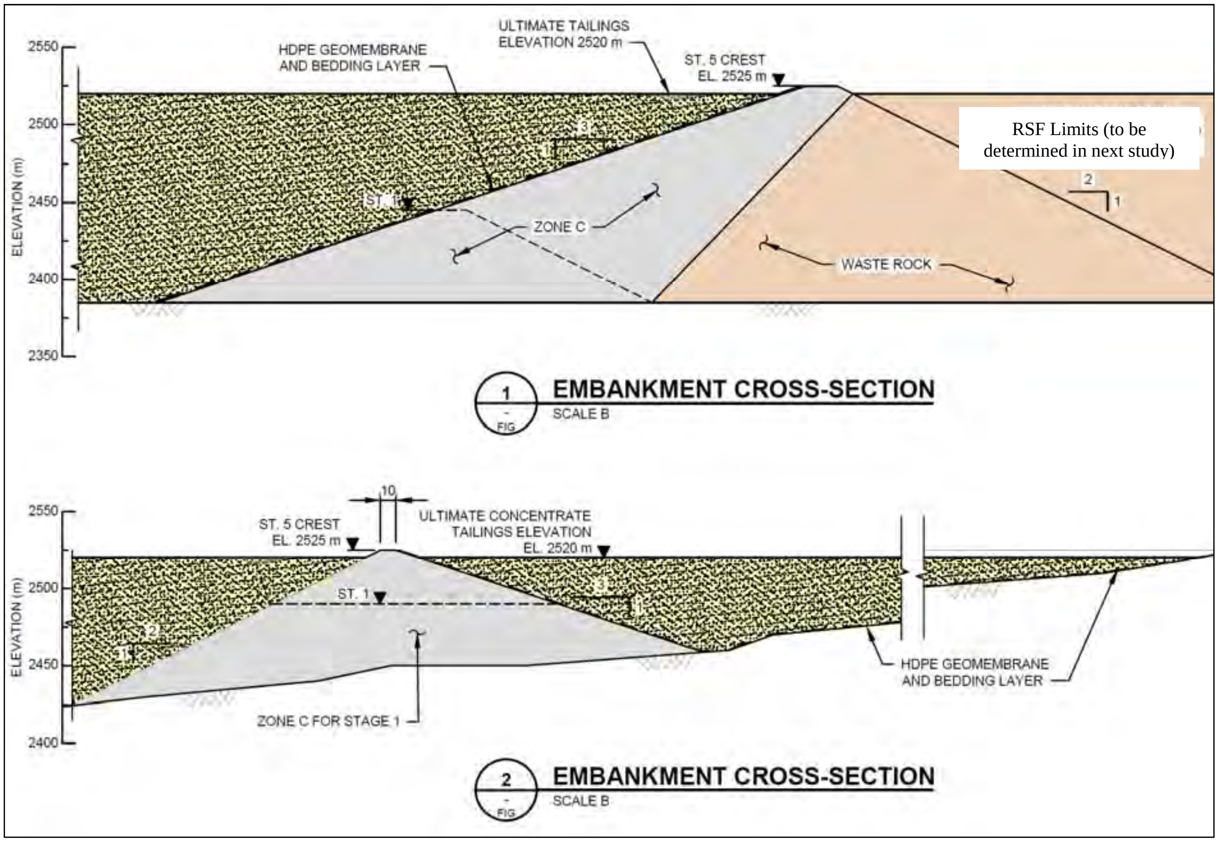


Figure 18-6 Stage 5 (Ultimate) Embankment Cross-sections for TMF, RSF, and CTMF (Base Case)

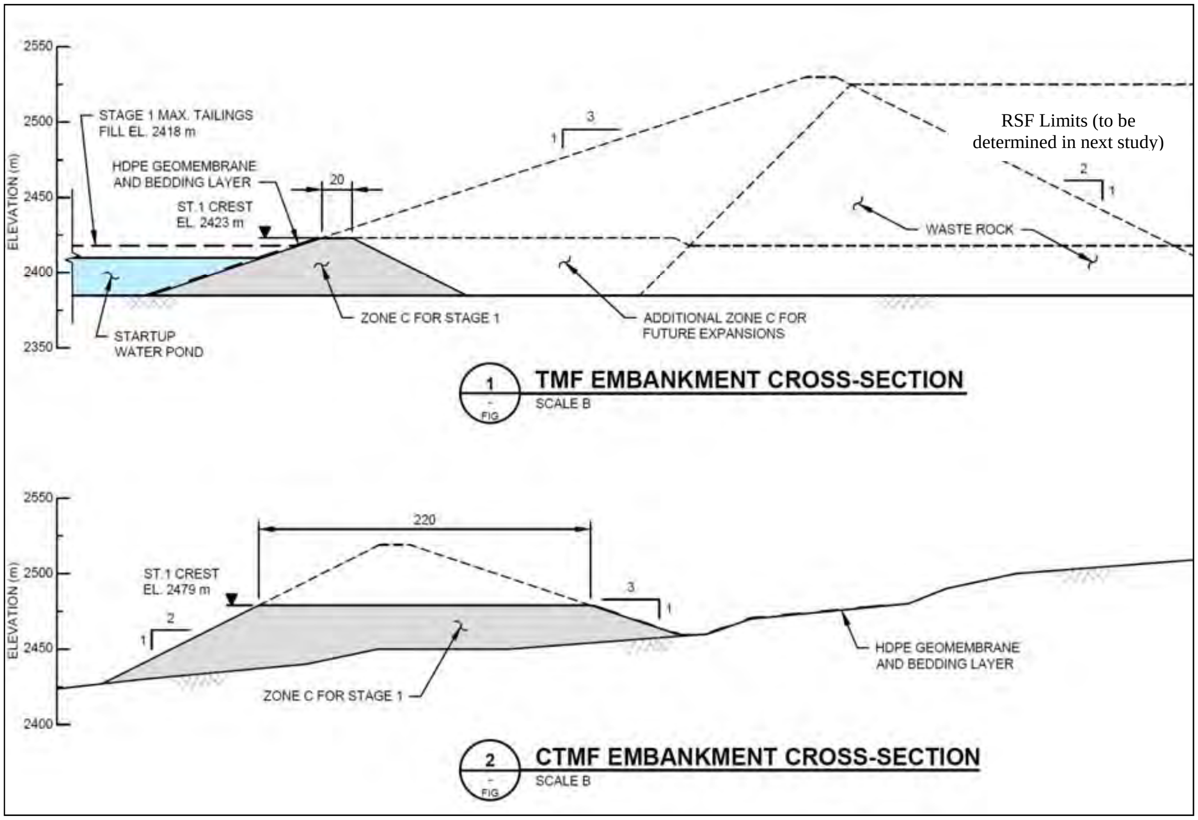


Figure 18-7 Stage 1 (Starter) Embankment Cross-sections for TMF, RSF and CTMF (Ramp-Up Case)

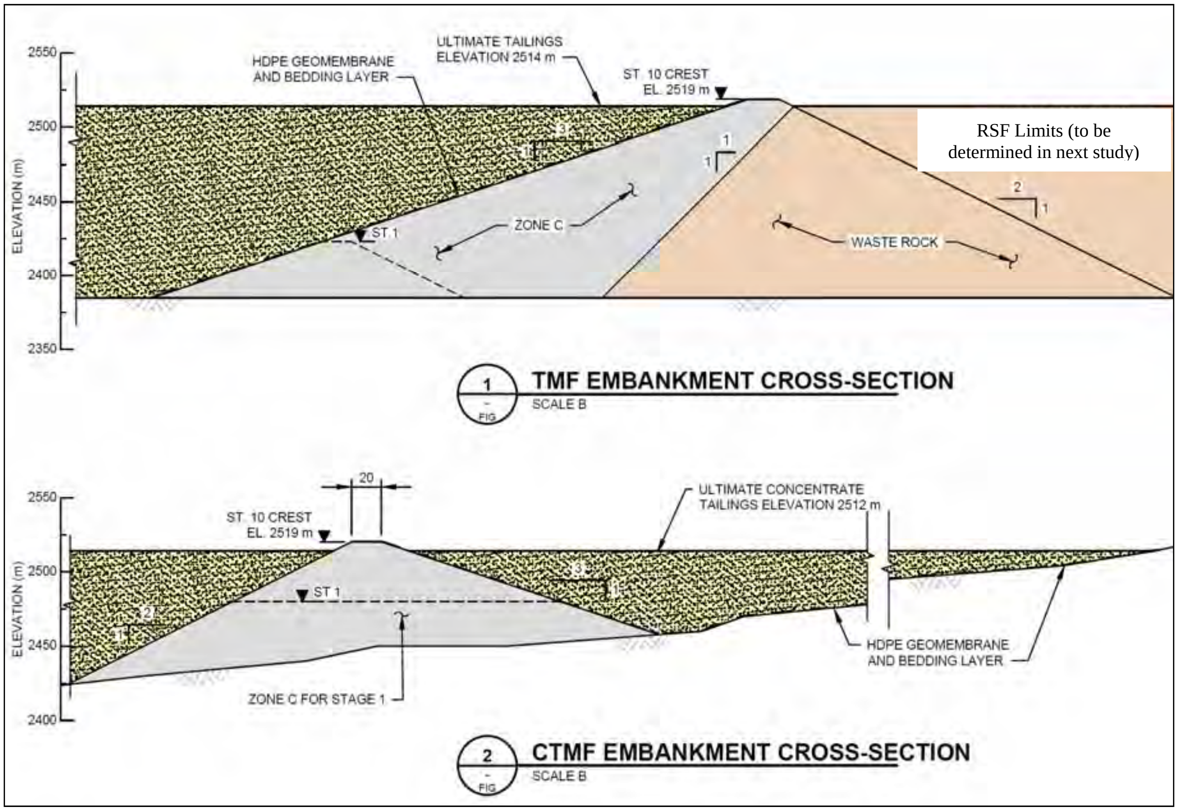


Figure 18-8 Stage 10 (Ultimate) Embankment Cross-sections for TMF, RSF and CTMF (Ramp-Up Case)

The tailings distribution system will deliver the tailings slurry in a pipeline from the mill to the TMF and CTMF. Tailings will be deposited from the perimeter of the embankments over the life of the operations. The higher mill location will allow for gravity feed of the tailings to the TMF throughout the mine life. However, tailings pumping may be required for ongoing development/operations. The tailings pipelines will be extended during operations, as the embankment is raised and expanded.

Runoff and supernatant water will accumulate in the TMF. A floating pump barge and pipeline will be installed to allow for water to be reclaimed from the supernatant pond for mill operations, to the maximum extent. The reclaim barge will be positioned in the location of the initial (start-up) pond and will retreat to the north of the supernatant pond as the tailings level rises, during operations, with the reclaim pipeline running around the north side of the facility, around the CTMF, to the plant site. Water accumulating within the CTMF will be required to be managed and will be recycled to the plant.

A seepage and runoff management system will be provided for the TMF. This system will include collection ditches that run along the toes of the embankment and RSF and discharge to collection ponds. Water from the embankment and foundation drains will also be discharged into the ponds. Recycle pumps and pipelines will return the collected water to the TMF. The ponds will also be used for water quality monitoring and sediment control.

18.7 Site Wide Water Management

A diversion system is required to prevent uncontrolled runoff from flowing into the open pit and to ensure that natural runoff from un-impacted areas is maintained to the greatest degree possible. The diversion system includes a diversion ditch that runs in an east-west direction north of the pit and isolates a significant portion of the upper catchment. Two diversion structures (water diversion dams) are required in the smaller drainages along the diversion alignment.

The water diversion dams will be constructed in a similar manner as the starter tailings embankments using select compacted rock with a geomembrane (HDPE) face liner for seepage control. The diversion ditch extends south to the existing drainage channel and the diverted water will ultimately rejoin its natural drainage course, downstream of the Project.

Additional water management measures will be implemented at the open pit and will include dewatering wells and sub-horizontal drains that may be required for open pit dewatering and to reduce pore water pressure in the pit walls. The pit water management systems will also include dewatering pumps and pipe works to remove precipitation during the rainy season and after storm events. This pit dewatering system has been developed by others, and KP has provided supplemental information relating to groundwater seepage control and pore pressure management relating to pit slope dewatering.

A preliminary water balance for the overall mine operations has been completed. The results indicate that the facility will have a net water surplus for average precipitation conditions. As a result, a separate make-up water supply system is not necessary under these conditions, although sufficient runoff must be captured to provide an adequate pond for mill startup. Baseline data collection activities currently underway, including a hydrometric and climate monitoring program, will refine water balance modelling as the project moves forward. Future modelling will include analyses of extreme conditions (wet and dry), and a detailed evaluation of potential make-up water needs.

18.8 Rock and Tailings Storage

The construction of the RSF and TMF from a mining perspective is dependent on the mining rate and availability of appropriate construction material. The TMF design summary as proposed by KP includes staged construction and filling of the TMF and CTMF for both the Base Case and the Ramp-Up Case. The stages correlate with the tailings filling schedules as shown on Figure 18-9 and Figure 18-10 below.

However, the building of the TMF and RSF are also driven by equipment hours and their respective haul cycles from the active mining bench to the destination lift of the RSF and/or TMF. Typically to smooth out the truck requirements, waste material required for the construction of the TMF requires longer hauls, and so the shorter hauls to the RSF are incorporated. The production schedule includes eight years of active mining of Phases 1 to 3, and so by the end of Phase 3, the TMF needs to be fully constructed to the final design and lift elevation requirement. This results in an accelerated construction of the TMF, and includes the minimum lift requirement for any given year of operation.

The Table 16-25 details the construction by year for the TMF the RSF, and the CTMF in cubic meters for the Base Case Life of Mine.

The production schedule distributes approximately 56% of the material to the TMF, 36% to the RSF, and 8% to the CTMF.

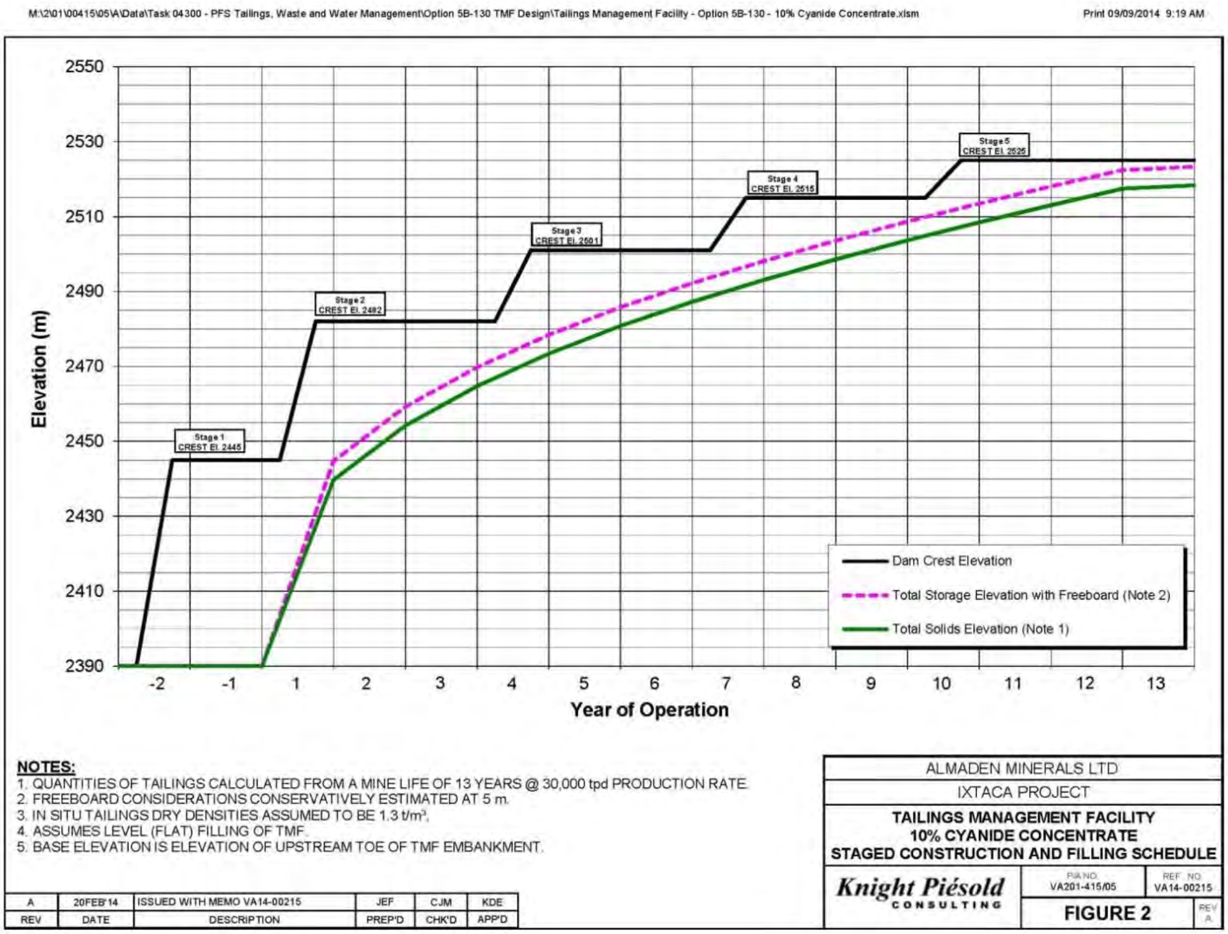


Figure 18-9 Tailings Management Facility Filling Schedule (Base Case)

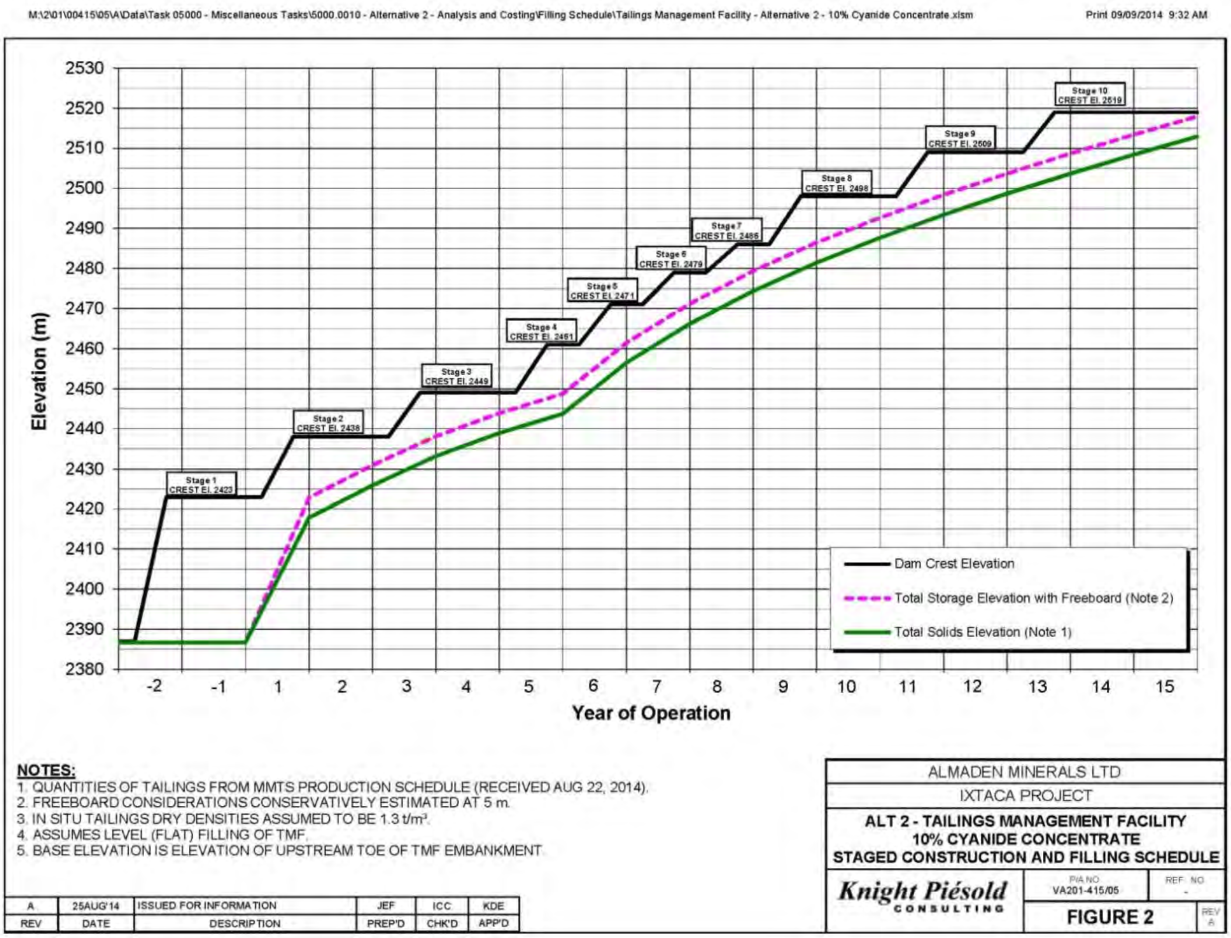


Figure 18-10 Tailings Management Filling Schedule (Ramp-Up Case)

19.0 MARKET STUDIES AND CONTRACTS

The Ixtaca Project is expected to produce a silver-gold bar assaying approximately 95% silver and 2% gold when assuming 98% purity; these are typical specifications for precious metals produced by the mining industry. The market for silver-gold bars is extensive with numerous buyers operating in the spot market as well as in long term contracts in North America, Europe, and Asia. Ixtaca has not yet entered into sales agreements with potential buyers.

The economic evaluations of Ixtaca are considering conservative refining terms for the sale of the silver-gold bar as follows:

- Payable gold = 99.8%
- Payable silver = 90%
- Refining charge for gold = $10.00/oz
- Refining charge for silver =$0.60/oz
- Transportation cost = $1.00/oz

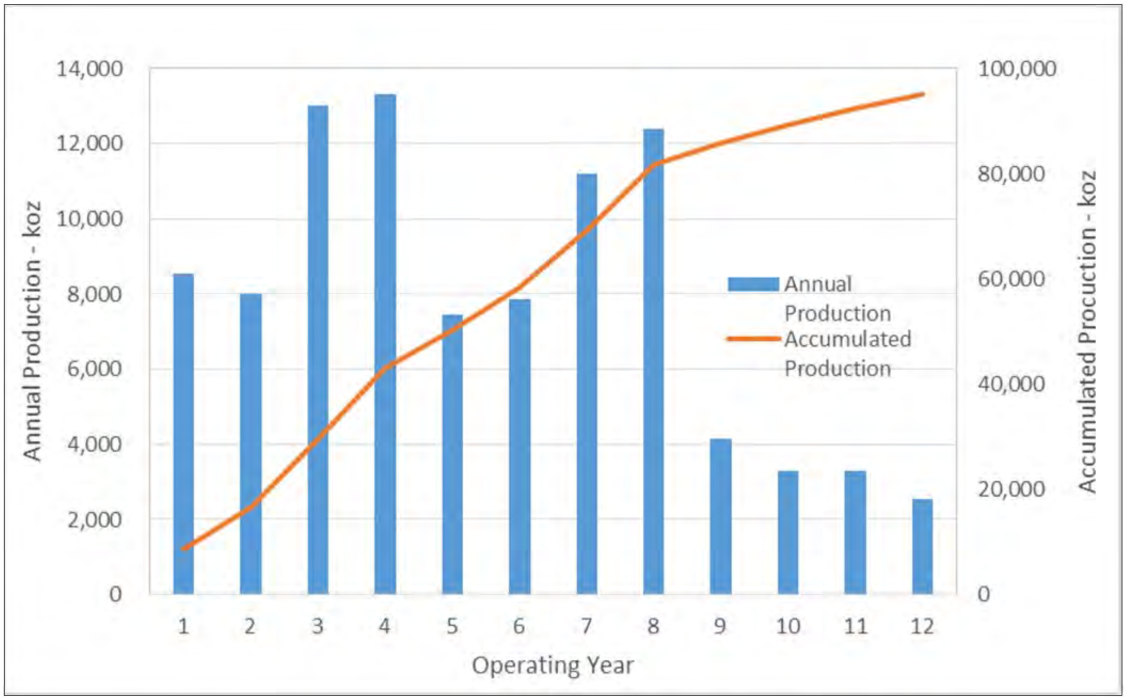


Figure 19-1 Base Case Silver-Gold Production (98% Purity)

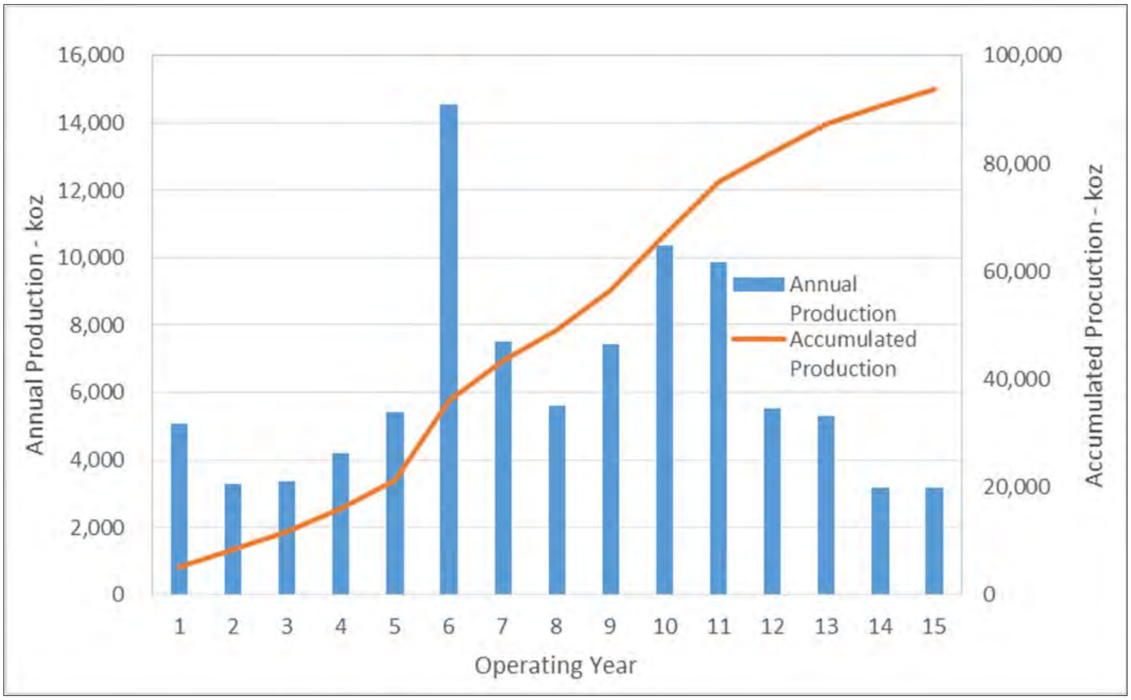


Figure 19-2 Ramp-Up Case Silver-Gold Production (98% Purity)

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Studies

20.1.1 Previous Environmental Studies

20.1.1.1 Meteorology

Knight Piésold has been retained by Almaden to initiate a climate baseline monitoring program at the Ixtaca Project. A climate station was commissioned in early April 2013 at the Project site. The station continually monitors the following parameters:

- Wind Speed (m/s),
- Wind Direction (Degrees from True North),
- Temperature (°C),
- Relative Humidity (%),
- Atmospheric Pressure (mbar),
- Incoming Solar Radiation (W/m^2),
- Net Solar Radiation (W/m^2), and
- Rainfall (mm).

Meteorological parameters are being collected from a variety of sensors, each connected to the CR1000 data logger. A customized data logger program dictates how often the sensors are monitored and in this case generates and stores both hourly and daily statistics.

The station is located at 14Q 616,641 E, 2,176,063 N, at 2,433masl and is sited to provide representative climate data for the Project.

Additionally, climate data are available from Government of Mexico regional meteorological stations; several of which are located within 35km of the Project, each with over 25 years of daily data on precipitation, evaporation, and minimum and maximum temperatures.

20.1.1.2 Water Quantity and Quality

In June 2009, prior to Almaden conducting any drilling activities, the Autonomous University of San Luis Potosi conducted the "Baseline Hydrogeological Study of a Mining Project in San Francisco Ixtacamaxtitlan, Pue."

The main purposes of the study were to:

- Establish baseline quality parameters in surface water and springs that are used primarily for domestic use by the people; and
- Establish the flow of the main springs and streams in the study area.

The specific objectives of the study were to:

- Conduct a water well census, including wells, springs, and streams;
- Characterize water quality baseline conditions by sampling water from springs and streams;
- Determine the flow of the main surface catchments; and
- Define the hydrogeological units of the area based on the information obtained.

The results obtained in this study were:

- Surface water is of great importance for the development of productive activities within the region, mainly for agriculture and human consumption.
- Hydraulic census data indicate that the main use of surface water relies on near-surface sources, with virtually no exploitation of deep groundwater.
- The flow of springs and streams are directly dependent on the amount of precipitation.
- Most of the trace elements were below the maxima presented in the Official Mexican Standard NOM-127-SSA1-1994 for comparison of water quality. Parameters which were elevated above the standards were pH (in 5% of total samples), sulphate (10%), aluminum (38%), and manganese (15%).
- The baseline study demonstrates that the chemical quality of the water within the study area is suitable for human consumption. There are currently no on-site anthropogenic impacts on surface water and natural concentrations of potentially toxic elements commonly associated with mineralization (including arsenic, cadmium, mercury, and lead) are, at, or below the reference standard.

In 2013, hydrogeological investigations were undertaken to understand groundwater flows and volumes in the project area. Also in 2013, ongoing surface quality sampling was initiated to serve as baseline information to support a future environmental assessment of the project. Water quality parameters include total and dissolved metals, anions, and nutrients.

20.1.1.3 Flora and Fauna

Almaden has engaged Consultores en Ecologia con Vision Integral S.A. de C.V. (COREVI) who have completed a flora and fauna study for the Project in order to develop the baseline conditions. Vegetation was studied to determine the structure and diversity of the existing communities and the geographic coverage of each characterized vegetation type. Terrestrial communities and waterfowl were also documented and assessed. The Project impacts will be developed and considered with respect to the biotic communities and, as appropriate, mitigation programs will be developed and implemented.

20.1.2 Known Environmental Issues

Water usage is an extremely important social and environmental consideration for all existing and potential mining operations. Mexico in general is an arid country however in the Ixtaca region; preliminary water balance studies show that a mining operation would have a net water surplus for average precipitation conditions. As a result, a separate make-up water supply system is not necessary under these meteorological conditions. If a net water surplus is realized, the project would have a minimal impact on water resources and would not require subsurface supply from aquifers. Baseline data collection activities currently underway, including a hydrometric and climate monitoring program, will refine water balance modelling as the project moves forward. Future modelling will include analyses of extreme conditions (wet and dry), and a detailed evaluation of potential make-up water needs.

Rock quality has been reviewed for the presence of Potential Acid Generating (PAG) material which necessitates a specific storage strategy. The limestone and carbonate host rock, which will constitute much of the waste rock of the Project, has a high natural buffering capacity. Static geochemical testing is currently underway to characterize this further. Based on the preliminary examination of geochemical data, a large proportion of the assumed waste rock (approximately 85%) is considered to be not Potentially-Acid-Generating (non-PAG), while 15% could be Potentially-Acid-Generated (PAG). More detailed waste characterization studies have been initiated to identify and define any acid generating potential. The results of this work will be incorporated into future engineering designs. It should be noted that if higher metal prices were realized, much of the PAG rock has the potential to be converted to run of mine material and would not be considered waste.

Vegetation and animal species protected under Mexican regulations can become disturbed during project construction. To mitigate this, flora and fauna “rescue” programs will be implemented to relocate any protected species from the Project footprint.

20.2 Permitting

20.2.1 Permitting Requirements

20.2.1.1 Mexican Legal Framework

Mine permitting in Mexico is administered by the federal government body Secretaría de Medio Ambiente y Recursos Naturales (SEMARNAT). Guidance for the federal environmental requirements is derived from the Ley General del Equilibrio Ecológico y la Protección al Ambiente (LGEEPA). Article 28 of the LGEEPA specifies that SEMARNAT must issue prior approval to parties intending to develop a mine and mineral processing plant. An Environmental Impact Assessment (Manifestación de Impacto Ambiental (MIA) by Mexican regulations) is the mechanism whereby approval conditions are specified where works or activities have the potential to cause ecological imbalance or have adverse effects on the environment. This is supported by Article 62 of the Reglamento de la Ley Minera. Article 5 of the LGEEPA authorizes SEMARNAT to provide the approvals for the works specified in Article 28.

The LGEEPA also contains articles that are relevant to conservation of soils, tailings management, water quality, flora and fauna, noise emissions, air quality, and hazardous waste management. The Ley de Aguas Nacionales provides authority to the Comisión Nacional de Agua (CONAGUA), an agency within SEMARNAT, to issue water abstraction concessions, and specifies certain requirements to be met by applicants.

Another important piece of environmental legislation is the Ley General de Desarrollo Forestal Sustentable (LGDFS). Article 117 of the LGDFS indicates that authorizations must be granted by SEMARNAT for land use changes to industrial purposes. An application for change in land use or Cambio de Uso de Suelo (CUS), must be accompanied by a Technical Supporting Study (Estudio Técnico Justificativo, or ETJ).

Guidance for implementation and adherence to many of the stipulations of environmental legislation is provided in a series of Normas Oficiales Mexicanas (NOM). These NOM provide specific procedures, limits, and guidelines, and carry the force of law. The relevant permit application will be developed as the Project progresses.

20.2.1.2 Land Use Plans

Voluntary surface land use agreements have been negotiated with landowners within the exploration prior to the start of exploration activities. Additional or revised land ownership negotiations are ongoing to accommodate alternate potential tailings storage areas, potential waste disposal areas, and potential processing plant sites. Mineral Claim owners have the right to obtain the temporary occupancy, or creation of land easements required to carry out exploration and mining operations, under the Federal Mining Law.

20.3 Social or Community Information

20.3.1 Potential Social or Community Requirements and/or Plans

Almaden has invested in the Tuligtic area since 2001 and has employed up to 70 people for its ongoing exploration program. Almaden is invested in the ongoing training of employees and is actively involved in the community's health and social welfare projects. Almaden's community involvement includes local construction and improvement projects, reforestation and recycling projects, and numerous educational initiatives. Almaden has led community meetings in schools and common areas to introduce the Project and have staffed personnel in the local area to lead community engagement. Almaden has implemented a comprehensive community relations and education program for employees and all local stakeholders to explain the exploration program underway as well as the potential impacts and benefits of any possible future mining operation at Ixtaca. This program includes regular tours of the Ixtaca site which are open to all local stakeholders and include visits to the core shack and drilling operations. Almaden has also started a general mining educational program in the form of tours to existing metal mines operated by third parties elsewhere in Mexico. To date the company has conducted seven such tours which have enabled over 200 local residents to gain first hand understanding of operating mines.

Impacts to the socio-economy of the area may occur as the Project is developed into a mine and becomes a source of jobs. Almaden plans to continue its open communication with the communities to provide for realistic expectations of any proposed mining operations and the social impacts of such a development.

20.4 Mine Closure

Mexico does not have detailed reclamation legislation, but has national environmental laws and is currently developing more specific mine closure requirements. Guidance for the construction, operation, and closure of tailings impoundments is included in a national regulation revealed in 2003 (NOM-141-SEMARNAT-2003). Post operation criteria are presented in Section 5.7 of NOM-141-SEMARNAT-2003 and include the following:

- Upon closure of the tailings impoundment, measures must be taken to ensure that:
  - Dust is not emitted into the atmosphere as a result of the loss of moisture from the surface of the tailings dam or from the curtain wall, among others;
  - Run-off does not affect surface water and groundwater; and
  - The tailings dam does not fail.
- The following aspects shall be complied with when tailings are potentially acid generating:
  - Cover with a mineral material in order to prevent the formation of acid drainage from the tailings;
  - Plant species that promote the acidification shall not be used;
  - When it is not appropriate to put measures in place to prevent the formation of acid drainage, measures must be put in place for its treatment to avoid harming water bodies, soils, and sediment, either because of its acidity or by pollution with toxic elements;
  - The surface of the dump shall be covered with the recovered soil, when applicable, or with materials that allow plant species to take root; and
  - The plant species that are used to cover the dump shall be native to the region, in order to guarantee their success and permanence with a minimum of conservation.

No mine reclamation bond is required in Mexico.

21.0 CAPITAL AND OPERATING COSTS

All currencies shown in this Section 21.0 are expressed in USD. The expected accuracy range of this estimate is in the order of +/-35% which is suitable for a PEA-level study.

Initial capital of $399 million is estimated for the Ixtaca Project Base Case scenario. The Ramp-Up Case is estimated to have an initial capital of $244 million.

21.1 Initial Capital Cost Estimate

Initial capital costs are factored estimates derived from a combination of MMTS experience in similar projects and consultation with contractors and equipment suppliers. The Table below shows the breakdown of initial capital costs for the Ixtaca Project. These cost estimates do not include taxes or duties.

Table 21-1 Capital Cost Estimates for Ixtaca Project

	Base Case	Ramp-Up Case
ITEM	**$M (USD)**	**$M (USD)**
Site Infrastructure, Power and Others	$20.4	$19.4
TMF and Water Management	$44.7	$29.0
Pre-Stripping	$64.5	$37.8
Mining Equipment	$8.0	$7.7
Processing and Plant	$194.5	$105.5
Indirects, EPCM, Contingencies and Owner’s Costs	$67.4	$44.1
TOTAL START-UP CAPITAL	**$399.4**	**$243.5**

Further descriptions and basis of estimates for these items are explained below.

21.1.1 Site Infrastructure

Site infrastructure costs include access road upgrades, maintenance facility, truck wash, administration, laboratory and mine dry building, fuel storage, security building and fencing. Costs based on MMTS experience and benchmarked unit costs such as $/kilometre or $/m^3 of material. Labour and equipment rates and hours are estimated as well as material costs.

21.1.2 Treatment Management Facility and Water Management

The TMF starter dam and CTMF starter dam capital cost requirements are derived from work done by Knight Piésold.

21.1.3 Pre-stripping

Contractor unit mining operating costs are applied against the pre-stripping tonnage detailed in the production schedule. Total pre-stripping material required to be moved in Year -2 and Year -1 is 40M tonnes in the Base Case and 24 million tonnes in the Ramp-Up Case. The estimated contractor mining operating costs are $1.57/tonne for volcanic material, $1.81/tonne for other rock material and $1.31/tonne for material conveyed to the mill (further detailed in Section 21.2)

21.1.4 Mining Equipment

The Project will be operated using contractor mining therefore the capital cost of the mining equipment will be covered by the contractor company. The project owner is expected to pay a mobilization fee to

the contractor mining company which is estimated as 10% of the total capital cost of purchasing the equipment new. This mobilization fee is listed as a mining equipment capital cost.
The capital cost for mining equipment is derived from MMTS consultation with equipment suppliers. Capital costs include purchase price, estimated delivery, assembly and a 5% spares factor. Estimated delivery costs are representative of rail and road delivery to projects in Western Canada. Actual delivery costs realized at the Ixtaca Project may be lower, therefore the delivery costs are considered to be conservative for this study. The spares factor is calculated as 5% of the purchase price.

The individual unit purchase cost for major components of the mining fleet are shown in the Table below:

Table 21-2 Mining Fleet Capital Cost

Unit/Description	Purpose	Capital - $M USD
177-tonne haul truck	Hauling mill feed and waste	$3.8
26m^3 diesel-hydraulic	Loading mill feed and waste	$10.2
Diesel drill – 270mm	Primary drill	$2.5
Water truck – 20,000 gallons	Haul Roads Water Truck	$2.2
Track dozer – 306 kW	Pit and RSF maintenance	$1.3
Grader – 221 kW	Road Maintenance	$1.0
Rubber Tire Dozer – 372 kW	Pit Clean-Up	$1.3
Stemming Loader – 80 kW	Stemming Blast Holes	$0.2
Front-end Loader – 373 kW	Mill and Stockpile Loader	$1.1
Excavator – 283 kW	Utility Excavator	$0.6
Fuel/Lube Truck – 3,500 gallons	Shovel/Truck service vehicle	$1.0
Crane – 40 tonne	Utility crane	$1.0
Crewcab Pickup	Supervision and crew transportation	$0.04
Light Plants	Lighting Plant	$0.03

The delivery, assembly and spares cost are included in the 10% mobilization fee paid to the contractor. Therefore the amounts included for these items in the capital estimate account for extra smaller auxiliary mining equipment not detailed in the above Table.

21.1.5 Processing and Plant

The capital expenditure for the Base Case processing facilities is estimated to be $194.5 million. This includes the 30,000 tonne/day process plant ($179 million), dore process plant ($8 million) and the conveyor from the pit rim area to the plant ($7.5 million). The initial capital expenditure for the Ramp-Up Case processing facilities is $105.5 million. This includes the 7,000 tonne/day process plant ($90 million), doré process plant ($8 million) and conveyor from the pit rim area to the plant ($7.5 million). MMTS estimated the construction of Ixtaca's processing facilities using a benchmark with comparable processing facilities that are adjusted to reflect Ixtaca's flowsheet, throughput and location.

21.1.6 Indirects, EPCM, Contingencies and Owner's Costs

A 30% contingency is applied to the TMF and Water Management item. A 20% allowance is applied to the Processing and Plant item for Contingency and Indirects. The Site Infrastructure, Power and other items has the following factors applied:

- 20% Indirects
- 15% project EPCM
- 10% owner's costs
- 25% contingency

The detailed components and costs of each item are listed in Appendix H.

21.1.7 Expansion Capital

Expansion capital includes the net cost of purchasing a 30,000 tonne per day mill for the Ramp-Up Case and the credit for the 7,000 tonne per day mill after decommissioning. A 70% credit is assumed for the 7,000 tonne per day mill.

21.1.8 Sustaining Capital

Sustaining capital includes the cost of raising the TMF and CTMF embankments, associated pumps and pipe works, water management from the TMF and open pit catchments, maintaining the process plant, and mobilization fees for expanding the fleet size. The Base Case sustaining capital for these items, with the exception of the process plant and mobilization fees, has been estimated by Knight Piésold to be $93.8 million, which includes a 30% contingency. The Ramp-Up Case sustaining capital for these items has been estimated by Knight Piésold to be $99.3 million. The sustaining capital costs are split equally over the mine life for each case. Sustaining capital for the process plant is estimated at $1 million per year for 30,000 tonne per day throughput and $0.25 million per year for 7,000 tonne per day throughput. Total sustaining capital over the life of the Project is $110.3 million for the Base Case and $110.8 for the Ramp-Up Case.

21.2 Operating Cost Estimate

The total life of mine operating costs for the Ixtaca Project are $14.48/tonne mill feed for the Base Case and $15.85/tonne mill feed for the Ramp-Up Case. Conveying mill feed is estimated to save approximately $0.50/tonne compared to hauling mill feed by trucks. Operating costs are detailed in the Tables below:

Table 21-3 Projected Operating Costs

	USD		
	30,000 tpd	**7,000 tpd**	
Contractor Mining – Rock	1.81		$/tonne mined
Contractor Mining - Volcanics	1.57		$/tonne mined
Mill Feed conveyed to mill	1.31		$/tonne mined
General Mine Expense	0.07		$/tonne mined
Stockpile re-handle	1.00		$/tonne stockpile material
Processing	9.00	14.00	$/tonne mill feed
G&A	10.0	4.2	$ million/yr
TMF Management	1.0	0.25	$ million/yr

Table 21-4 Projected Operating Costs per mill feed tonne

	USD		
	Base Case	**Ramp-Up Case**	
Contractor Mining	3.89	$4.34	$/tonne mill feed
Processing	9.00	$9.60	$/tonne mill feed
Stockpile re-handle	0.34	$0.52	$/tonne mill feed
TMF Management	0.10	$0.09	$/tonne mill feed
Reclamation	0.18	$0.19	$/tonne mill feed
G&A	0.80	$0.92	$/tonne mill feed
GME	0.17	$0.18	$/tonne mill feed
Total LOM Operating Cost	**14.48**	**$15.85**	**$/tonne mill feed**

**Numbers may not add up due to rounding*

G&A costs are reduced to $5M/yr. (approximately $0.47/tonne mill feed) when stockpile re-handling and processing are the only activities occurring and there is no mining activity in the pit.

21.2.1 Contractor Mining

The estimated owner-operated mining operating costs (including fuel, operators, maintenance labour and components) are $1.45/tonne. This estimate is based on benchmarks of similar sized operations in Mexico and scaled to represent the size of mining equipment used. Contractor mining costs are estimated to have a 25% mark-up applied to the expected owner operating costs. The 25% mark-up is comprised of a 20% portion for the contractor margin as well as 5% portion to cover finance costs. Detailed long-term contracts with a contractor mining company can result in a lower mark-up. For this study 25% is assumed to be a conservative estimate.

21.2.2 Processing

The processing operating costs are estimated from a benchmark of similar processing operating plants, including operations in Mexico, and adjusted to reflect local electrical energy cost, and labour cost.

21.2.3 Stockpile Re-handling

Stockpile re-handling costs are estimated to be $1/tonne stockpile material. The re-handling costs include loading and haulage costs from the temporary stockpile location up to the mill site.

21.2.4 Tailings Management Facility Water Management

An allowance of $1 million/yr. is estimated by Knight Piésold for the operating costs dealing with water management (TMF and CTMF) for a 30,000 tpd operation and $0.25 million/yr. for a 7,000 tpd operation.

21.2.5 Reclamation

Reclamation costs have been estimated by Knight Piésold to be approximately $23M (including a 15% contingency). Included in this estimate are reclamation costs for the Tailings Storage Facility and RSF buttress, plant site, open pit, water diversion structures and environmental monitoring for 5 years after production ceases.

21.2.6 General & Administration (G&A)

An allowance of $10 million/yr. has been added for G&A costs for a 30,000 tpd operation and $4.2 million/yr. for a 7,000 tpd operation. This estimate is based on a benchmark of operations of similar size, including operations in Mexico, to the Ixtaca Project. G&A costs are reduced during the later portions of the mine life when stockpile re-handling and processing are the only activities occurring and there is no mining activity in the pit.

21.2.7 General Mine Expense

GME costs are estimated to be $0.17-0.18/tonne mill feed. The breakdown of positions and yearly salaries are shown in the Table below. A 25% payroll burden is applied to the base salaries to estimate the total yearly cost.

Table 21-5 Annual General Mine Expense Costs - USD

MINE OPERATIONS	Base Salary ($)	Burden	Loaded Salary ($)	Quantity	Annual Cost ($)
Operations General Foreman	61,500	25%	76,875	1	76,875
Training General Foreman	45,000	25%	56,250	1	56,250
Drilling & Blasting Foreman	53,400	25%	66,750	1	66,750
Maintenance General Foreman	61,500	25%	76,875	1	76,875
Maintenance Planning Clerk	26,400	25%	33,000	2	66,000
Electrical Foreman	45,000	25%	56,250	2	112,500
Administration Assistant	26,400	25%	33,000	1	33,000
Technical Services					
Senior Geologist	53,400	25%	66,750	1	66,750
Pit Geologist	45,000	25%	56,250	2	112,500
Ore Grade Technicians	35,400	25%	44,250	1	44,250
Project Engineer	45,000	25%	56,250	1	56,250
Mine Engineer	45,000	25%	56,250	1	56,250
Drilling & Blasting Engineer	45,000	25%	56,250	1	56,250
Drilling & Blasting Technician	35,400	25%	44,250	1	44,250
Surveyor	35,400	25%	44,250	1	44,250
Engineering Clerk	26,400	25%	33,000	1	33,000
Senior Geotechnical Engineer	53,400	25%	66,750	1	66,750
Chief Engineer	61,500	25%	76,875	1	76,875
Clerk (part, supplies, ordering)	21,000	25%	26,250	2	52,500
Mine Manager	72,000	25%	90,000	1	90,000
Safety and LPO	26,400	25%	33,000	3	99,000
Environmental department	28,800	25%	36,000	2	72,000
Accounting and Payroll	21,000	25%	26,250	4	105,000
IT department	18,000	25%	22,500	2	45,000
Maintenance Superintendent	90,000	25%	112,500	1	112,500
Operations Superintendent	90,000	25%	112,500	1	112,500
Total Yearly GME					**$ 1,708,875**

The average mill feed tonnes per year for the Base Case is 10,650kt resulting in average GME costs of $0.17/tonne.

The total owner operating costs and contractor operating costs during the mine production life are estimated to be $1,814 million for the Base Case and $1,918 million for the Ramp-Up Case.

21.3 Cost of Production

The total life of mine cost of production for the Base Case and Ramp-Up Case are shown in Table 21-7 below.

The Base Case metal prices used for economic analysis are $1320/oz for gold and $21/oz for silver which means that silver is valued at 1:62.9 gold equivalency. The life of mine doré production is detailed in Table 21-6 below.

Table 21-6 Life of Mine Doré Production

	Base Case	Ramp-Up Case
Gold ounces	1,562,000	1,539,000
Silver ounces	93,461,000	92,218,000
Gold-Equivalent ounces	3,048,000	3,005,000

Table 21-7 Life of Mine Cost of Production

	Base Case		Ramp-Up Case	
	$ million	**$/oz AuEq**	**$ million**	**$/oz AuEq**
Operating Costs	$1,814	$595	$1,918	$638
Sustaining Capital	$110	$36	$111	$37
All-in Sustaining Costs	**$1,924**	**$631**	**$2,029**	**$675**
Initial Capital	$399	$131	$244	$81
Expansion Capital	$0	$0	$116	$39
All-in Costs	**$2,324***	**$762**	**$2,389**	**$795**

**Numbers may not add due to rounding*

22.0 ECONOMIC ANALYSIS

22.1 Introduction

An economic analysis is prepared using the pre and after tax financial models for the Base Case and the Ramp-Up Case. The Base Case has a twelve year mine life and approximately 125 million tonnes of in-pit resources while the Ramp-Up Case has a fifteen year mine life and approximately 121 million tonnes of in-pit resources.

The economic analysis is based on production schedules that include Inferred Mineral Resources, which are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the PEA forecast will be realized or that Inferred Mineral Resources will ever be upgraded to Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

All currencies referring to total project analysis are in USD unless specified otherwise.

NPV values are calculated to the start of the Project financing in Year -3 for the Base Case and Year -2 for the Ramp-Up Case. The Ramp-Up Case requires two years of pre-production while the Base Case requires three.

Base Case metal prices remain the same as the May 2014 PEA report and were based on a combination of spot prices to date in 2014 and current common peer usage. Base Case metal prices are as follows:

- Gold - $1320/oz
- Silver - $21/oz

This study considers leaching the combined gravity/flotation concentrate to produce a gold and silver doré on site. The doré terms estimated are as follows:

- Doré Au payable – 99.8%
- Doré Ag payable – 90%
- Au Refining - $10/oz
- Ag Refining - $0.60/oz
- Transport - $1/oz

Yearly cashflows for the Base Case and Ramp-Up Case are detailed in Appendix I.

The pre-tax cashflows are shown in the Figures below:

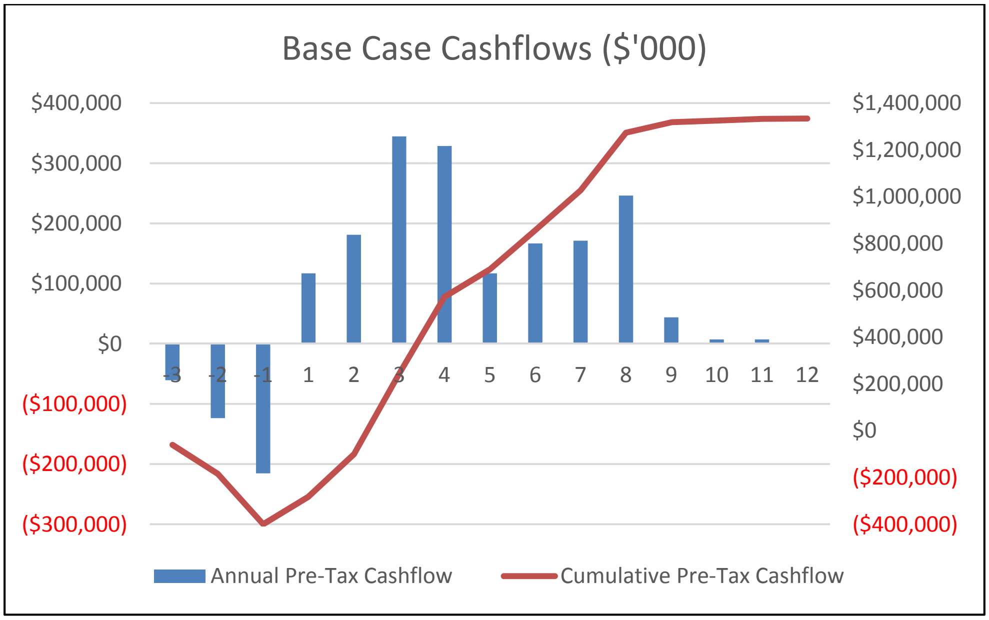


Figure 22-1 Base Case Pre-Tax Cashflows (undiscounted)

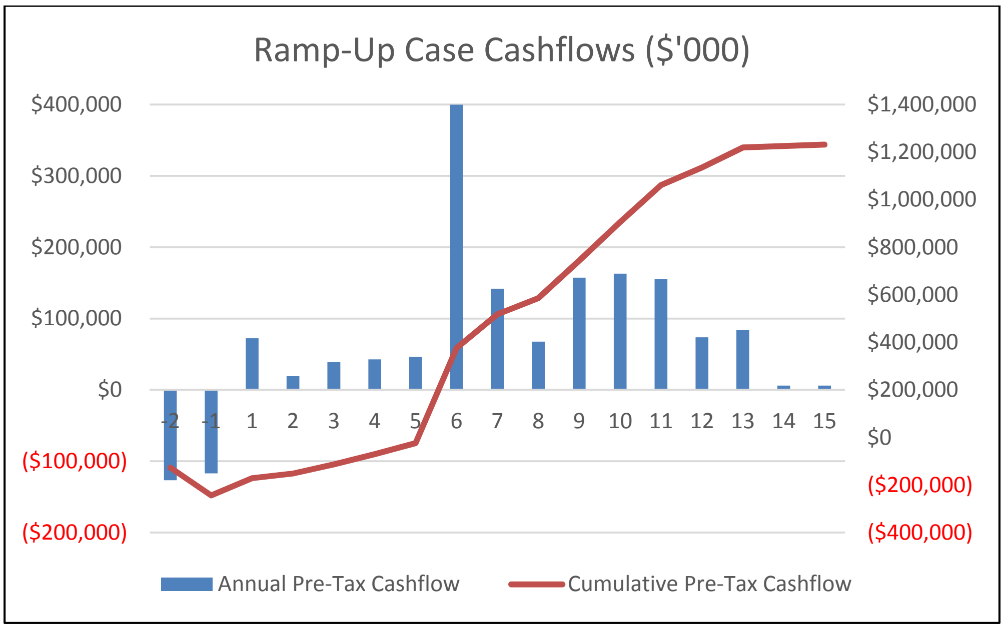


Figure 22-2 Ramp-Up Case Pre-Tax Cashflows (undiscounted)

In both cases during the later years of production, the low-grade stockpile material is processed. Due to reduced operating costs (no mining activity) during those years, this material is able to be processed economically on a marginal basis.

The following pre-tax financial results are calculated using the Base Case metal prices for the Base Case mining scenario:

- 37.2% IRR
- 2.3 year payback on $399 million capital
- $842 million NPV at a 5% discount rate
- $640 million NPV at a 8% discount rate

The following pre-tax financial results are calculated using the Base Case metal prices for the Ramp-Up mining scenario:

- 28.9% IRR
- 4.2 year payback on $244 million initial capital
- 0.3 year payback on $116 million expansion capital
- $699 million NPV at a 5% discount rate
- $497 million NPV at a 8% discount rate

22.2 Taxes and Mining Duties

Effective January 1, 2014, the Mexican Tax Reform increased corporate income tax rate from 28% to 30% and introduced two new mining duties. The Tax Reform includes the implementation of a 7.5% Special Mining Duty (SMD) and a 0.5% Extraordinary Mining Duty (EMD) on gross revenue from the sale of gold, silver and platinum. The SMD is applicable to earnings before income tax, depreciation, depletion, amortization and interest. The SMD and EMD are tax deductible for income tax purposes. The total income taxes and mining duties for the life of the Project amount to $484 million for the Base Case and $443 million for the Ramp-Up Case.

The annual after-tax cash flow results are shown in the Figures below:

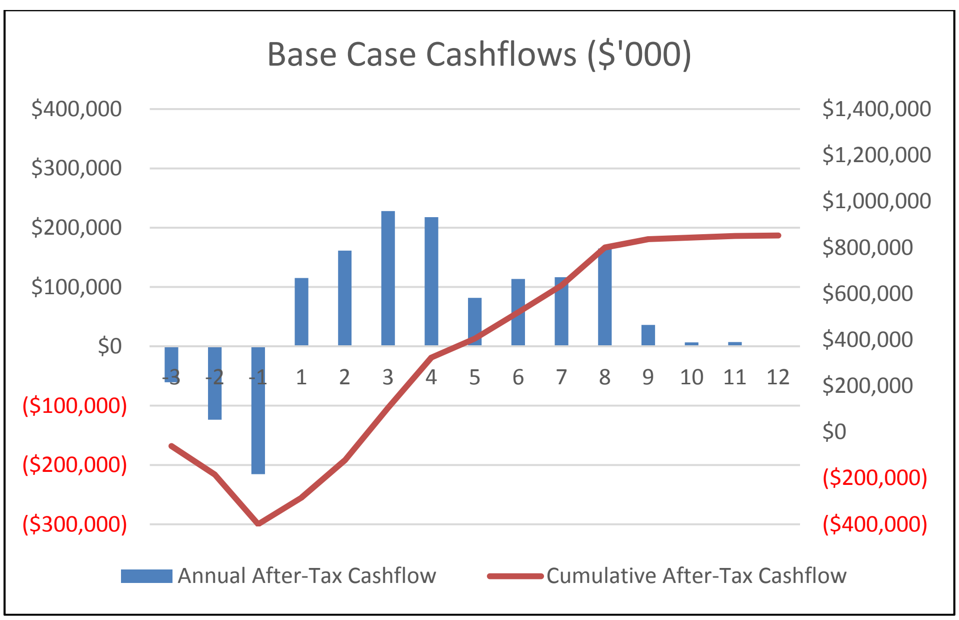


Figure 22-3 Base Case After-Tax Cashflows (undiscounted)

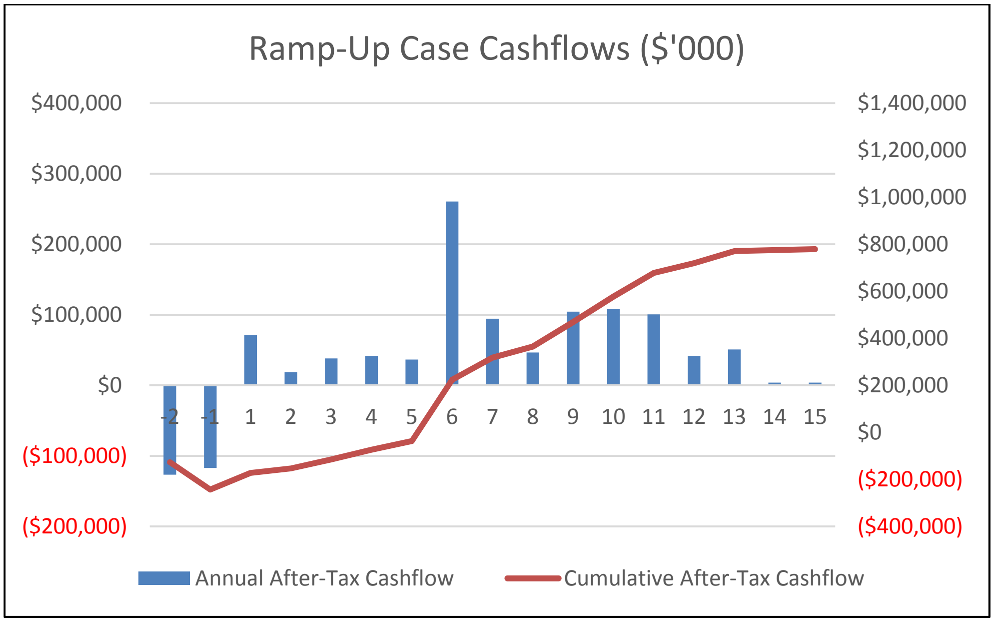


Figure 22-4 Ramp-Up Case After-Tax Cashflows (undiscounted)

The following after-tax financial results are calculated using the Base Case metal prices for the Base Case mining scenario:

- 28.3% IRR
- 2.5 year payback on $399 million capital
- $515 million NPV at 5% discount rate
- $378 million NPV at 8% discount rate

The following after-tax financial results are calculated using the Base Case metal prices for the Ramp-Up Case mining scenario:

- 23.2% IRR
- 4.5 year payback on $244 million initial capital
- 0.4 year payback on $116 million expansion capital
- $427 million NPV at 5% discount rate
- $294 million NPV at 8% discount rate

The total taxes paid over the life of the mine schedules are shown in the Table below:

Table 22-1 Taxes Paid Over Life of Mine ($ Million)

	Base Case	Ramp-Up Case
Income Tax	$365.1	$333.8
Extra-ordinary mining duty	$18.3	$18.0
Special mining duty	$100.1	$91.3
Total Taxes	**$483.5**	**$443.1**

For the Base Case, the taxes decrease the IRR from 37.2% to 28.3%% and reduce the NPV by $327 million at a 5% discount rate and $262 million at an 8% discount rate. For the Ramp-Up Case, the taxes decrease the IRR from 28.9% to 23.2% and reduce the NPV by $272 million at a 5% discount rate and $203 million at an 8% discount rate

22.3 Metal Price Scenarios

In addition to the Base Case price, two additional metal price scenarios have been examined. The first is an Alternate Case which represents the lowest sustained metal prices over the last three year period during which gold and silver prices retreated from their previous highs in 2011. The second is a three year trailing average price. This price case scenario represents the upside potential should metal prices regain their previous strength. The summary of Base Case scenario and the two additional scenarios are presented in the chart below:

Table 22-2 Base Case Mining Schedule with Metal Price Scenarios ($ million)

	Alternate Case*		Base Case		3 Year Trailing Average	
	Pre-Tax	After-Tax	Pre-Tax	After-Tax	Pre-Tax	After-Tax
Gold Price ($/ounce)	$1200		$1320		$1530	
Silver Price ($/ounce)	$18		$21		$29	
Net Cash Flow	$889	$558	$1,334	$852	$2,334	$1,496
NPV (5% discount rate)	$538	$315	$842	$515	$1,514	$950
NPV (8% discount rate)	$395	$216	$640	$378	$1,179	$727
Internal Rate of Return	28.1%	20.8%	37.2%	28.3%	53.0%	41.4%
Payback	2.7 yrs	3.0 yrs	2.3 yrs	2.5 yrs	1.7 yrs	2.0 yrs

* The lowest grade stockpile material processed at the end of the mine life is below break-even cut-off grade at the Ramp-Up Case metal prices. In the Ramp-Up Case this material is not processed and is counted as waste. This in turn shortens the mine life to 9 years (from 12 years)

Table 22-3 Ramp-Up Case Mining Schedule with Metal Price Scenarios ($ million)

	Alternate Case**		Base Case		3 Year Trailing Average	
	Pre-Tax	After-Tax	Pre-Tax	After-Tax	Pre-Tax	After-Tax
Gold Price ($/ounce)	$1200		$1320		$1530	
Silver Price ($/ounce)	$18		$21		$29	
Net Cash Flow	$792	$494	$1,231	$779	$2,218	$1,415
NPV (5% discount rate)	$424	$246	$699	$427	$1,314	$826
NPV (8% discount rate)	$284	$151	$497	$294	$972	$603
Internal Rate of Return	21.5%	16.7%	28.9%	23.2%	42.5%	34.8%
Initial Capital Payback*	5.0 yrs	5.2 yrs	4.2 yrs	4.5 yrs	2.9 yrs	3.2 yrs
Expansion Capital Payback	0.4 yrs	0.5 yrs	0.3 yrs	0.4 yrs	0.2 yrs	0.3 yrs

*Cash Flows, NPV and IRR numbers reflect the larger mill expansion capital being financed internally from production revenue. Payback is calculated without including the mill expansion capital in order for a relative understanding of the timing of revenue streams

** The lowest grade stockpile material processed at the end of the mine life is below break-even cut-off grade at the Ramp-Up Case metal prices. In the Ramp-Up Case this material is not processed and is counted as waste. This in turn shortens the mine life to 13 years (from 15 years)

22.4 Sensitivity Analysis

Cash flows have been estimated for the Ixtaca Project to test the Project NPV sensitivity to:

- Operating Cost
- Initial Capital Cost
- Metal Price

The production schedule, stockpiling and in-pit resources are kept constant for these sensitivity analyses. The pre-tax NPV at a 5% discount rate and IRR sensitivities are summarized in the Tables below:

Table 22-4 Base Case Mining Schedule Pre-Tax NPV (5%) and IRR Sensitivities ($ million)

	Operating Cost Sensitivity		Initial Capital Sensitivity		Metal Price Sensitivity	
Variance	**NPV (5%)**	**IRR**	**NPV (5%)**	**IRR**	**NPV (5%)**	**IRR**
-20%	$1,081	43.1%	$913	45.0%	$334*	20.6%*
-10%	$962	40.2%	$878	40.8%	$573*	29.7%*
Base	$842	37.2%	$842	37.2%	$842	37.2%
+10%	$708*	33.5%*	$807	34.0%	$1,100	43.8%
+20%	$584**	29.7%**	$771	31.3%	$1,357	49.8%

** Mine life is shortened to 9 years in this scenario*

***Mine life is shortened to 8 years in this scenario*

Table 22-5 Ramp-Up Case Mining Schedule Pre-Tax NPV (5%) and IRR Sensitivities ($ million)

	Operating Cost Sensitivity		Initial Capital Sensitivity		Metal Price Sensitivity	
Variance	**NPV (5%)**	**IRR**	**NPV (5%)**	**IRR**	**NPV (5%)**	**IRR**
-20%	$934	34.5%	$744	34.0%	$234*	15.1%*
-10%	$817	31.8%	$722	31.2%	$457*	22.4%*
Base	$699	28.9%	$699	28.9%	$699	28.9%
+10%	$572*	25.6%*	$676	26.8%	$934	34.4%
+20%	$464*	22.4%*	$654	25.0%	$1,169	39.4%

** Mine life is shortened to 13 years in this scenario*

The marginal grade stockpile material becomes un-economic if the operating costs increase by 10% or more or the metal prices drop by 10% or more. However, the mine production schedules are designed to process this material at the end of the mine life and removing this material does not have a great impact on the IRR, NPV or payback period. The mine production sequence targets the best material first, thereby limiting the Project exposure to changes in operating or initial capital costs as well as a drop in metal prices.

The Project NPV (5%) has the largest sensitivity to metal prices and is least sensitive to initial capital cost changes. The Project pre-tax NPV at 5% discount rate sensitivity is shown in the Figures below:

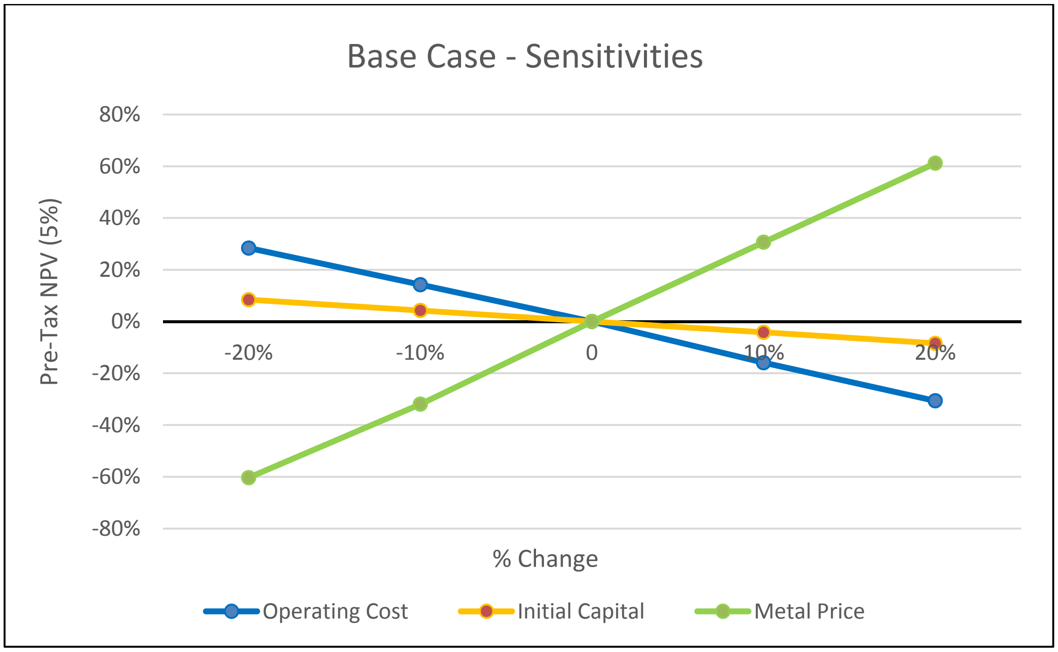


Figure 22-5 Base Case Mining Schedule Pre-Tax NPV (5%) Sensitivities

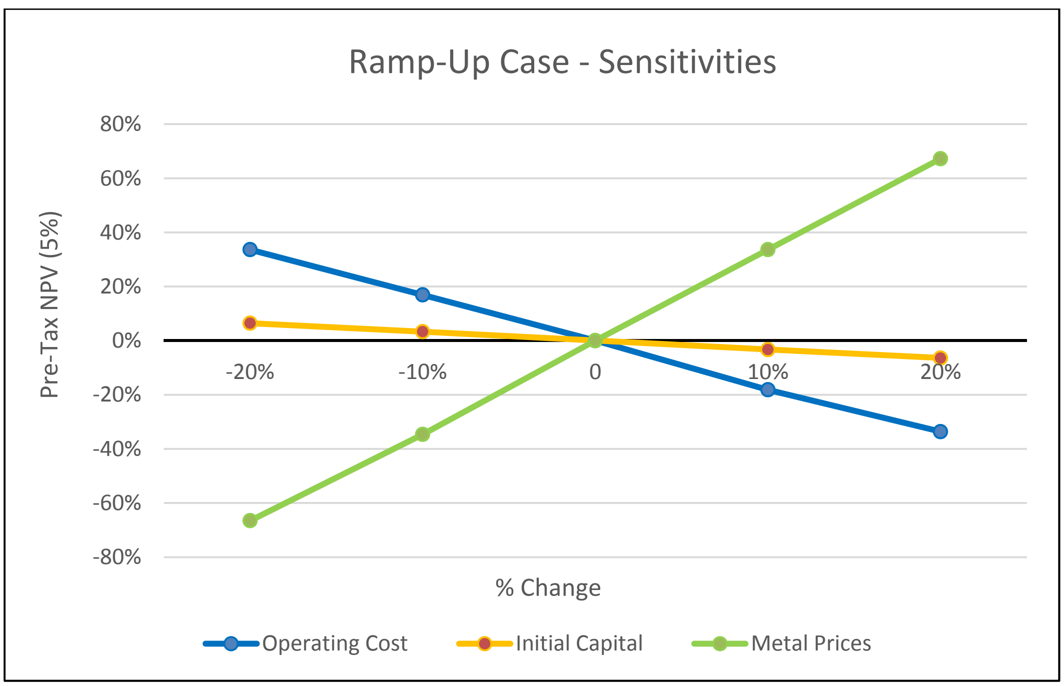


Figure 22-6 Ramp-Up Case Mining Schedule Pre-Tax NPV (5%) Sensitivities

23.0 ADJACENT PROPERTIES

23.1 Santa Fe Metals Corp. Cuyoaco Property

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

23.1.1 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 sixteen documented, historical workings on the Pau Project, many of which are believed to be as old as 16th century. The largest workings include the 170m x 200m 'El Magistral' open pit, three levels of underground workings at 'California' as well as 'Lincon' (two 50m 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 10m 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. has returned high-grades of copper, silver, gold, lead and zinc from the exposed rock within workings, and mapping in 2011 has found that many of the adits ended in mineralization. Soil and rock sampling by Santa Fe Metals in 2011 has focused on further exploration of the northern part of the Pau Claim and mapping skarn mineralization between known adits. Highlights include a 7.21g/t Au, 27.7g/t Ag skarn sample in the El Magistral zone. Low grade gold (0.32g/t Au) is found within the granodiorite itself, and a previously unknown skarn showing has been discovered in the north of the Property, a further 1km north of the La Juanita adits.

23.1.2 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 1km long and 800m wide. In 2011, a parallel dyke and sill system 200m wide and 600m in length has been 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 1m and 10m wide and form a 1km by 800m 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 has delineated a large gold rich envelope called the Santa Anita zone. Mineralization is found to be coarse, free metallic-gold and electrum in calcite stringers associated with narrow dacitic dikes hosted in a skarn-hornfels-limestone sequence. A limited chip sampling program of the underground workings returned an average grade of 3g/t. Fifty-eight (58) samples were collected in total.

The drilling of five shallow holes (607 metres in total) in 2005/2006 intersected gold mineralization with one hole intersecting 12 metres of 2.45g/t Au and another hole intersecting 4 metres of 2.54g/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 200m and a strike length of 600m. To date, Santa Fe Metals has collected 29 channel samples from dykes to the north of the Property that has returned values greater than 0.1g/t.

23.2 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 7km 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.

24.0 OTHER RELEVANT DATA AND INFORMATION

The authors are not aware of any other relevant information with respect to the Tuligtic Property that is not disclosed in the Report.

25.0 INTERPRETATION AND CONCLUSIONS

A PEA open pit mine plan has been developed for the Ixtaca Gold-Silver deposit using a NI 43-101 compliant Resource Estimate. The PEA mine plan is updated with two scenarios both of which show the strong economic viability of the Ixtaca deposit and it is recommended that Almaden proceed with a pre-feasibility study (PFS).

The following Table discusses the potential risks to future development of the Project and the current mitigation measures as well as possible future mitigation measures.

Table 25-1 Risk factors and Mitigation

Risk	Explanation	Potential Outcome	Current Mitigation	Possible Future Mitigation
Metal Prices	-Metal prices have the highest impact on the economic viability of the Project.	-A drop in the metal prices has a large reduction in project NPV and shortens the mine life to 9 years	-Pit phases are designed to target areas of highest economic return first -Production schedule utilizes stockpiling strategy to attain quick payback and return on capital investment inside the shortened mine life	
In-fill drilling	-Not all of the Inferred resources get upgraded to Measured or Indicated	-Ultimate pit size used in the PFS study has less resources than the PEA study and the Project life is shorter	-Resource has been "over-drilled" for PEA level study (only 16% of in-pit resources are Inferred) -Pit phasing is designed to initially target areas that are unaffected by the "loss" of Inferred resources on the ultimate economic pit. Areas with greater sensitivity to the inclusion of Inferred resources are mined later in the production schedule. (The 80% MI pit loses 18% and 23% of the contained gold and silver respectively, with a total loss of 31% of the insitu mill feed tonnes. The loss is predominately in the North East deposit area.) -Stockpile strategy has been incorporated into the mine schedule	-Target deeper north-east area of deposit with infill drilling -Combine post underground mining potential with open pit
Land Use	-Private ownership of land in the Project area	-Current land owners may reject the proposed infrastructure locations	-An alternate TMF location (3B) north of the open pit has been examined and found to be economically viable solution -The majority of land in the pit area is held under agreements with Almaden	-Negotiations of for land required for the Project will be done by Almaden
Dilution	-Extra mining dilution will lower the mill head grade	-Project NPV is reduced by ~35% if mill feed head grades are reduced by 10%	-An extra 3% mining dilution is added to the internal dilution already included in the block model	-Detailed in-pit assay program to define mill feed/waste boundaries at higher level of detail -Selective mining plan detailed in

Risk	Explanation	Potential Outcome	Current Mitigation	Possible Future Mitigation
				future studies
Recovery	-Project economics are sensitive to recovery	-A drop in recovery has a large reduction in project NPV and shortens the mine life	-Extensive metallurgical testing has been done to provide detailed results and support the PEA recovery assumption -Doré final product is assumed for PEA study (increases metal recoveries to concentrate when compared to producing a concentrate only)	-Additional testing on variability samples by ore type. Leaching of rougher concentrates
Operating Costs	-Operating cost has the next largest impact after metal prices	-Base Case Mine life is shortened to 9 years if operating costs increase by 10% or 8 years if operating costs increase by 20%	-A conservative contractor mark-up is assumed for the PEA -Stockpiling strategy is incorporated so payback and positive economics are still generated when mine life is shortened	-Calculate operating cost from first principles
Initial Capital Costs	-Financing is difficult to obtain and the capital “hurdle” may be too high for potential investors	-Project is unable to be started due to financing restrictions	-A viable Ramp-Up Case is examined which reduces the initial capital from \$399 million to \$244 million (39% reduction) and still mines to same ultimate pit limit as the Base Case	-Examine opportunities to purchase used equipment

The authors conclude that the Ixtaca deposit is well suited for proceeding to a PFS based on the following:

a) The Ixtaca deposit is well suited for open pit mining with higher grade material near surface, easy access to infrastructure and close access to the regional power grid.
b) Previous social community work done by the client has allowed for a social license to explore in the area.
c) The Project demonstrates strong economic viability at a variety of metal prices with a significant upside potential should metal prices regain previous strengths seen in the three year trailing average.
d) The resources are well defined inside the chosen ultimate economic pit limit with 84% classified as Measured or Indicated.
e) The Project has strong economics even with a shortened mine life with an after-tax payback between 2 and 3 years, depending on the metal price used.
f) The Project can be started with a much lower initial capital than the Base Case mining scenario and still demonstrates good economic viability.

26.0 RECOMMENDATIONS

The PEA of the Ixtaca deposit indicates its potential as an economically viable mining operation. The Qualified Persons recommend that the Project should proceed to a pre-feasibility study (PFS).

The following activities are recommended to progress the Project forward.

26.1 Geotechnical/Hydrogeological Recommendations

26.1.1 Pit Slope Design

Further development of geomechanical data collection program, which will include four oriented core drillholes along with in-situ hydrogeological testing and instrumentation and laboratory rock strength testing, is suggested. Basic pit geomechanical and hydrogeological models will be developed for pit slope design. Slope stability analyses will be conducted to refine the recommended pit slope configurations and a PFS level Pit Slope Design Report will be compiled. A conceptual pit dewatering plan will be developed under this task. A preliminary assessment of quantities and costs associated with pit dewatering will be provided to support the PFS.

Total engineering cost of the above study is between $100,000 and $150,000, which includes on site drilling supervision and quality assurance, but excludes the costs for drilling and testing.

26.1.2 Tailings, Rock, and Water Management Recommendations

A geotechnical field data program will be carried out for the tailings, waste rock storage and planned infrastructure sites. Geotechnical data collected from the 2014 geotechnical field investigation and laboratory testing programs will be compiled and incorporated into a data report for PFS level designs.

A geochemical static testing program to better assess the tailings and major deposit materials with respect to their neutralising, Acid Rock Drainage (ARD) and Metal Leaching (ML) potential will be carried out. This task will include review of the geological database, selection and preparation of samples for ABA, total metals and SFE testing, evaluation of analytical results, and preparation of letter report on static testing.

PFS level engineering designs will be completed for the preferred TMF and RSF including (but not limited to):

- Confirmation of preferred TMF location
- Perform climate and hydrological studies to provide estimates of surface water flows within the TMF catchment
- Perform a hydrogeological study to estimate the groundwater inflow within the mine site
- Complete geotechnical investigations to support the TMF embankment design
- Development of a TMF design basis and operating strategy for tailings disposal
- Development of a filling schedule to define initial and ongoing staged embankment construction requirements for both tailings impoundments (main TMF and CTMF)
- Confirm the tailings and rock characteristics
- Review of site characteristics data including foundation conditions
- Preliminary layouts and operating requirements for tailings delivery, water reclaim and seepage recycle systems

- Preliminary seepage analyses and considerations of seepage recovery and control systems
- Preliminary design of site water management facilities
- Review of foundation conditions of the RSF site
- Stability analyses for the TMF (static and seismic loading conditions)
- Conceptual closure and reclamation plan for tailings and rock storage facilities

A review of the regional hydrometeorology data to develop preliminary site specific precipitation evaporation and runoff estimates for the Project which will be used in developing water management requirements for the preferred mine development concept.

Production of a Tailings, Waste, and Water Management Report which will detail the results of the design work is needed.

Estimation of the costs and quantities associated with the above waste and water management. Initial capital and sustaining capital/operating costs through the mine life will be estimated and incorporated into the PFS report.

Total engineering cost is estimated between $350,000 and $500,000, which includes field supervision and quality assurance, but excludes the costs for drilling and testing.

26.2 Mining Recommendations

The pit limit, pit phase designs, mining method/equipment, and production schedule will be further optimized and detailed at a design level to support a PFS. These recommendations are not necessarily contingent on positive results from previous phases but reflect the ongoing level of detail required to advance the Project.

Activities involved in updating the mining section include (but are not limited to):

- Further examination of a concentrate as final product option using results from ongoing metallurgical studies
- Updating the ultimate economic pit limits using Measured and Indicated Resources only from the updated block model based on the latest in-fill drilling
- Optimize the pit phase designs and mining equipment based on updated ultimate economic pit limit and updated geotechnical criteria
- Examine potential backfilling strategies to reduce waste haul costs
- Optimize the production schedule through examining various stockpiling scenarios and stockpile locations as well as RSF locations
- Update the capital and operating cost estimates using budgetary quotes from established vendors in the area
- Update the financial model using equipment hour and cost calculations and detailed auxiliary fleet requirements
- Examine the possibility of using the conveyor for hauling waste as well as mill feed
- Examine the possibility of additional deposits being included into the mine plan (pending sufficient drilling results and enough additional resources included in an updated resource report)

Total costs estimated between $300,000 and $400,000 depending on the results of future exploration and geological modeling, pit geotechnical studies, waste and water management studies, mine reclamation and mine closure planning.

26.3 Process Recommendations

Additional metallurgical tests are recommended to assess process recovery variability throughout the Ixtaca Deposit as well as to further test the option of concentrate as a final product. The results of this testwork will be used for process plant engineering design at pre-feasibility level.

Recommended testwork includes:

- Ball Mill Bond Work Index tests on core samples at variable head grade.
- Crushing Bond Work Index tests on core samples at variable head grade.
- Rougher flotation testing on fresh core samples at variable head grade in order to confirm the selection of grind size, residence time, mass pull, and target recovery of metals.
- Leaching of flotation concentrates to improve understanding of leaching response to grind size, lithology, slurry concentration, reagent consumption, and metal recovery.
- If producing a mineral concentrate as final product is considered a valid option, then Ixtaca will need to execute locked-cycle tests to prove that commercial grades are achievable, identify potentially deleterious metals, and support ultimate recovery estimates. It will be also necessary to perform sedimentation and filtration tests on final concentrates.

Above metallurgical testing work and process plant engineering design is estimated to cost between $350,000 and $400,000 to complete.

26.4 Environmental Recommendations

It is also recommended to continue with the long lead environmental baseline studies to support later permitting and feasibility study requirements. The initiation of these studies has been based on positive results from the PEA study.

Total estimated cost is $300,000.

26.5 Infrastructure Recommendations

It is recommended to proceed with infrastructure details to support a PFS. Capital cost estimates for the process plant and infrastructure should be performed.

Total estimated cost is $250,000.

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