



Los Reyes Project

NI 43-101 Technical Report and Preliminary Economic Assessment

Sinaloa and Durango, Mexico

Effective Date: April 25, 2026

Report Date: July 7, 2026

Prepared for

Torex Gold Resources Inc.
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Prepared by

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List of Qualified Persons

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Daniel Ruane, P.Eng., Knight Piésold Ltd
Alex Hemmingsen, P.Eng., Torex Gold Resources Inc
Fernando Cornejo, P.Eng., Torex Gold Resources Inc
Chantal Jolette, P.Geo., Qualitica Consulting Inc
Benjamin Peacock, P.Eng., Knight Piésold Ltd

Certificate of Qualified Person

Peter Mehrfert, P.Eng.

I, Peter Mehrfert, P.Eng., certify that:

- 1) I am employed as a Principal Process Engineer with Ausenco Engineering Canada ULC, (Ausenco), with an office address of 1050 West Pender St., Vancouver, BC, V6E 3S7.
- 2) This certificate applies to the technical report titled *“Los Reyes Project: NI 43-101 Technical Report and Preliminary Economic Assessment, Sinaloa and Durango, Mexico”* (the “Technical Report”), prepared for Torex Gold Resources Inc. (the “Company”), with a report date of July 7, 2026 and an effective date of April 25, 2026 (the “Effective Date”).
- 3) I graduated from the University of British Columbia with a Bachelor of Applied Science in Mining and Mineral Process Engineering in 1996.
- 4) I am a Professional Engineer registered and in good standing with the Engineers and Geoscientists British Columbia (No. 24527).
- 5) I have practiced my profession continuously for 30 years with experience in the design, evaluation and operation of mineral processing facilities during that time. Approximately half of my professional practice has been the supervision and management of metallurgical test work related to feasibility and pre-feasibility studies of projects involving flotation and leaching technologies. Previous projects that I have worked on that have similar features to Los Reyes are: Panuco, Lemhi, Lawyers/Ranch, Springpole, and Spanish Mountain Gold.
- 6) I have read the definition of “Qualified Person” set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for those sections of the Technical Report that I am responsible for preparing.
- 7) I have visited the project site on June 1, 2025.
- 8) I am responsible for Sections 1.1, 1.2, 1.10, 1.13, 1.14, 1.16-1.18, 2.1, 2.2, 2.4.1, 2.5-2.9, 12.5, 13, 17, 18.1, 18.2, 18.6.1-18.6.3, 18.6.6-18.6.8, 18.7-18.9, 19, 21.1, 21.2.1, 21.2.2, 21.2.4, 21.2.5.1, 21.2.5.3, 21.3, 21.4.1, 21.4.2, 21.4.3.2, 21.4.4, 21.4.5, 22, 23, 24, 25.1, 25.5, 25.8-25.10, 25.12, 25.13, 25.14.1.2, 25.14.1.5, 25.14.1.6, 25.14.1.8, 25.14.1.9, 25.14.2.2, 25.14.2.5, 25.14.2.6, 25.14.2.8, 26.4, 26.5, 26.6.1, 27 of the Technical Report.
- 9) I am independent of the Company as independence is defined in Section 1.5 of NI 43-101.
- 10) I have not been previously involved with the Los Reyes Project.
- 11) I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: July 7, 2026

“Signed and sealed”

Peter Mehrfert, P.Eng.

Certificate of Qualified Person

John Sims, CPG

I, John Sims, CPG, certify that:

- 1) I am employed as the President of Sims Resources LLC, (Sims), with an office address of 945 Wyoming St. Unit 214, Missoula, MT, 59801.
- 2) This certificate applies to the technical report titled *“Los Reyes Project: NI 43-101 Technical Report and Preliminary Economic Assessment, Sinaloa and Durango, Mexico”* (the “Technical Report”), prepared for Torex Gold Resources Inc. (the “Company”), with a report date of July 7, 2026 and an effective date of April 25, 2026 (the “Effective Date”).
- 3) I graduated from University of Montana with a Bachelor of Applied Science (B.A.Sc.) in Geology in 1993.
- 4) I am a Certified Professional Geologist registered with the American Institute of Professional Geologists (AIPG) (CPG No. 10924).
- 5) I have practiced my profession continuously for 38 years with experience including:
 - a) 20 years as a site and corporate Qualified Person which includes positions as a Senior Project Mine Geologist, and Director of Technical Services for Coeur d'Alene Mines Corporation, and as Director, VP, and SVP of Technical Services and Brownfield Exploration for Kinross Gold Corporation.
 - b) Experience with respect to mineral resources and reserves includes resource exploration geologist in Chile, Honduras, Mexico, Tanzania and USA; exploration project manager in Nicaragua; mine site project manager and geologist at various underground and open pit mines in western USA, Central and South America; 20 years of resource modelling and reserve optimization experience for deposits in Argentina, Australia, Chile, Bolivia, Ecuador, Ghana, Mauritania, Mexico, Russia, Tanzania and USA.
 - c) Involved in the Los Reyes property since 2023 as an independent consultant and QP.
 - d) Additional work history:
 - i) Senior Vice President, Mineral Resource Geology & Brown Fields Exploration (June 2018 – January 2021)
 - ii) Vice President of Technical Services, Resource Geology and Brownfield Exploration (June 2013 – June 2018)
 - iii) Director of Technical Services Corp Reserves, Kinross Gold Corporation (April 2010 – June 2013)
 - iv) Director of Technical Services, Coeur d'Alene Mines Corporation (March 2005 – April 2010)
 - v) Senior Underground Project Geologist, Coeur d'Alene Mines Corporation (April 2000 – March 2005)
- 6) I have read the definition of “Qualified Person” set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for those sections of the Technical Report that I am responsible for preparing.
- 7) I have visited the project site on October 8, 2024, for a total of 7 days.
- 8) I am responsible for Sections 1.4-1.8, 1.11, 2.4.2, 5-10, 12.1, 14, 25.3, 25.4, 25.6, 25.14.1.1, 25.14.1.3, 25.14.2.1, 25.14.2.4, 26.2, 27 of the Technical Report.
- 9) I am independent of the Company as independence is defined in Section 1.5 of NI 43-101.

- 10) I have been previously involved with the Los Reyes Project as the QP for Prime Mining prior to the Torex Gold Resources Inc. acquisition.
- 11) I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: July 7, 2026

"Signed"

John Sims, CPG

Certificate of Qualified Person

Rodrigo Capel Pasqua, FAusIMM

I, Rodrigo Capel Pasqua, FAusIMM, certify that:

- 1) I am employed as a Principal Mining Consultant with AGP Mining Consulting Inc., (AGP), with an office address of 132 Commerce Park Drive, Unit K #246, Barrie, Ontario L4N 0Z7, Canada.
- 2) This certificate applies to the technical report titled *“Los Reyes Project: NI 43-101 Technical Report and Preliminary Economic Assessment, Sinaloa and Durango, Mexico”* (the “Technical Report”), prepared for Torex Gold Resources Inc. (the “Company”), with a report date of July 7, 2026 and an effective date of April 25, 2026 (the “Effective Date”).
- 3) I graduated from the Escola Politécnica, Universidade de São Paulo (USP), Brazil with a Bachelor of Engineering (Mining Engineering) in 2012.
- 4) I am a Fellow of the Australasian Institute of Mining and Metallurgy (FAusIMM), Member No. 326234, which is a “recognized professional association” as defined in NI 43-101.
- 5) I have practiced my profession continuously for 16 years with experience in underground and open-pit mine design, production scheduling and mining studies for gold and base-metal projects.
- 6) I have read the definition of “Qualified Person” set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for those sections of the Technical Report that I am responsible for preparing.
- 7) I visited the project site from May 31 to June 2, 2025.
- 8) I am responsible for Sections 1.12, 2.4.3, 12.3, 15, 16.1, 16.3-16.12, 18.3, 18.6.3, 18.6.4, 18.6.5, 21.2.3, 21.4.3.1, 25.7, 25.14.1.4, 26.3, 27 of the Technical Report.
- 9) I am independent of the Company as independence is defined in Section 1.5 of NI 43-101.
- 10) I have had prior involvement with the Property that is the subject of this Technical Report. Since April 2025 I have acted as the consulting mining engineer responsible for the underground mine planning, initially for the previous owner, Prime Mining Corp., and subsequently for Torex Gold Resources Inc. following its acquisition of the Property in October 2025, including the mine designs and mining studies that support the mining sections of this Technical Report.
- 11) I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: July 7, 2026

“Signed”

Rodrigo Capel Pasqua, FAusIMM

Certificate of Qualified Person

Daniel Ruane, P.Eng.

I, Daniel Ruane, P.Eng., certify that:

- 1) I am employed as a Specialist Engineer with Knight Piésold Ltd, (KP), with an office address of Suite 1400-750 West Pender Street, Vancouver, British Columbia, V6C 2T8, Canada.
- 2) This certificate applies to the technical report titled *“Los Reyes Project: NI 43-101 Technical Report and Preliminary Economic Assessment, Sinaloa and Durango, Mexico”* (the “Technical Report”), prepared for Torex Gold Resources Inc. (the “Company”), with a report date of July 7, 2026 and an effective date of April 25, 2026 (the “Effective Date”).
- 3) I graduated from the National University of Ireland, Galway with a Bachelor of Engineering in Civil Engineering in 2010 and University of Strathclyde and the University of Glasgow with a Master of Science in Geotechnics in 2011.
- 4) I am a professional engineer registered with the Professional Engineers Ontario (No. 100546532), and Engineers and Geoscientists British Columbia (No. 42894).
- 5) I have practiced my profession continuously for 15 years with experience in geotechnical engineering, mine waste and water management, heap leaching, environmental compliance, mine development with experience in feasibility studies, detailed engineering, permitting, construction, operations and closure.
- 6) I have read the definition of “Qualified Person” set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for those sections of the Technical Report that I am responsible for preparing.
- 7) I visited the project site on January 29, 2024.
- 8) I am responsible for Sections 2.4.4, 18.4, 18.5, 21.2.5.2, 26.6.2, 27 of the Technical Report.
- 9) I am independent of the Company as independence is defined in Section 1.5 of NI 43-101
- 10) I have not been previously involved with the Los Reyes Project.
- 11) I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: July 7, 2026

“Signed and sealed”

Daniel Ruane, P.Eng.

Certificate of Qualified Person

Alexander Hemmingsen, P.Eng.

I, Alexander Hemmingsen, P.Eng., certify that:

- 1) I am employed as a Principal Engineer, Mine Design with Torex Gold Resources Inc. (Torex), with an office address of 130 King Street West, Suite 740, Toronto, Ontario, Canada, M5X 2A2.
- 2) This certificate applies to the technical report titled *“Los Reyes Project: NI 43-101 Technical Report and Preliminary Economic Assessment, Sinaloa and Durango, Mexico”* (the “Technical Report”), prepared for Torex Gold Resources Inc. (the “Company”), with a report date of July 7, 2026 and an effective date of April 25, 2026 (the “Effective Date”).
- 3) I graduated from Queen’s University with a Bachelor of Applied Science (B.A.Sc.) in Mining Engineering in 2007.
- 4) I am a Professional Engineer registered with the Professional Engineers Ontario (No. 100215402).
- 5) I have practiced my profession continuously for 19 years with experience in underground and open pit hard rock mining, with extensive experience in precious metals mining. Experience regarding permitting, water monitoring and mining claims include engineering support at the development of an underground silver mine in Sonora, Mexico; deep-well dewatering and discharge program in Durango, Mexico and the permitting of a mine expansion and waste dump in Guerrero, Mexico.
- 6) I have read the definition of “Qualified Person” set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for those sections of the Technical Report that I am responsible for preparing.
- 7) I have not visited the project site.
- 8) I am responsible for Sections 1.3, 1.15, 3, 4, 20, 25.2, 25.11, 25.14.1.7, 25.14.2.7, 26.7, 27 of the Technical Report.
- 9) I am not independent of the Company as independence is defined in Section 1.5 of NI 43-101. I am responsible for mine design and engineering studies at Torex Gold Resources Inc.
- 10) I have not been previously involved with the Los Reyes Project.
- 11) I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: July 7, 2026

“Signed and sealed”

Alexander Hemmingsen, P.Eng.

Certificate of Qualified Person

Fernando Cornejo, P.Eng.

I, Fernando Cornejo, P. Eng., certify that:

- 1) I am employed as a Vice President of Torex Gold Resources Inc., (Torex), with an office address of 130 King St. West, Suite 740 Toronto, ON, Canada M5X 2A2.
- 2) This certificate applies to the technical report titled *"Los Reyes Project: NI 43-101 Technical Report and Preliminary Economic Assessment, Sinaloa and Durango, Mexico"* (the "Technical Report"), prepared for Torex Gold Resources Inc. (the "Company"), with a report date of July 7, 2026 and an effective date of April 25, 2026 (the "Effective Date").
- 3) I graduated from Ecole Polytechnique de Montreal with a master's degree in chemical engineering in 2005, and from Universidad Nacional de San Agustin with a bachelor's degree in chemical engineering in 2001.
- 4) I am a professional engineer registered with the Professional Engineers Ontario (No. 100170042).
- 5) I have practiced my profession continuously for 25 years with experience in project development, mining operations and technical services.
- 6) I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
- 7) I have not visited the project site.
- 8) I am responsible for Sections 1.19, 2.3 and 26.1 of the Technical Report.
- 9) I am not independent of the Company as independence is defined in Section 1.5 of NI 43-101. I am responsible for all capital projects which includes mining studies and project execution in Torex Gold.
- 10) I have not been previously involved with the Los Reyes Project.
- 11) I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: July 7, 2026

"Signed and sealed"

Fernando Cornejo, P.Eng.

Certificate of Qualified Person

Chantal Jolette, P.Geol.

I, Chantal Jolette, P.Geol., certify that:

- 1) I am employed as a Principal Geologist with Qualitica Consulting Inc., (Qualitica), with an office address of 1300 Kelly Lake Road, Sudbury, Ontario, Canada, P3E 5P4.
- 2) This certificate applies to the technical report titled *“Los Reyes Project: NI 43-101 Technical Report and Preliminary Economic Assessment, Sinaloa and Durango, Mexico”* (the “Technical Report”), prepared for Torex Gold Resources Inc. (the “Company”), with a report date of July 7, 2026 and an effective date of April 25, 2026 (the “Effective Date”).
- 3) I graduated from University of Ottawa with a Bachelor of Science (B.Sc.) in Geology in 2001.
- 4) I am a professional geologist registered with the Geoscientists of Ontario (No. 1518), and with Ordre des géologues du Québec (No. 2214).
- 5) I have practiced my profession as a geologist for 25 years, including more than 20 years specializing in analytical quality assurance and quality control for mineral exploration and mining projects. My experience includes supervision of drilling and sampling programs, geological logging and core sampling, review and auditing of commercial and mine analytical laboratories, development and implementation of QA/QC protocols, evaluation of control sample performance, and validation of analytical data. Since 2017, I have provided consulting services focused on analytical QA/QC, including the review of sample preparation and analytical procedures and the assessment of laboratory performance. This experience qualifies me to take responsibility for the sample preparation, analytical methods, and QA/QC sections of this report. I have reviewed the quality assurance and quality control procedures, as well as the results of the control samples, for the 2021–2024 drilling at the Los Reyes Project.
- 6) I have read the definition of “Qualified Person” set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for those sections of the Technical Report that I am responsible for preparing.
- 7) I have not visited the project site.
- 8) I am responsible for Sections 1.9, 11, 12.2, 27 of the Technical Report.
- 9) I am independent of the Company as independence is defined in Section 1.5 of NI 43-101.
- 10) I have been previously involved with the Los Reyes Project. I conducted an annual review of analytical quality control and procedures in 2022. I was the Qualified Person for Section 11 for the 2023 and 2024 Technical Reports on behalf of Prime Mining.
- 11) I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: July 7, 2026

“Signed and sealed”

Chantal Jolette, P.Geol.

Certificate of Qualified Person

Benjamin Peacock, P.Eng.

I, Benjamin Peacock, P.Eng., certify that:

- 1) I am employed as a Specialist Engineer with Knight Piésold Ltd., (KP), with an office address of 200 – 1164 Devonshire Avenue, North Bay, Ontario, P1B 6X7, Canada.
- 2) This certificate applies to the technical report titled *“Los Reyes Project: NI 43-101 Technical Report and Preliminary Economic Assessment, Sinaloa and Durango, Mexico”* (the “Technical Report”), prepared for Torex Gold Resources Inc. (the “Company”), with a report date of July 7, 2026 and an effective date of April 25, 2026 (the “Effective Date”).
- 3) I graduated from the University of Waterloo with a Bachelor of Applied Science in Civil Engineering in June 2008.
- 4) I am a professional engineer registered with the Professional Engineers Ontario (No. 100141409), and NWT and Nunavut Association of Professional Engineers and Geoscientists (No. L3788) and with Professional Engineers and Geoscientists Newfoundland and Labrador (No. 09001).
- 5) I have practiced my profession continuously for 18 years with experience in both open pit and underground mining rock mechanics at operating mines and projects worldwide, including Mexico.
- 6) I have read the definition of “Qualified Person” set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for those sections of the Technical Report that I am responsible for preparing.
- 7) I have not visited the project site.
- 8) I am responsible for Sections 12.4, 16.2, 27 of the Technical Report.
- 9) I am independent of the Company as independence is defined in Section 1.5 of NI 43-101.
- 10) I have not been previously involved with the Los Reyes Project.
- 11) I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: July 7, 2026

“Signed and sealed”

Benjamin Peacock, P.Eng.

Important notice

This technical report was prepared in accordance with National Instrument 43-101 – Standards of Disclosure for Mineral Projects for Torex Gold Resources Inc (Torex) by Qualified Persons listed herein from Ausenco Engineering Canada ULC (Ausenco), Sims Resources LLC (Sims), AGP Mining Consulting, Inc. (AGP), Knight Piésold Ltd. (KP), and Qualitica Consulting, Inc (Qualitica), collectively the Report Authors. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the Report Authors' services, based on i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. Except for the purposes legislated under Canadian provincial and territorial securities law, any other uses of this report by any third party is at that party's sole risk.

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1 Summary

1.1 Introduction

Ausenco Engineering Canada ULC (Ausenco) has prepared a Preliminary Economic Assessment (PEA) and associated technical report for (Torex) on the Los Reyes Project located in Sinaloa and Durango, Mexico. The Technical Report was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101) and Form 43-101 F1.

The responsibilities of the engineering consultants are as follows:

- Ausenco was responsible for managing and coordinating the work related to the technical report and developed PEA-level design, including capital and operating cost estimates for the process plant, general site infrastructure, tailings storage facility (TSF) design and site-wide water management measures.
- AGP Mining Consultants Inc. (AGP) was responsible for designing the open-pit and underground mine, mine production schedule, mining fleet and mining capital and operating costs. AGP also developed the mineral resource shapes for the project.
- Sims Resources LLC (Sims) was responsible for completing the work related to the geological setting, deposit type, drilling, explorations work, sample preparation and analysis and data verification.
- Knight Piésold Ltd. (KP) was responsible for designing and providing capital and operating costs for the tailings and water management facilities for the project, they were also responsible for the geotechnical assessment and design.
- Qualitica Consulting Inc (Qualitica) was responsible for the review of the QA/QC for diamond drilling programs completed by Prime during their ownership.

1.2 Terms of reference

Torex is using this Technical Report to support disclosure of a PEA level study of the Project, with an updated MRE. The Technical Report conforms to NI 43-101. The effective date of this report is April 25, 2026. There were no material changes to the project between this date and the publication of this accompanying NI 43-101 Technical Report.

All measure units used in this Technical Report are metric unless otherwise noted. Currency is expressed in United States dollars (US\$ or USD) unless otherwise noted. This technical report uses English.

Mineral resources and mineral reserves are reported in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (CIM, 2019).

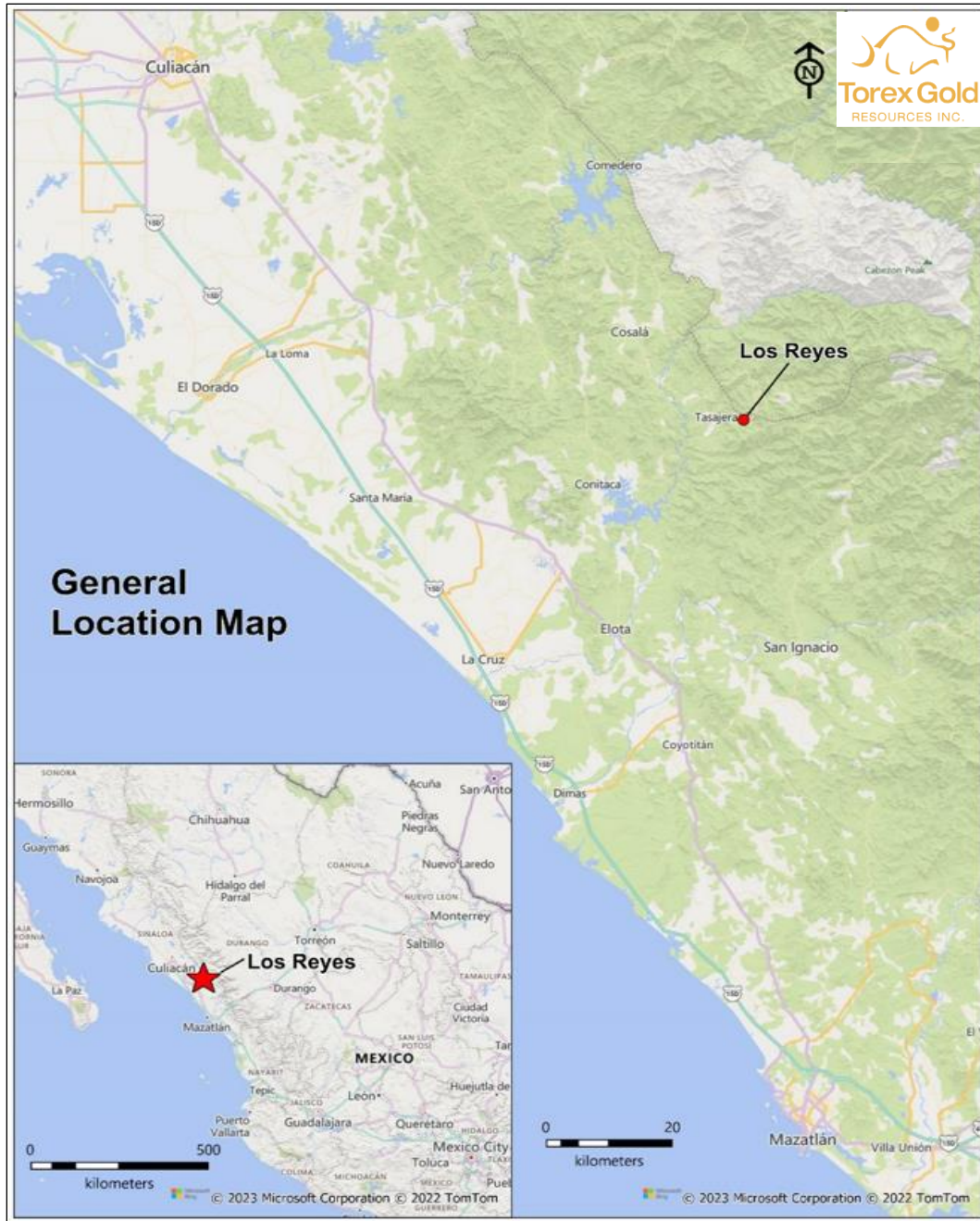
Readers are cautioned that the PEA is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized.

1.3 Project description

The Property is located north of the coastal city of Mazatlán, Sinaloa, approximately 110 km by air and 200 km by paved highway (Figure 1-1). The Property is within the municipality of Cosalá (population 17,012)

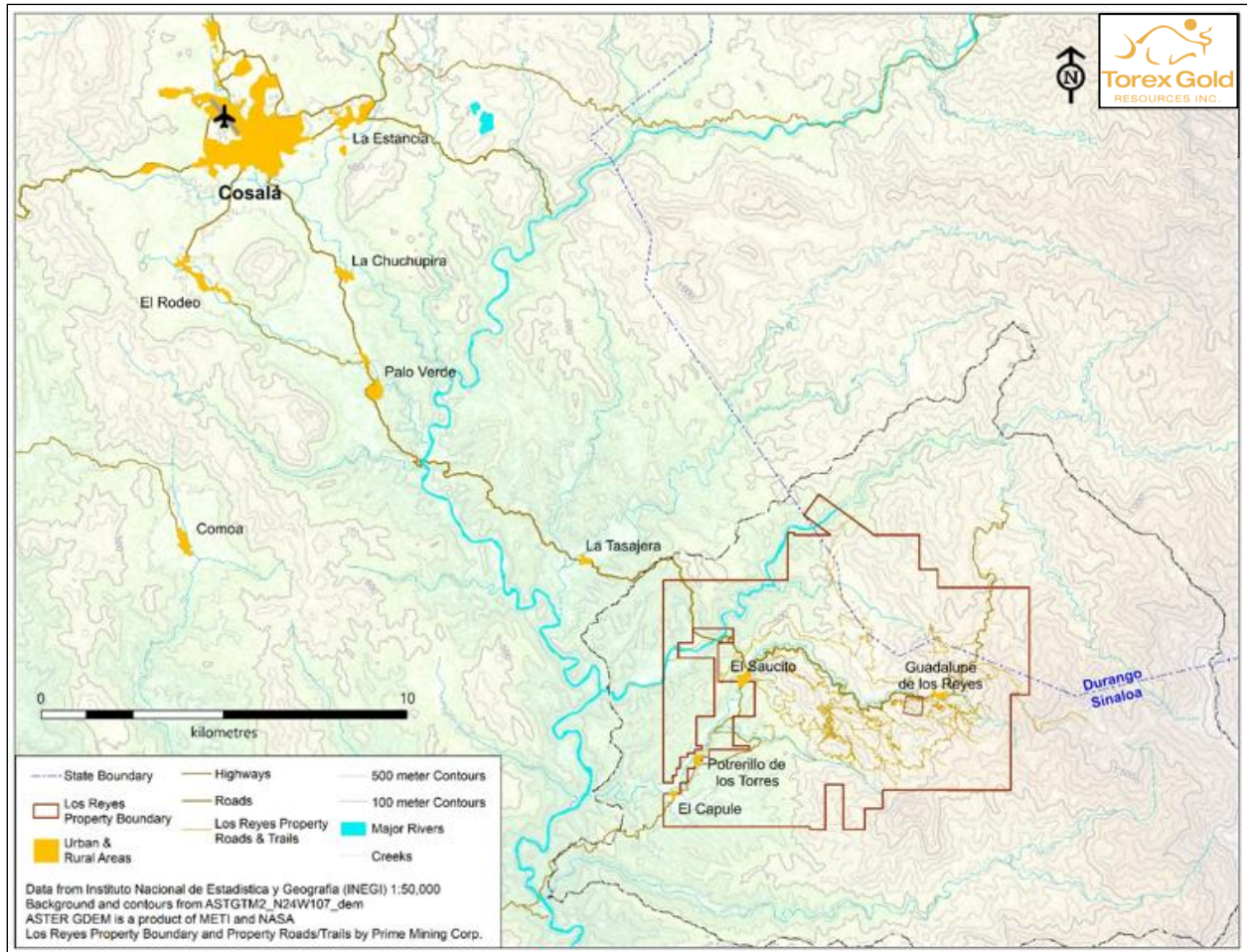
and the closest city to the Property is Cosalá (population 7,888, INEGI 2020) which is located 27 km to the northwest of the proposed plant site location and connected by a gravel road. En route to the Property from Cosalá are the villages of Palo Verde and La Tasajera. The village of Guadalupe de Los Reyes is on the Property and was the site of Spanish colonial mining (Figure 1-2). The general geographic coordinates of the Property are N-24°17' and W-106°32' (UTM Zone 13 North 344250E, 2686400N). Coordinates are in WGS 84.

Figure 1-1: General location map



Source: Microsoft and modified by Torex, 2023

Figure 1-2: Property location map

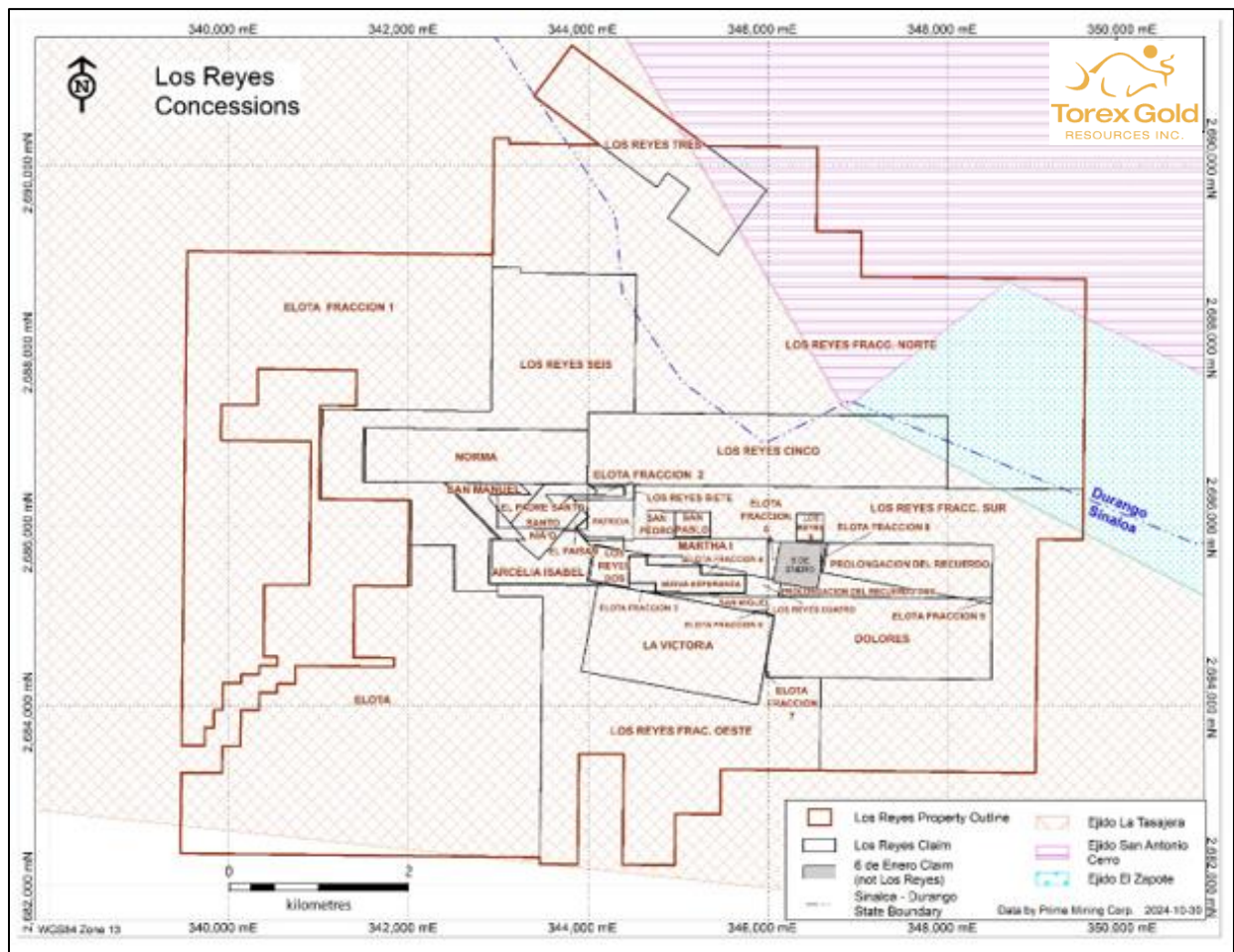


Source: Torex, 2024

1.3.1 Ownership and History

On October 22, 2025, Torex Gold Resources Inc. acquired all issued and outstanding shares of Prime Mining Corp., and its wholly owned subsidiary Minera Amari, resulting in the indirect ownership of 100% of the Los Reyes project, subject to various royalties and/or NSRs (as defined below). The Project is comprised of 37 contiguous mining concessions that have a combined area of 6,273 hectares as of the effective date of this report (Figure 1-3).

Figure 1-3: Los Reyes Concessions



Source: Torex, 2024

Several previous owners completed surface mapping, surface sampling, drilling, and various study work on the Property. It is believed that mining in the Project area dates back to the 18th century, if not earlier.

1.4 Accessibility, climate, local resources, infrastructure and physiography

The Property is located within the Sierra Madre Occidental mountain range in Sinaloa, Mexico, and is characterized by variable topography ranging from steep mountainous terrain in the east to river valleys in the west, with elevations between approximately 230 and 1,400 metres above sea level (masl). Vegetation varies from tropical shrubs at lower elevations to evergreen cover at higher elevations, with surrounding areas partially developed for agriculture. The Property is situated approximately 200 km by road from Mazatlán and 27 km southeast of the town of Cosalá and is accessible by an all-weather road. The climate is classified as tropical wet and dry, with distinct seasonal rainfall concentrated in the summer months, which may result in localized flooding and temporary access constraints.

The Property benefits from established regional infrastructure, including access to multiple international airports within several hours of travel, a regional airport in Cosalá, and a maintained road network providing primary access. Local communities provide basic services and may support part of the workforce. Electrical power is available in nearby communities, and water is seasonally available from local sources; however, additional water supply development would be required to support mining operations. Surface rights agreements with local ejidos are reported to be in place and are considered sufficient to support project development.

1.5 History

Exploration and mining on the Property date back to the initial discovery of gold and silver at Guadalupe de Los Reyes in approximately 1772, with historical underground mining continuing through the 19th and early 20th centuries and smaller-scale operations persisting into the late 20th century. In total, historical production is estimated at approximately 1 Moz of gold and 60 Moz of silver from about 2.7 Mt of material, derived from multiple vein systems and mining areas across the Property.

Prior to the current ownership, the Property has been explored by several companies, including Luismin, Northern Crown Mine (NCM), Vista Gold, and Great Panther, and more recently Prime. These operators conducted exploration programs from approximately the early 1990s through to 2024, generating a substantial historical dataset that has since been validated and incorporated into the current database. Prime was acquired by Torex in October 2025.

Historical work programs included geological mapping, soil and rock geochemical sampling, limited geophysical surveys (including VLF electromagnetics and ground magnetics), and drilling campaigns, particularly by NCM, Vista Gold, Great Panther, and Meridian. These programs identified numerous mineralised zones across the Property and defined multiple deposits and targets, supported by extensive sampling and drilling datasets that underpin the current mineral resource evaluation.

1.6 Geology and mineralisation

The Los Reyes Property is situated within the Sierra Madre Occidental of the North American Cordillera, where a late Cretaceous to middle Tertiary volcanic sequence unconformably overlies a late Cretaceous batholith. On the Property, the lower volcanic sequence is approximately 1 km thick and is dominated by intermediate volcanic rocks with dacitic to rhyolitic units, while the upper sequence comprises younger rhyolitic to dacitic ash-flow and ash-fall tuffs preserved on higher topography. Several generations of felsic dykes cut the lower volcanic sequence and are interpreted to be temporally associated with mineralisation, with gold and silver bearing quartz veins commonly linked to the earlier rhyolitic to dacitic intrusions.

Gold and silver mineralisation is hosted predominantly within three principal silicified structural corridors: the Mariposa-Zapote-Tahonitas trend, the San Miguel-Noche Buena Central trend, and the Guadalupe trend, together with several subordinate structures such as Las Primas and Fresnillo. Mineralisation is characteristic of a low-sulphidation epithermal system and occurs as silicified host rock, hydrothermal and stockwork breccias, and quartz vein networks displaying saccharoidal, drusy, crustiform, colloform, and crystalline textures. The mineralised zones are developed along northwest- to west-northwest-oriented structures, commonly with strong structural and lithological control near intrusive contacts, and individual zones range from several hundred metres to more than 2.5 km in strike length with variable true widths from a few metres to locally more than 60 m. Historical underground workings and surface exposures across several trends support the continuity of these mineralised corridors and confirm the district-scale nature of the epithermal system.

1.7 Exploration

Geological mapping initiated in late 2020 now covers more than 60 km² and has identified several high-potential target areas, including Tahonitas, Las Primas, Fresnillo, (Zapote West), and the Orito trend, as well as additional mineralised structures north of San Miguel East and Guadalupe East. Surface programs have included approximately 5,000 m of trench and road-cut sampling, 9,164 total rock samples, and 4,251 soil samples plus 157 duplicates. Geochemical results show strong Au-Ag-As-Sb associations in the Z-T and Central zones and elevated Au, Ag, and Cu at Guadalupe East. Airborne magnetic-radiometric and VLF-EM surveys completed in 2021 highlighted structural trends and radiometric anomalies that correlate with known mineralisation and additional targets such as Zapote 300, Orito, La Verde/El Tule, and Las Palmitas. Spectral mineralogy, magnetic susceptibility, and remote sensing data further support a low-sulphidation epithermal Au-Ag system and continue to refine vectors toward mineralisation.

Work completed since 2019 includes grid control and surveys, geological mapping, rock and soil geochemistry, airborne geophysics, spectral mineralogy, magnetic susceptibility studies, and remote sensing. The Project is referenced to WGS84 UTM Zone 13 North and is tied to INEGI control. LiDAR data were collected in 2021 over 1,050 line-kilometres at 50 m spacing, and additional underground LiDAR surveys were completed in 2022 and 2024 for more than 45 adits, shafts, and workings, with results incorporated into the geological and resource model.

Exploration on the Property has been completed by Prime and previous owners; Torex has initiated a drilling program in 2026.

1.8 Drilling

The Project has been drilled extensively by multiple operators since 1993, with historical programs completed by NCM, Meridian Gold, Vista Gold, Great Panther Silver Limited, and, most recently, Prime Mining Corporation (Prime). Historical drilling from 1993 to 2015 totaled 493 holes for 51,733 m, while Prime's 2020 to 2024 campaigns added 744 holes for 218,819.4 m, substantially expanding the drilling database across the Guadalupe, Zapote-Tahonitas, Central, and other target areas. Drilling methods included both reverse circulation and diamond core, with modern programs incorporating downhole surveys, oriented core in selected holes, systematic geological logging, secure sample handling, and density measurements. Based on the Independent QP's review, the quantity and quality of lithological, geotechnical, collar, and downhole survey data are sufficient to support mineral resource and mineral reserve estimation, with no apparent drilling, sampling, or recovery issues that would materially affect reliability. Following the acquisition of Prime, Torex has initiated data validation and relogging activities, they have also initiated a drilling program in 2026.

1.9 Sample preparation, analyses, and security

Since acquiring Prime in 2025, Torex has not conducted additional drilling, sample preparation, or analytical work. Historical samples collected by Prime were processed at ALS Global and Bureau Veritas laboratories in Mexico using standard industry preparation methods, including crushing, splitting, pulverizing, fire assay for gold, and multi-element analysis by four-acid digestion with ICP finish. Sample handling and security procedures were documented from collection at the drill rig through transport, logging, storage, and laboratory submission, with chain-of-custody controls in place. Quality assurance and quality control measures included the regular insertion of blanks, certified reference materials, coarse duplicates, pulp duplicates, and check assays at secondary laboratories. Results from both ALS and Bureau Veritas indicate

acceptable performance for contamination control, analytical accuracy, and precision, with no material concerns identified that would compromise the reliability of the assay database used to support the project.

1.10 Metallurgical testwork

Five metallurgical test programs have been conducted on the Los Reyes property. Column leach tests were conducted in 1998, subsequent test programs in 2012 and 2023-2025 were conducted at RDI, Forte Analytical and Kappes, Cassiday & Associates (KCA). The more recent test programs evaluated gravity concentration, froth flotation and bottle roll cyanide leach testing, as well as some comminution testing. The total number of metallurgical samples included 25 composites and 47 variability samples and were obtained from 6 of the main deposits in the project.

The mineralogy of the material is understood to be a quartz hosted vein mineralisation with low sulphur content. Gold occurs as native gold or electrum, and silver is primarily present as argentite. Gold grains are described as being quite fine, sub 20µm, whereas argentite grains range in size from 2µm to 250µm.

A limited amount of comminution testing was completed in the test programs, 4 samples for SMC testing and 8 samples for Bond ball mill work index tests. Drop weight SMC tests returned Axb values averaging 38, indicating that the material is of moderate competence with respect to impact breakage in a SAG mill. Bond ball mill Wi values averaged 17.7 kWh/t, and ranged from 15.8 to 19.2 kWh/t.

The material showed a modest response to gravity concentration, averaging 23% gold and 10% silver recovery on Knelson plus Gemini table upgrading tests conducted on samples ground to 80% passing 150µm.

Froth flotation response was varied, tests on Guadalupe East samples returned gold and silver recoveries to rougher concentrates as high as 90-95% while Tahonitas and Zapote tests recovered only 85% and 70% or lower of the feed gold and silver, respectively, to similar concentrates. Froth flotation can be investigated further in conjunction with cyanide leaching to reduce grinding energy requirements, but a flotation only processing method yields lower gold and silver recoveries compared to cyanide leaching for the group of deposits.

A limited amount of column leach testing was completed on samples from a portion of the deposits, indicating that the materials are amenable to heap leaching for gold, however silver extractions were quite low. A total of 7 samples were tested at a crush size of 12.5mm, results averaged 67% gold and 17% silver extractions from the average feed grades of 1.5 g/t Au and 23 g/t Ag.

The materials responded well to cyanide bottle roll leach tests, which indicated that levels of gold and silver extraction were sensitive to primary grind size. Precious metal extraction tended to level off at grind sizes around 100µm P₈₀, and a grind size of 75µm P₈₀ was determined to be most economical for treating the series of deposits. High levels of gold extraction, approximately 95%, were consistently achieved across all the deposits within 48 hours at this primary grind size. Silver kinetics were slower, requiring generally 72 hours of residence time and higher NaCN dosages to achieve maximum extractions. High silver extractions in the range of 90-95% were measured on samples from the Guadalupe and Noche Buena deposits, whereas lower extractions in the 60-80% range were measured on San Miguel and Tahonitas deposits. The Zapote samples consistently returned silver extractions in the 40% range, despite longer leaching times or increased sodium cyanide tenors. The reason for this current limitation is not entirely clear.

Extraction models as a function of feed grades were developed from the collection of metallurgical data for each deposit. Life of mine grades for each deposit were applied to the models and weighted by contained

ounces in each to obtain a project wide LOM recovery estimate of 95.8% for gold and 82.1% for silver. Plant design includes a 1% deduction on these recoveries to account for solution losses.

1.11 Mineral resource estimate

The Los Reyes resource model was prepared under the supervision of Sims Resources LLC (Independent QP). Geologic and estimation domains were constructed using Leapfrog Geo v.2024.1.1, including input from geochemical analyses completed in ioGAS v.8.0. Geostatistical evaluations and Exploratory Data Analysis (“EDA”), including topcut selection, declustering, variography, and Sequential Gaussian Simulation (“SGS”) were completed using Snowden Supervisor v.8.15. Resource estimation was prepared using Leapfrog EDGE v.2024.1.1.

Two sets of block models were prepared for this resource estimate - (1) 5x5x5m block models were generated for use in Open Pit optimization, and (2) 2.5x2.5x2.5m block models were generated for use in underground Mineable Shape Optimization. The 5x5x5m and 2.5x2.5x2.5m models are estimated independently, and employ distinct compositing, domaining, and estimation strategies which are suitable for their respective mining methods and Selective Mining Units (SMU). Gold and silver grades were interpolated into both sets of models using inverse distance cubed (“ID3”) estimation techniques. Search ellipse orientation and radii were selected based on variogram models for each individual estimation domain, with variable search orientation applied according to the nearest vein midpoint surface in the quartz vein and breccia model. Blocks were classified under the categories of “Indicated” and “Inferred” mineral resources, in accordance with the 2014 Canadian Institute of Mining, Metallurgy and Petroleum Standards for Mineral Resources and Mineral Reserves, Definitions and Guidelines, May 2014 (the “CIM Definition Standards”). The “Measured” resource category was not used in either model because no modern mining has been undertaken at the Project and it is therefore not possible to reconcile the models against production or tightly spaced data such as grade control drilling.

Mineral resources (“Mineral Resources”) were reported below the most recent light detection and ranging (“LiDAR”) topographic surface and are contained within economically constrained pit shells generated using the Hochbaum Pseudoflow algorithm implemented in Datamine’s Studio NPVS or underground stope shapes generated using Datamine’s Mineable Shape Optimizer (“MSO”). Open pit Mineral Resources are reported estimated using a 0.17 g/t Au-only cutoff grade, and underground Mineral Resources are reported from economic longhole open stopes generated using Datamine’s Mineable Shape Optimizer (MSO). The Mineral Resources are classified as Indicated or Inferred based on drill spacing and geological continuity. Two processing methodologies were assumed: a plant to process the higher-grade blocks, and a heap leach. Mineral resources that are not mineral reserves do not have demonstrated economic viability. The Los Reyes Mineral Resource Statement is summarized in Table 1-1.

Table 1-1: Mineral Resource Statement

Mining Method and Process	Class	Tonnage (kt)	Au (g/t)	Au (koz)	Ag (g/t)	Ag (koz)	AuEq (g/t)	AuEq (koz)
Open pit plant	Measured	-	-	-	-	-	-	-
	Indicated	26,448	1.13	961	30.8	26,192	1.47	1,251
	Measured and Indicated	26,448	1.13	961	30.8	26,192	1.47	1,251
	Inferred	8,056	0.93	240	34.4	8,917	1.31	339
Underground plant	Measured	-	-	-	-	-	-	-
	Indicated	5,361	2.48	428	129.0	22,231	3.91	674
	Measured and Indicated	5,361	2.48	428	129.0	22,231	3.91	674
	Inferred	5,430	1.84	322	72.7	12,685	2.65	463
Total Plant	Measured	-	-	-	-	-	-	-
	Indicated	31,809	1.36	1,388	47.3	48,423	1.88	1,925
	Measured and Indicated	31,809	1.36	1,388	47.3	48,423	1.88	1,925
	Inferred	13,487	1.30	562	49.8	21,602	1.85	801
Open Pit Heap Leach	Measured	-	-	-	-	-	-	-
	Indicated	19,441	0.25	154	11.9	7,437	0.30	187
	Measured and Indicated	19,441	0.25	154	11.9	7,437	0.30	187
	Inferred	7,413	0.24	57	12.6	3,013	0.30	71
Total	Measured	-	-	-	-	-	-	-
	Indicated	51,250	0.94	1,542	33.9	55,860	1.28	2,111
	Measured and Indicated	51,250	0.94	1,542	33.9	55,860	1.28	2,111
	Inferred	20,899	0.92	620	36.6	24,615	1.30	872

Notes:

1. Mineral resources were prepared in accordance with the CIM Definition Standards (2014) and Estimation of Mineral Resource and Mineral Reserve Best Practice guidelines (2019).
2. Mineral resources are inclusive of mineral reserves. Mineral resources that are not mineral reserves do not have demonstrated economic viability.
3. Numbers may not add due to rounding.
4. Mineral resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.
5. The effective date of the estimate is April 25, 2026
6. Mineral resources for Los Reyes are based on underlying metal prices of \$1,950/oz Au and \$25.24/oz Ag, unless otherwise noted.
7. The preparation of the mineral resource estimate was supervised by John Sims, President of Sims Resources LLC, an independent contractor and QP as a CPG member with AIPG.
8. Los Reyes – Open Pit (Plant and Heap Leach):
 - a. Open Pit Resource estimates are based on economically constrained revenue factor one (RF1) open pits generated using the Hochbaum Pseudoflow algorithm in Datamine's Studio NPVS and the following optimization parameters:
 - b. Assumes plant recoveries of 94.8% for Au and 81.1% for Ag and heap leach recoveries of 73% Au and 25% Ag.
 - c. Pit slopes by area ranging from 42-47 degrees overall slope angle.
 - d. 5% mineralized material loss and 5% dilution factor applied to the 5 m x 5 m x 5 m open pit resource block models.

- e. Mining costs of \$2.00 per tonne of waste mined and \$2.50 per tonne of mineralized material mined. Processing costs of \$16.81 per tonne processed. Heap leach costs of \$5.53 per tonne processed. G&A cost of \$2.00 per tonne of material processed.
 - f. A 0.17 g/t gold only cutoff was applied to ex-pit processed material (which is above the heap-leaching NSR cutoff).
 - g. $\text{AuEq (Open Pit Plant)} = \text{Au (g/t)} + (\text{Ag (g/t)} * 0.0111)$ and $\text{AuEq (Open Pit Heap Leach)} = \text{Au (g/t)} + (\text{Ag (g/t)} * 0.0044)$.
9. Los Reyes – Underground (Plant):
- a. Underground Resource estimates are based on economically constrained stopes generated using Datamine's Mineable Shape Optimizer (MSO) algorithm and the following optimization parameters:
 - b. Diluted to a minimum 3 m stope width with a 95% mining recovery.
 - c. Assumes plant recoveries of 94.8% for Au and 81.1% for Ag.
 - d. Longhole Open Stope mining with a \$50.00 per tonne cost. Processing costs of \$17.24 per tonne processed. G&A cost of \$4.50 per tonne of material processed.
 - e. $\text{AuEq (Underground Plant)} = \text{Au (g/t)} + (\text{Ag (g/t)} * 0.0111)$.

1.12 Mining methods

The Los Reyes Project is a combined underground and open-pit mining operation feeding a single 5,000 t/d (1.83 Mt/a) processing plant over a life of mine of approximately 15 years. The PEA mining inventory totals 26.1 Mt at 1.68g/t Au (1,411 koz Au) and 63.4g/t Ag (53,122 koz Ag), comprising 14.8 Mt of underground mineralised material at 2.12g/t Au (1,011 koz Au) and 92.00g/t Ag (43,842 koz Ag) and 11.3 Mt of open-pit mineralised material at 1.11g/t Au (400 koz Au) and 25.7g/t Ag (9,280 koz Ag); approximately 37% by tonnage (~39% by contained AuEq ounces) is derived from Inferred Mineral Resources. The underground deposits are mined by long-hole open stoping with paste backfill; the open-pit complexes are mined by conventional drill-and-blast, load-and-haul methods at an average strip ratio of 6.8:1. A 60-tonne articulated-truck haulage fleet is standardised across the underground and open-pit operations, with central-valley open-pit material routed through the underground haulage network to the plant. The mine is owner-operated, with the mobile fleet and selected fixed equipment leased. Mining-specific capital is estimated at US\$513 million of direct cost (US\$98M initial, US\$415 million sustaining), and life-of-mine mining operating cost at US\$1,122 million, equivalent to US\$43/t of material processed (underground US\$57/t of mineralised material; open-pit US\$4.3/t of total material moved). Cut-off grades of 1.50 g/t AuEq (underground) and 0.37 g/t AuEq (open pit) were applied. The operation reaches a peak workforce of approximately 780 personnel and a peak mobile fleet of approximately 15 haulage trucks.

Readers are cautioned that the PEA is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

The four underground deposits are mined predominantly by longitudinal long-hole open stoping — transverse only at Guadalupe East where the mineralised width exceeds about 20 m — at full extraction with cemented paste backfill. The rock mass is poor to fair (RMR89 of about 25 to 70), with the Estaca fault the principal control on dilution and sequencing; deposit-wide PEA modifying factors are about 14% underground dilution and a flat 5% open-pit dilution, at 95% mining recovery on stoped material and 100% on development. The schedule holds the plant at nameplate from Year 3 to Year 11, with open-pit material and low-grade stockpile reclaim carrying the early and tail years until Year 15. The mobile fleet is leased, so equipment cost sits in operating cost, and mining capital is dominated by capitalised development and fixed infrastructure. The PEA mining inventory by classification is given in Table 1-2 and the integrated mine layout in Figure 1-4.

Table 1-2: PEA mining inventory by Mineral Resource classification

Classification	Tonnes (Mt)	AuEq (g/t)	Au (g/t)	Ag (g/t)	AuEq (koz)	Au (koz)	Ag (koz)
Measured	–	–	–	–	–	–	–
Indicated	16.5	2.37	1.62	59.5	1,254	861	31,545
Measured + Indicated	16.5	2.37	1.62	59.5	1,254	861	31,545
Inferred	9.6	2.55	1.79	70.0	786	551	21,577
Total mining inventory	26.1	2.43	1.68	63.4	2,040	1,411	53,122

Note: AuEq is calculated based on Au Price of \$1950/oz, Ag Price of \$25.24/oz and recoveries of 94.5% Au and 78.5% Ag, reported out inclusive of modifying factors on a per-task basis in the schedule.

1.13 Recovery methods

The process design is based on mineralised material from the Los Reyes deposit zones. Material is processed by crushing, grinding, whole-ore cyanide leaching, counter-current decantation, and precious metals recovery via the Merrill Crowe process. Tailings materials are detoxified using the SO₂/O₂ cyanide destruction process, thickened, and filtered before being dry-stacked on a filtered tailings storage facility. When cemented paste is required for backfilling underground, detoxified tailings are diverted to the paste plant near the underground mine portal and cemented paste is prepared before pumping to the underground reticulation system.

The design is based on completed test work programs performed on the deposit, Ausenco's database of reference projects, and in-house process modelling. The process plant has been designed with assumed availabilities of 65% for the crushing plant, 85% for tailings filtration, 65% for the paste plant, and 92% for all other processing circuits, based on industry-proven industry values. The plant will operate with two 12-hour shifts per day, 365 days per year.

The process design consists of the following major components:

- Three-stage crushing of run of mine (ROM) material
- Ball milling in closed circuit with a classifying cyclone cluster
- Whole-ore leaching of the cyclone overflow
- Counter-current decantation (CCD)
- Merrill-Crowe circuit
- Retorting and smelting to produce doré
- Tailings cyanide detoxification
- Tailings thickening and filtering
- Paste backfill plant.

A simplified process flowsheet diagram for the project is shown in Figure 1-4.



1.14 Project infrastructure

There are two key accesses to the site. The main site entry is in the north-west of the site and is connected to the township of Cosalá by a 36 km main road. It is proposed to upgrade this road in the initial stages of the project. This access is close to the proposed process plant and site camp facilities. The other site access is on the eastern side of the project and is connected by an existing public road that supports the nearby community of Guadalupe. This access is close to the proposed paste plant and Guadalupe East underground portal.

Infrastructure at the Los Reyes Project includes on-site infrastructure such as earthworks development, TSF and buildings, on-site roads, water management systems, and site electrical power facilities. Off-site infrastructure includes site access roads, raw water supply, power supply, and piping. The project infrastructure will include:

- mine facilities, including truck shop & wash bay and mine office, explosive storage facilities, warehouse, paste backfill facilities, waste rock storage and low-grade stockpiles, ROM (run-of-mine) pad, pits and Underground portals
- bulk earthwork required to develop the process plant pad, paste plant pad, interconnecting roads and additional laydown areas to facilitate temporary facilities and storage
- infrastructure for power distribution
- fuel storage for mine, process, and support fleets
- sewage and effluent treatment for site
- offices, workshops, warehouses, security buildings, accommodations as well as additional support buildings
- surface mobile fleet for non-mining site crew and staff
- process plant including coarse crushed materials storage, grinding and classification, leaching and gold processing, detox, filtering, storing and handling, tailings thickening, a reagent mixing and distributing facility, assay laboratory and process plant workshop and warehouse
- tailings management including filtration, haulage and a TSF
- water management and balance for the entire site.

The overall site layout was developed using the following criteria and factors:

- The processing facilities described above must be located to minimize mine haul costs, initial capital costs (earthworks, roads, power, water), environmental and community impacts.
- The TSF should be located as close to the process plant as possible, taking advantage of the sloped natural terrain, down-gradient from tailings filtering facilities, and surface runoff managed accordingly.
- The arrangement of the administration buildings, mine workshops, processing plant, and additional offices should be optimized for foot and vehicle traffic.
- The project is located at a high altitude with very steep terrain. Consideration has been given to these conditions when locating buildings and access roads.

1.15 Environmental, permitting and social considerations

1.15.1 Environmental considerations

The Project area is defined by sedimentary and extrusive igneous geology, warm to semi-warm sub-humid climatic conditions, and distinct seasonal precipitation patterns typical of the region. Surface hydrology is associated with the Habitas River basin. Underground hydrology is associated with the Elota River.

The Project area is set in a mountainous region and supports a diverse biological environment, including numerous flora and fauna species. Cultural heritage considerations are also present, notably the town of Guadalupe de Los Reyes.

1.15.2 Permitting considerations

Current Project activities are limited to exploration, and permitting to date reflects this stage of development. An Environmental Impact Statement (Manifestación de Impacto Ambiental, MIA) covering exploration activities was submitted to SEMARNAT on September 30, 2025, and subsequently approved on December 9, 2025. Advancement of the Project beyond exploration will require additional environmental and operational permits in accordance with applicable Mexican regulatory frameworks.

1.15.3 Social considerations

The Project is located in the municipality of Cosalá, Sinaloa, and is situated across lands held by three ejidos, with Ejido La Tasajera representing the majority of the Project area. Agreements have been established with local ejido communities to secure land access and support ongoing activities, including a 15-year renewable agreement with Ejido La Tasajera and a similar surface rights agreement with Ejido San Antonio. These agreements form the basis of the Project's social license and enable continued exploration and future development activities.

1.15.4 Closure and reclamation considerations

Closure planning for the Project is currently at a conceptual level consistent with a PEA study and is based on standard industry practices and applicable Mexican regulatory requirements. The closure strategy is designed to return the site to a stable condition, with post-closure land uses anticipated to be compatible with surrounding environments and communities. Key closure activities are expected to include decommissioning and removal of infrastructure, stabilization and recontouring of disturbed areas, rehabilitation of mine-related facilities, and revegetation using native species where practicable.

A conceptual closure cost estimate of approximately US\$35 million has been developed, with the largest cost components associated with rehabilitation of waste rock and tailings storage facilities, followed by plant decommissioning and site restoration.

1.16 Markets and contracts

Project economics were estimated based on long-term metal prices of US\$3600/oz Au, and US\$50/oz Ag, which was based on consensus forecasts from various financial institutions.

The Company does not currently have contracts for the gold and silver doré production at Los Reyes. The company will leverage its experience and existing relationships in the gold doré market to enter into future contracts for the sale of gold and silver doré. We expect the terms in the contract to be market-based terms.

1.17 Capital and operating cost estimates

1.17.1 Capital Cost Estimate

The capital cost estimate was developed in Q2 2026 to Class 5 estimate in accordance with the Association for the Advancement of Cost Engineering International (AACE international) for an accuracy of $\pm 50\%$. The estimate was developed with Ausenco's in-house database of projects and studies, including processing, onsite infrastructure, offsite infrastructure, project indirect costs, project delivery, owner's costs and provisions. Mining costs was built up by AGP Mining, tailings and water management was developed by Knight Piesold.

A summary of the capital costs is presented below in Table 1-3.

Table 1-3: Capital Cost Summary

WBS	Description	Capital Cost (US\$M)	Sustaining Cost (US\$M)	Total Cost (US\$M)
1000	Mining	\$98.2	\$414.9	\$513.1
2000	Process Plant	\$113.3	\$89.4	\$202.7
3000	Additional Process Facilities	\$30.9	\$25.2	\$56.0
4000	On-Site Infrastructure	\$54.1	\$2.0	\$56.1
5000	Off-Site Infrastructure	\$16.5	\$0	\$16.5
Total Direct Costs		\$313.0	\$531.4	\$844.4
6000	Project Preliminaries	\$36.5	\$0	\$36.5
7000	Indirect Costs Owners Costs	\$44.2	\$0	\$44.2
Total Indirect Costs		\$80.7	\$0	\$80.7
Total Direct + Indirect Costs		\$393.6	\$531.4	\$925.0
8000	Owner's Costs	\$18.5	\$0	\$18.5
9000	Contingency	\$102.6	\$47.2	\$149.8
Total Capital Cost		\$514.8	\$578.6	\$1,093.4

1.17.2 Operating Cost Estimate

The costs considered on-site operating costs are those related to processing, tailings handling, maintenance, power and general and administrative (G&A) activities.

A summary of the operating costs is presented below in Table 1-4. The unit operating cost is US\$79.93/t processed material, and G&A cost of US\$8.43/t processed.

Table 1-4: Operating Cost Summary

Cost Area	US\$/t Processed
Mining	37.36
Process	23.39
G&A	8.43
Total	79.93

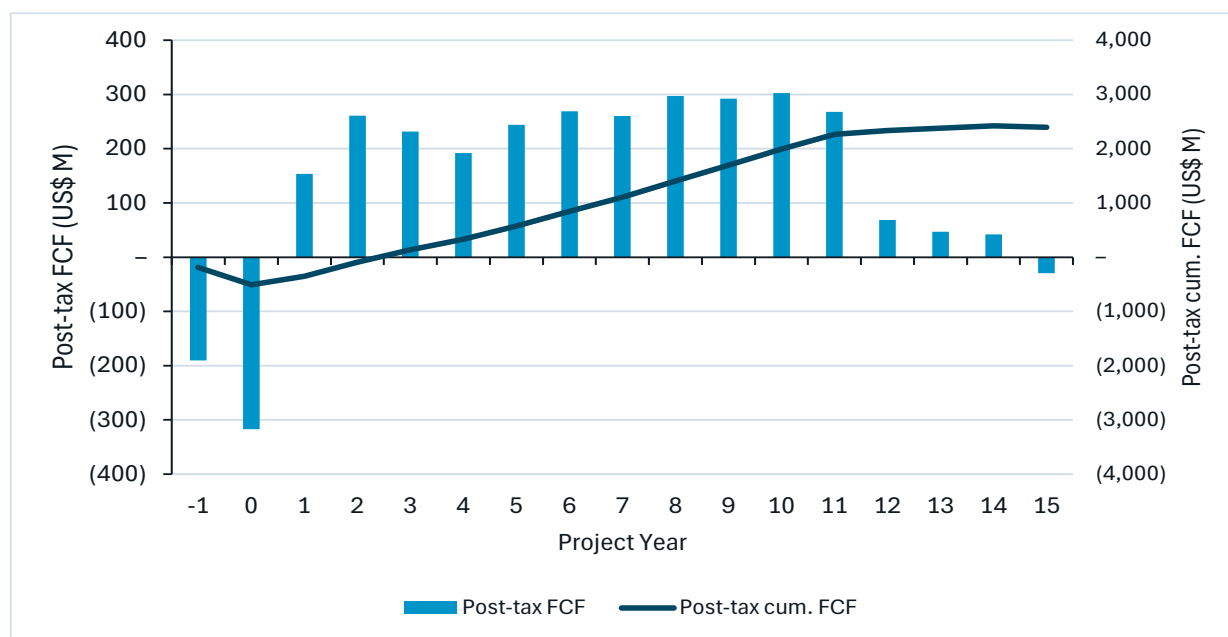
Note: Total may not add up due to rounding. Unit rates exclude PTU.

1.18 Economic analysis

The economic analysis was performed assuming a 5% discount rate. On a pre-tax basis the net present value (NPV) discounted at 5% is US\$2,098 million; the internal rate of return (IRR) is 46.7%, and payback period is 1.4 years. On a post-tax basis, the NPV discounted at 5% is \$1,491 million; the IRR is 37.3%, and the payback period is 1.9 years. A summary of project economics is shown graphically in Table 1-5.

The analysis was performed on a quarterly cashflow basis; the cashflow output is shown graphically in Figure 1-5.

Figure 1-5: LOM post-tax free cashflow¹



Notes:

1. Post-tax free cashflow is a non-GAAP financial measure or ratio. This measure has no standardised meaning under IFRS Accounting Standards and may not be comparable to similar measures used by other issuers. As the Project is not in production, Torex does not have historical non-GAAP financial measures nor historical comparable measures under IFRS, and therefore the foregoing prospective non-GAAP financial measure or ratio may not be reconciled to the nearest comparable measures under IFRS. Refer to the "Non-GAAP Financial Measures" in Section 2.8 of this Technical Report for more information, including a detailed description of post-tax free cashflow.

Source: Ausenco, 2026

Readers are cautioned that the PEA is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Table 1-5: Economic analysis summary

Item	Units	Total/Avg
Macro		
Gold price	US\$/oz	3,600.0
Silver price	US\$/oz	50.0
Discount Rate	%	5.0%
Production		
Project Life	yrs	14.4
Mine life at nameplate capacity	yrs	10.0
Total Resource Mined	kt	26,074
LOM Au Grade	g/t	1.68
LOM Ag Grade	g/t	63.37
Total Waste Mined	kt	76,034
Average Annual Mined Resource	ktpa	1,808
Total ROM feed	kt	26,074
Gold recovery	%	94.8%
Silver recovery	%	81.1%
Total Recovered Au	koz	1,338
Total Recovered Ag	koz	43,082
Total Recovered Au Eq	koz	1,936
Average Annual Recovered Au Eq at nameplate capacity	koz/a	166
Total Payable Au	koz	1,337
Total Payable Ag	koz	42,651
Total Payable Au Eq	koz	1,929
Average Annual Payable Au Eq at nameplate capacity	koz/a	166
Average Annual Payable Au Eq	koz/a	134
Revenue/Costs		
Total Revenue	US\$ M	6,945
Average Annual Revenue	\$M/a	482
EBITDA ⁽⁵⁾	US\$ M	4,439
Average Annual EBITDA ⁽⁵⁾	\$M/a	308
"Total On-Site Operating Costs (Mining, Process, G&A)"	US\$ M	2,131
Open Pit Unit Cost*	\$/t mined OP	2.89

Item	Units	Total/Avg
Open Pit Unit Cost*	\$/t processed	9.67
Underground Unit Cost*	\$/t mined UG	54.15
Underground Unit Cost*	\$/t processed	30.81
Paste Plant Unit Cost*	\$/t mined UG	11.52
Paste Plant Unit Cost*	\$/t processed	6.55
Mining + Paste Plant Unit Cost*	\$/t mined UG	65.68
Mining + Paste Plant Unit Cost*	\$/t processed	37.36
Process Unit Cost*	\$/t processed	23.39
G&A Unit Cost*	\$/t processed	8.43
Total operating unit cost ⁽⁴⁾	\$/t processed	79.93
"Total Off Site Operating Costs (Transport, Treatment & Refining)"	US\$ M	28
Cash costs		
Total cash costs - gold equivalent ⁽¹⁾	US\$/oz Au Eq	1,260
Mine-site all-in sustaining costs - gold equivalent ⁽²⁾	US\$/oz Au Eq	1,578
Capital costs		
Initial CAPEX	US\$ M	515
Sustaining CAPEX	US\$ M	579
Closure CAPEX	US\$ M	35
Salvage Value	US\$ M	–
Pre-production OPEX	US\$ M	19
Pre-Tax Economics		
NPV (5%)	US\$ M	2,098.0
IRR	%	46.7%
Payback	yr	1.4
NPV/Initial Capital	-	4.1
Post-Tax Economics		
NPV (5%) ⁽³⁾	US\$ M	1,491.0
IRR	%	37.3%
Payback	yr	1.9
NPV/Initial Capital	-	2.9

Notes

- 1) Total cash costs include operating cost, offsite costs and royalties excluding PTU profit share.
- 2) Mine-site AISC include total cash costs, closure costs and sustaining capital expenditures.
- 3) The effective tax rate is 27.3% (This combines corporate tax and mineral tax).
- 4) Total operating unit cost includes mine, mill, paste plant, G&A and offsite costs (excludes PTU profit share).
- 5) EBITDA, Average Annual EBITDA, AISC, total cash costs, and sustaining capital expenditures are non-GAAP financial measures or ratios. These measures have no standardised meaning under IFRS Accounting Standards and may not be comparable to similar measures used by other issuers. As the Project is not in production, Torex does not have historical non-GAAP financial measures nor historical comparable measures under IFRS, and therefore the foregoing prospective non-GAAP financial measures or ratios may

not be reconciled to the nearest comparable measures under IFRS. Refer to the “Non-GAAP Financial Measures” in Section 2.8 of this Technical Report for more information, including a detailed description of each of these measures.

1.18.2 Sensitivity Analysis

A sensitivity analysis was conducted on the base case pre-tax and post-tax NPV and internal rate of return (IRR) of the project, using the following variables: commodity prices, discount rate, operating costs, initial capex, metal recovery, and head grade.

A summary of post-tax economic sensitivities to commodity prices, operating costs, initial capex, metal recovery, and head grade is shown in Figure 1-6. The sensitivity analysis showed that the project is most sensitive to head grade, gold price, and recovery.

Figure 1-6: Post-tax sensitivity summary



Note: Head grade and recovery overlap. Source: Ausenco, 2026

1.19 Conclusions and recommendations

The Los Reyes Project demonstrates positive economics, as illustrated by the results presented in this Technical Report. It is recommended that the Project be advanced through to Pre-feasibility study. The following sections detail the recommended future work for the Project. The estimated costs are summarized in Table 1-6.

Table 1-6: Recommendations Cost Summary

Program component	Estimated total cost (US\$M)
Geotechnical Drilling	8.80
Metallurgical Testwork	1.50
Geotechnical & Hydrological drilling & field work	3.05
Mining & Geotechnical Studies, incl. backfilling	1.70
Process & Infrastructure Engineering	1.45
Tailings Storage Facility	0.45
Environmental Studies	1.20
Others	1.75
Total	19.90

2 Introduction

2.1 Overview

Ausenco Engineering Canada ULC (Ausenco) has prepared a Preliminary Economic Assessment (PEA) and associated technical report for (Torex) on the Los Reyes Project located in Sinaloa and Durango, Mexico. The Technical Report was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 (NI 43-101) and Form 43-101 F1.

The responsibilities of the engineering consultants are as follows:

- Ausenco was responsible for managing and coordinating the work related to the technical report and developed PEA-level design, including capital and operating cost estimates for the process plant, general site infrastructure, TSF design and site-wide water management measures.
- AGP Mining Consultants Inc. (AGP) was responsible for designing the open-pit and underground mine, mine production schedule, mining fleet and mining capital and operating costs. AGP also developed the mineral resource estimate (MRE) for the project.
- Sims Resources LLC (Sims) was responsible for completing the work related to the geological setting, deposit type, drilling, explorations work, sample preparation and analysis and data verification.
- Knight Piésold Ltd. (KP) was responsible for designing and providing capital and operating costs for the tailings and water management facilities for the project, they were also responsible for the geotechnical assessment and design input for the open pit and underground mine.
- Qualitica Consulting Inc (Qualitica) was responsible for the review of the QA/QC for diamond drilling programs completed by Prime Mining during their ownership.

2.2 Terms of reference

Torex is using this Technical Report to support disclosure of a PEA level study of the Project, with an updated MRE. The Technical Report conforms to NI 43-101. The effective date of this report is April 25, 2026. There were no material changes to the project between this date and the publication of this accompanying NI 43-101 Technical Report.

All measure units used in this Technical Report are metric unless otherwise noted. Currency is expressed in United States dollars (US\$ or USD) unless otherwise noted. This technical report uses English.

Mineral resources and mineral reserves are reported in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (CIM, 2019).

Readers are cautioned that the PEA is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized.

2.3 Qualified persons

The following serve as the qualified persons for this Technical Report as defined in National Instrument 43-101, Standards of Disclosure for Mineral Projects, and in compliance with Form 43-101F1:

Table 2-1: Report contributors

Qualified person	Professional designation	Position	Employer	Independent of Torex Gold Resources Inc.
Peter Mehrfert	P.Eng.	Principal Process Engineer	Ausenco Engineering Canada ULC	Yes
John Sims	Certified Professional Geologist (CPG)	President	Sims Resources LLC	Yes
Rodrigo Capel Pasqua	FAusIMM	Principal Mining Consultant	AGP Mining Consulting, Inc.	Yes
Daniel Ruane	P.Eng.	Specialist Engineer	Knight Piésold Ltd.	Yes
Alexander Hemmingsen	P.Eng.	Principal Engineer, Mine Design	Torex Gold Resources Inc.	No
Fernando Cornejo	P.Eng.	Vice-President Projects	Torex Gold Resources Inc.	No
Chantal Jolette	P.Geo.	President and Principal Geologist	Qualitica Consulting Inc.	Yes
Benjamin Peacock	P.Eng.	Specialist Engineer	Knight Piésold Ltd.	Yes

2.4 Personal inspections

2.4.1 Personal inspection details by qualified person Peter Mehrfert, P.Eng.

Peter Mehrfert visited the Los Reyes site on June 1, 2025 during which time he travelled along the site access road and observed a view of the Zapote and Mariposa deposits, topography and location of the proposed processing facility site and dry stack tailings impoundment. The project office in Cosalá was visited and some of the drill core samples were observed.

2.4.2 Personal inspection details by qualified person John Sims, CPG

John Sims visited the Property for 7 days starting October 8, 2025. The visit included a review of the Mineral Resource Estimate and a site inspection. Mr. Sims inspected core and sample cutting and logging areas; discussed geology and mineralisation; reviewed geological interpretations and Mineral Resource modeling procedures with Prime's technical staff.

2.4.3 Personal inspection details by qualified person Rodrigo Capel Pasqua, P.Eng.

Rodrigo Capel Pasqua conducted a site visit to the Property from May 31 to June 2, 2025. During the inspection, which lasted one and a half days, Mr. Pasqua examined the surface footprint of the proposed open pits, the access roads, the proposed portal locations for the plant site and the underground tunnel network, and the core storage and geological sample facilities at the Cosalá office. Observations from the visit were incorporated into the mining design, schedule and risk assessment.

2.4.4 Personal inspection details by qualified person Daniel Ruane, P.Eng.

Daniel Ruane visited the Los Reyes Project on January 29, 2024. During the site inspection, Mr. Ruane visited the proposed TSF locations and associated water management infrastructure areas to assess site topography, access conditions, foundation exposures, and potential facility layouts. The inspection included observation of local geological and drainage conditions. Observations from the site visit were incorporated into the assessment of tailings management alternatives, site selection and facility layout development presented in this report.

2.5 Effective dates

The report contains the following effective dates:

- Effective date of Mineral Resource Estimate: April 25, 2026.
- Effective date of the Report is: April 25, 2026.

2.6 Information sources and references

2.6.1 Previous reports

The following report has been completed previously for the project:

- NI 43-101 Technical Report for Los Reyes Project, Update of Mineral Resource Estimate, November 27, 2024.

2.6.2 Other sources

Documents listed in Section 3 and Section 27 were used to support the preparation of the technical report. The sources of information supplied by Torex include historical data and reports compiled by previous consultants and researchers of the project as well as other documents cited throughout the technical report and referenced in Section 27.

2.7 Currency, units, abbreviations and definitions

All units of measurement in this report are metric, and all currencies are expressed in United States dollars (symbol: US\$ or currency: USD) unless otherwise stated. Exchange rates are included in Section 21.2.2. Contained gold metal is expressed as troy ounces (oz), where 1 oz = 31.1035 grams (g). All material tonnes are expressed as dry tonnes (t) unless stated otherwise. A list of abbreviations and acronyms is provided in Table 2-1, and units of measurement are listed in Table 2-2.

Table 2-2: Abbreviations and acronyms

Abbreviation	Description
AA	atomic absorption spectroscopy
ASD	Analytical Spectral Services, Inc.
AuEq	gold equivalent
BTW	B-size thin-wall core diameter
BV	Bureau Veritas (Mexico)
CAD:USD	Canadian-American exchange rate

Abbreviation	Description
CAF	cut-and-fill
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIM Definition Standards	CIM Definition Standards for Mineral Resources and Mineral Reserves 2014
CoG	cut-off grade
CONAGUA	National Water Commission (<i>Comisión Nacional del Agua</i>)
DDH	diamond drill hole
ELOS	Equivalent Linear Overbreak/Slough
EM	electromagnetic
ETW	extended thin wall
FA	fire assay
FAR	fresh-air raise
FET	federal excise tax
FS	feasibility study
FTE	full-time equivalent
FW	footwall
G&A	general and administration
GE	Guadalupe East
GPR	gross production royalty
GPS	global positioning system
HQ	H-size wireline core diameter
HSE	Health, Safety and Environment
HW	hanging wall
ICMM	International Council on Mining and Metals
ICP-OES	inductively coupled plasma - optical emission spectrometry
IMSS	Mexican Social Security Institute (<i>Instituto Mexicano del Seguro Social</i>)
INAH	National Institute of Anthropology and History (<i>Instituto Nacional de Antropología e Historia</i>)
INFONAVIT	Institute of the National Housing Fund for Workers (<i>Instituto del Fondo Nacional de la Vivienda para los Trabajadores</i>)
ISO	International Organization for Standardization
LHD	load-haul-dump (underground mining equipment)
LHOS	long-hole open stoping
LIMS	Laboratory Information Management System
LOM	life-of-mine
MRE	mineral resource estimate
NAD 83	North American Datum of 1983
NCM	Northern Crown Mine
NGI	Norwegian Geotechnical Institute
NI 43-101	National Instrument 43-101 (Regulation 43-101 in Quebec)
NQ	N-size wireline core diameter
NSR	net smelter return
NTW	N-size thin-wall core diameter
NWT	N-size wireline thin-wall core diameter
OEE	overall equipment efficiency
OP	open pit

Abbreviation	Description
OK	ordinary kriging
PEA	preliminary economic assessment
PFS	prefeasibility study
PoF	probability of failure
QA/QC	quality assurance/quality control
QP	qualified person (as defined in National Instrument 43-101)
RAR	return-air raise
RC	reverse circulation
ROM	run of mine
RPD	relative percent difference
RQD	rock quality designation
SCC	Standards Council of Canada
SD	standard deviation
S _d -BWI	micro hardness or bond ball mill work index on SAG ground material
SE	Ministry of Economy (<i>Secretaría de Economía</i>)
SEDENA	Ministry of National Defense (<i>Secretaría de la Defensa Nacional</i>)
SEMARNAT	Ministry of Environment and Natural Resources (<i>Secretaría de Medio Ambiente y Recursos Naturales</i>)
SG	specific gravity
SLC	sub-level caving
TSF	tailings storage facility
UG	underground
UTM	Universal Transverse Mercator coordinate system
WRSFs	waste rock storage facilities
Z-T	Zapote-Tahonitas

Table 2-3: Units of measurement

Abbreviation	Description
%	percent
% solids	percent solids by weight
C\$	Canadian dollar (as symbol)
\$/t	dollars per metric ton
°	angular degree
°C	degree Celsius
µm	micron (micrometre)
cm	centimetre
cm ³	cubic centimetre
ft	foot (12 inches)
g	gram
g/cm ³	gram per cubic centimetre
g/L	gram per litre
g/t	gram per metric ton (tonne)
h	hour (60 minutes)
ha	hectare
kg	kilogram
kg/t	kilogram per tonne

Abbreviation	Description
km	kilometre
km ²	square kilometre
koz	kilo-ounces (troy)
kW	Kilowatt
kWh/t	kilowatt-hour per tonne
L	litre
lb	Pound
m, m ² , m ³	metre, square metre, cubic metre
M	Million
Ma	million years (annum)
masl	metres above mean sea level
mm	millimetre
Moz	million (troy) ounces
Mt	million tonnes
MW	megawatt
oz	troy ounce
oz/t	ounce (troy) per tonne
oz/ton	ounce (troy) per short ton (2,000 lbs)
ppb	parts per billion
ppm	parts per million
s	second
t	metric tonne (1,000 kg)
ton	short ton (2,000 lbs)
t/d	tonnes per day
USD	US dollars (currency)
US\$	US dollar (as symbol)

2.8 Use of non-GAAP Financial Measures

Certain financial measures referred to in this Technical Report are not measures recognized under IFRS and are referred to as non-GAAP financial measures or ratios. These measures have no standardized meaning under IFRS and may not be comparable to similar measures presented by other companies. These measures are intended to provide additional information and should not be considered in isolation or as a substitute for measures prepared in accordance with IFRS.

The non-GAAP financial measures used in this Technical Report and common to the mining industry are defined below:

- Earnings Before Interest, Taxes, Depreciation, and Amortization (EBITDA) is gross revenue less offsite charges, operating costs and royalties.
- Average Annual EBITDA is EBITDA divided by project life.
- All-in Sustaining Costs ("AISC") includes total cash costs, closure costs and sustaining capital expenditures.
- Total Cash Costs includes operating cost excluding PTU profit share, offsite costs and royalties.
- Sustaining Capital Expenditures is capital expensed during the mine life to sustain operations and not deemed part of an expansion.

- Pre-tax Free Cashflow is EBITDA less pre-production operating costs, initial capital, sustaining capital, closure costs and changes in working capital.
- Post-tax free cashflow is EBITDA less pre-production operating costs, initial capital expenditure, sustaining capital expenditure, closure costs, changes in working capital and taxes paid.

Los Reyes is not currently in operation; therefore, there are no historical equivalent measures to compare to, and a reconciliation of these non-IFRS financial performance measures cannot be carried out.

2.9 Forward-Looking Information

This Technical Report contains “forward-looking information” within the meaning of applicable Canadian securities legislation. Forward-looking information includes, but is not limited to: the timing of the potential advancement of the Los Reyes Project towards pre-feasibility and feasibility, the completion of environmental studies, the infill and expansion of the known mineralization at the Los Reyes Project, the planned drilling metres, the capex requirements, the funding of activities in 2026 and beyond, IRR, NPV, AISC, the future price of gold and silver, cash flow, payback period, life-of-mine and other future financial or operating performance of the Los Reyes Project, the advancement of the Los Reyes Project towards ‘shovel-ready’ status, long-life and large-scale potential of the Los Reyes Project and exploration upside of the land packages. Generally, forward-looking information can be identified by the use of forward-looking terminology such as “accelerate”, “add” or “additional”, “advancing”, “anticipates” or “does not anticipate”, “appears”, “believes”, “can be”, “conceptual”, “confidence”, “continue”, “convert” or “conversion”, “deliver”, “demonstrating”, “estimates”, “encouraging”, “expand” or “expanding” or “expansion”, “expect” or “expectations”, “fast-track”, “forecasts”, “forward”, “goal”, “improves”, “increase”, “intends”, “justification”, “leading”, “plans”, “potential” or “potentially”, “pro forma”, “promise”, “prospective”, “prioritize”, “reflects”, “re-rating”, “robust”, “scheduled”, “stronger”, “suggesting” or “suggests”, “support”, “updating”, “upside”, “will be” or “will consider”, “work towards”, or variations of such words and phrases or state that certain actions, events or results “may”, “could”, “would”, “might”, or “will be taken”, “occur”, or “be achieved”. Forward-looking information is based on the opinions and estimates of the QPs at the date the information is made, and is based on a number of assumptions and is subject to known and unknown risks, uncertainties and other factors that may cause the actual results, level of activity, performance or achievements of the issuer to be materially different from those expressed or implied by such forward looking information, including risks associated with required regulatory approvals, the exploration, development and mining such as economic factors as they effect exploration, future commodity prices, changes in foreign exchange and interest rates, global inflationary pressures, actual results of current exploration activities, government regulation, political or economic developments, the ongoing wars and their effect on supply chains, tariffs, environmental risks, pandemic risks, permitting timelines, capex, operating or technical difficulties in connection with development activities, employee relations, the speculative nature of gold exploration and development, including the risks of diminishing quantities of grades of reserves, contests over title to properties, and changes in project parameters as plans continue to be refined as well as those risk factors discussed in Torex’s Annual Information Form for the year ended December 31, 2025, available on www.sedarplus.ca.

Although the QPs have attempted to identify important factors that could cause actual results to differ materially from those contained in forward-looking information, there may be other factors that cause results not to be as anticipated, estimated or intended. There can be no assurance that such information will prove to be accurate, as actual results and future events could differ materially from those anticipated in such information. Accordingly, readers should not place undue reliance on forward-looking information.



The QPs do not undertake to update any forward-looking information, except in accordance with applicable securities laws.

3 Reliance on other experts

3.1 Introduction

The QPs have relied upon the following other expert reports, which provided information regarding mineral rights, surface rights, property agreements, royalties, environmental, permitting, social licence, closure, taxation, and marketing for sections of this Report.

3.2 Property agreements, mineral tenure, surface rights, and royalties

The QPs have not independently reviewed ownership of the Project area and any underlying property agreements, mineral tenure, surface rights, or royalties. The QPs have fully relied upon information derived from Torex and legal experts retained by Torex for this information through the following documents:

- Juan Carlos Moreles de Teresa, Legal Opinion - Minera Amari, S.A. de C.V., Torex Gold Resources Inc., July 24, 2025, 33 pp.
- Sergio Biebrich Guevara, Review of Ejido Agreements and Terms, Torex Gold Resources Inc., June 22, 2026, 6 pp.

This information is used in Sections 4 and 5 of the Report. The information is also used in support of Section 22.

3.3 Environmental, permitting, closure, and social and community impacts

The QPs have fully relied upon information supplied by Torex and experts retained by Torex for information related to environmental (including tailings and water management) permitting, permitting, closure planning and related cost estimation, and social and community impacts as follows:

- Patricia Aguayo, July 30, 2025: Environmental and Permitting Section Los Reyes PEA, Prime Mining Corp., 15 pp.
- John Sebastian Vaughan, May 28, 2026: Los Reyes Closure Costs PEA, Torex Gold Resources, 1 p.
- Rafael Andres Curra, May 23, 2026: PEA Environmental & Permitting, Torex Gold Resources, 4 pp.
- Rafael Andres Curra, June 23, 2026: Chapter 20 references, Torex Gold Resources, 1 p.

This information is used in Section 20 of the Report.

3.4 Taxation

The QPs have fully relied upon information derived from and by experts retained by Torex for information related to taxation and applied to the financial model as follows:

- David Mendez Estrada & Ajay Kukreja, June 1, 2026: Minutes of Meeting – Tax Approach Discussion, Torex Gold Resources, 2 pp.

This information is used in Section 22 of the Report.

3.5 Markets

The QPs have not independently reviewed the marketing or metal price information. The QPs have fully relied upon information derived from Torex and experts retained by Torex for this information through the following documents:

- Dan Rollins, May 23, 2026: Los Reyes PEA – Discussion – Suggested Metal Prices for Economics: Torex Gold Resources Inc., 1 p.

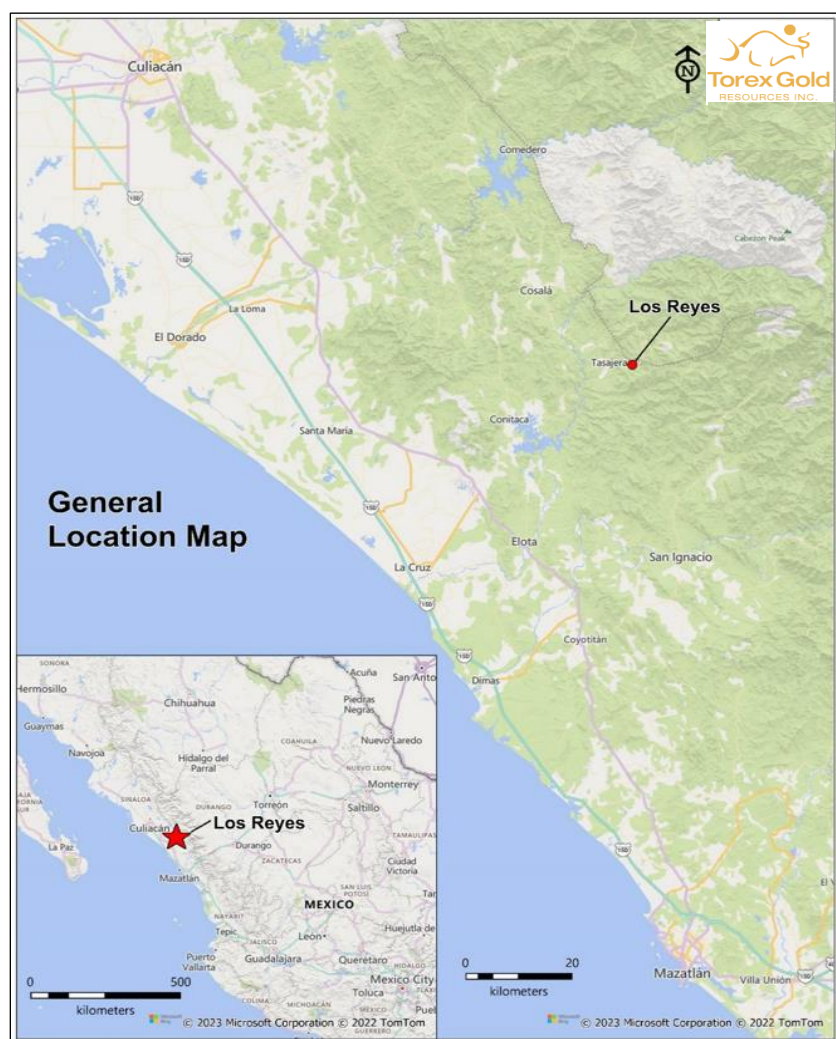
This information is used in Section 19 of the Report. The information is also used in support of Section 22.

4 Property description and location

4.1 Introduction

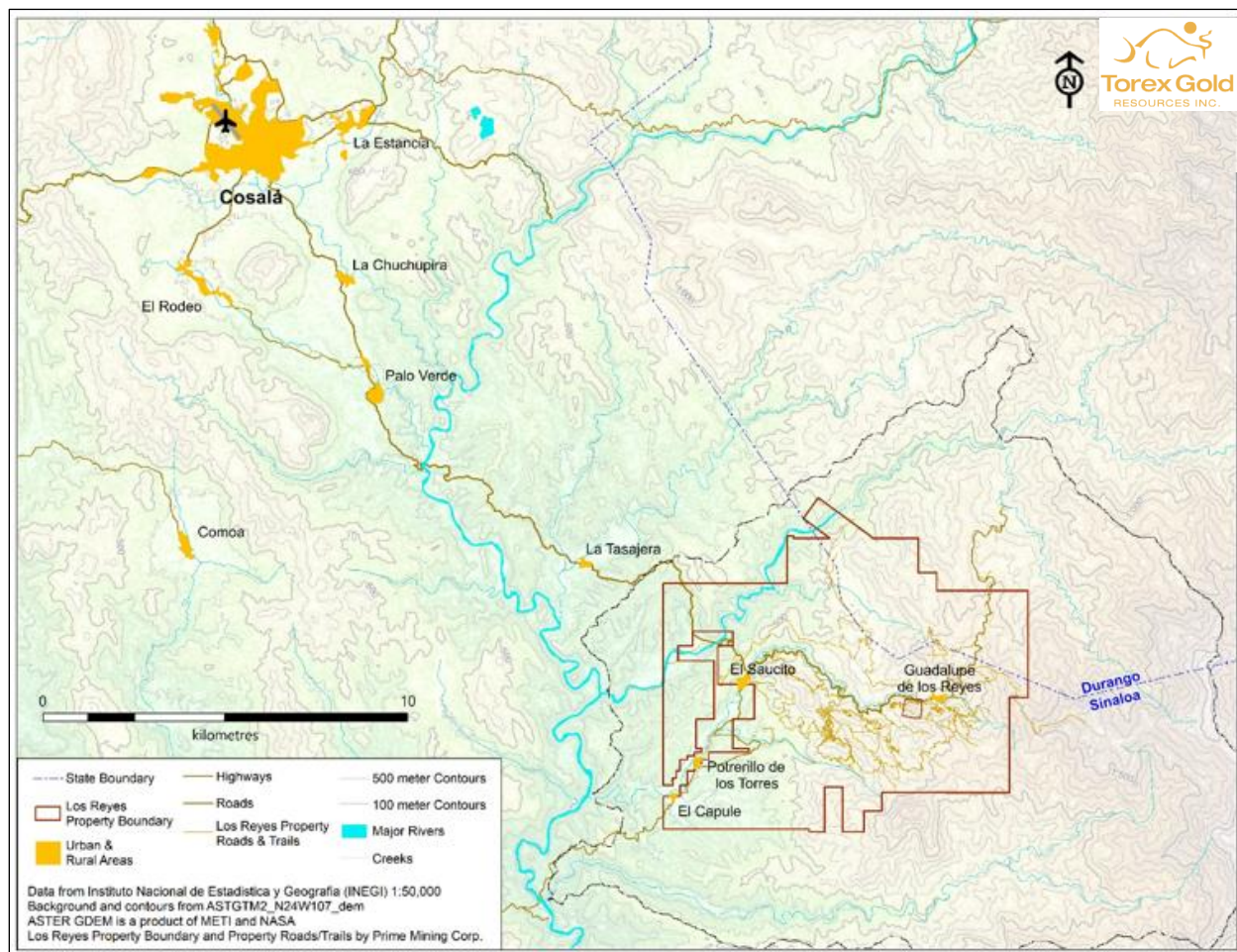
The Property is located in the Guadalupe de Los Reyes mining district in the western foothills of the SMO mountain range, Sinaloa and Durango states, Mexico (Figure 4-1). The Property is north of the coastal city of Mazatlán, approximately 110 km by air and 200 km by paved road. The Property is within the municipality of Cosalá (population 17,012, INEGI 2020) and the closest city to the Property is Cosalá (population 7,888, INEGI 2020), which is located 27 km to the northwest of the proposed plant site location. Enroute to the Property from Cosalá are the villages of Palo Verde and La Tasajera. The village of Guadalupe de los Reyes is on the Property and was a site for Spanish colonial mining (Figure 4-2). The general geographic coordinates of the Property are N-24°17' and W-106°32' (UTM Zone 13 North 0344250E, 2686400N). Coordinates are in WGS 84.

Figure 4-1: General location map



Source: Torex, 2024

Figure 4-2: Property location map



Source: Torex, 2024

4.2 Los Reyes Property concessions and area

The Property is composed of 37 contiguous concessions that have an area of 6,287.2 hectares (ha); however, due to overlap between some of the concessions, the actual area is 6,273 ha. Figure 4-3 shows the concessions and their associated overlap.

Bello Gallardo Bonequi y García, S.C., a law firm located in Mexico City, was commissioned in 2025 to provide a title opinion and related matters that are associated with the mining concessions of the Property. This title opinion was completed on July 25, 2025, and stated that the Property was in good standing.

Minera Amari, S.A. de C.V. ("Minera Amari"), an indirect wholly owned subsidiary of Torex, is the registered holder of the concessions and has negotiated surface rights agreement for 1,800 ha of land use with the local community, Ejido La Tasajera, located in Cosalá, Sinaloa, Mexico. These payments are in good standing, and are progressive, based on time and project status. See Section 4.4 below for further details.

4.3 Mineral Tenure, Option Agreements, Royalties and Encumbrances

Torex, through its indirect wholly owned subsidiary, Minera Amari, holds the unencumbered rights to 100% of the Los Reyes mining concessions, free of any liens, charges or third-party claims or rights, and subject only to royalties, some of which include a buyout option. Outlined below is the history of the Property concessions ownership.

Portions of the claim block have been owned by several entities over time. As a result, there are several royalty agreements in place. The entities that hold or held agreements are listed below:

- Minera Alamos Inc. ("MAI"), a company incorporated under the laws of the Province of Ontario
- Minera Alamos de Sonora S.A. de C.V. ("MAI Mexico"), a company incorporated under the laws of the United Mexican States
- ePower Metals Inc. (subsequently Prime Mining Corp. ("Prime"), which is a company incorporated under the laws of the Province of British Columbia
- Vista Gold Corp. ("Vista Gold") is a corporation existing under the laws of the Province of British Columbia
- Minera Gold Stake, S.A. de C.V. ("MGS"), a corporation existing under the laws of the United Mexican States
- Minera Gold Stake Holdings Corp. ("MGS Canada"), a corporation existing under the laws of British Columbia
- Granges Inc. ("Granges"), a corporation existing under the laws of British Columbia.

4.3.1 Previous Ownership Agreements

4.3.1.1 Vista Gold Agreement

Vista Gold, MGS Canada and Granges together owned 100% of the outstanding common shares of MGS. MGS acquired a 100% interest in portions of the Zapote zone on August 1, 2003, from Sr. Enrique Gaitán Maumejean. The final payment of the purchase option, which also included acquisition of a data package associated with the project, was completed in 2009. In January 2008, MGS further consolidated the remaining mining concessions, subsequently known as the Guadalupe de Los Reyes Project (Los Reyes Project), except for the 6 de Enero claim, which is 23.7 ha in size. Following this consolidation, and the acquisition of ten new claims, including fractionals, MGS's land position included 37 contiguous concessions. The consolidation of the mineral rights was completed through agreements with Grandcru Resources Corporation ("Grandcru"), Goldcorp Inc., and the San Miguel Group.

In addition to securing the mineral tenure, MGS also negotiated access agreements to the lands held by Ejido La Tasajera. These agreements provided access to the Property.

4.3.1.2 Minera Alamos Option Agreement

MAI and its wholly owned Mexican subsidiary, MAI Mexico, subsequently entered into an option agreement (the "Vista Gold Option Agreement") dated October 23, 2017, with Vista Gold, MGS, MGS Canada and Granges (collectively, the "Vista Gold Parties"). This option agreement granted to MAI an option to acquire 100% of the issued and outstanding common shares of MGS and therefore acquire full ownership the Los Reyes mining concessions.

This Vista Gold Option Agreement was subject to a 49% Back-In Right on underground resources, and required the payment of US\$6 million, payable in four installments of US\$1.5 million each. The first installment stipulated payment at the time of execution of the agreement, with the following two installment payments being made on the 12th month and 24th month of execution of the agreement. Payment of the last installment, termed the purchase price payment, was to be made on or before the end of the option period. In the event that MAI announced a positive decision to take the Los Reyes project into construction, MAI agreed to make this final payment within 30 days following the date of such announcement. The date of the announcement would then be the closing date. In addition to these cash payment requirements, MAI agreed for the duration of the option agreement to pay for storage of core in Hermosillo and to pay for 100% of the maintenance costs to keep the mining concession in good standing.

In order for the Alamos group of companies (MAI and MAI Mexico) to comply with Mexican law, MGS and MAI Mexico entered into an exploration agreement to conduct exploration work on the mining concessions, to cover the costs of the core storage facility in Hermosillo, and to fulfill all of the obligations with the Ejido La Tasajera. An Ejido is communal land used for agriculture in which the community members have usufruct rights rather than ownership rights to the land. Obligations to the Ejido La Tasajera and other landowners in the Property area include obtaining a temporary occupancy agreement to obtain access to the Property, along with other ancillary agreements, as required.

4.3.1.3 Prime Mining Corp. Purchase Agreement Terms

Pursuant to an assignment and assumption agreement made effective June 25, 2019, MAI and MAI Mexico transferred all of their rights and interests in the Los Reyes project to Prime (formerly ePower Metals Inc.) and its wholly owned subsidiary, Minera Amari (formerly ePower Metals, S.A. de C.V.). To meet the contractual obligations of the Vista Option Agreement, Prime agreed to the following conditions:

- Payment of US\$1.5 million to MAI as reimbursement for the installment payment made by MAI to Vista Gold on April 23, 2019 (which represented the second option payment under the October 23, 2017, agreement).
- Assume MAI's remaining option payments of US\$3 million in favour of Vista Gold: US\$1.5 million due on October 23, 2019, and US\$1.5 million due on the earlier of October 23, 2021 or a production decision.
- Issuance of 9,450,000 post-consolidation common shares of Prime and 3,350,000 common share purchase warrants of Prime to MAI, entitling MAI to acquire further post-consolidation common shares at a price of \$0.50 per share for a period of 24 months. Prime completed this transaction, as stated in its August 28, 2019, news release.

With this agreement, Prime and Minera Amari assumed all obligations and liabilities of MAI and MAI Mexico under the Vista Gold Option Agreement, as amended.

Pursuant to an agreement (the "Amended Option Agreement") made on June 12, 2020, the Vista Gold Option Agreement was amended to provide for the cancellation of all ongoing NSRs and Back-In Rights held by Vista Gold, in consideration for (i) accelerating the final US\$1.5 million option payment owing to Vista Gold (the "Option Payment"), and (ii) paying US\$1.1 million no later than six months following the acquisition date and US\$1 million no later than 12 months following the acquisition date.

According to the terms of the Amended Option Agreement, once Prime made the Option Payment, Vista Gold would no longer retain a capped NSR on production from open-pit mining or a perpetual NSR on production from underground mining. In addition, Vista Gold would no longer have the Back-in Rights to assume a 49% non-carried interest in any underground mining project developed at the Property. If the Company failed to make the US\$1.1 million and US\$1 million payments, Vista Gold would have the right to reinstate its NSRs and Back-in Rights.

In summary, to acquire the Property, Prime:

- paid US\$1.5 million to MAI, to reimburse MAI for the cost of an option payment required to be made to Vista Gold in April 2019.
- assumed MAI's remaining Option Payments of US\$3 million in favor of Vista Gold of which US\$1.5 million was paid in October 2019 and US\$1.5 million was paid in July 2020.
- issued to MAI 9,450,000 common shares and 3,350,000 common share purchase warrants entitling MAI to acquire further common shares at a price \$0.50 per share for a period of 24 months.
- paid to Vista Gold US\$1.1 million in January 2021 and a further US\$1 million in July 2021, which together satisfied the conditions of the Option Payment within the required timeframe, thereby terminating the NSRs and Back-In Rights previously held by Vista Gold.

As of July 20, 2020, Minera Amari filed a deed with the Public Registry of Property and Commerce to record the transfer of the 37 Los Reyes mining concessions. Registration of these concessions with the Mines General Directorate's Mining Public Registry was completed in November 2024.

Effective October 22, 2025, Torex acquired all the issued and outstanding shares of Prime by way of a plan of arrangement. Upon the effective date of the arrangement, Prime and Minera Amari became wholly owned subsidiaries of Torex, and Torex thereby acquired indirect ownership of the Los Reyes Project.

4.3.2 Summary of Los Reyes Property Concessions and Royalties

A summary of the royalties applicable each mining concession that forms part of the Property, as currently understood by Torex based on documentation held by Prime, is included in Table 4-1, below. Note that due to slight overlap among some of the 37 individual Los Reyes concessions, the total area for the concessions shown in Table 4-1 sums to 6,287.2 ha. The actual surface area of the Los Reyes property is 6,273 ha. The Property dispositions are shown on Figure 4-4.

Table 4-1: Mineral tenure, royalty agreement summary by mining concession within the property

Concession Name	Title Number	Surface Area (Ha)	Expiration Date	Location	Royalty %	
					With repurchase option	Without repurchase option
Los Reyes Dos	214131	17.4	9-Aug-51	Cosalá, Sinaloa		4%: NSRs 1 and 4
Los Reyes Tres	214302	197.0	5-Sep-51	Tamazula, Durango		4%: NSRs 1 and 4
Los Reyes Cuatro	217757	11.2	12-Aug-52	Cosalá, Sinaloa		4%: NSRs 1 and 4
Los Reyes Cinco	216632	320.0	16-May-52	Cosalá, Sinaloa		4%: NSRs 1 and 4
Los Reyes Seis	225122	427.7	21-Jul-55	Cosalá, Sinaloa		1%: NSR 4
Los Reyes Siete	225123	4.8	21-Jul-55	Cosalá, Sinaloa		1%: NSR 4
Los Reyes 8	226037	9.0	14-Nov-55	Cosalá, Sinaloa		4%: NSRs 1 and 4
Los Reyes Fracc. Oeste	210703	476.9	17-Nov-49	Cosalá, Sinaloa		4%: NSRs 1 and 4
Los Reyes Fracc. Sur	212758	598.1	7-Oct-49	Cosalá, Sinaloa		4%: NSRs 1 and 4
Los Reyes Fracc. Norte	212757	1334.5	7-Oct-49	Cosalá, Sinaloa		4%: NSRs 1 and 4
Norma	177858	150.0	28-Apr-36	Cosalá, Sinaloa	2%: NSR 3	1%: NSR 5
Nueva Esperanza	184912	8.53	5-Dec-39	Cosalá, Sinaloa	2%: NSR 3	1%: NSR 5
San Miguel	185761	11.8	13-Dec-39	Cosalá, Sinaloa	2%: NSR 3	1%: NSR 5
San Manuel	188187	55.8	21-Nov-40	Cosalá, Sinaloa	2%: NSR 3	1%: NSR 5
El Padre Santo	196148	50.0	15-Jul-43	Cosalá, Sinaloa	2%: NSR 3	1%: NSR 5
El Faisán	211471	2.6	30-May-50	Cosalá, Sinaloa	2%: NSR 3	1%: NSR 5
Santo Niño	211513	44.1	30-May-50	Cosalá, Sinaloa	2%: NSR 3	1%: NSR 5
San Pablo	212752	11.2	21-Nov-50	Cosalá, Sinaloa	2%: NSR 3	1%: NSR 5
San Pedro	212753	9.0	21-Nov-50	Cosalá, Sinaloa	2%: NSR 3	1%: NSR 5
Patricia	212775	26.2	30-Jan-51	Cosalá, Sinaloa	2%: NSR 3	1%: NSR 5
Martha I	213234	46.7	9-Apr-51	Cosalá, Sinaloa	2%: NSR 3	1%: NSR 5
Elota	237661	947.7	19-Apr-61	Cosalá, Sinaloa Tamazula, Durango	NIL	NIL
Elota Fracción 1	237662	905.6	19-Apr-61	Cosalá, Sinaloa Tamazula, Durango	NIL	NIL
Elota Fracción 2	237663	3.3	19-Apr-61	Cosalá, Sinaloa Tamazula, Durango	NIL	NIL
Elota Fracción 3	237664	2.7	19-Apr-61	Cosalá, Sinaloa Tamazula, Durango	NIL	NIL

Concession Name	Title Number	Surface Area (Ha)	Expiration Date	Location	Royalty %	
					With repurchase option	Without repurchase option
Elota Fracción 4	237665	8.1	19-Apr-61	Cosalá, Sinaloa Tamazula, Durango	NIL	NIL
Elota Fracción 5	237666	4.2	19-Apr-61	Cosalá, Sinaloa Tamazula, Durango	NIL	NIL
Elota Fracción 6	237667	0.5	19-Apr-61	Cosalá, Sinaloa Tamazula, Durango	NIL	NIL
Elota Fracción 7	237668	0.2	19-Apr-61	Cosalá, Sinaloa Tamazula, Durango	NIL	NIL
Elota Fracción 8	237669	0.7	19-Apr-61	Cosalá, Sinaloa Tamazula, Durango	NIL	NIL
Elota Fracción 9	237670	1.0	19-Apr-61	Cosalá, Sinaloa Tamazula, Durango	NIL	NIL
Diez de Mayo	223401	0.2	10-Dec-54	Cosalá, Sinaloa	2%: NSR 2	2-3%: NSR 6
Prolongación del Recuerdo	210497	91.5	7-Oct-49	Cosalá, Sinaloa	2%: NSR 2	2-3%: NSR 6
Prolongación del Recuerdo Dos	209397	26.7	8-Apr-49	Cosalá, Sinaloa	2%: NSR 2	2-3%: NSR 6
Arcelia Isabel	193499	60.4	18-Dec-41	Cosalá, Sinaloa	2%: NSR 2	2-3%: NSR 6
Dolores	180909	222.0	5-Aug-37	Cosalá, Sinaloa	2%: NSR 2	2-3%: NSR 6
La Victoria	210803	199.9	29-Nov-49	Cosalá, Sinaloa	2%: NSR 2	2-3%: NSR 6
Total Surface Area		6,287.2				

A description of the various royalties identified in table 4-1 is as follows:

NSR 1: A 3% NSR to be obtained as a result of producing and selling gold, silver and other ores covering the mining concessions: "Los Reyes Dos", title 214131; "Los Reyes Tres", title 214302; "Los Reyes Cuatro", title 217757; "Los Reyes Cinco", title 216632; "Los Reyes 8", title 226037; "Los Reyes Fracc. Oeste", title 210703; "Los Reyes Fracc. Sur", title 212758; and "Los Reyes Fracc. Norte", title 212757, in favor of Corporación Turística San Luis, S.A. de C.V. or its designee.

NSR 2: (Re-purchase option) Enrique Gaitán's 2% NSR (Minera Tatemás S.A. de C.V.) can be re-purchased for US\$1 million before July 31, 2053. The Gaitán NSR came from Vista Gold's acquisition of the Gaitán claims Jan 3, 2003, for US\$1.4 million plus 2% NSR.

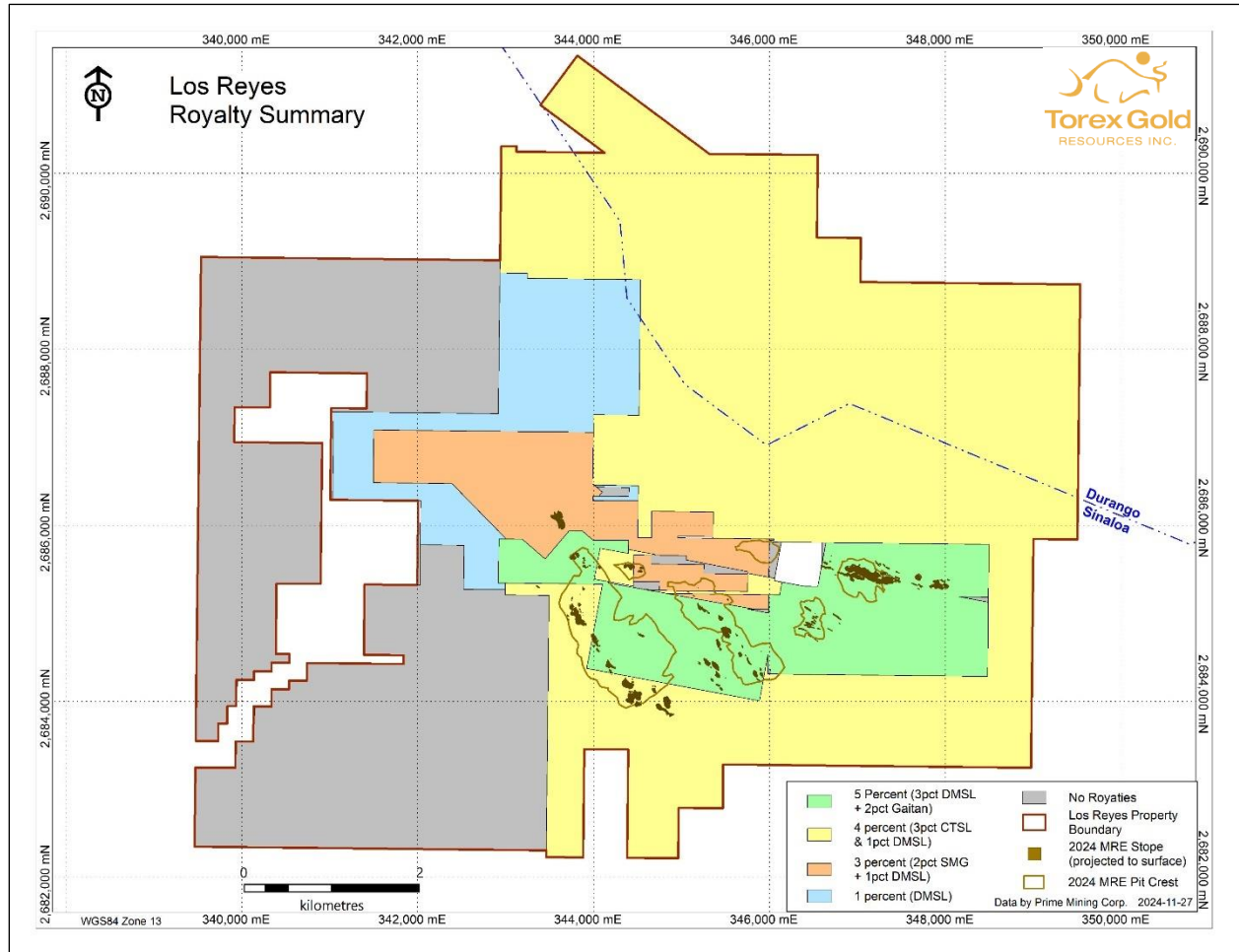
NSR 3: (Re-purchase option) San Miguel Group's (SMG) 2% NSR can be re-purchased for US\$1 million at any time. The SMG NSR is owed to Genssler Investment Partnership, LLP, Doug Foote, and Synergy Group Limited (San Miguel Group). GrandCru acquired the San Miguel Group claims in 2004 for \$650,000 plus the 2% NSR.

NSR 4: A 1% NSR to be obtained as a result of producing and selling gold, silver and other ores covering the mining concessions "Los Reyes Dos", title 214131; "Los Reyes Tres", title 214302; "Los Reyes Cuatro", title 217757; "Los Reyes Cinco", title 216632; "Los Reyes Seis", title 225122, "Los Reyes Siete", title 225123; "Los Reyes 8", title 226037; "Los Reyes Fracc. Oeste", title 210703; "Los Reyes Fracc. Sur", title 212758; and "Los Reyes Fracc. Norte", title 212757, in favor of Desarrollos Mineros San Luis, S.A. de C.V. or its designee.

NSR 5: A 1% NSR to be obtained as a result of producing and selling gold, silver and other ores covering the mining concessions "Norma", title 177858; "Nueva Esperanza", title 184912; "San Miguel", title 185761; "San Manuel", title 188187; "El Padre Santo", title 196148; "El Faisán", title 211471; "Santo Niño", title 211513; "San Pablo", title 212752; "San Pedro", title 212753; "Patricia", title 212775; and "Martha I", title 213234, in favor of Desarrollos Mineros San Luis, S.A. de C.V. or its designee.

NSR 6: A royalty between 2% and 3%, based on the prevailing gold price as indicated below, on the NSR from producing and selling gold, silver from the lots covering the mining concessions: "Prolongación del Recuerdo", title 210497; "Prolongación del Recuerdo Dos", title 209397; "Arcelia Isabel", title 193499; "Dolores", title 180909; and "La Victoria", title 210803, in favor of Desarrollos Mineros San Luis, S.A. de C.V. or its designee. For gold prices US\$499 per Troy ounce or less, the royalty is 2.00% and for gold prices US\$500 or more, it is 3.00%.

Figure 4-4: Property royalties by concession map

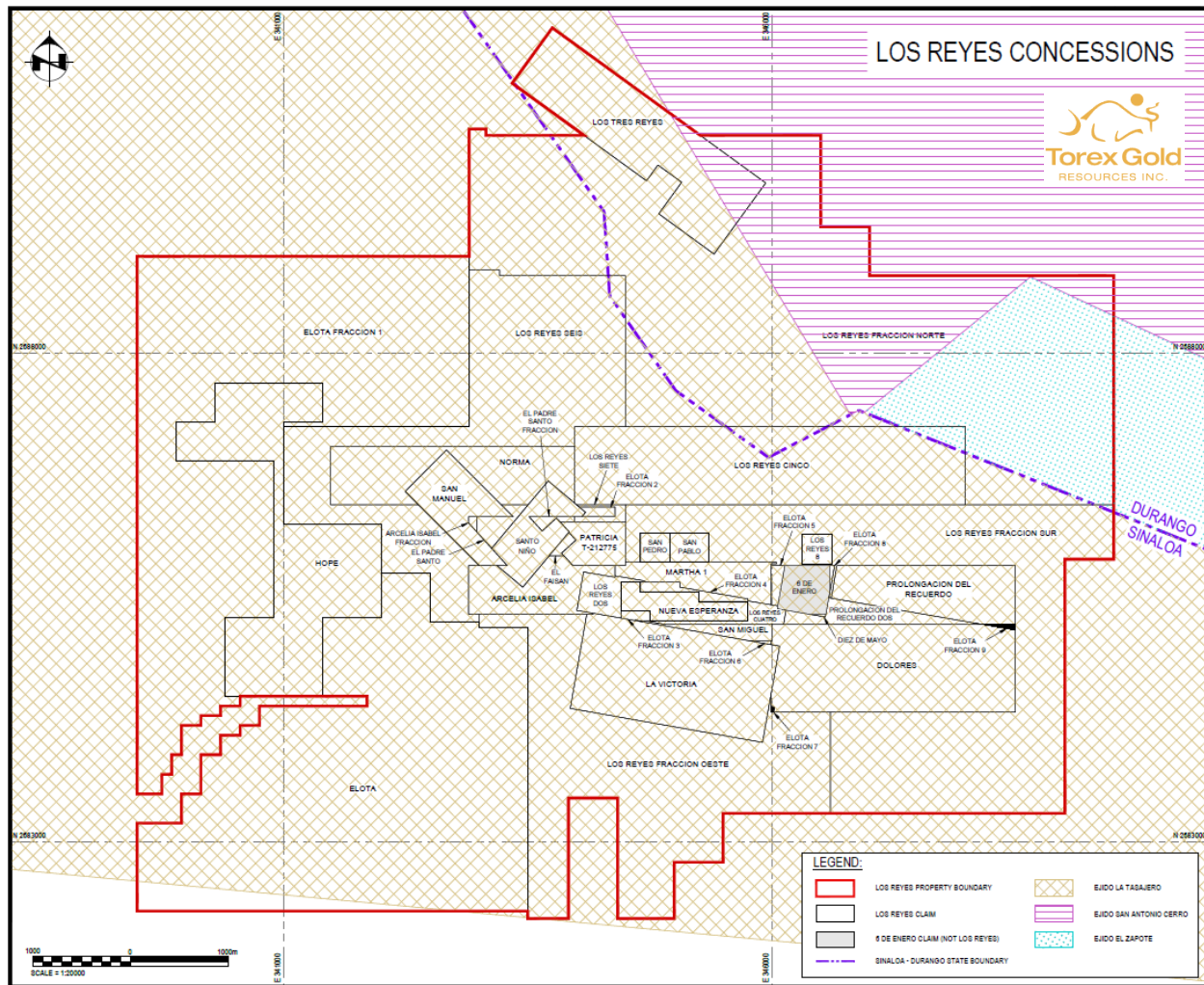


Source: Torex, 2024

4.3.3 Subsequent Concession Acquisition

Subsequent to the effective date of this report, Torex completed the purchase of the Hope Concession, which sits on the western side of the Property and abuts the existing concessions as displayed in Figure 4-5, below. As this concession was acquired after the effective date of this report, it is not included in and does not form part of any of the scientific and technical information disclosed herein. Any reference herein to the Los Reyes project, property, concessions, claims or claim blocks (including any reference in tables, figures and diagrams), does not include the Hope concession and should not be interpreted to include the Hope concession unless otherwise stated.

Figure 4-5: Hope concession



Source: Torex, 2024

4.4 Surface Use and Disturbance Agreement

On March 3, 2020, Prime announced the signing of a long-term agreement (the "Agreement") with the representatives of the Ejido la Tasajera for surface use and compensation for disturbance of the Project area. The Agreement, which has an initial term of 15 years and can be extended for an additional 15-year period, includes access and land use for exploration, engineering, construction, commissioning, and commercial operations. Upon execution of this agreement, Minera Amari, a wholly-owned subsidiary of Prime made an initial payment of US\$38,300 to Ejido la Tasajera. In September 2023, the area covered by this Agreement was doubled from 900 to 1,800 ha under a second agreement covering an additional 900 ha (the "2023 Agreement, and together with the 2020 Agreement, the "Agreements"). The principal terms and conditions of the 2020 Agreement are as follows:

- For an initial period of three years while conducting exploration, Minera Amari agreed to pay an upfront fee of MXP\$600,000 (US\$32,967), as noted above, that included the exploration work completed over the preceding six months and a three-year prepayment of three MXP\$200,000 annual payments.
- Minera Amari has exercised its right to extend the exploration period and makes annual payments of US\$30,000 during the exploration phase.
- Minera Amari has the right to initiate construction of a mine at any time.
- Upon commencement of commercial production, the annual payment increases to US\$200,000, paid in semi-annual installments of US\$100,000. The payments are subject to customary indexing for inflation. Payments due to the Ejido La Tasajera during commercial production are subject to adjustments based on Unidad de Medida de Actualización, the official Mexican index for the adjustment of government pensions, social security payments, taxes, etc.
- During commercial production, Minera Amari will also make, collectively, a US\$15,000 annual “apoyo” or gift distributed to the local families of Ejido La Tasajera.
- During the dry season months, Minera Amari will arrange to haul water to the La Tasajera village if requested.
- Minera Amari also intends to maximize employment of qualified local and Ejido La Tasajera residents in its activities with individuals having the necessary skill levels and capability.
- The Agreement is fully transferable without further approval of the Ejido La Tasajera.

The principal terms and conditions of the Agreements are substantially similar, except for the payment terms. The payment terms in the 2023 Agreement provide as follows:

- During the exploration period, Minera Amari must pay MXN \$600,000 annually, and during the exploitation phase, USD \$60,000 annually. Minera Amari will also make nominal payments for drilling activities.

In addition to the Agreement, an arrangement has also been completed that establishes specific, non-material payments to local individuals whose traditional land use within the Ejido becomes affected by Miner Amari’s exploration, construction, and production activities.

4.5 Environmental Liabilities

There are no known significant environmental liabilities on the property. Existing environmental liabilities on the property are limited and include mine adits, roads, small waste rock piles and one cyanidation vat, which operated in the 1950s, near the village of Guadalupe de Los Reyes (Borrastero, López, & Stevens, 2003; López 2009). No acid mine drainage from the existing adits or underground mine has previously been reported.

4.6 Permitting Requirements

It is understood that environmental permitting will involve environmental impact assessment, obtaining permission to utilize natural resources, and change of land use permit. Approval of these permits is a prerequisite for obtaining a construction permit, which is the final permit that must be approved prior to commencement of mining activities.

The primary law legislating environmental protection in Mexico is the *Ley General del Equilibrio Ecológico y la Protección al Ambiente* (General Law of Ecological Balance and Environmental Protection, (LGEEPA). This environmental law is administered by the Secretaría de Medio Ambiente y Recursos Naturales (Ministry

of Environment and Natural Resources, SEMARNAT), which is a branch of the federal government. SEMARNAT is also responsible for issuing land-use change permits for properties such as the Property that involve alteration of forested areas. SEMARNAT representatives in each state administer and address environmental impact issues as they are familiar with local issues and concerns.

The Procuraduría Federal de Protección al Ambiente (Federal Attorney for Environmental Protection, PROFEPA) is the agency responsible for enforcing SEMARNAT regulations. PROFEPA's main activities are to deal with complaints, conduct inspections, and in general verify compliance with all federal environmental laws and regulations. It imposes penalties for violations of environmental laws and regulations, and monitors compliance with any preventive and mitigating measures issued by it. PROFEPA also conducts environmental audits.

Water use and infrastructure, water quality, and the right to discharge process water (collectively referred to as water rights) related to the Property would be handled by Comisión Nacional del Agua (National Water Commission, CONAGUA). Land use permits are handled by local agencies in charge of the zoning and registration of land ownership.

Permitting is ongoing for exploration on the Property. Environmental permitting is supported by ongoing water quality monitoring and baseline studies.

4.7 Baseline Environmental Studies

Torex has not completed any baseline environmental studies for the project at the writing of this report and bases its assumptions on the study previously completed by Prime and its subsidiaries at the time of its acquisition.

The objective of the previous study was to analyze, characterize and describe the physical, environmental, and biological conditions of the environmental system where the Project is located. Another additional objective resulting from this study was to identify the most relevant physical-environmental attributes at the Property.

The Los Reyes concession area is not located within any designated *Área Natural Protegida* (Natural Protected Area, NPA) nor within areas identified as important for bird conservation, as recognized by the "*Sección Mexicana del Consejo Internacional para la Preservación de las Aves*" ("*Mexican Section of the International Council for Bird Preservation*"), and no *regiones terrestres prioritarias* ("priority terrestrial regions, RTP") are located within the Los Reyes area.

The authors of the Technical Report note the disturbance of the area due to previous mining activities, as well as agricultural and livestock impact.

4.8 Other Significant Factors and Risks

The authors are unaware of significant factors or risks that may materially restrict Torex from its right and ability to perform work on the Property.

5 Accessibility, climate, local resources, infrastructure, and physiography

The Property is located within the Sierra Madre Occidental (SMO) mountain range that trends north-northwest along the western coast of Mexico. The topography varies from steep mountain terrain in the east to river valley in the west. Elevations range from 230 to 1,400 masl.

The vegetation includes tropical bushes and shrubs within the river valleys and evergreens at higher elevations within the mountainous regions. The majority of the land surrounding the villages is developed for agriculture.

5.1 Property Access and Proximity to Population Centres

The Property is approximately 110 km by air and 200 km by road from the coastal city of Mazatlán, Sinaloa. The Property is within the municipality of Cosalá (population 17,012). The proposed plant site location is 27 km southeast of the city of Cosalá (population 7,888, INEGI 2020) by an all-weather road. The village of Guadalupe de Los Reyes is on the Property and was the site of Spanish colonial mining (Figure 5-1).

5.2 Climate

According to the Köppen-Geiger Climate Classification, this region is classified as Aw – Tropical Wet and Dry (De Jesus, A., Brena-Naarnjo, J.A., Pedrozo-Acuna, A., & Yamanaka, V.H.A., 2016). This region's weather is characterized by distinct wet and dry seasons, with most of the precipitation occurring in the high-sun ('summer') season (Britannica, 2020). Annual temperatures range from 16°C to 29°C. Precipitation reaches a peak in July, with 212.6 mm of rainfall in 2019 (World Weather online, 2020, paras.2-3). This can cause flooding along the river that can limit access from the Property to Cosalá (López, 2009). To improve access to the Property, widening and upgrading the road, as well as reinforcing the river crossings, will be necessary (Turner and Hunter, 2020).

The Property is accessible year-round. Recent improvements to the area's infrastructure, such as the high bridge over the Las Habitas River, have enhanced accessibility.

5.3 Infrastructure

Cosalá is proximal to four international airports. Mazatlán International Airport is located approximately 2.5 hours from Cosalá and has regular flights to and from many North America centres. The other three airports are: Culiacán International Airport, also located approximately 2.5 hours from Cosalá; Mochis International Airport, located approximately five hours from Cosalá; and General Guadalupe Victoria International Airport, also known as the Durango International Airport, located approximately 6 hours from Cosalá.

Cosalá has a regional airport, Aeropuerto de Cosalá, located northwest from the city centre. The road from Cosalá through Guadalupe de Los Reyes is the only maintained land access to the southeastern mountains in this part of the country.

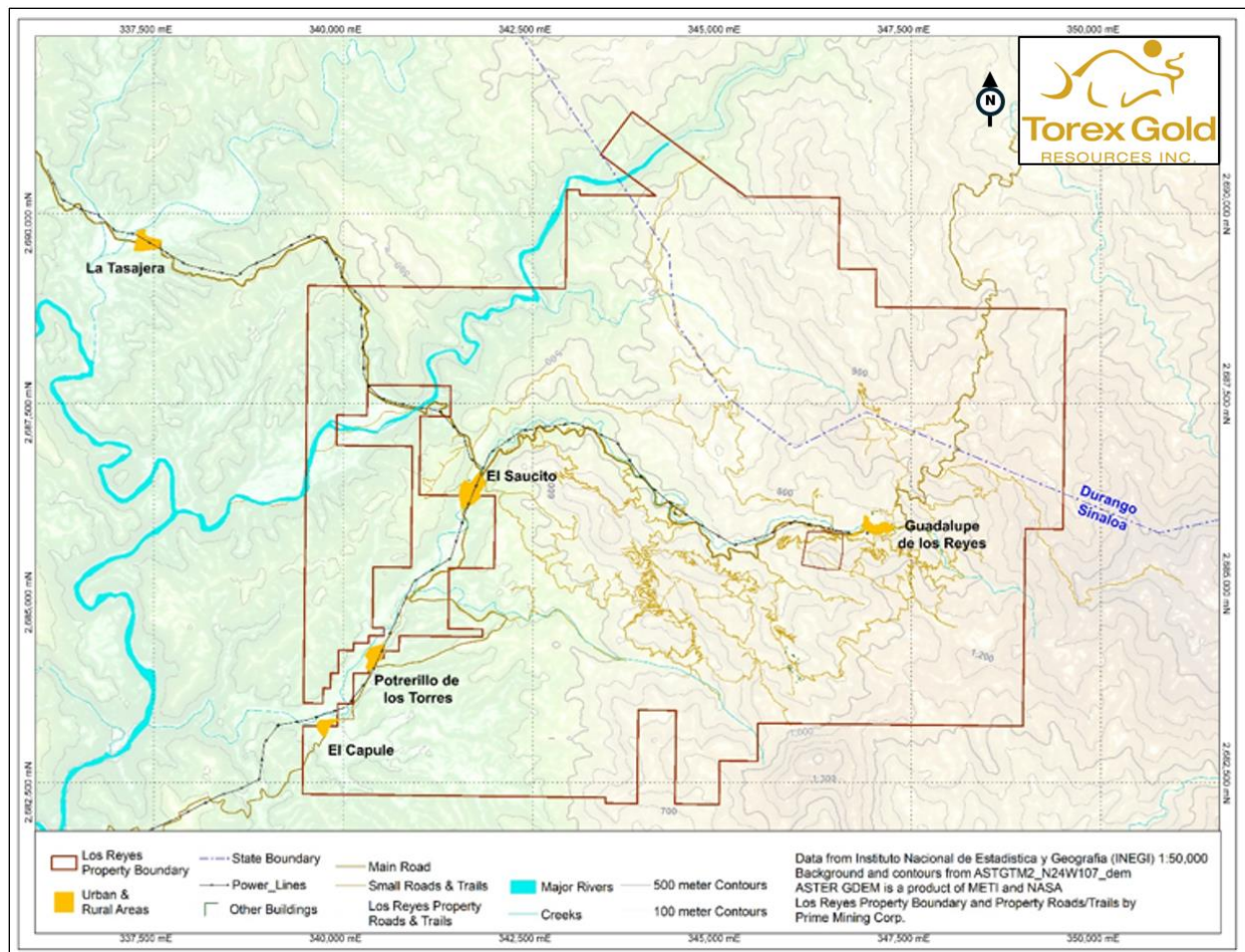
Local facilities include a hospital and health clinics; schools; banks; retail stores; hotels; restaurants; and tourism companies (sport utility vehicles).

The surrounding communities can provide labor, but a skilled workforce would have to be sourced from larger cities such as Mazatlán, Culiacán, and Durango, or imported from other countries. However, there is mining and minerals processing industry in and around Cosalá.

Local grid power is available from nearby towns and throughout the area villages. Water is available seasonally from the Ejido and surface run-off. Additional water sources would need to be developed to support mining operations (refer Fig 5-1).

The surface rights and agreements with the Ejido are sufficient for mining operations.

Figure 5-1: Infrastructure Map



Source: Prime, 2024

6 History

6.1 Introduction

The original data from NCM (as defined below), Vista Gold, Great Panther Silver Limited ("Great Panther") and Prime Mining ("Prime") is currently located in the Torex offices in Toronto, Canada. All of this data has been checked, compared and verified with the Prime digital files received in the acquisition of the Company. Meridian Gold ("Meridian") data is incomplete with only the reverse circulation ("RC") drill holes on the Zapote deposit completed, and these were the only drill holes from Meridian used in the Resource estimate. All the Vista Gold and Great Panther drill collars have been located and resurveyed previously by a surveyor contracted through Prime. Twenty-seven NCM drill collars were surveyed and another approximately 35 NCM drill collars were visually verified with handheld GPS coordinates collected. All NCM drill holes were collared on drill roads as shown on NCM maps and these locations compared well with Prime's LiDAR topography, indicating that the NCM drill collars are probably within 10 metres of their original coordinates.

6.2 Prior ownership and ownership changes on the property

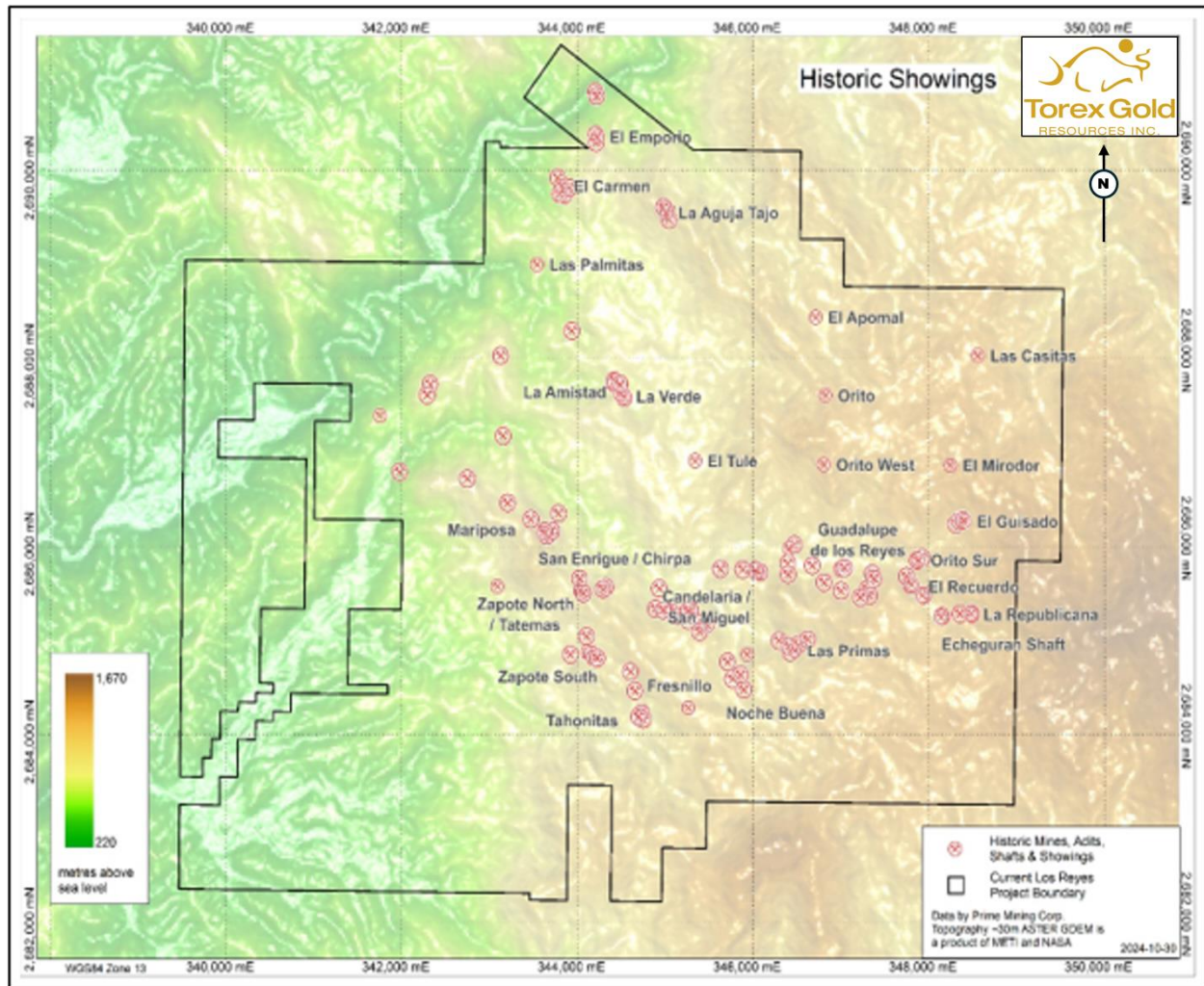
Since the discovery of gold and silver on the Property in approximately 1772, there have been several changes in ownership. The current ownership of the concessions that make up the Property are reviewed in detail in Section 4.

6.3 Exploration type, amount, quantity and results

6.3.1 Property Overview

Several mineralised areas have been identified, including Mariposa; Zapote North; Zapote South; Tahonitas; San Miguel West (previously La Chiripa and San Enrique); San Miguel East; Fresnillo; Guadalupe West; Guadalupe East (previously Laija); Las Primas; Noche Buena; El Apomal; Orito; Las Casitas; Mina 20/21 and El Mirador. Figure 6-1 shows the location of all the known historical mines, adits, shafts and showings over topography. Previous technical reports mention two additional areas: Tatemas and Candelaria (Borrastero, López, & Stevens, 2003; López, 2009). It was concluded during this study that Tatemas was an open stope in the northern extent of Zapote North, while Candelaria was in the current extents of San Miguel East.

Figure 6-1: Los Reyes historical showings



Source: Prime, 2024

6.3.2 Historical surface exploration

In the last three decades, several companies completed surface mapping, sampling (soil and rock), and geophysics exploration campaigns. These companies included Luismin (Gold Corp's Mexican subsidiary), NCM, Vista Gold, and Great Panther. A summary of the historical surface exploration completed before the acquisition of Prime by Torex is provided in Table 6-1 and further explained in Section 9.

Table 6-1: Summary of exploration (1992-2024)

Company name	Year	Surface sampling and/or exploration method (sample total)	Mineralised areas
Luismin	1990-2000	Unknown	Orito, La Palmita, El Mirador, Las Casitas, El Apomal
NCM	1992-2000	Soil (4,640 samples) Rock (1,448 samples) very low frequency electromagnetics and ground magnetics	Guadalupe, Zapote, San Miguel, Noche Buena, Tahonitas, Orito, Mariposa
Vista Gold	2011-2012	Rock (271 samples)	Guadalupe, Zapote, San Miguel, Noche Buena
Great Panther	2014	Rock (275 samples)	Zapote, San Miguel, Las Primas
Prime	2019-2023	Soil (4,251 samples) Rock (9,164 samples)	Zapote, San Miguel, Nocha Buena, Las Primas, Guadalupe

6.4 Historical Production on the Property

Historical production, estimated from several reports and memorandums, is approximately 1 Moz of gold and 60 Moz of silver from 2.7 Mt. The main historical mines were the Estaca and Descubridora mines and ancillary veins at Guadalupe de Los Reyes, La Candelaria mine (San Miguel East), Tatemas and La Chiripa mines (Zapote North and San Miguel West), and Pachuca and Zapote mines (Zapote South). Small operations in more recent times include the Gaitan mine (Zapote South) and Mariposa mine. There are numerous exploration adits and shafts that tested a variety of mineralised veins, some of which may have produced limited ore, but the history on these is sparse. Figure 6-1 shows the locations of the recognized mines, adits and shafts and Table 6-2 shows the estimated historical production.

Table 6-2: Historical production at the property

Main Areas Mined	Years Mined	Production Mined (tonnes)	Estimated Au Grade (g/t)	Estimated Au (oz)	Estimated Ag Grade (g/t)	Estimated Ag (oz)
Guadalupe De Los Reyes	1772 - 1871	1,500,000	12	578,713	900	43,403,445
Guadalupe De Los Reyes	1871-1938	875,000	8.8	247,464	521	14,650,972
La Candelaria	1930s	100,000	11	35,366	200	643,014
Tatamas/La Chiripa	1935-1944	170,000	12	65,587	250	273,281
Zapote South/Pachuca	1930s	NA	NA	NA	NA	NA
Zapote South/Gaitan	1988-1989	31,500	6	5,975	NA	NA
Mariposa	1980s	10,000	5	1,672	NA	NA
Total Estimate		2,686,500		934,776		58,970,712

7 Geological setting and mineralisation

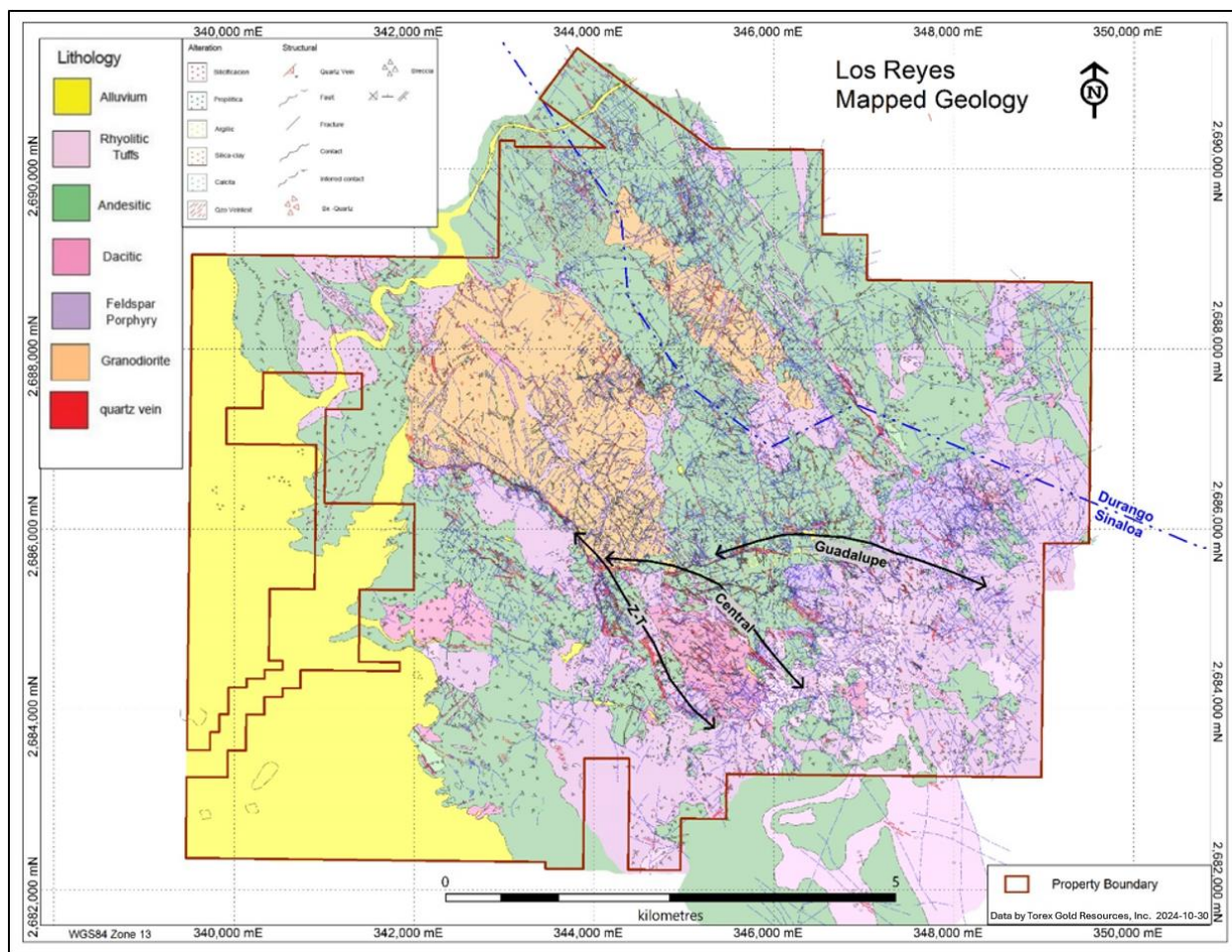
7.1 Regional geology

The Property is within the Sierra Madre Occidental (SMO) mountain range of the North American Cordillera that extends for hundreds of kilometres from central to northern Mexico in the Basin and Range province (Rossotti, Ferrari, López-Martínez, & Rosas-Elguere, 2002). The SMO is a large continuous sequence of volcanics from late Cretaceous to middle Tertiary in age (McDowell & McIntosh, 2012).

Near the Property, the volcanic sequence unconformably overlies a late Cretaceous- aged batholith. This overlying volcanic package is subdivided into Lower and Upper sequences that are separated by an angular unconformity. The Lower sequence spans from late Cretaceous-early Tertiary, is approximately 1 km thick, and is predominantly composed of intermediate (andesite) volcanics and more felsic units that are mostly dacitic to rhyolitic in composition. Intercalated sandstone and volcanic conglomerates of the Lower volcanic sequence are not significant on the property but do increase north of the Property towards Cosalá. The Upper Volcanic sequence which is deposited unconformably on the Lower sequence is composed of ash-flow and ash-fall tuffs that are rhyolitic to dacitic in composition. This sequence is over 1 km thick in higher elevation areas to the east (López & Ramírez, 2019; Turner and Hunter, 2020). Figure 7-1 shows significant detailed mapping of the Property. Within the Property the Upper sequence is found to be generally gently dipping and caps higher hilltops and ridges within and surrounding the district.

Several generations of felsic dykes cut the Lower Volcanic sequence. They include probable feeders for the rhyolitic to dacitic volcanic rocks in the upper part of the Lower sequence and some that are strongly flow-banded may be associated with the Upper Volcanic sequence eruptions. Gold and silver bearing quartz veins are commonly associated with the earlier rhyolitic to dacitic dykes suggesting that the mineralisation is temporally associated with the culmination of Lower Volcanic sequence volcanism (Allen et al., 2001).

Figure 7-1: Los Reyes property geology



Source: Prime, 2024

7.2 Mineralisation

Gold and silver mineralization at Los Reyes is hosted in Low Sulphidation Epithermal (LSE) quartz-adularia veins, stockworks, and breccias. Mineralization in the district occurs predominantly along three northwest to west-northwest oriented silicified structural corridors that form a horsetail-like structural complex. These mineralized structural corridors are named after the mineralized areas that they host, which include: 1) the Mariposa-Zapote-Tahonitas (ZT) trend; 2) San Miguel-Noche Buena (Central) trend, and 3) the Guadalupe trend. Several other subsidiary mineralized structures have been identified between the main mineralized structures (Las Primas and Fresnillo). Gold at Los Reyes is primarily hosted in electrum, while silver can be hosted in electrum, acanthite, and sulfosalt minerals. Sulfide content, consistent with the LSE mineralization style, is generally less than 2% in most mineralized zones. Elevated gold and silver grades are most commonly associated with fine-grained black acanthite or sulfosalt bands or clots within colloform-crustiform banded quartz-adularia veins. Additional "boiling zone" vein textures, such as bladed quartz after calcite, hydrothermal brecciation, and cockade textures, are also frequently observed, and an association of higher-grade mineralization with greenish quartz has also been noted throughout the district. Characteristic alteration includes vein-proximal quartz-sericite-adularia alteration, intermediate

illite-smectite alteration, and distal chlorite-epidote alteration. Intermediate and distal alteration generally occurs as selective replacement of phenocrysts rather than pervasive, texturally-destructive alteration

Table 7-1

7.2.1 Mariposa-Zapote-Tahonitas (Z-T) Trend

The Mariposa-Zapote-Tahonitas (Z-T) structure strikes to the north-northwest and dips at approximately 50° to the southwest. This trend shows evidence of previous workings, such as stopes, trenches, and adits.

Mariposa is the northernmost portion of the Z-T mineralised trend and has a length of approximately 1100m. The structure at Mariposa is at the contact between andesite and granodiorite with the rhyolitic stock. This structure is exposed on surface intermittently along the road that provides access to the historical mine workings (López, 2009). Mineralisation is associated with hydrothermal breccias ((Turner and Hunter, 2020); Table 7-1).

Zapote, which is divided into the North and South, is drill tested and is approximately 1.6 km in length and widths vary from 7 m to 66 m (Table 7-1). The hanging wall mineralisation of Zapote is gradual and consists of quartz veining, brecciation, moderate silicification and argillization. Mineralisation along the footwall is variable and consists of weak silicification and propylitic alteration. Mineralisation within the main zone is strongly associated with silicified breccia and contains quartz, calcite, and adularia veins (López, 2009; Turner and Hunter, 2020).

Tahonitas, which is southernmost along the mineralised trend, is over 950 m in length. The widths vary from 15 m to 45 m. The mineralised structure at Tahonitas dips between 45° and 60° to the southwest (López, 2009). Tahonitas is hosted at the contact between rhyolite dikes and andesites of the Lower Volcanic Sequence.

7.2.2 San Miguel-Noche Buena (Central) Trend

The San Miguel West, San Miguel East and Noche Buena areas occur along a regional northwest – southeast striking structure called the Central trend that moderately dips between 50° and 70° to the southwest. The San Miguel West and East areas have a combined length of approximately 1,450 m and widths vary from 9 m to 75 m. This trend also shows evidence of previous workings, such as stopes, trenches, and adits. The host rocks in the area are andesites of the Lower Volcanic sequence that were subsequently intruded by an argillic altered feldspar-hornblende-biotite porphyry dike. The mineralised zone is associated with the brecciated zones proximal to the dike, as well as along the structure (López, 2009; Turner and Hunter, 2020).

The Noche Buena zone is the southernmost extension of the Central trend and is hosted in a system of sub-parallel quartz veins and veinlets. This zone has an approximate length of 800 m and widths vary from 12 m to 55 m (Turner and Hunter, 2020).

7.2.3 Guadalupe Trend

The Guadalupe trend, which bifurcates to the east from the San Miguel West deposit, is subdivided into the Guadalupe East and Guadalupe West deposits. The trend, which has an east-west orientation, is strongly silicified and contains zones of silica stockworks. Guadalupe East and Guadalupe West have a cumulative length of 2,500 m and a true width that varies from 4 m to 68 m. The area was previously mined, with significant underground working that extend laterally by approximately 1.5 km and vertically to 400 m. The host rocks to mineralisation are intermediate volcanics (in Guadalupe West this is andesitic flows and tuffs, while in Guadalupe East it is predominantly feldspathic porphyry) of the Lower Volcanic sequence.

In Guadalupe East the main structure contains a steeply south dipping to vertical vein called Estaca which has seen historical mining. North dipping antithetic structures in the hanging wall of the Estaca vein are present within a 500 m long block bounded by 2 northwest structures. This block was historically called Laija and more recently described as a “flower structure”. East of Laija there are north dipping antithetic structures in the footwall of the Estaca vein; the larger antithetic veins were called San Manual and San Nicolas and these also reported historical mining.

7.2.4 Additional Mineralised Areas

Two mineralised areas: Fresnillo and Las Primas occur along subordinate subparallel structures to these main structures described above. Fresnillo is located between the Z-T and Central structures and the host rocks to mineralisation are rhyolitic and dacitic rocks. In Las Primas, which is east of the Central structure and south of the Guadalupe structure, the host rocks are andesites of the Lower Volcanic sequence that were subsequently intruded by a feldspathic porphyry and rhyolite/dacite intrusives.

Additional mineralised areas are also identified to the north and northeast of the main area of mineralisation. These mineralised areas include Palmitas, La Verde, Las Agujas, Orito, and Mina. Table 7-1 is a summary of mineralised areas and their characteristics. Table 7-1 is modified from Turner and Hunter, 2020.

Table 7-1: Summary of mineralised areas

Mineralised Area	Characteristics
Mariposa	Structure: Z-T Trend; Strike: From 320° in the southeast to 280° in the northwest; Dip: ~50° SW; Length: 1,100 m Structure Width (estimated): 15m to >40m; Mineralisation Style: Silicified host rock, border breccias, stockwork and veins; Lithological Contacts: Breccias along faults proximal to andesite to rhyolitic intrusive bodies, border breccias, and veins.
Zapote (North & South)	Structure: Z-T Trend; Strike: 290° to 350° in the south, 280° to 360° in the north; Dip: 48-56° W; Length: 1,600 m; Width (true): varies from 7 m to 66 m; Mineralisation Style: Silicified host rock, border breccias, stockwork and veins; Contacts: Sharp footwall contact and faulted stockwork and breccias to the hanging wall.
Tahonitas	Structure: Z-T Trend; Strike: 315° in the NW to 350 in the S°; Dip: 45° to 60° SW; Length: 950 m; Width (true): varies from 15m to 45m; Mineralisation Style: Silicified host rock and veins border breccias, stockwork and veins; Contacts: Sharp footwall, some veining to the hanging wall.
San Miguel West	Structure: Central Trend; Strike: 275°; Dip: 70° SW; Length: 600 m; Width (true): varies from 9 m to 75 m; Mineralisation Style: Silicified host rock, stockwork breccias and veins; Contacts: Sharp both walls
San Miguel East	Structure: Central Trend; Strike: 295°; Dip: 70° SW; Length: 850 m; Width (true): varies from 9 m to 75 m; Mineralisation Style: Silicified host rock, stockwork breccias and veins; Contacts: Sharp both walls
Noche Buena	Structure: Central Trend; Strike: 320° to 335°; Dip: 50° to 60° SW; Length: 800 m; Width (true): varies from 12 m to 55m; Mineralisation Style: Silicified host rock, breccias and veins; Contacts: stockwork footwall and stockwork to the hanging wall.
Las Primas	Structure: between Central and Guadalupe Trends Strike: 310° Dip: 65° - 80°SW to 80° NE; Length: 480 m (drill defined); Width (true): 1 m to 10 m based on drilling to date; Mineralisation Style: a number of parallel to divergent structures across a 450 m width

Mineralised Area	Characteristics
	with stockwork Silicified host rock and breccias, stockwork veins; Contacts: Breccia to stockwork zone with quartz veining.
Fresnillo	Structure: between Z-T and Central Trends; Strike: 315-340°; Dip: 65°- 70° SW; Length: 750 m (drill defined); Width (true): 1 m to 10 m based on drilling to date; Mineralisation Style: Dacite host rock, breccias and veins(estimated): unknown; Contacts: Breccia zone with quartz veining.
Guadalupe East	Structure: Guadalupe Trend; Strike: 290° to 300°; Dip: 70° to 90° SW; Length: 1,500 m (drill defined); Width (true): varies from 4 m to 68 m; The widest area of mineralisation is called Laija, it has a strike length of 500m and is characterized by a trans-tensional flower structure in which multiple NE-dipping veins emanate from the SW-dipping Estaca vein, which is hypothesized to be the principal controlling structure in Guadalupe East. Mineralisation Style: Silicified host rock and breccias, stockwork veins; Contacts: Sharp in veins within breccia zone.
Guadalupe West	Structure: Guadalupe Trend; Strike: 270° to 280°; Dip: 65° to 80° SW; Length: 500 m (drill defined); Width (true): varies from 4 m to 55 m; Mineralisation Style: Silicified host rock and breccias, stockwork veins; Contacts: Sharp in veins within breccia zone.
Orito	Structure: Orito Trend; Strike: 330°; Dip: 66°- 85° NE; Length: 3,000 m; Width (estimated): 1 m to 10 m based on the width of brecciated zones; Contacts: Breccia zone with quartz veining and iron oxide alteration, kaolinization.
Gavilanes	Structure: Outcroppings (structure not defined); Strike: 270°; Dip: 63°-70° S; Length: 500 m; Width (estimated): from 1 m to 8 m based on mapping; Contacts: Breccias in hanging wall of Rhy Dk.
La Verde/ El Tule	Structure: Fault Zone; Strike: 330° Dip: ~60° SW; Length: 1000 m Structure Width (estimated): 1-10 metres based on mapping; Mineralisation Style: border breccias, stockwork and veins; Lithological Contacts: Breccias along faults proximal to andesite to rhyolitic intrusive bodies or granodiorite contact border breccias, and veins
Catanos/Republicana	Structure: San Manuel Vein Trend part of Guadalupe Trend; Strike: 270° to 300°; Dip: 55° to 70° NE; Length: 1,500 m (cumulative); Width: varies from 4 m to 10 m; Mineralisation Style: Silicified host rock and breccias, stockwork veins; Contacts: Sharp in veins. NE trend cut the E-W system.
Las Palmitas (El Carmen Mines)	Structure: Fault Zone; Strike: 310°; Dip: NE; Length: 350m; Width (estimated): unknown; Contacts: Quartz vein associated to Rhy Dk.
Las Agujas	Structure: Orito Trend; Strike: 330°; Dip: 60-75° NE; Length: part of the Orito trend (3000m); Width (estimated): 1-3 m; Contacts: Quartz vein associated to Rhy Dk.
Mina	Structure: on strike with Orito Trend; Strike: 340-355°; Dip: 70-85° NE; Length: (unknown); Width (estimated): unknown; Mineralisation Style: border breccias, stockwork and veins; Contacts: Breccias along faults proximal to rhyolitic intrusive bodies.

8 Deposit types

8.1 Deposit model

The mineralised zones are characterized by a LSE system containing silica veins, stockworks, and breccias. These zones are generally formed in felsic subaerial complexes in extensional strike slip structural settings. Low-sulphidation gold deposits are associated with magmas where mineralisation occurs several kilometres above the intrusion, and display certain alteration assemblages (Cooke & Simmons, 2000). The intrusions are likely the result of tectonic activity, such as plate subduction and extension. Deep hydrothermal fluid flow systems comprised of meteoric water, as well as near surface systems such as hot springs, are the sites of mineralisation. Mineral deposition takes place as the fluids undergo cooling by fluid mixing, boiling and decompression (Cooke & Simmons, 2000).

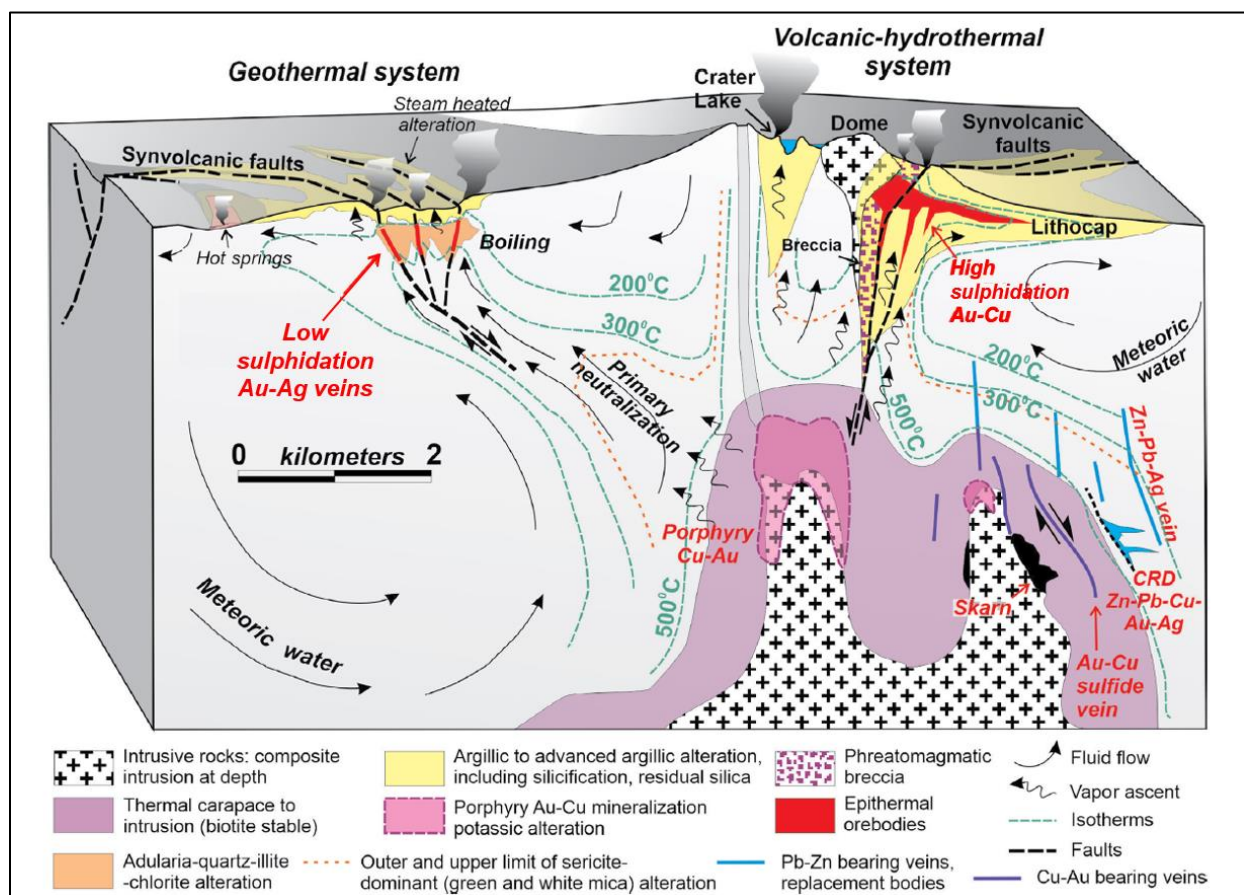
Distinguishing characteristics of LSE deposits are shown in Table 8-1 (Cooke & Simmons, 2000). A schematic cross-section diagram of a LSE system is shown in Figure 8-1 (Rhys et al, 2020).

Table 8-1: Characteristics of low-sulphidation epithermal deposits

Characteristics	Descriptions
Size of largest deposit	~90 t gold in Florida Canyon
Age	Oilocene, Miocene
Ore Bodies	Veins, stockwork, disseminations
Vein Textures	Brecciates, crustiform, colloform, lattice
Tectonic setting	Magmatic arc with transtensional faults or Basin and Range faults
District setting	Volcanic centres localized by crustal fractures
Igneous association	Calc-alkaline, subduction related or bimodal, mantle derived
Igneous composition	Mafic-intermediate, intermediate-felsic
Host rocks	Calcareous and siliceous sedimentary rocks, metasedimentary rocks, & volcanic rocks
Depth of formation	Shallow, 0 to 2 km
Mineralisation	Discordant $\pm$ strata bound; sinters are stratiform
Alteration types	Phyllic, argillic and opaline silica near surface
Open-space filling minerals	Quartz, adularia, bladed calcite, fluorite, pyrite-marcasite, sulfides
Ore minerals	Pyrite/marcasite, sulfosalts, base metal sulfides, electrum
Residence of gold	Free, inclusions and solid solution in pyrite-marcasite, sulfosalts
Landscape geochemistry	Along fracture zone near magmatic centre
Geochemical signature	Gold, silver, arsenic, antimony, mercury, selenium, barite, manganese, $\pm$ base metals
Iron mobility	Introduced, generally weak
Gold – Silver ratio	Low, variable, up to 2

Characteristics	Descriptions
Base metal content	Low or variable, 100 ppm to 3%
Formation temperature	~250°C to 100°C
Ore fluid chemistry	Low to moderate salinity, low CO ₂ , H ₂ S
pH of mineralizing fluid	Near neutral
Gold transport	Bisulfide complex
Source of H ₂ O	Meteoric, ± magmatic
Source of CO ₂	Igneous or carbonate rocks
Source of H ₂ S	Magmatic, sedimentary rocks
Depositional mechanisms	Boiling and mixing, ± sulphidation, ± oxidation

Figure 8-1: Schematic of epithermal deposits



Source: adapted from Rhys et al., 2020

9 Exploration

9.1 Introduction

Exploration activities have been undertaken by the previous owners of the Property. Torex has not completed any exploration activities on the Property. Below is a summary of the exploration activities undertaken by Prime between 2020-2024.

9.2 Grids and surveys

The Coordinate System used is WGS84 UTM Zone 13 North. The Project's control points are tied into control points of the National Institute of Statistics and Geography (INEGI), the National Institute of Statistics and Geography, an autonomous agency of the Mexican Government. In March 2021 accompanying a helicopter supported geophysical survey, LiDAR digital elevation data was collected by Pioneer Exploration Consultants Ltd., along 1,050 line-kilometres, with 50-metre line spacing over most of the Property, resulting in 20–50-centimetre (cm) resolution data. Actual elevations occurring in the survey block ranged from 325 to 1475 masl. In April 2022 and May 2024, a contractor (Unmanned Aerial Services Incorporated) performed LiDAR surveys with an Elios 3 drone, a handheld and remote controlled small, tracked unit to the extent possible of +45 adits, shafts, and underground workings within the main area of interest at the Project. This data was tied into control points and the data added to the geological and resource model as historically mined voids.

9.3 Geological mapping program

Prime personnel undertook geological mapping in late 2020 to early 2024, with mapping covering more than 60 km². Regional geological mapping (1:5,000 scale) worked towards covering the complete claim group and complimented with detailed geological mapping (1:2,000 scale) over areas previously identified as containing mineralisation and new areas identified during the regional mapping.

High-potential areas identified through mapping are:

- **Tahonitas (southeast)** – This area is at the southernmost end of the Z-T Trend. Mapping has defined the extension of the structure more than 500 m further to the southeast than where drill defined at the end of 2024. The structure contains faulting, quartz breccias, quartz veins and veinlets with strike like the main Z-T Trend.
- **Las Primas** – Mapping has better defined the prospect with a rhyolitic dike with two quartz veins inside the dike and quartz stockwork zone on the border of dike in contact with the volcanic andesite rock.
- **Fresnillo** – This area is between the Z-T and Central Trends. Mapping defined structures as quartz breccias, quartz veins and veinlets with strike like Zapote and Noche Buena, the difference is the structures/mineralisation are hosted in a dacite intrusive. Surface sampling returned elevated gold and silver values.
- **300 Trend (Zapote West)** – Mapping west of Mariposa identified a structural system with quartz/breccia veins/veinlets trending 300 degrees, with a strike length of more than 500 metres. It is located on the west side and joins into the Z-T trend at Zapote South. Surface sampling returned elevated gold and silver values up to 0.811 g/t Au.

- **Orito trend** – Mapping defined 2-3 regional structural systems NW40-50SE, with zones of quartz breccia, quartz vein-veinlets and stockwork, evidence mineralisation like those encountered in other areas. This area appears to be higher in elevation, but further exploration is required.

On the north-central part of the Project, north of San Miguel East and Guadalupe East, there are structures demonstrating evidence of mineralisation, such as quartz breccias quartz vein-veinlets with strike similar to the Orito trend and others associated to rhyolitic dikes with strike E-W. Further exploration is required on these structures.

9.4 Rock and soil sample programs

In late 2019, Prime personnel began systematically trenching and sampling road-cuts over approximately 5,000 m. The program focused on sampling across outcrops along known mineralised structures. The program was designed to obtain continuous surface grades in select areas that, when combined with historical drill hole data, assisted to advance the geological model. Prime continued to collect rock samples which include adit, channel, chip, float and grab samples as part of the geological mapping program. Table 9-1 shows the total number of rock samples taken and the number of samples ≥ 0.2 g/t Au and ≥ 20.0 g/t Ag. All trench and roadcut samples were 1.5 m in length. Adit samples varied from 0.5m to 2.0m. QA/QC (Section 11.1.4) samples include standards, blanks, and check assays on selected mineralised samples and the shoulders to mineralised zones. Figure 9-1 shows the locations of the trench, road-cut and adit samples and Figure 9-2 shows the locations and range of Au concentrations of the mapping samples.

In addition, 4,251 soil samples were collected by Prime along with 157 duplicates, Figure 9-3 shows the locations. Sampling points were defined as equidistant locations arranged along parallel lines, with variable spacing determined by the project manager. Each assigned sampling point contained a pit or hole excavated to a specified depth until reaching the "B Horizon,"; approximately 0.5 to 1 kg of material was collected from this horizon. The material was sieved using a -80 mesh to retain the finer fraction, with organic material discarded. Subsamples were placed in small plastic containers for pre-laboratory analysis using the TerraSpec Halo and KT-10 (Minera Amari, 2025). A geochemical study was undertaken by Heberlein Consulting at the request of Prime in November 2021 to review historical soil samples along with Prime rock and soil samples in the main area at the Property. This study of historical and new soil and rock geochemistry successfully identified the known mineralised structures and identified potential targets for further investigation, including the Fresnillo and Las Primas targets (Heberlein, 2021).

In addition to engaging external consultants, Prime evaluated geochemical data internally to refine exploration targeting in the Los Reyes district. Data collected showed a strong spatial correlation between Au and Ag, As, and Sb in the Z-T and Central Zones, with As and Sb anomalism observed at lateral distances up to 100m from Au-Ag mineralisation. A systematic increase in Cu, Pb, and Zn with increasing depth typical of many low sulphidation epithermal Au-Ag deposits was not confirmed in the Z-T and Central zones; this is likely due to the relative lack of deep drilling in these target areas. However, mineralisation at Guadalupe East does show a tight spatial correlation between Au, Ag, and Cu, in addition to displaying similar broad As and Sb anomalism to the Z-T and Central Zones. The presence of elevated Cu, along with generally higher Ag/Au ratios at Guadalupe East, may suggest that mineralisation in this zone formed at higher temperatures than in the Z-T and Central zones. All zones of potentially economic Au-Ag mineralisation discovered to date show evidence of K addition and corresponding Na-Ca depletion within and immediately adjacent to precious metals mineralisation. This is consistent with the presence of adularia in many areas of high-grade Au-Ag mineralisation and the observed alteration of feldspars to illite and sercite in the surrounding wallrock.

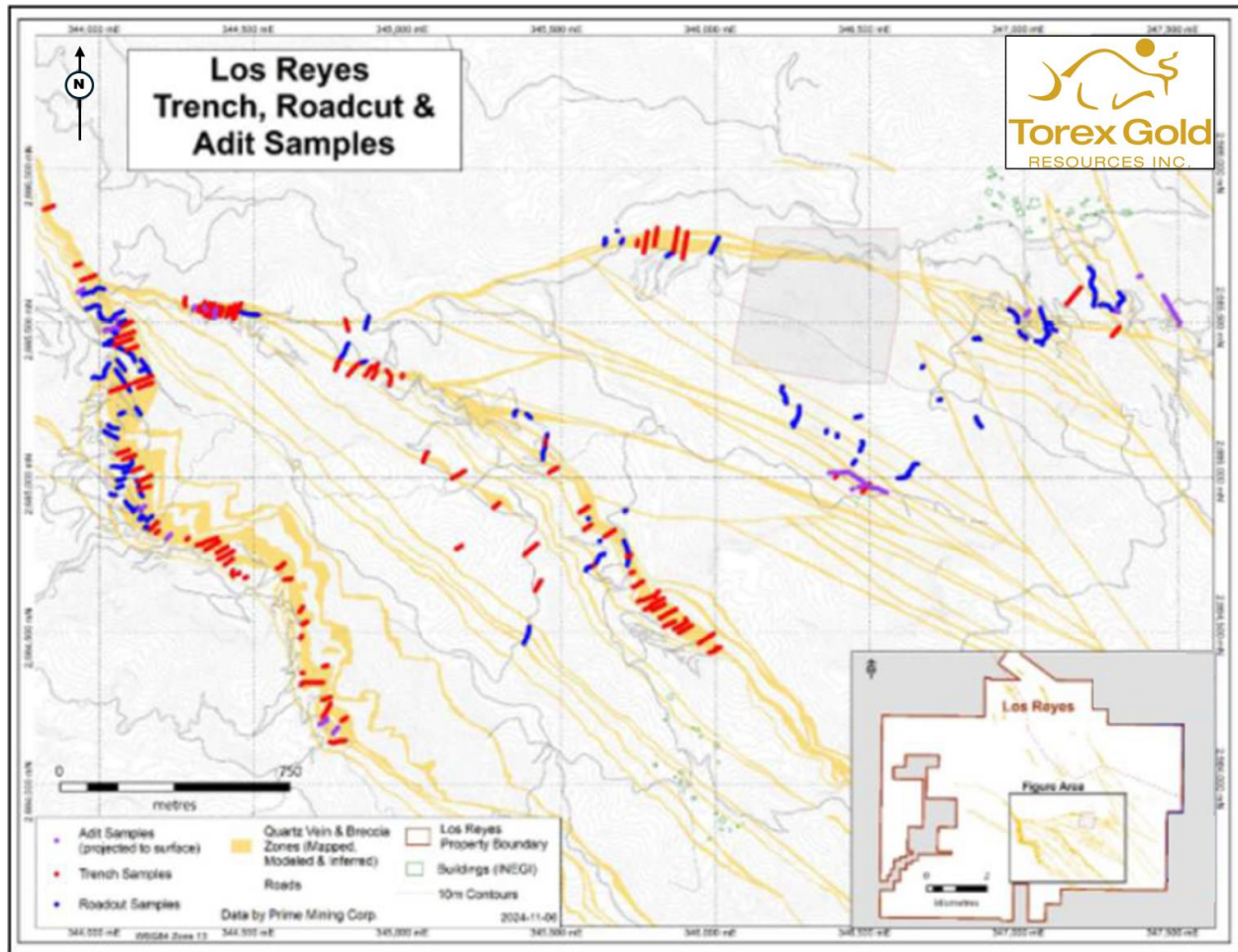
9.5 Geophysics

An airborne magnetic-radiometric survey was completed in March 2021 and interpreted by Campbell and Walker Geophysics Ltd. A total of 1,056 line-kilometres of combined magnetics, gamma-ray spectrometry and VLF-EM were successfully acquired and processed from a survey comprised of a 50 x 500 m grid, with traverses flown between 30 and 210 degrees and control/tie lines flown perpendicular or 120 – 300 degrees. Mean terrain clearance was 85.1 m. A series of transformations and derivatives applied to the magnetics highlighted the structural fabric of the Property. Northwest- to west-northwest-striking magnetic linears or trends correlate in large part to the known mineralised structures. Additional linears and/or inferred structures are apparent southwest from the Zapote–Tahonitas trend, and may cut through, extending further northwest; one such feature is provisionally termed the ‘Zapote 300’ trend. A number of pronounced Th/K ratio ‘low anomalies’ are mapped by the radiometrics; three of these correlate very well to the known mineralisation. Three other Th/K lows appear to coincide with established target zones, Orito, La Verde/El Tule and Las Palmitas zones. Overall, the gamma-ray spectrometry mapped positive correlations to anomalous mineralisation (Campbell, 2021). The VLF-EM results highlighted several prominent shear zones, with associated offsets, breaks. Power transmission lines cutting across/through the survey area presented issues, but overall, the VLF-EM provided some benefit and confirmation of regional structural trends.

Table 9-1: Rock sampling program

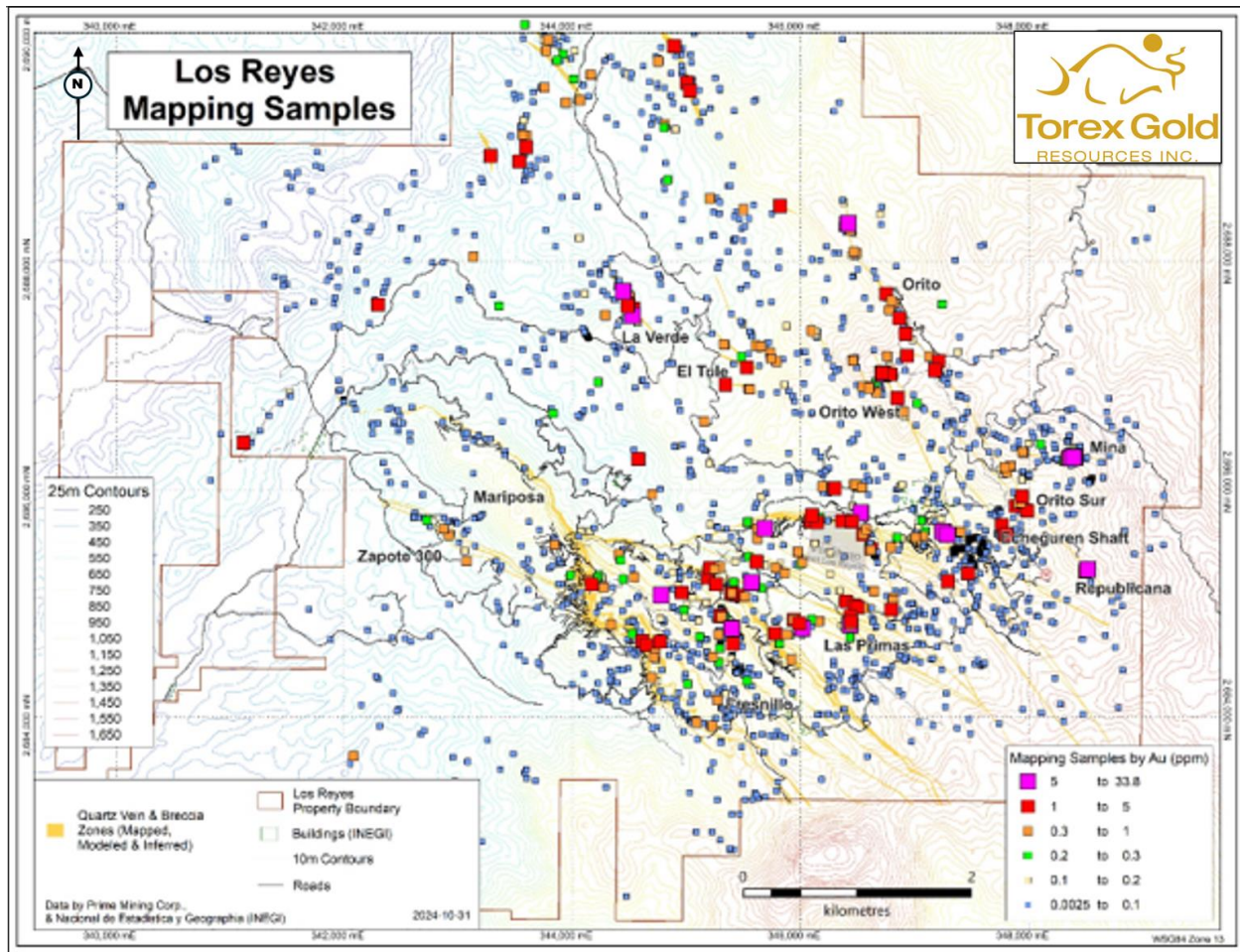
Sample Type	Sample Count	>= 0.2 g/t Au	>= 20.0 g/t Ag
Trenching	2894	827	408
Roadcut	2083	522	240
Adit	585	174	105
Chip/channel	2749	276	120
Grab	41		
Float	19		
Undefined	3		
QA/QC	790	N/A	N/A
Totals	9164	1799	873

Figure 9-1: Los Reyes trench, roadcut & adit sampling



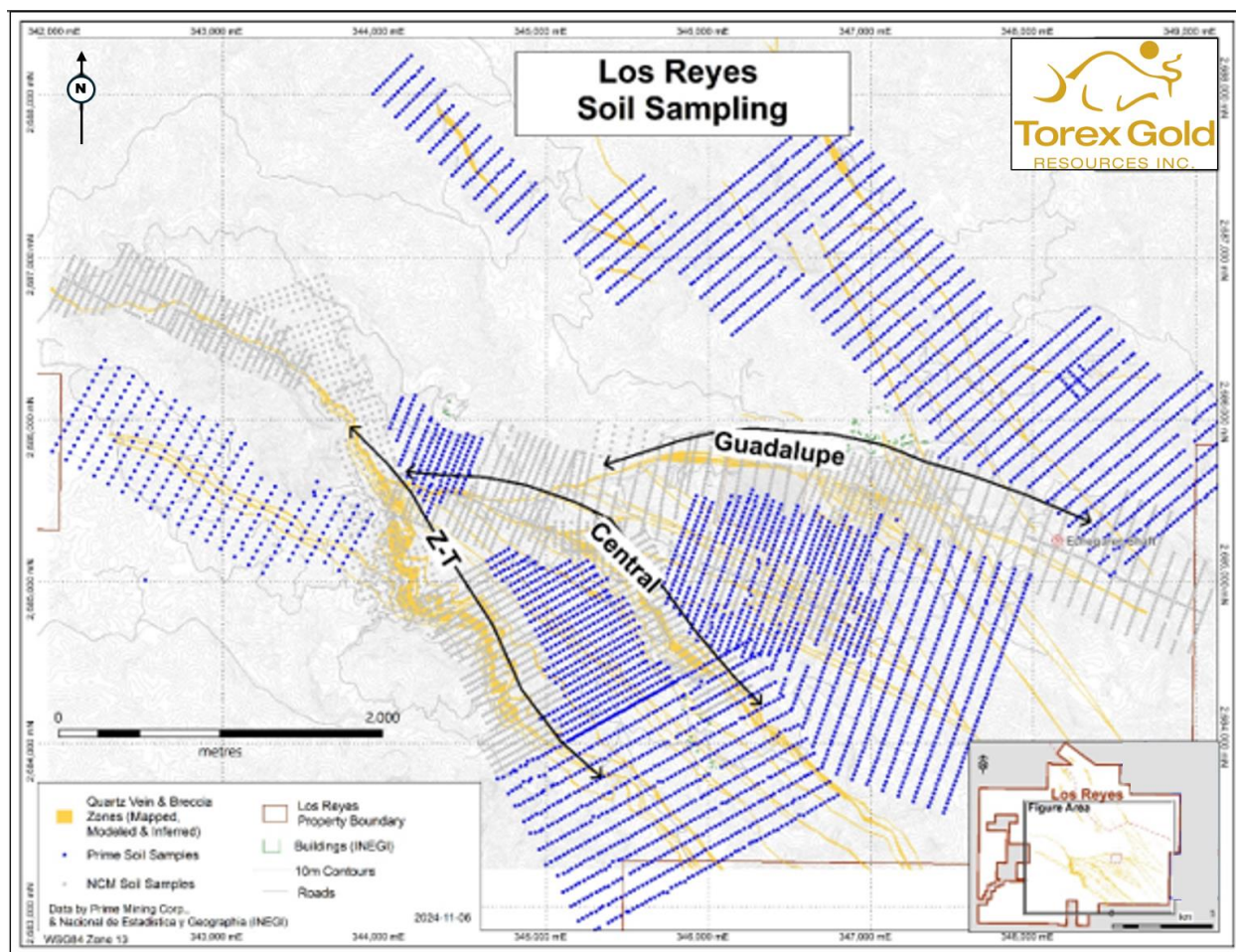
Source: Prime, 2024

Figure 9-2: Los Reyes mapping samples locations



Source: Prime, 2024

Figure 9-3: Los Reyes Soil Sampling



Source: Prime, 2024

9.6 Spectral mineralogy and magnetic susceptibility

Prime used a TerraSpec Halo mineral identifier, an instrument developed by ASD, to systematically collect mineralogical data from drill core and surface samples across the Property. The instrument measures the visual, near-infrared, and short-wave infrared ("VIS-NIR-SWIR") regions (350-2500 nanometres) and is used to assist in exploration vectoring by identifying minerals associated with hydrothermal alteration. Spectral data are collected from cleaned, dry drill core at regular intervals, typically at 3 to 6-metre increments in non-mineralised host rock and at 0.7 to 1.5-metre increments in intervals showing visible alteration or quartz veining. The raw spectral data are then processed using the Spectral Geologist software, developed by CSIRO for analysis of VIS-NIR-SWIR data, to determine principal alteration minerals and to calculate additional scalars such as illite crystallinity and kaolinite crystallinity. The December 31, 2022, TerraSpec dataset includes analyses from 44 drillholes and 1,056 surface samples.

Spectral data collected to date suggests the Property is characterized by an alteration zonation pattern typical of many low-sulphidation epithermal Au-Ag deposits (Hedenquist, et. al., 2000). Quartz-adularia-sericite (muscovite) alteration is observed within hydrothermal upflow zones and is often present at lateral distances up to 10-30 m from Au-Ag mineralisation. Illite alteration is dominant at intermediate distances

(generally 30-75 m), typically with increasing illite crystallinity and a transition to more muscovitic compositions as the mineralised zones are approached. Distal alteration is typically characterized by a transition from illites to smectites with increasing distance from mineralisation, and chlorite-epidote is dominant at distances greater than 100 m from the mineralised zone. While near-paleosurface expressions of hydrothermal alteration such as sinter and opaline silica are generally absent at the Property, kaolinite is occasionally observed at high elevations and may reflect the remnants of steam-heated alteration associated with the Los Reyes system.

Magnetic susceptibility measurements were also collected from cleaned, dry drill core on the same intervals as the TerraSpec Halo data, using a KT-10 handheld magnetic susceptibility metre developed by Terraplus Inc, and Georadis S.R.O. Andesites and andesitic tuffs across the Property are generally weakly magnetic, and magnetite destruction associated with hydrothermal alteration can therefore be an important vector for mineralisation in these host rocks. Magnetic susceptibility data collected suggests that de-magnetization in andesitic host rocks can be observed at lateral distances up to 20-30 m from mineralised structures.

9.7 Remote sensing

Remote Spectral Geology is the measurement and analysis-interpretation of spectral satellite data to identify different rock types and surface materials, their mineralogy, and their mineralisation-alteration signatures. Prime engaged an external consultant to generate a set of spectral images to reflect the local geology, showing variations in rock types, mineral associations, mineral zoning/vectoring. Results to date have confirmed the regional geological trends and alteration.

10 Drilling

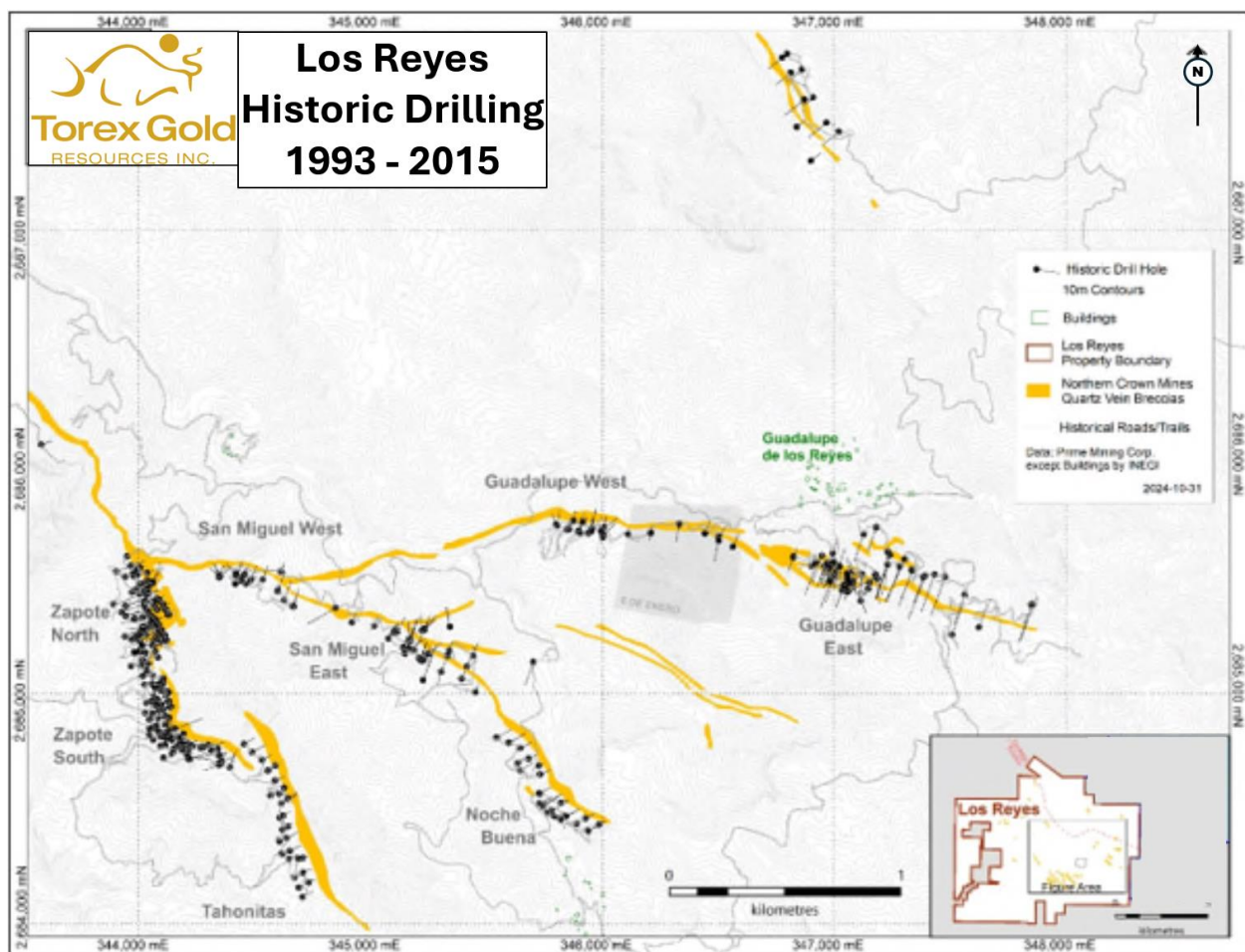
10.1 Introduction

The Project has been drilled by several operators since 1993 including NCM, Meridian Gold, Vista Gold, Great Panther, and Prime. Following the acquisition of Prime, Torex commenced with a data validation and relogging program. Torex has initiated a drilling program in 2026.

10.2 Historical Drilling

Historical drilling was conducted throughout the Property from 1992 to 2015 and is summarized by company in Table 10-1 and Figure 10-1 shows historical drill hole locations within the Property.

Figure 10-1: Map of Historic drillholes



Source: Prime, 2024

Table 10-1: Drilling summary by year and company

Mineralised Zone	Company	NCM					Meridian	Vista			Great Panther	Total
	Year	1993	1994	1996	1997	sub-total	2001	2011	2012	sub-total	2015	
Guadalupe	drillholes	-	-	48	31	79	-	10	8	18	9	106
	metres	-	-	6783.3	3765.2	10548.5	-	1470.1	1481.6	2951.7	1493.6	14993.8
Zapote North & South	drillholes	28	38	22	113	201	7 ¹	-	15	15	11	234
	metres	2336.6	3375.7	1991.3	8024.6	15728.2	1082.0	-	1886.5	1886.5	1156.8	19853.4
San Miguel	drillholes	-	-	-	34	34	13 ²	-	11	11	17	75
	metres	-	-	-	3674.4	3674.4	1243.5	-	1852.8	1852.8	2313.1	9083.7
Noche Buena	drillholes	-	4	9	12	25	-	-	4	4	4	33
	metres	-	246.9	1016.7	1328.9	2592.6	-	-	729.0	729.0	541.6	3863.1
Tahonitas	drillholes	-	-	33	-	33	-	-	-	-	-	33
	metres	-	-	2258.0	-	2258.0	-	-	-	-	-	2258.0
Orito	drillholes	-	-	8	-	8	3 ²	-	-	-	-	11
	metres	-	-	1140.1	-	1140.1	374.9	-	-	-	-	1515.0
Mariposa	drillholes	-	-	-	1	1	-	-	-	-	-	1
	metres	-	-	-	166.1	166.1	-	-	-	-	-	166.1
Total	drillholes	28	42	120	191	381	23	10	38	48	41	493
	metres	2336.6	3622.6	13189.4	16959.2	36107.8	2700.4	1470.1	5949.8	7419.8	5505.1	51733.1

Notes:

- Two of the seven Meridian drill holes have no assay or geological information.
- There is no available data, which includes assay and geological information, for these drill holes.

10.2.1 1993-1997 Northern Crown Mine (“NCM”)

NCM conducted RC drilling in 1993, 1994, 1996 and 1997. In 1993 drilling was contracted to Tonto Drilling Services Ltd. of Hermosillo, Sonora, Mexico. In 1994, the contract changed to Dateline International, S.A. de C.V. of Hermosillo. In 1996 and 1997 drilling was contracted to Dateline International, S.A. de C.V. and Layne de Mexico, S.A. de C.V., both based in Hermosillo. NCM Drilled 381 holes totaling 36,108 metres.

Most of the RC drilling was dry and used air for a medium to recover the drill chips but when ground water was encountered, a water recovery medium was occasionally used.

The processing and geological logging of the samples was conducted at the various drill sites by qualified geologists and geotechnicians who were either independent contractors or employed by NCM. The physical features and lithologic composition of each sample were recorded onsite, including alteration, mineralisation, and any observable structural evidence. Any underground workings intersected by the drilling were also documented on the drill log forms.

Collar locations, originally spotted by chain and pacing, were later located with a handheld GPS. The RC drill holes were not surveyed for down hole deviation. Samples were picked up from the NCM field camp by the assay laboratory and transported to their facility in Hermosillo, Sonora, Mexico (Allen and Thurston, 1997).

10.2.2 2001 Meridian Gold

Meridian entered into an agreement with NCM in late 2000 and conducted an RC drilling program in 2001. Meridian contracted the drilling to Layne de Mexico, of Hermosillo, Mexico, and completed 23 RC drill holes on the Property (Table 6-2). The drill tested sites at Guadalupe East, San Miguel West, Zapote North and Zapote South. Meridian collected samples from approximately 15 m above the mineralised/alter zone (hanging wall) and generally to the end of the hole. Bonder Clegg picked up samples at the field camp site and transported them to their laboratory in Hermosillo. Only the drill holes in the Zapote south area have geological logs and sample assay data, which includes 313 intervals.

10.2.3 2011-2012 Vista Gold

During 2011 and 2012, Vista Gold completed 48 diamond drill core holes on the Property. Drill collar locations within Zapote, Noche Buena, Guadalupe, and San Miguel were placed along section lines that spanned 25 m to 100 m apart, depending upon the area.

The diamond core drilling program was conducted under the supervision of Minera Cascabel. Vista Gold core drilling in 2011 and 2012 utilized HQ core to depths of approximately 110-150 m (rarely as shallow as 50 m) switching to NTW size core and occasionally BTW size core around 130 m depending on conditions. Vista Gold drilled 7,420 m in 48 core holes. The drill contractor was Energold de Mexico, S.A de C.V., of Mexico City, Mexico.

All collar locations were collected using a handheld GPS unit. Downhole surveys were completed on all drill holes. All the samples were kept in the secure area until picked up by ALS Chemex and delivered to Hermosillo for sample preparation and assay. Vista Gold submitted, 5,396 split core samples. All remaining core, and pulps, are currently stored in one of Prime’s warehouses in Cosalá, Mexico.

10.2.4 2015 Great Panther Silver Limited

In 2015, Great Panther drilled 41 confirmatory core holes for a total of 5,505 m. This includes 11 holes in the Zapote area, four holes in the Noche Buena area, 9 holes in the Guadalupe areas, and 17 holes in the San Miguel area. The drill program was supervised by Great Panther personnel. The drilling was carried out by Maza Diamond Drilling of Mazatlán, Mexico using a track mounted HTM 2500 drill rig generating HQ or NQ core.

The objectives of the drill program were to test the continuity of the mineralised structures and the associated gold-silver mineralisation with infill and confirmation holes, and to expand the mineralised zones with select step-out drill holes.

10.2.5 2021-2024 Prime Mining Corporation

From late 2020 to December 31, 2024, Prime drilled 218,819.4 metres in 744 completed drill holes. Total drill samples submitted are 133,440 representing approximately 192,486 metres of drilled rock. Some holes were not sampled or assayed in their entirety, particularly holes that were drilled from coincident collars that were fully assayed or areas with sufficient sampling of wall rock. The drilling was carried out by Maza Diamond Drilling of Mazatlán, Mexico using a track mounted HTM 2500 drill rig generating HQ or NQ core.

Table 10-2 summarizes the Prime drilling. The two drill holes started in 2020 are included in the figure showing the 2021 drill holes.

Table 10-2: Summary of prime drilling

Mineralised Zone		Prime Mining Corp.				Sub-total
		2021 ¹	2022	2023	2024 ²	
Guadalupe	Core drill holes	37	89	21	16	163
	Core metres	10,169.30	32,818.50	7187.25	5516.5	55691.55
	RC drillholes	5	16	0	0	21
	RC metres	1,127.35	3553.26	0	0	4680.61
	Total drill holes	42	105	21	16	184
	Total metres	11,296.65	36,371.76	7187.25	5516.5	60,372.16
Z-T	Core drill holes	56	75	81	67	279
	Core metres	11,472.55	18,039.80	27251.7	22819.9	79583.95
	RC drillholes	0	19	0	0	19
	RC metres	0	3,652.37	0	0	3652.37
	Total drill holes	56	94	81	67	298
	Total metres	11,472.55	21,692.17	27251.7	22819.9	83236.32
Central ³	Total drill holes	31	59	40	51	181
	Total metres	6,768.55	14,182.65	12676.6	20361.9	53989.7
Other	Total drill holes	24	9	41	8	82
	Total metres	5,301.98	2,314.50	11429.2	2175.55	21221.23
Total	Total drill holes ⁴	153	266	183	142	744
	Total metres	34,839.73	74,561.08	58544.75	50873.85	218819.41

Notes:

1. 2021: Two drill holes were started in December 2020, one of which was completed in 2020.
2. 2024: December 31.
3. Central: Drilling in the Central areas and other areas were core drill holes only.
4. Total Drill Holes: Drill hole count and metres are counted in the year the drill hole was completed.

Core holes, all started as HQ size core and forty-five drill holes were reduced to NQ size core. Forty of the drill holes (8,333.0 metres) were RC drill holes. Twenty-one RC drill holes were drilled in the Guadalupe area (4680.6 metres) and 19 RC drill holes were drilled in the Z-T area (3652.4 metres). Two drill holes (799.4 metres) were collared as RC holes, drilled to a predetermined depth and finished as HQ size core.

The first 321 drill holes were sampled in their entirety submitting sawed half core to the assay laboratory to minimize sample bias. Starting in August 2022, core intervals were selectively submitted for assay analysis. Drill collar alignment of drill rigs was done using a Reflex TN14 Gyrocompass™. A Sokkia™ i-X600 total station is used to survey the location. Downhole surveys were done every 20 metres on the way down and from the bottom of the hole back to the collar with a DeviShot™, multishot survey tool. Some core drill holes were selected for oriented core measurements using a wireless Devicore BBT tool.

Drill core is placed directly into the core boxes by the driller's helper. The core was logged by geologists into the software recording a complete description of the rock including lithology, alteration, key mineralisation, textures, veining, oxidation state, colour and rock quality designation ("RQD").

Prior to August 2022, maximum sample intervals were typically 1.5 metres in length. Subsequently, maximum intervals were typically 2.0 metres. Minimum sample intervals are typically 0.5 metres with rare exceptions. Prior to the core being cut and sampled it was photographed. Once sawn the half core with the red line is returned to the core box and the half core marked with the blue line is submitted for assay.

The RC drilling provided rock chips up to 3.0 centimetres in size and pulverized rock material. Drill runs are 1.525 metres in length (5.0 feet) which makes up each sample. A full run recovers an average of 27.3 kilograms of rock. Rock cuttings are collected straight from the cyclone in a bucket which is then separated into two parts using a splitter. One half is discarded, and the second half is split again, collecting each half in a pre-labeled micropore sample bag containing a portion of the sample tag. One sample is retained at a secure storage facility and the other submitted for analysis.

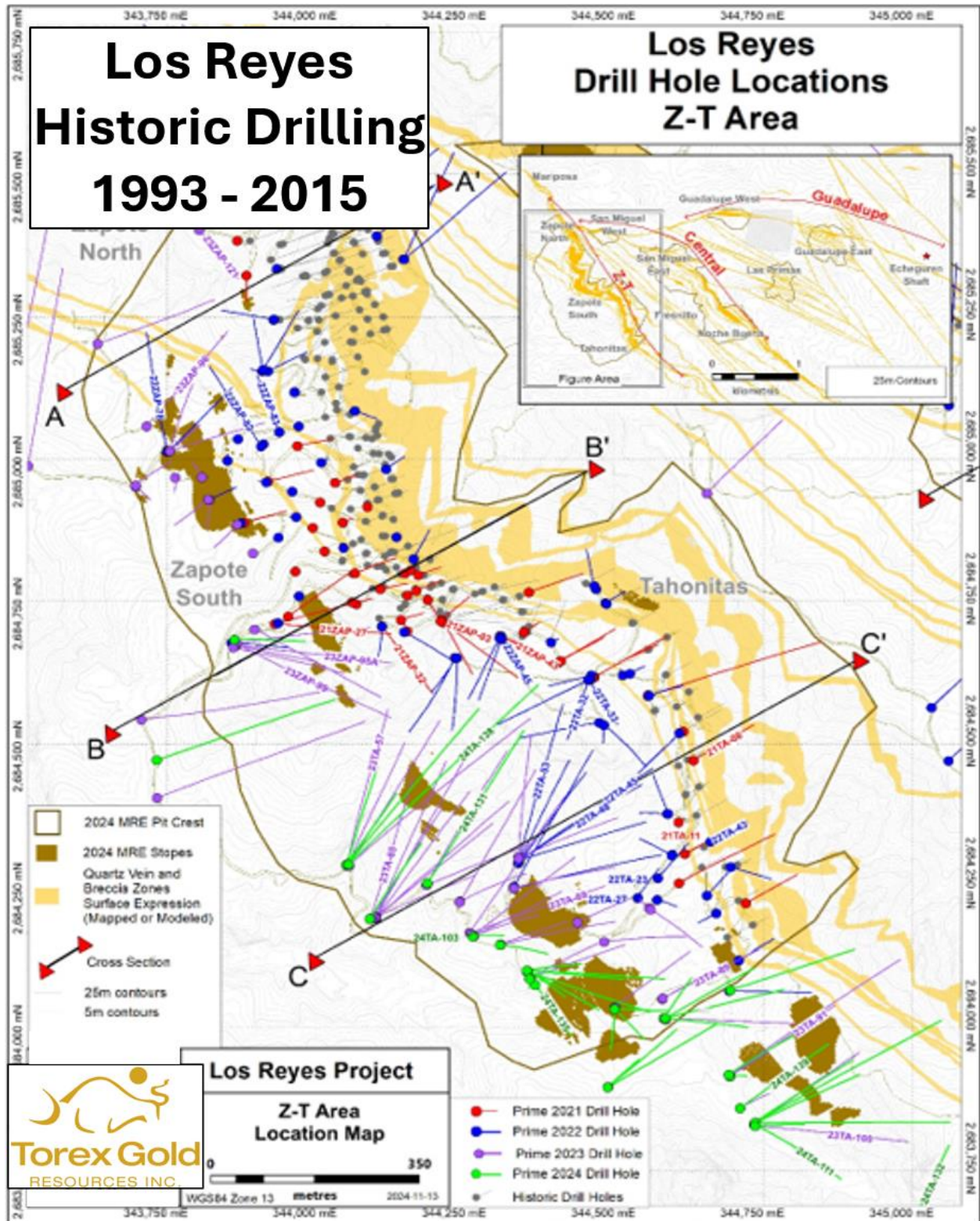
Logging of the RC chips is done at the drill site by a geologist using a sampling of the chips collected from the entire drill run. Chips were washed and a binocular microscope is used to identify the lithology and any other rock characteristics discernable such as alteration, textures, general colour, percentage of key minerals, estimated quartz vein content and characteristics, oxidation characteristics and any specific comments.

Prime collected 4,912 density measurements on core pucks from mineralised zones and a variety of country rock. Density measurements were filtered by lithology and alteration.

Additional information regarding Prime's drilling and sampling procedures can be found in Prime's technical report "Prime Mining Corp. Los Reyes Project", (Sims et al, June 2025).

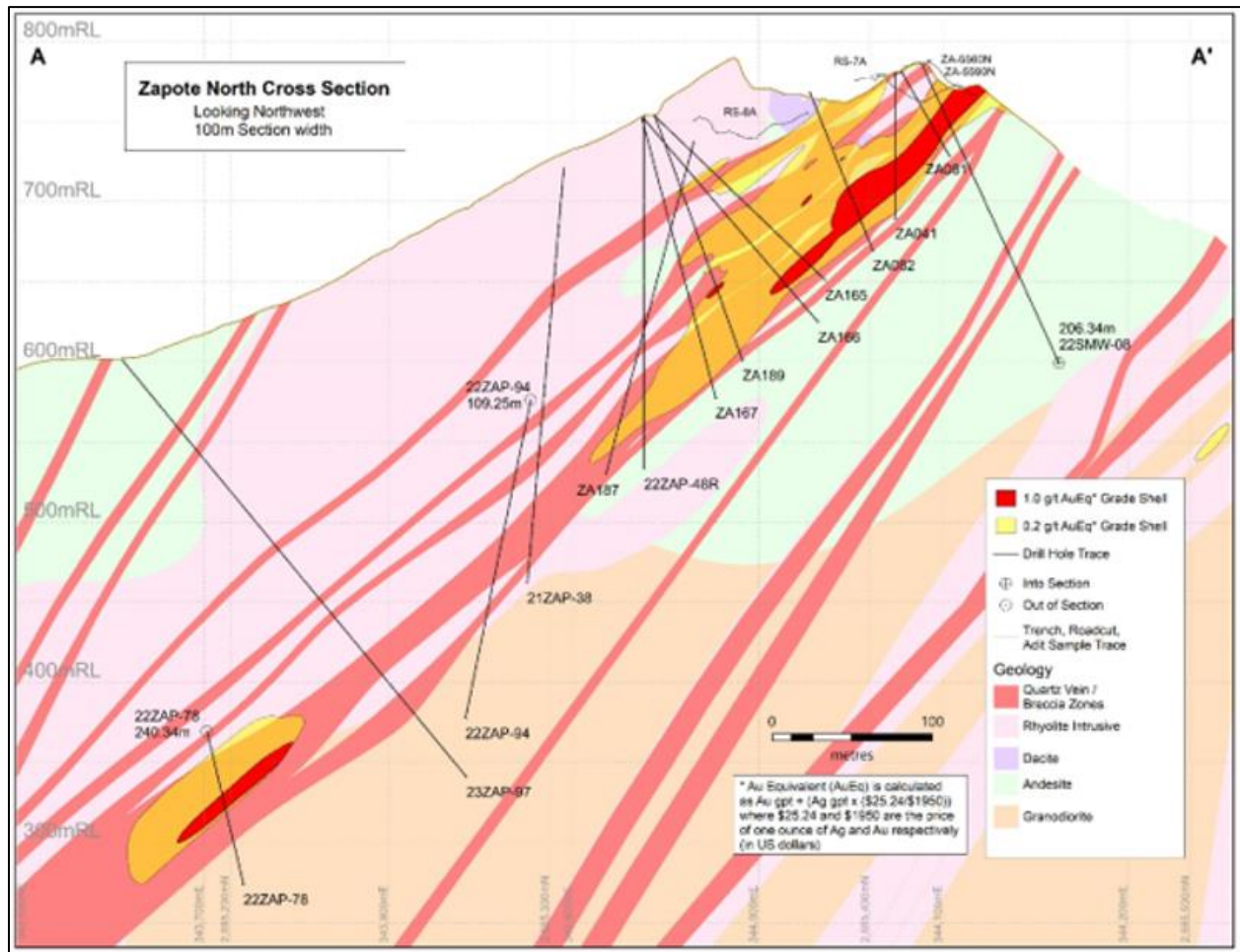
Figure 10-2 shows the Zapote-Tahonitas drill hole locations with representative cross sections lines. The representative cross sections are in Figure 10-3 to Figure 10-5.

Figure 10-2: Los Reyes Z-T Trend Drill Hole Locations



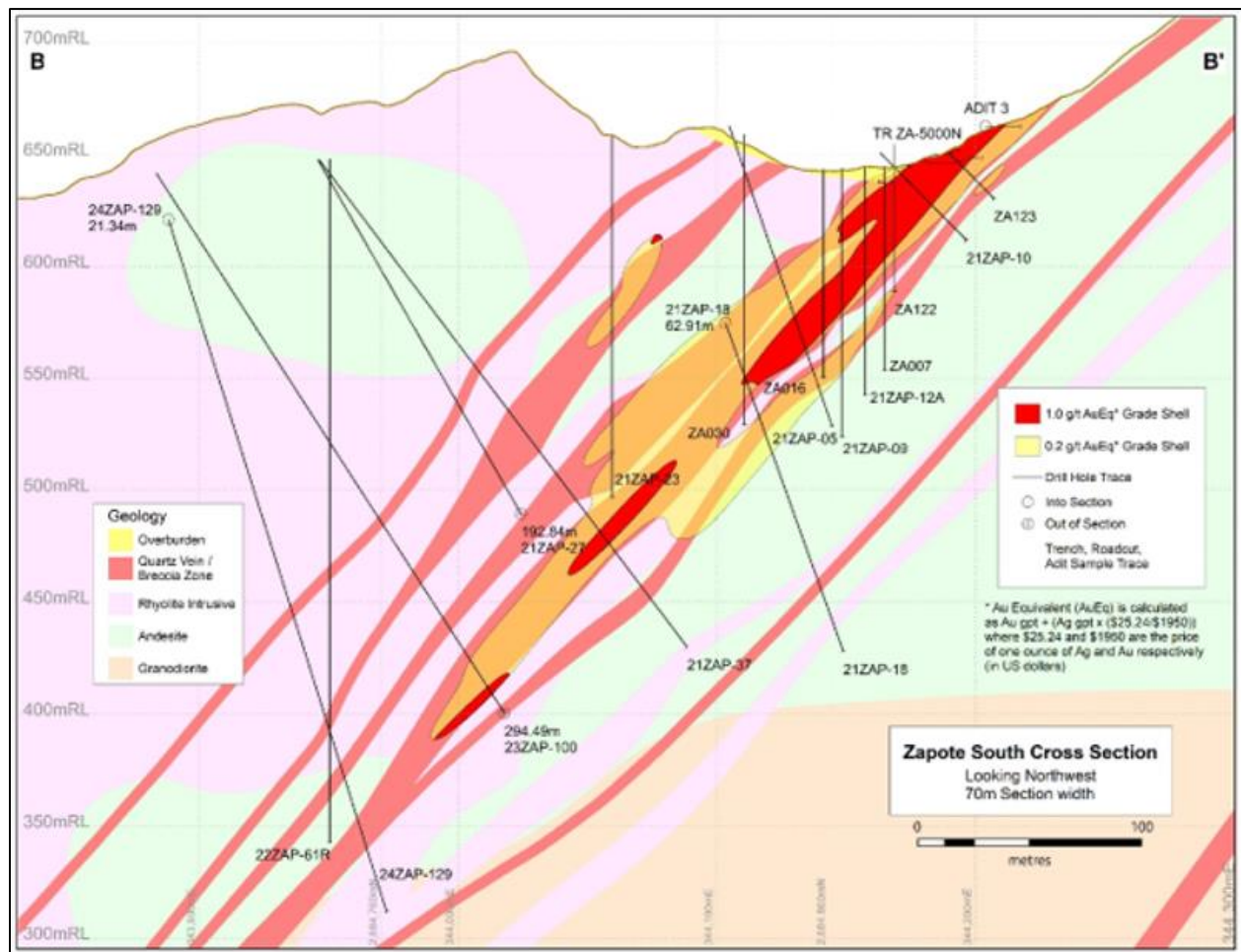
Source: Prime, 2024

Figure 10-3: Zapote North Cross Section



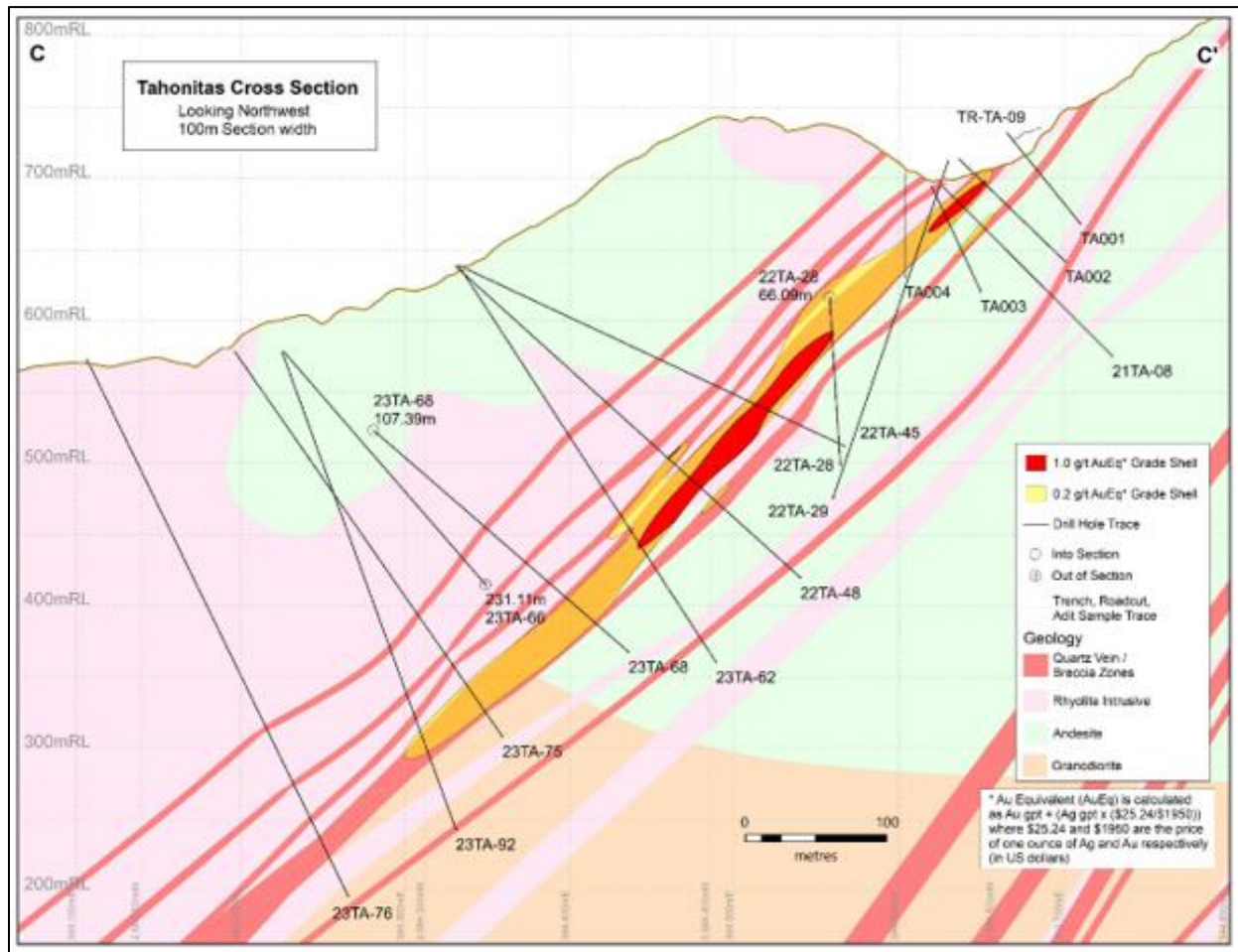
Source: Prime, 2024

Figure 10-4: Zapote South Cross Section



Source: Prime, 2024

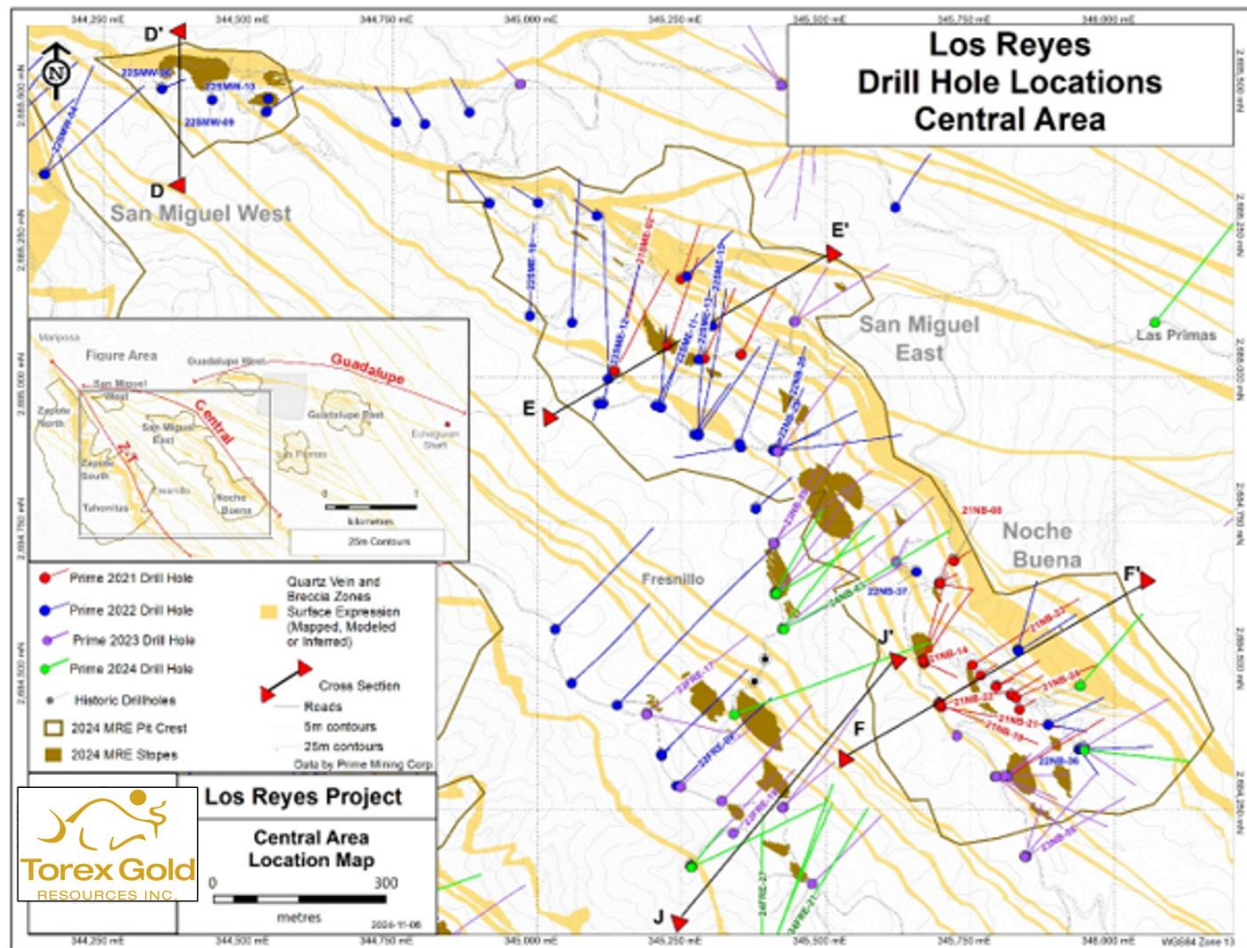
Figure 10-5: Tahonitas Cross Section



Source: Prime, 2024

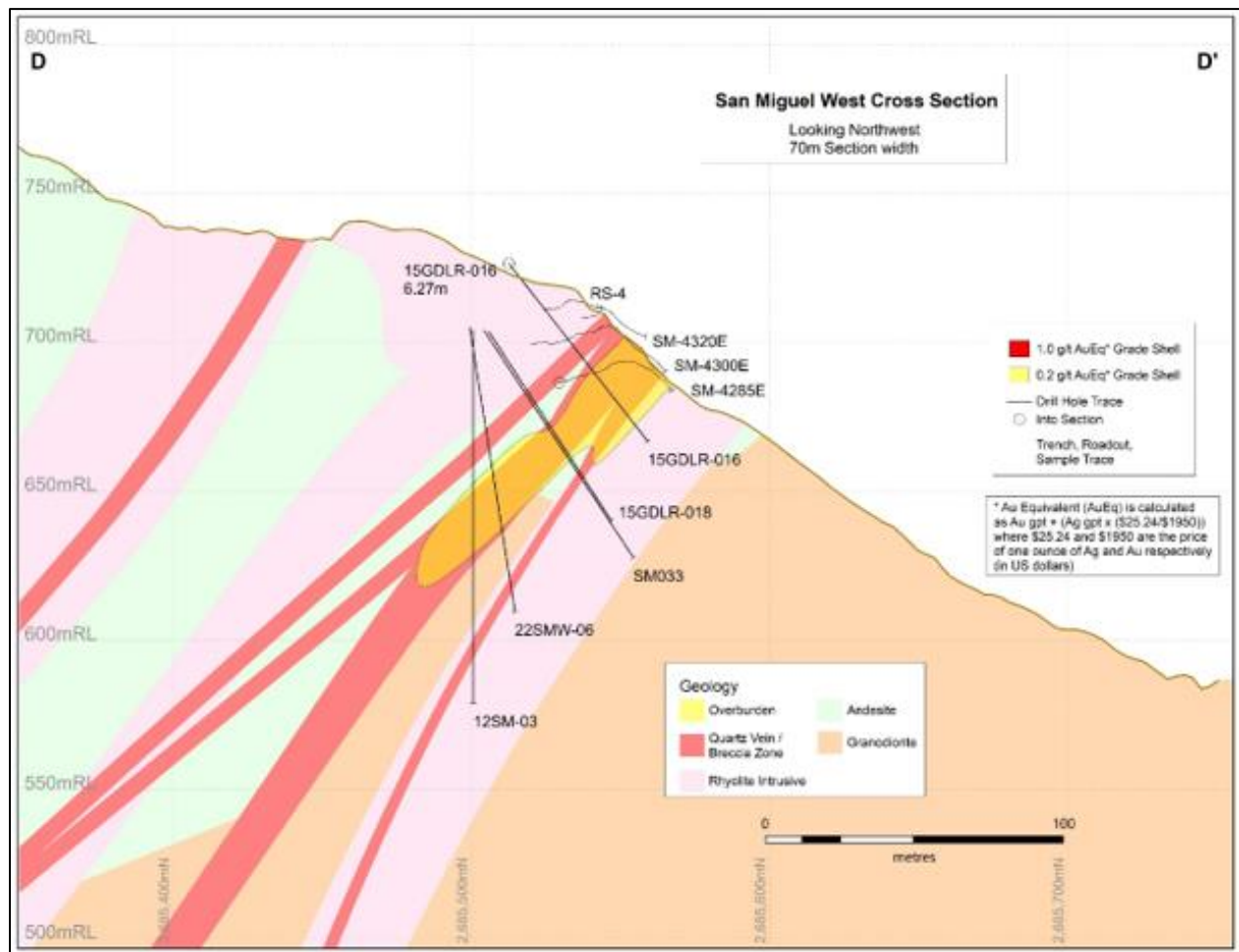
Figure 10-16 shows all the Central drill hole locations and representative cross sections are in Figure 10-7 to Figure 10-9.

Figure 10-6: Los Reyes Central Trend Drill Hole Locations



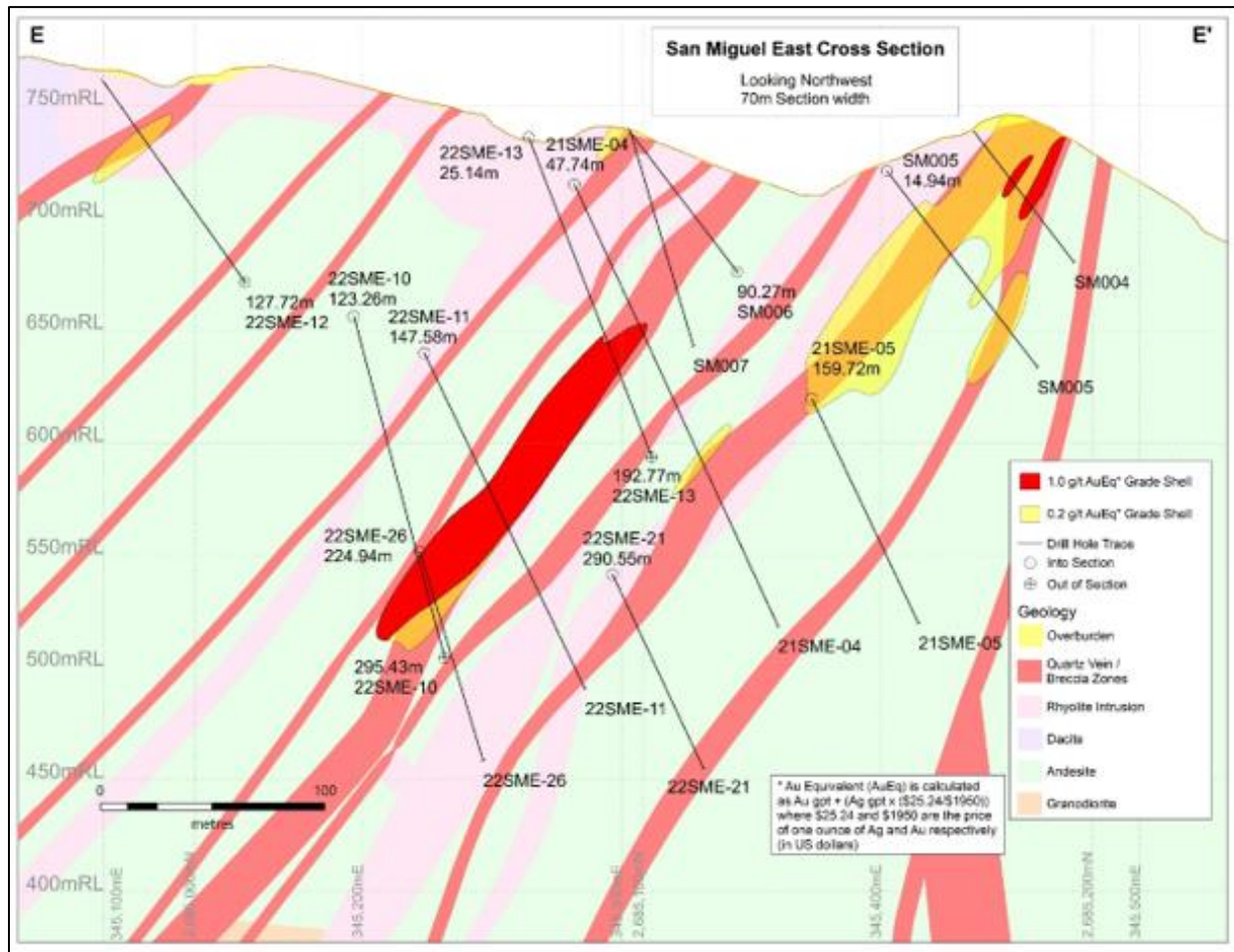
Source: Prime, 2024

Figure 10-7: San Miguel West Cross Section



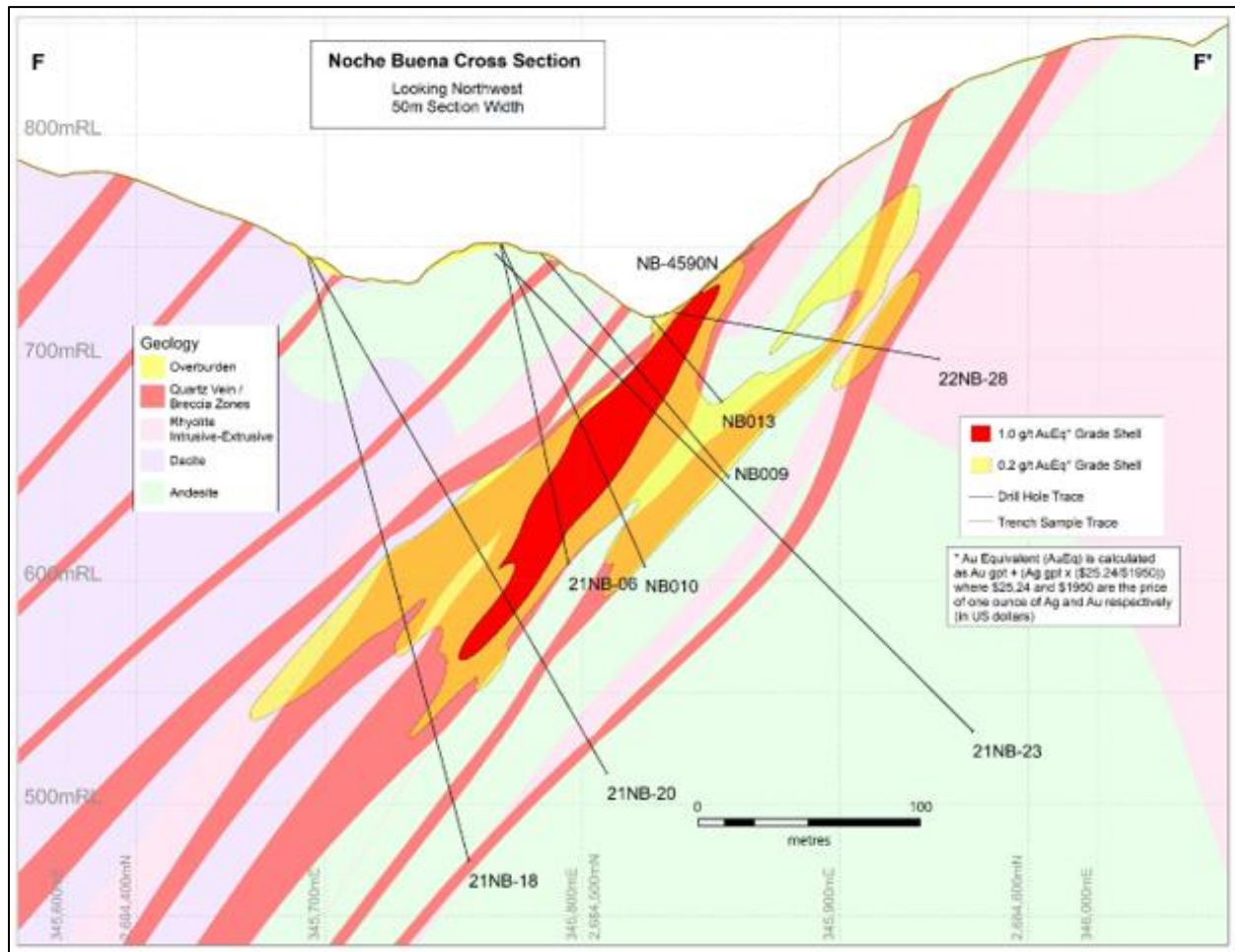
Source: Prime, 2024

Figure 10-8: San Miguel East Cross Section



Source: Prime, 2024

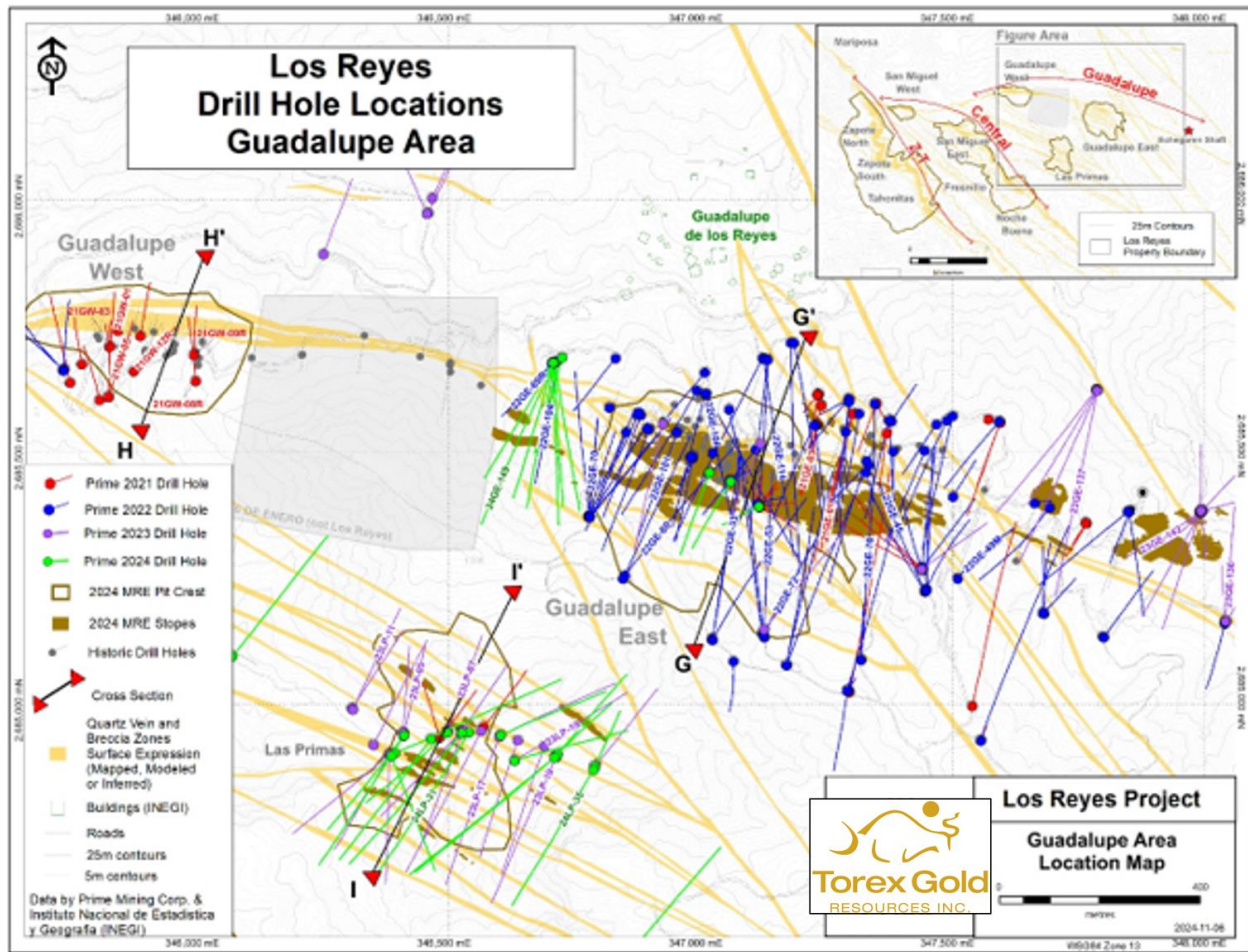
Figure 10-9: Noche Buena Cross Section



Source: Prime, 2024

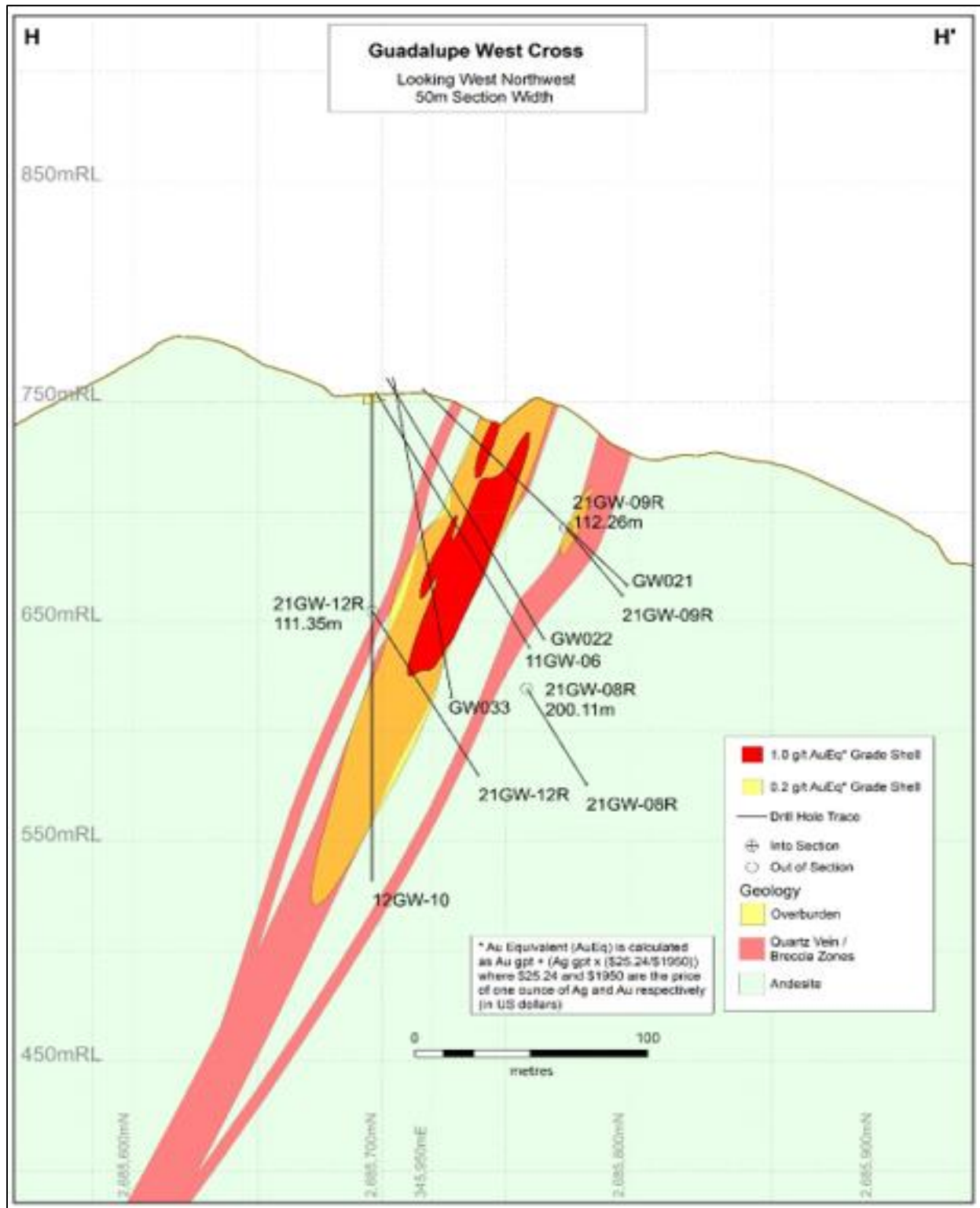
Figure 10-10 shows all the Guadalupe trend hole locations and representative cross sections are in Figure 10-11 to Figure 10-12.

Figure 10-10: Los Reyes Guadalupe Trend Drill Hole Locations



Source: Prime, 2024

Figure 10-12: Guadalupe West Cross Section



Source: Prime, 2024

Las Primas Cross Section
Looking Northwest
50m Section Width

900mRL
700mRL
500mRL
300mRL

Drill Holes:
24LP-40: 128.29m
23LP-12: 106.54m
22LP-12: 222.55m
23LP-14: 42.61m
24LP-30: 194.53m
24LP-36: 246.0m
24LP-37: 198.16m
21LP-01: 57.22m
21LP-02: 194.34m
24LP-33: 182.20m

Geology:
Quartz Vein / Breccia Zones
Felsic Intrusive, Rhyolite Intrusive Extrusive
Andesite

Scale: 0 to 100 metres

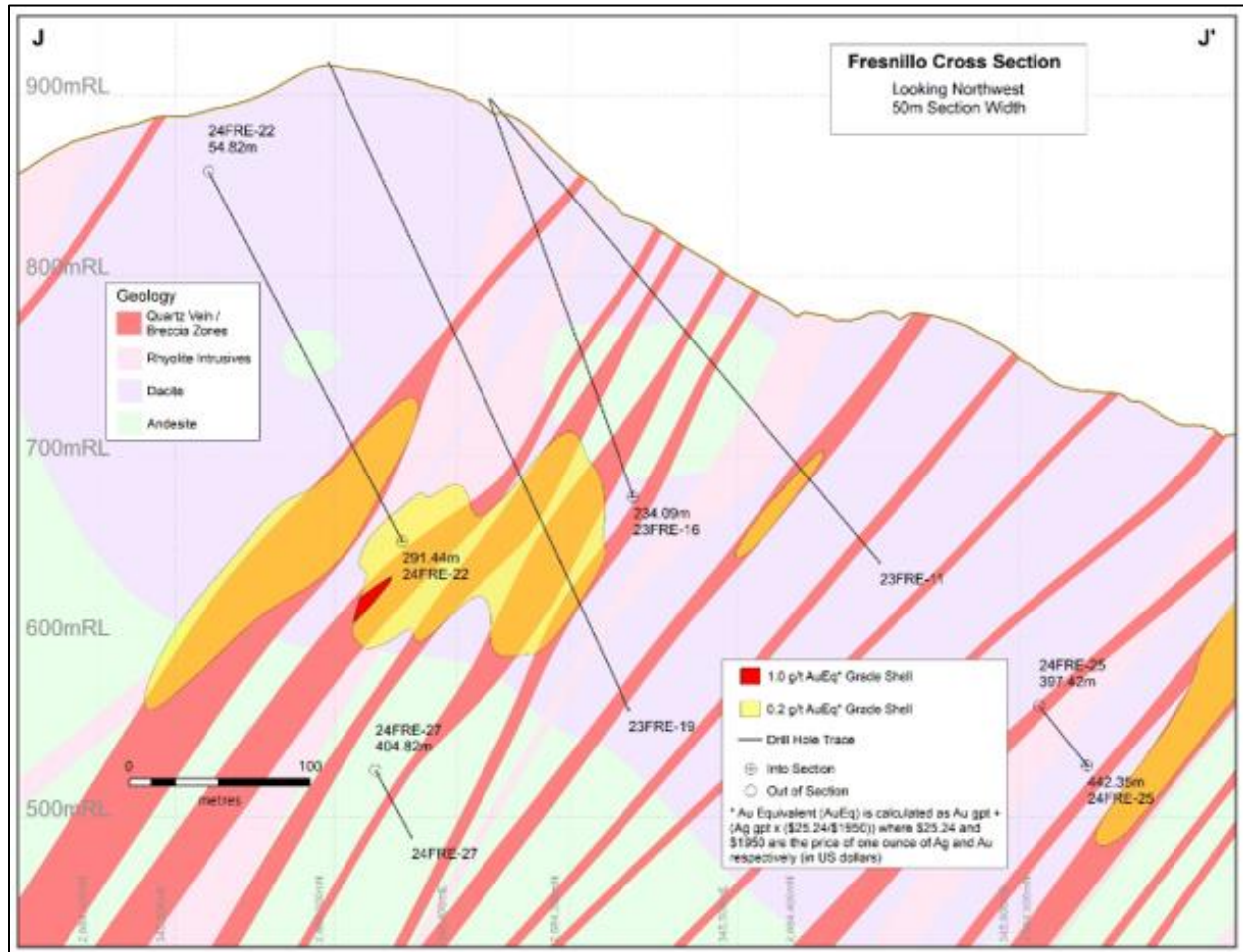
Legend:
1.0 g/t AuEq* Grade Shell
0.2 g/t AuEq* Grade Shell
Drill Hole Trace
Into Section
Out of Section
Trench, Roadcut, Adit Sample Trace

* Au Equivalent (AuEq) is calculated as $Au\ gpt + (Ag\ gpt \times (\$25.24 / \$1950))$ where \$25.24 and \$1950 are the price of one ounce of Ag and Au respectively (in US dollars)

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Fresnillo hole locations are shown on Figure 10-6, along with the other drill holes in Las Primas. A representative cross section through Fresnillo is shown in Figure 10-14 and its location is shown on Figure 10-6.

Figure 10-14: Los Reyes Fresnillo Cross Section

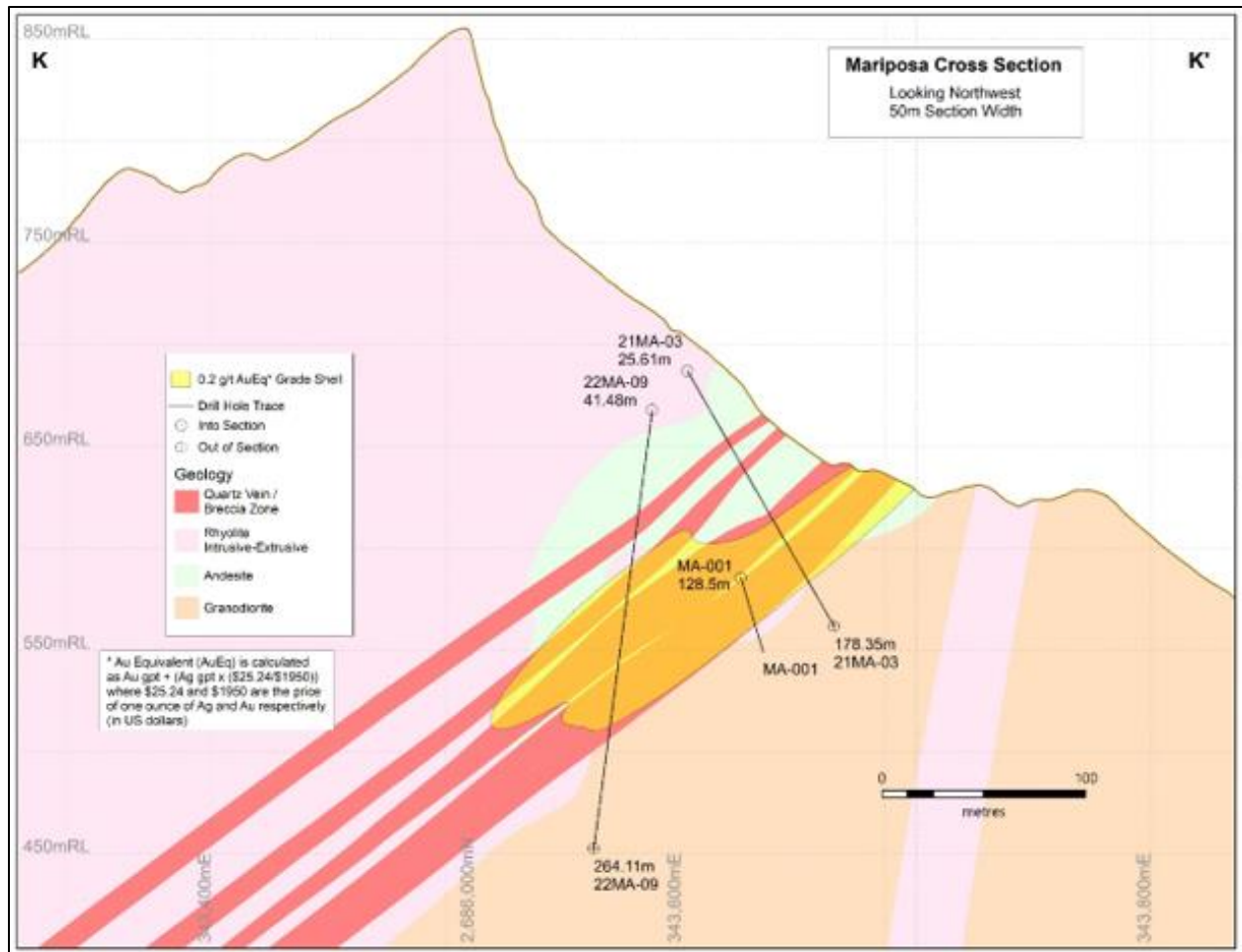


Source: Prime, 2024

Mariposa hole locations are shown on Figure 10-15. A representative cross section is shown on Figure 10-16.

Source: Prime, 2024

Figure 10-16: Los Reyes Mariposa Cross Section



Source: Prime, 2024

10.3 2025 Torex

Following the acquisition of Prime, Torex commenced with a data validation and relogging program. Torex initiated a drilling program in 2026.

10.4 Comments on Drill Programs

In the opinion of the Independent QP, the quantity and quality of the lithological, geotechnical, collar and down hole survey data collected in the drill programs are sufficient to support mineral resource estimation as follows:

- Core logging meets industry standards for gold exploration.
- Collar surveys have been performed using industry-standard instrumentation.
- Downhole surveys have been performed using industry-standard instrumentation.

- Recovery data from core drill programs are acceptable.
- There are no apparent drill, sampling or recovery factors that could impact the accuracy and reliability of the drilling results.
- Geotechnical logging of drill core meets industry standards.

Drilling is normally perpendicular to the strike of the mineralisation but due to topographical conditions and road access, oblique intersections are common. Depending on the dip of the drill hole, and the dip of the mineralisation, drill intercept widths are typically greater than true widths.

Drill hole orientations are appropriate for the mineralisation style and have been drilled at orientations that are optimal for the orientation of mineralisation for the bulk of the deposit area. Drill hole orientations are shown in plan figures and representative cross sections above and can be seen to test the mineralisation appropriately.

11 Sample preparation, analyses, and security

Torex has initiated a drilling program in 2026, but has not completed any sample preparation or analysis since acquiring Prime in 2025. Prior to the acquisition, Prime samples were submitted for crushing and pulverization to the ALS Global Geochemistry facilities (ALS) and Bureau Veritas (BV) facilities in Mexico. Both labs hold an ISO 9001 certification and are independent from both Prime and Torex.

11.1 Sample Preparation and Analysis

11.1.1 ALS Global, Zacatecas/Guadalajara, Mexico

Prime submitted diamond drill core samples to ALS for preparation and analysis.

The samples were prepared using ALS's standard sample preparation procedure PREP-31 which consists of logging the sample in the ALS internal tracking system and attaching a barcode to each sample. The sample is weighed, and if necessary, will be dried. The entire sample is then crushed to better than 70% passing two millimetres. The crushed sample is then riffle split to obtain a 250-gram subsample. The subsample is pulverised to better than 85% passing 0.075 millimetres (mm).

A 30-gram aliquot of the prepared sample is fused with a mixture of lead oxide, sodium carbonate, borax, silica, and other reagents as required, inquarted with six milligrams of gold-free silver and then cupelled to yield a precious metal bead. The bead is digested in 0.5 millilitres (mL) dilute nitric acid in the microwave oven, 0.5 mL concentrated hydrochloric acid is then added, and the bead is further digested in the microwave at a lower power setting. The digested solution is cooled, diluted to a total volume of four milliliters with de-mineralised water, and analyzed for gold by atomic absorption spectroscopy against matrix-matched standards (ALS procedure document).

If the gold results are greater than ten ppm, a new aliquot of the prepared sample is fused, the bead is parted to remove the silver and the bead weighed for gold, known as gravimetric finish.

In addition to the gold results, a 0.25-gram aliquot of the 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 inductively coupled plasma-atomic emission spectrometry (ICP) for a suite of thirty-four elements (ALS procedure document). The suite of elements is listed in Table 11-1.

No aspect of the sample preparation process was conducted by an employee, officer, director, or associate of Prime.

Table 11-1: List of Analytes, ME-ICP61, ALS Global

Analyte	Units	Lower Limit	Upper Limit	Analyte	Units	Lower Limit	Upper Limit
Silver	ppm	0.5	100	Manganese	ppm	5	100000
Aluminum	%	0.01	50	Molybdenum	ppm	1	10000
Arsenic	ppm	5	10000	Sodium	%	0.01	10
Barium	ppm	10	10000	Nickel	ppm	1	10000
Beryllium	ppm	0.5	1000	Phosphorus	ppm	10	10000
Bismuth	ppm	2	10000	Lead	ppm	2	10000
Calcium	%	0.01	50	Sulphur	%	0.01	10
Cadmium	ppm	0.5	1000	Antimony	ppm	5	10000
Cobalt	ppm	1	10000	Scandium	ppm	1	10000
Chromium	ppm	1	10000	Strontium	ppm	1	10000
Copper	ppm	1	10000	Thorium	ppm	20	10000
Iron	%	0.01	50	Titanium	%	0.01	10
Gallium	ppm	10	10000	Thallium	ppm	10	10000
Potassium	%	0.01	10	Uranium	ppm	10	10000
Lanthanum	ppm	10	10000	Vanadium	ppm	1	10000
Lithium	ppm	10	10000	Tungsten	ppm	10	10000
Magnesium	%	0.01	50	Zinc	ppm	2	10000

11.1.2 Bureau Veritas, Durango, Mexico

Prime submitted diamond drill core samples and reverse circulation samples to BV for preparation and analysis.

The samples were prepared using BV's PRP70-250 preparation procedure. The samples are logged in the BV Laboratory Information Management System (LIMS), weighed, and if necessary dried before being crushed to greater than 70% passing 2 mm. The crushed sample is then riffle split to obtain a 250-gram subsample. The subsample is pulverised to better than 85 percent passing 0.075 mm.

Like the samples submitted to ALS, the samples are analysed for gold by fire assay and for multiple elements by four-acid digestion with ICP finish. The lower detection limit for gold is 0.005 ppm and the lower detection limit for silver is 0.5 ppm.

No aspect of the sample preparation process was conducted by an employee, officer, director, or associate of Prime.

11.1.3 Security

The core is boxed at the drill rig (RC samples are bagged) and picked up daily by Prime personnel for direct transport to Prime's fenced-in logging facility. The core remains in the fenced yard until it has been logged, photographed, and split. All sample intervals are recorded in Prime's Access Capture software and a sample submittal inventory form is prepared for the laboratory.

Split samples are bagged in pre-labeled sample bags (with a sample tag inside the bag) and double bagged in a larger rice bag and labelled with sample from-to and bag number. These are stored in the secure logging area until personnel from the laboratory collects them. Sample collection usually occurs once or twice per week. The laboratory personnel check the sample count and transport the samples directly to the

laboratory facility. Confirmation is sent to Prime from the laboratory that all samples submitted as per the submittal form have been received. If the laboratory sample count differs from Prime's submittal form, the differences will be flagged for review.

Core samples (or RC samples) not submitted for analysis are stored in their original core box (or RC cuttings bag) and transported to a locked warehouse for storage.

11.1.4 Analytical Quality Control

Prime has submitted half drill core samples and reverse circulation samples. The samples were assayed for gold by fire assay, and multiple elements, including silver, by four-acid digest.

Prime has maintained a quality control program that includes the insertion of blank materials, certified reference materials, coarse duplicates and the submission of primary pulps to a secondary laboratory for check assaying.

Prime submitted samples to both ALS and BV for sample preparation and analysis. The samples were prepared and analysed in Zacatecas, Guadalajara, and Durango, Mexico, as well as some analyses in Vancouver, British-Columbia, Canada.

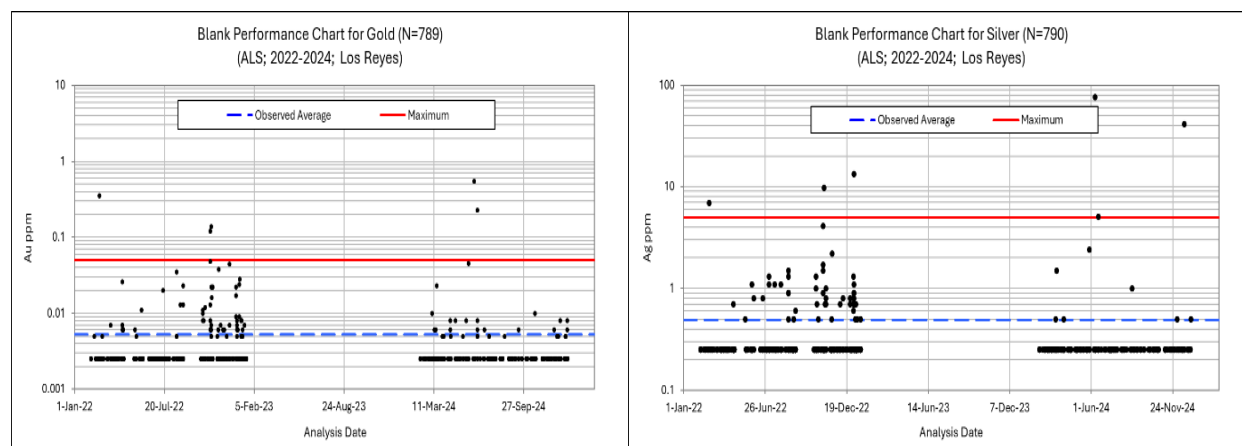
Blanks materials and certified reference materials were inserted at a rate of one in fifty samples.

11.1.4.1 ALS Global, Mexico

11.1.4.1.1 Blank Materials

A locally sourced basalt was inserted as a blank material 789 times in regular sequence with samples submitted to ALS. The blank materials are determined to have failed when they assay more than ten times the detection limit: 0.05 ppm gold or 5 ppm silver. There are five blank material failures for gold and six failures for silver, Figure 11-1. The failure rate for the blank materials is less than 1% for gold and silver, and there is no evidence of systematic contamination. The blank material results are acceptable.

Figure 11-1: Blank performance chart, 2022-2024, ALS



Source: Qualitica, 2025

11.1.4.1.2 Certified Reference Materials

Eleven different reference materials were inserted 1,270 times in regular sequence with the samples and analysed for gold by fire assay and multiple elements, including silver, by four-acid digest. The performance for gold and silver are summarized in Table 11-2 and Table 11-3.

A quality control failure is defined as an assay result for a certified reference material that is outside plus or minus three standard deviations of the expected value. A total of 36 quality control failures were identified for gold and 38 for silver, Figure 11-2. This represents a failure rate of less than 3% for both gold and silver. The overall failure rate is acceptable.

The failures were removed to calculate the Percent of Expected. The Percent of Expected demonstrates that there are no significant biases in the results but that on average the gold and silver results are reporting higher than the expected values.

Based on the results of the certified reference materials, it can be concluded that the accuracy of the results from ALS are acceptable.

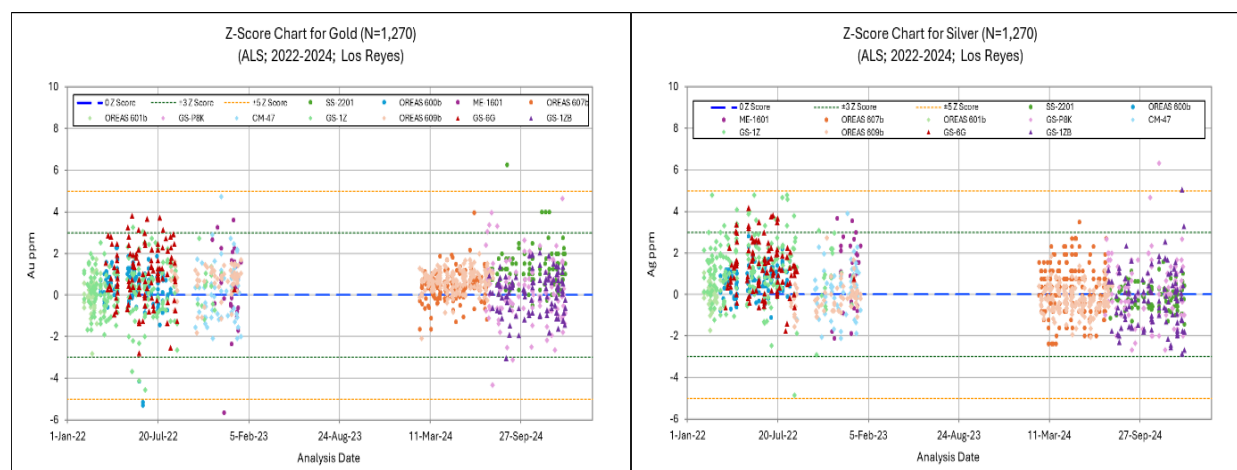
Table 11-2: Summary reference material statistics for gold, ALS Global

RM	N	Failures Excluded	Au ppm		Observed Au ppm		Percent of Expected
			Expected	Std. Dev.	Average	Std. Dev.	
CDN-SS-2201	90	4	0.074	0.004	0.078	0.003	106%
OREAS 600b	116	3	0.204	0.007	0.208	0.005	102%
CDN-ME-1601	27	3	0.613	0.023	0.623	0.029	102%
OREAS 607b	170	1	0.696	0.025	0.706	0.016	101%
OREAS 601b	43	-	0.775	0.021	0.775	0.017	100%
GDN-GS-P8K	99	7	0.829	0.045	0.834	0.052	101%
CDN-CM-47	56	1	1.13	0.055	1.13	0.071	100%
CDN-GS-1Z	184	7	1.16	0.048	1.17	0.051	101%
OREAS 609b	250	1	4.97	0.260	5.15	0.150	104%
CDN-GS-6G	102	8	6.30	0.150	6.42	0.184	102%
CDN-GS-1ZB	97	1	6.47	0.280	6.50	0.266	100%
Total	1234	36			Weighted Average		102%

Table 11-3: Summary reference material statistics for silver, ALS Global

RM	N	Failures Excluded	Ag ppm		Observed Ag ppm		Percent of Expected
			Expected	Std. Dev.	Average	Std. Dev.	
OREAS 607b	170	1	6.11	0.255	6.17	0.316	101%
CDN-GS-P8K	103	3	8.10	0.300	8.12	0.368	100%
OREAS 609b	251	-	24.6	1.03	24.4	0.673	99%
OREAS 600b	118	1	25.1	1.00	25.8	0.672	103%
CDN-SS-2201	94	-	31.6	1.40	31.5	0.823	100%
CDN-ME-1601	28	2	39.6	0.900	40.3	1.36	102%
OREAS 601b	43	-	50.1	1.74	50.1	1.09	100%
CDN-CM-47	56	1	69.0	3.00	69.5	3.31	101%
CDN-GS-1ZB	96	2	81.0	3.50	80.0	4.36	99%
CDN-GS-6G	100	10	84.0	2.50	86.6	2.48	103%
CDN-GS-1Z	173	18	89.5	2.20	91.7	2.48	102%
Total	1232	38			Weighted Average		101%

Figure 11-2: Z-Score chart, 2022-2024, ALS



Source: Qualitica, 2025

11.1.4.1.3 Quarter Core Duplicates

The field duplicate results received from ALS consist of quarter core duplicates. The quarter core duplicates are taken for every 50th and 100th sample, with the duplicate result reported in the 55th and 105th sample.

During 2022, the submission of quarter core duplicates was discontinued. Prime has already obtained a significant quantity of quarter core duplicate results for the Los Reyes Project, and understand the variability associated with sampling the core.

Precision, by definition, is about $\pm 100\%$ at ten times the detection limit. Assays close to the detection limit are not included in calculations of precision. This applies to all discussions of precision in this section.

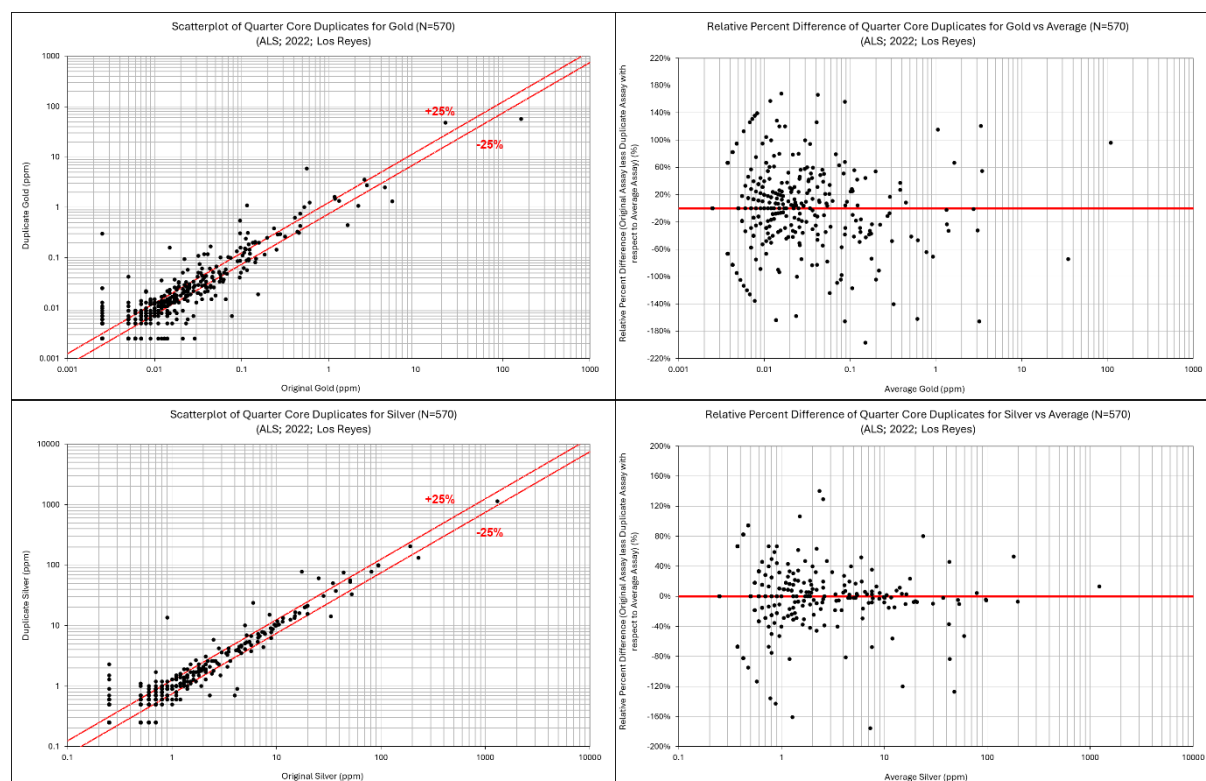
As shown in Table 11-4, a total of 570 quarter core duplicates were collected and submitted for analyses. A total of 73 quarter core duplicates out of 570 reported above 0.05 ppm gold, and 60 reported above 5 ppm silver. Twenty-nine percent of the duplicate pairs are reporting within $\pm 25\%$ for gold, and seventy-five percent for silver, Figure 11-3. This is considered acceptable for quarter core duplicates.

Table 11-4: Summary of quarter core duplicate results for gold and silver

Analyte	# of Pairs	# of Pairs above 10x d.l.	% of Sample Pairs (>10x d.l.) Reporting within			
			$\pm 5\%$	$\pm 10\%$	$\pm 20\%$	$\pm 50\%$
Au	570	73	7%	14%	29%	63%
Ag	570	57	33%	56%	75%	84%

Source: Qualitica, 2025

Figure 11-3: XY and RPD Chart for Gold and Silver in Field Duplicates



Source: Qualitica, 2025

11.1.4.1.4 Coarse Duplicates

Coarse duplicates are created by splitting a second cut of the crushed sample in the same way and for the same weight as the original sample.

Prime select samples ending in '25', or one sample in one hundred samples as a coarse duplicate. It is expected that the coarse duplicate pairs will be more similar than the quarter core duplicates due to the smaller particle size during sub-sampling.

A total of 788 coarse duplicates were analyzed for gold Table 11-5. A total of 133 duplicate pairs out of 788 reported above 0.05 ppm gold. The coarse duplicates for gold have 61% of the duplicate pairs reporting within $\pm 25\%$, Figure 11-4. These results are considered acceptable.

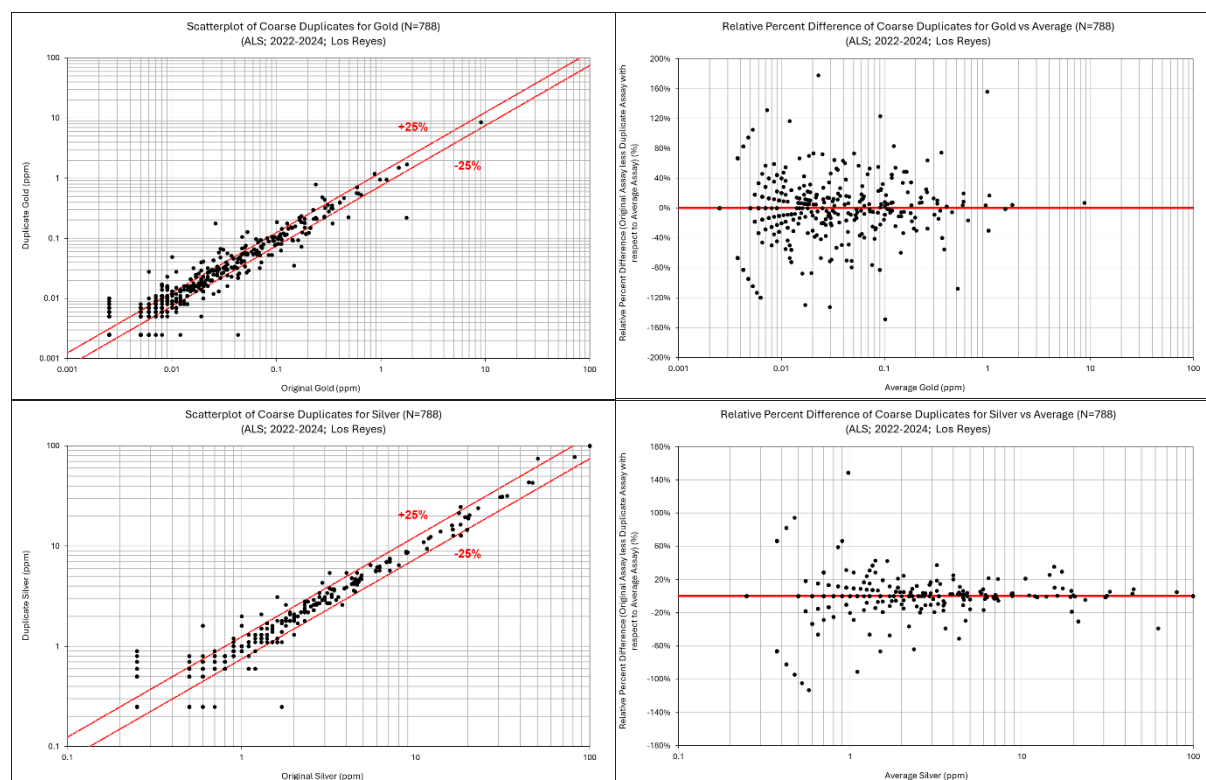
A total of 788 preparation duplicates were analyzed for silver. Sixty-four duplicate pairs out of 788 reported above 5 ppm silver. The coarse duplicates for silver have 86% of the duplicate pairs reporting within $\pm 25\%$. These results are considered acceptable.

Table 11-5: Summary of coarse duplicate results for gold and silver, 2022-2024, ALS

Analyte	# of Pairs	# of Pairs above 10x d.l.	% of Sample Pairs (>10x d.l.) Reporting within			
			$\pm 5\%$	$\pm 10\%$	$\pm 20\%$	$\pm 50\%$
Au	788	133	16%	41%	61%	87%
Ag	788	64	55%	72%	86%	100%

Source: Qualitica, 2025

Figure 11-4: XY and RPD chart for gold and silver in coarse duplicates



Source: Qualitica, 2025

11.1.4.1.5 Pulp Duplicates

The assays for pulp duplicates provide an estimate of the reproducibility related to the uncertainties inherent in the analytical method and the homogeneity of the pulps. The precision or relative percent difference calculated for the pulp duplicates indicates whether pulverizing specifications should be changed and/or whether alternative methods should be considered.

Commercial laboratories routinely assay a second aliquot of the sample pulp, for approximately one in ten samples, these are pulp duplicates. The pulp duplicate results are used by the laboratory for their internal quality control monitoring. Prime selects every sample ending in '75', or one sample in one hundred samples, to be analysed in duplicate. Only the Prime pulp duplicates are plotted below.

Pulp duplicate results for gold are available for 824 samples, Table 11-6. A total of 167 duplicate pairs out of the 824 reported above 0.05 ppm gold. The pulp duplicates for gold have 70% of the duplicate pairs reporting within $\pm 25\%$, Figure 11-5. Precision for the pulp duplicates is acceptable.

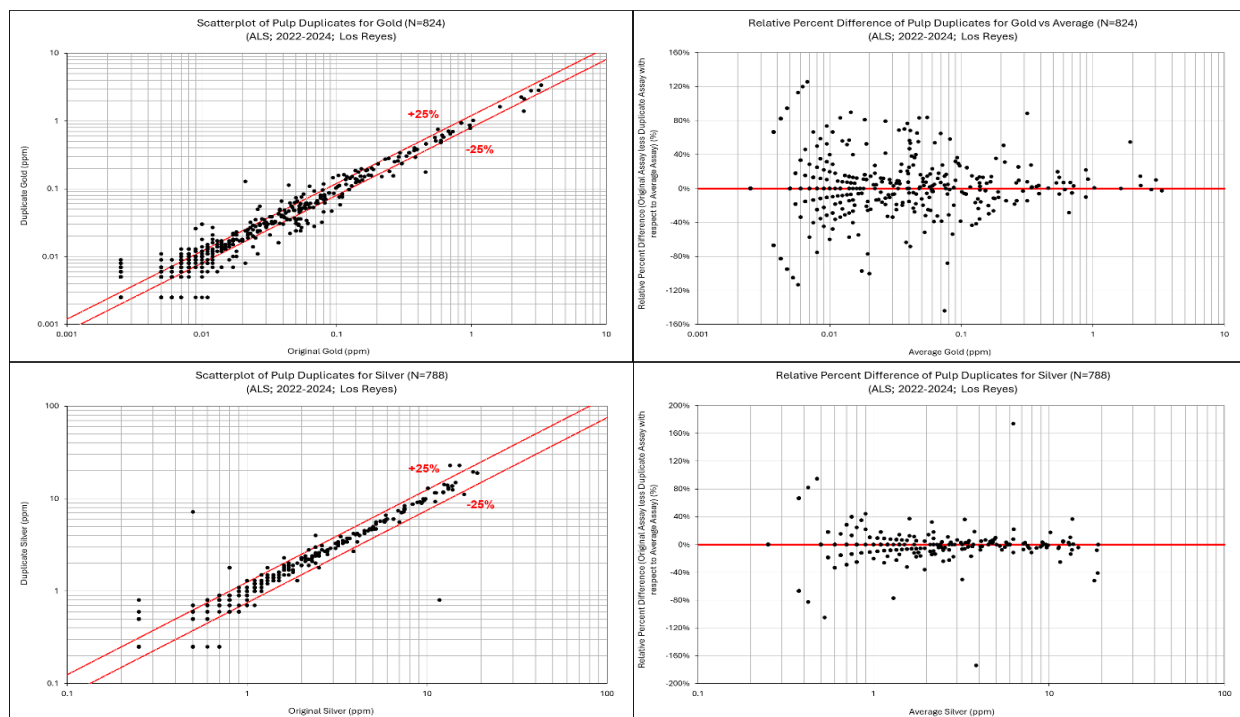
Pulp duplicate results for silver are available for 788 samples, Table 11-6. A total of 64 duplicate pairs out of the 788 reported above 5 ppm silver. The pulp duplicates for silver have 91% of the duplicate pairs reporting within $\pm 25\%$, Figure 11-5. Precision for the pulp duplicates is acceptable.

Table 11-6: Summary of pulp duplicate results for gold and silver

Analyte	# of Pairs	# of Pairs above 10x d.l.	% of Sample Pairs (>10x d.l.) Reporting within			
			$\pm 5\%$	$\pm 10\%$	$\pm 20\%$	$\pm 50\%$
Au	824	167	26%	43%	70%	89%
Ag	788	64	59%	78%	91%	97%

Source: Qualitica, 2025

Figure 11-5: XY and RPD chart for gold and silver in pulp duplicates



Source: Qualitica, 2025

11.1.4.1.6 Check Assays

Check assays are recommended where the same pulp that was assayed originally is submitted to a different laboratory for the same analytical procedures primarily to augment the assessment of bias based on the reference materials and in-house control samples submitted to the original laboratory.

A total of 260 pulps were selected from the samples analysed at ALS, Table 11-7. The samples were submitted to BV for check assaying. The methods from both laboratories are comparable.

Seventy-five percent of the check assay results for gold are within $\pm 25\%$ of the two sets of laboratory results; this is acceptable agreement. In fifty-eight percent of cases ALS is higher than BV. The average RPD for gold between ALS and BV is 6%, this indicates that on average the ALS results are higher than BV results by about 6%. Based on the charts in Figure 11-6, the results are well distributed above and below zero percent relative difference, especially for results above 0.5 ppm gold.

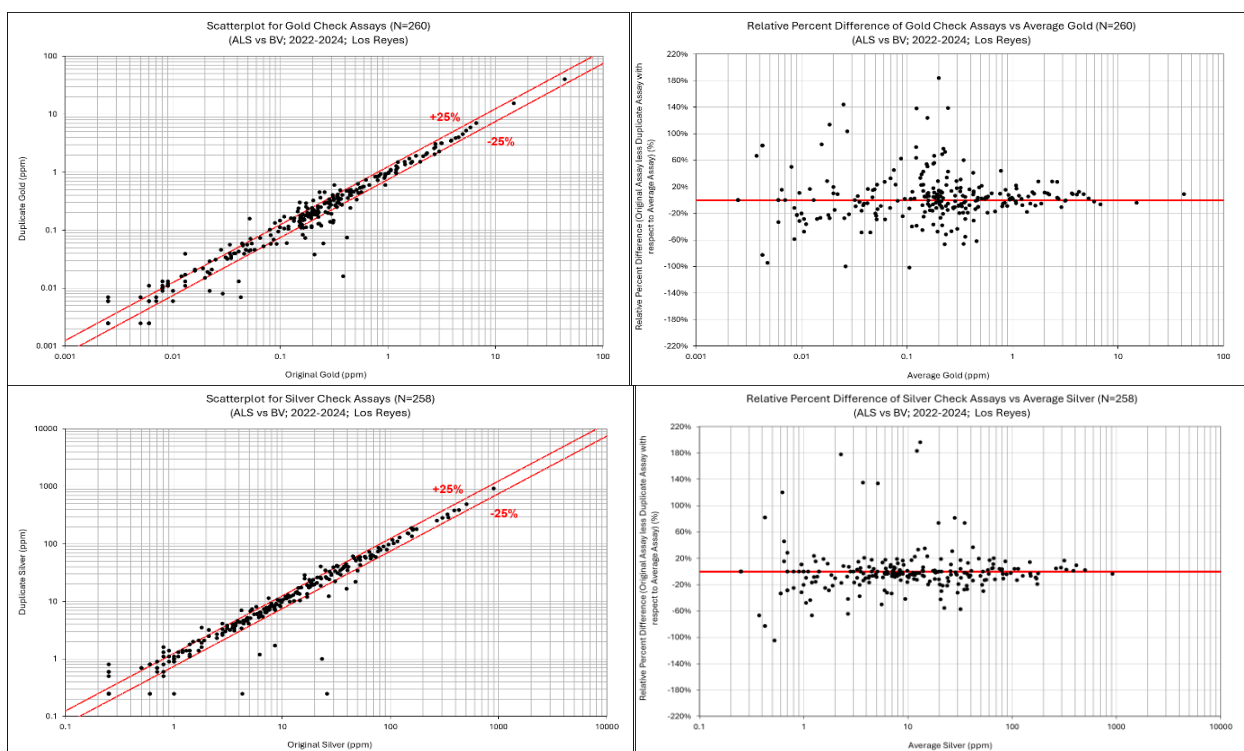
For the silver results, 85% are within $\pm 25\%$ of the two sets of laboratory results. The results are well distributed above and below zero relative percent difference. The average difference between the two laboratories is 3%, with the results being higher at BV for 56% of the case.

Table 11-7: Summary of Check Assay Results for Gold and Silver, ALS vs BV

Analyte	# of Pairs	# of Pairs above 10x d.l.	Average RPD	% of Sample Pairs (>10x d.l.) Reporting within			
				±5%	±10%	±25%	±50%
Au	260	198	6%	20%	46%	75%	89%
Ag	258	158	3%	41%	58%	85%	94%

Source: Qualitica, 2025

Figure 11-6: XY and RPD for gold and silver in check assays, ALS vs BV



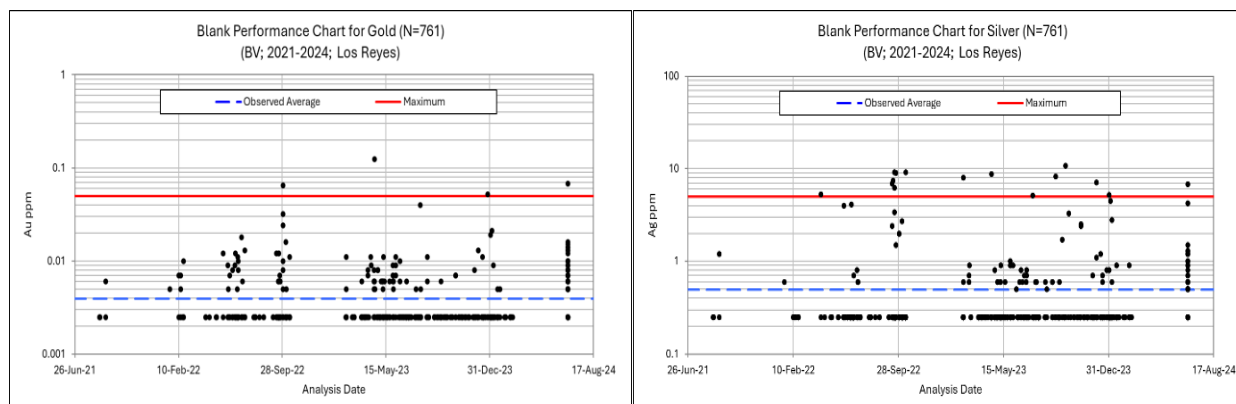
Source: Qualitica, 2025

11.1.4.2 Bureau Veritas, Mexico

11.1.4.2.1 Blank Materials

A locally sourced basalt was inserted as a blank material 761 times in regular sequence with samples submitted to BV. Blank materials were determined to have failed when they assayed more than ten times the detection limit or 0.05 ppm gold and 5 ppm silver, Figure 11-7. There is a failure rate of less than 1% for blank materials and no evidence of systematic contamination.

Figure 11-7: Blank performance chart; 2021-2024, BV



Source: Qualitica, 2025

11.1.4.2.2 Certified Reference Materials

Ten different reference materials were inserted 1,609 times in regular sequence with the samples and analysed for gold by fire assay and multiple elements, including silver, by four-acid digest. The performance for gold and silver are summarized in Table 11-8 and Table 11-9.

A total of 90 quality control failures were identified for gold and 44 for silver, Figure 11-8. This represents a failure rate of 5% for gold and 3% for silver. The overall failure rate is acceptable.

The Percent of Expected demonstrates that on average there are no significant biases in the overall results. Individually, some of the reference materials show a bias, for example OREAS 609b is reporting 4% high for gold and 2% low for silver, compared to the expected value.

Based on the overall results of the certified reference materials, it can be concluded that the accuracy of the results for gold and silver from BV are acceptable.

Table 11-8: Summary reference material statistics for gold, BV

RM	N	Failures Excluded	Au ppm		Observed Au ppm		Percent of Expected
			Expected	Std. Dev.	Average	Std. Dev.	
OREAS 600b	50	5	0.204	0.007	0.207	0.008	101%
CDN-ME-1101	5	1	0.564	0.028	0.616	0.022	109%
CDN-ME-1601	194	14	0.613	0.023	0.615	0.027	100%
OREAS 607b	120	1	0.696	0.025	0.714	0.021	103%
OREAS 601b	283	17	0.775	0.021	0.757	0.023	98%
CDN-CM-47	137	10	1.13	0.055	1.14	0.070	101%
CDN-GS-1Z	203	18	1.16	0.048	1.16	0.060	100%
CDN-GS-1P5T	125	20	1.75	0.085	1.75	0.130	100%
OREAS 609b	345	-	4.97	0.260	5.17	0.127	104%
CDN-GS-6G	57	4	6.30	0.150	6.34	0.201	101%
Total	1,519	90			Weighted Average		101%

Figure 11-8: Z-Score chart, 2021-2024, BV



The field duplicates results received from BV consist of quarter core duplicates and reverse circulation duplicates. For reverse circulation, the duplicate is taken from the original sample split, one mineralised interval and one un-mineralised interval, they are selected after the hole is completed.

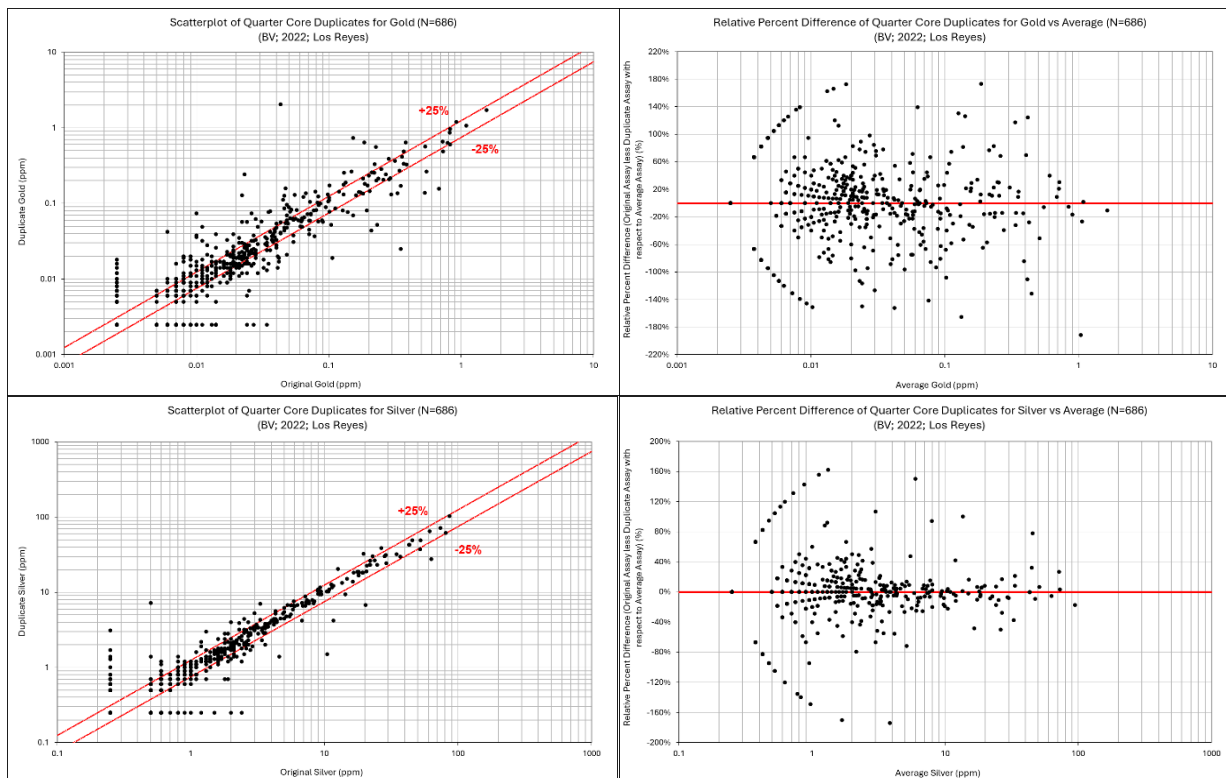
A total of 686 quarter core duplicates were collected and submitted for analyses, Table 11-10. A total of 108 quarter core duplicates out of 686 reported above 0.05 ppm gold and 81 reported about 5 ppm silver. Forty-eight percent of the duplicate pairs are reporting within $\pm 25\%$ for gold and 84% for silver, Figure 11-9. This is considered acceptable for quarter core duplicates.

Table 11-10: Summary of Quarter Core Duplicate Results for Gold and Silver

Analyte	# of Pairs	# of Pairs above 10x d.l.	% of Sample Pairs (>10x d.l.) Reporting within			
			±5%	±10%	±25%	±50%
Au	686	108	10%	18%	48%	73%
Ag	686	81	25%	59%	84%	95%

Source: Qualitica, 2025

Figure 11-9: XY and RPD chart for gold and silver in quarter core duplicates



Source: Qualitica, 2025

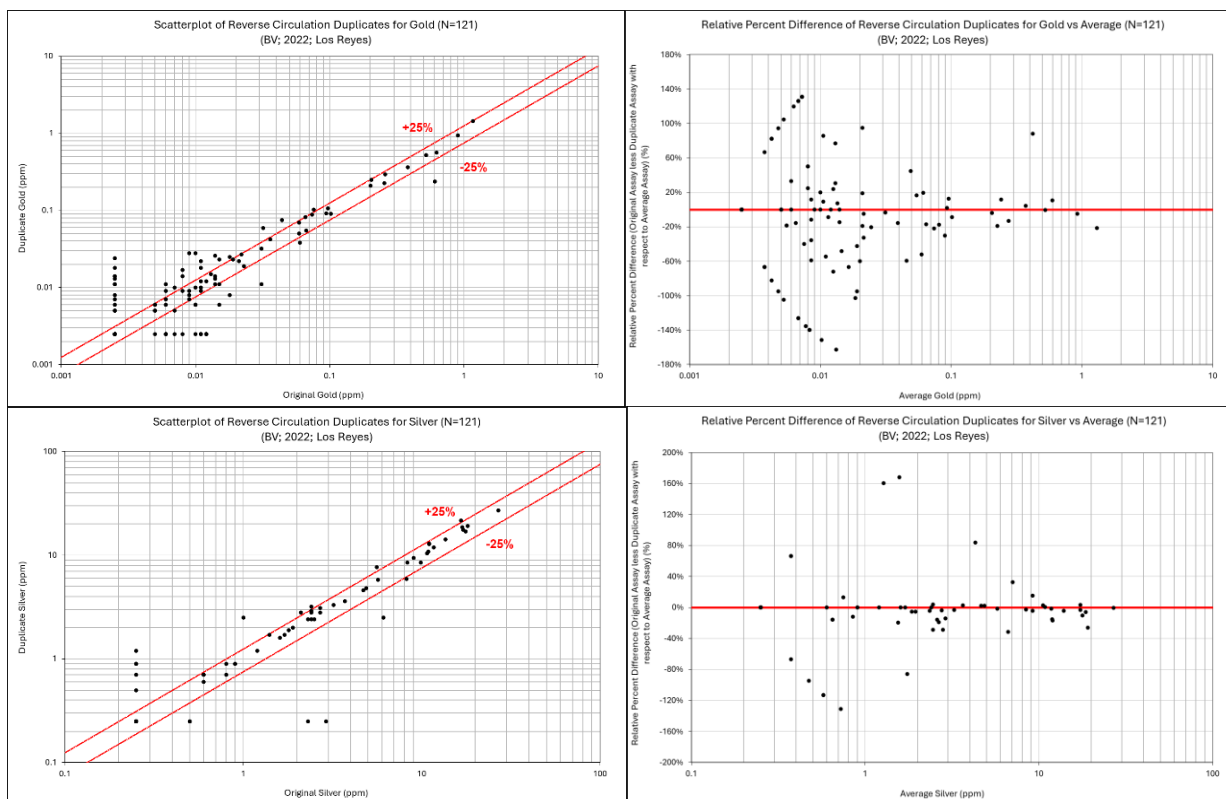
A total of 121 reverse circulation duplicates were collected and submitted for analyses, Table 11-11. A total of 20 reverse circulation duplicates out of 121 reported above 0.05 ppm gold and 19 reported above 5 ppm silver. Eighty-five percent of the duplicate pairs are reporting within $\pm 25\%$ for gold and 74% for silver, Figure 11-10. This is considered acceptable for reserve circulation duplicates.

Table 11-11: Summary of RC duplicate results for gold and silver

Analyte	# of Pairs	# of Pairs above 10x d.l.	% of Sample Pairs (>10x d.l.) Reporting within			
			±5%	±10%	±25%	±50%
Au	121	20	25%	30%	85%	95%
Ag	121	19	47%	53%	74%	84%

Source: Qualitica, 2025

Figure 11-10: XY and RPD chart for gold and silver in reverse circulation, BV



Source: Qualitica, 2025

11.1.4.2.4 Coarse Duplicates

Coarse duplicates are created by splitting a second cut of the crushed sample in the same way and for the same weight as the original sample.

Prime select samples ending in '25', or one sample in one hundred samples as a coarse duplicate. It is expected that the coarse duplicate pairs will be more similar than the quarter core duplicates due to the smaller particle size during sub-sampling.

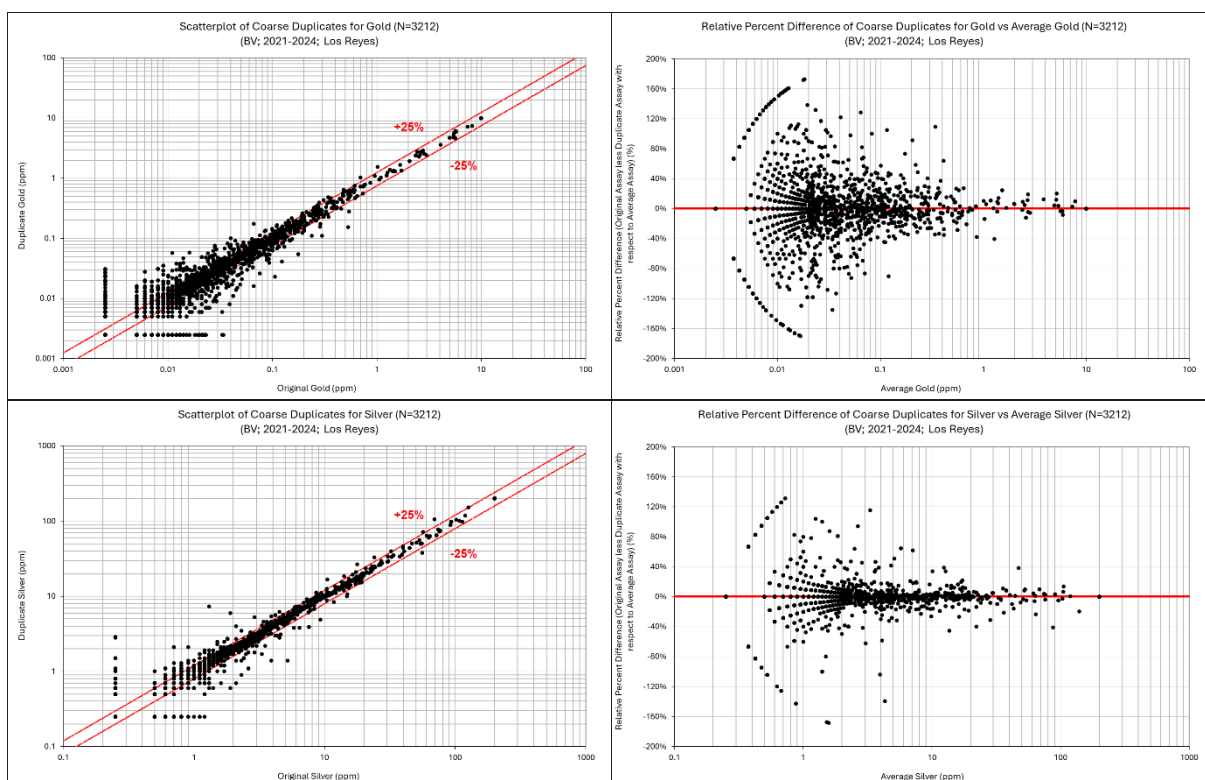
A total of 817 coarse duplicates were analyzed for gold and silver, Table 11-12. A total of 141 duplicate pairs out of 817 reported above 0.05 ppm gold. The coarse duplicates for gold have 74% of the duplicate pairs reporting within ±25%. A total of 83 duplicate pairs out of 817 reported above 5 ppm silver. The coarse duplicates for silver have 93% of the duplicate pairs reporting within ±25%, Figure 11-11. These results are considered acceptable.

Table 11-12: Summary of coarse duplicate results for gold and silver

Analyte	# of Pairs	# of Pairs above 10x d.l.	% of Sample Pairs (>10x d.l.) Reporting within			
			±5%	±10%	±25%	±50%
Au	3212	537	24%	42%	76%	96%
Ag	3212	329	63%	77%	95%	98%

Source: Qualitica, 2025

Figure 11-11: XY and RPD chart for gold and silver in coarse duplicates, 2021-2024, BV



Source: Qualitica, 2025

11.1.4.2.5 Pulp Duplicates

A pulp duplicate is created by taking a second aliquot of the already pulverized sample material in the same way and for the same weight as the original sample. The assays for the pulp duplicate provide an estimate of the reproducibility related to the uncertainties inherent in the analytical method and the homogeneity of the pulp sample.

Prime selects every sample ending in '75', or one sample in one hundred samples to be analysed in duplicate.

There are 3,412 pulp duplicate results for gold, Table 11-13. A total of 532 duplicate pairs out of the 3,412 reported above 0.05 ppm gold. The pulp duplicates for gold have 78% of the duplicate pairs reporting within ±25%. Precision for the gold pulp duplicates is acceptable.

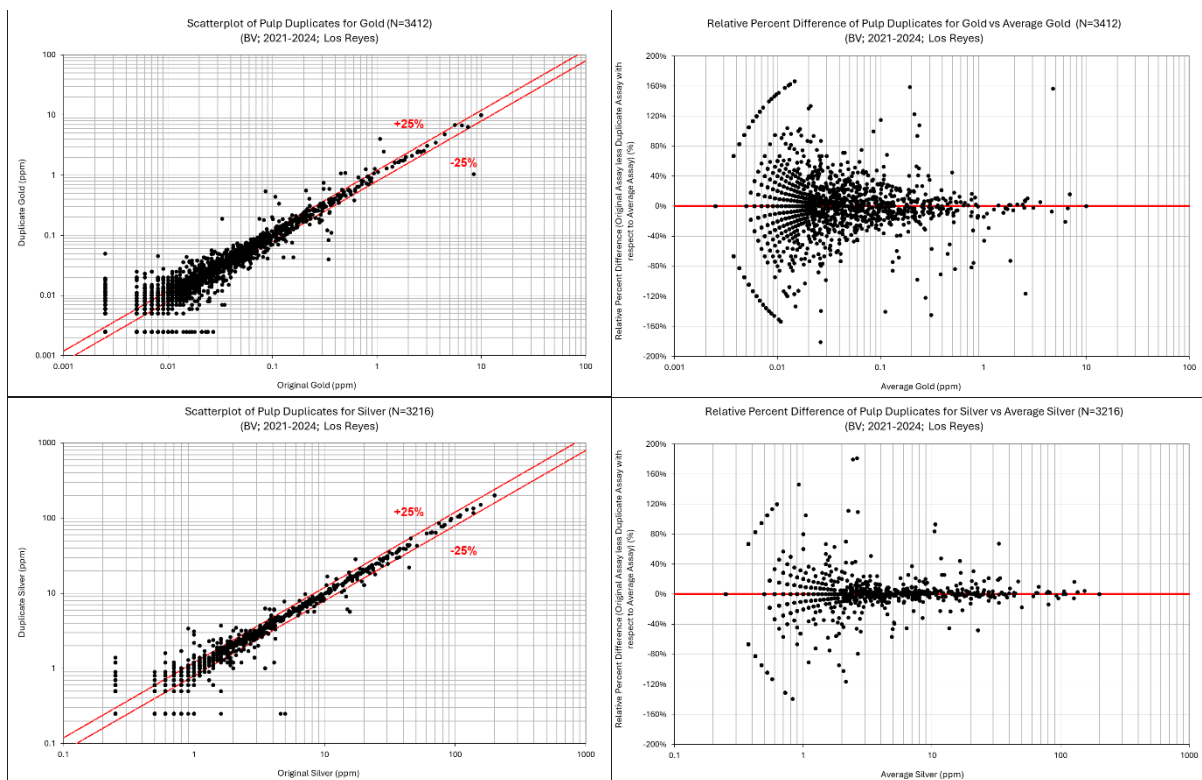
There are 3,216 pulp duplicate results for silver. A total of 307 duplicate pairs out of the 3,216 reported above 5 ppm silver. The pulp duplicates for silver have 93% of the duplicate pairs reporting within $\pm 25\%$, Figure 11-12. Precision for the silver pulp duplicates is acceptable.

Table 11-13: Summary of pulp duplicate results for gold and silver

Analyte	# of Pairs	# of Pairs above 10x d.l.	% of Sample Pairs (>10x d.l.) Reporting within			
			$\pm 5\%$	$\pm 10\%$	$\pm 25\%$	$\pm 50\%$
Au	3412	532	23%	45%	78%	93%
Ag	3216	307	69%	82%	93%	98%

Source: Qualitica, 2025

Figure 11-12: XY and RPD Chart for Gold and Silver in Pulp Duplicates, 2021-2024, BV



Source: Qualitica, 2025

11.1.4.2.6 Check Assays

BV vs SGS

A total of 654 pulps were selected from the samples analysed at BV, Table 11-14. The samples were submitted to SGS for check assaying. The methods from both laboratories are comparable.

Seventy-three percent of the check assay results for gold are within $\pm 25\%$ of the two sets of laboratory results; this is acceptable agreement. The pairs are evenly distributed with 49% being higher at BV and 49% being higher at SGS. The average RPD for gold between BV and SGS is 3%, this indicates that on average

the BV results are higher than SGS results by about 3%. Based on the charts in Figure 11-13, the results are well distributed above and below zero percent relative difference.

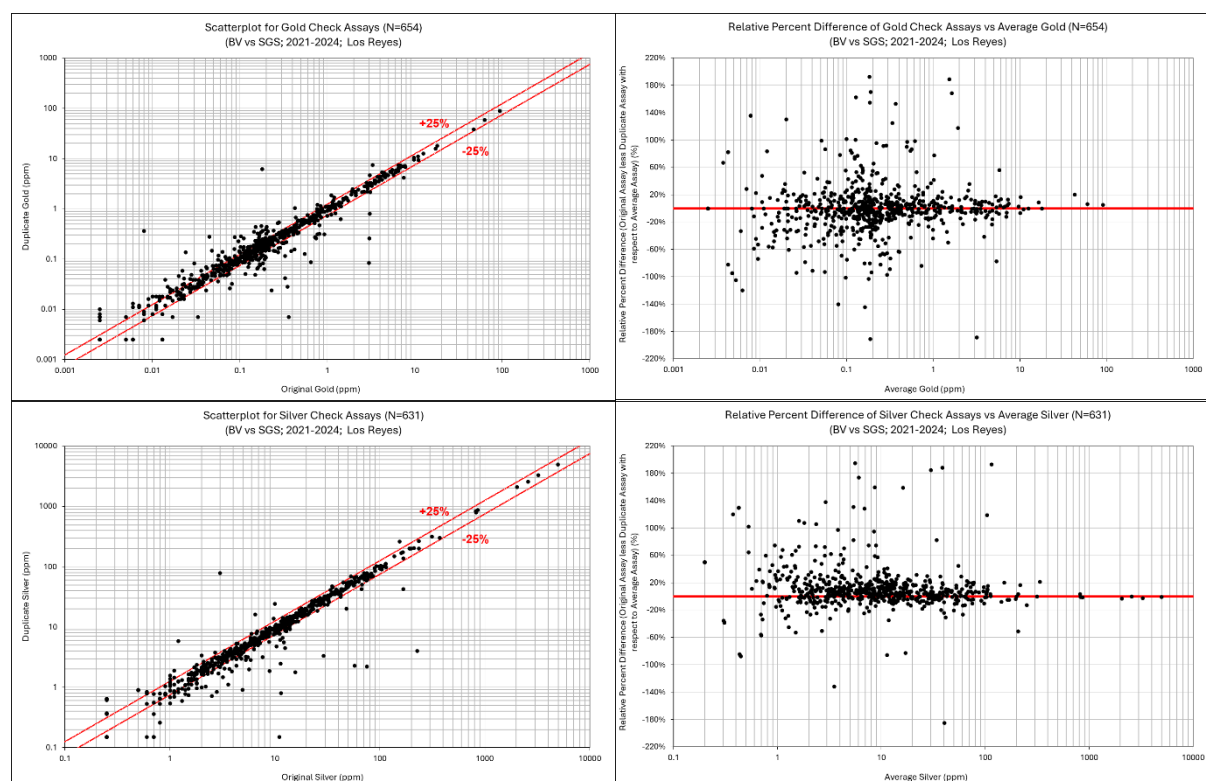
For the silver results, 85% are within $\pm 25\%$ of the two sets of laboratory results. The results tend to report higher at BV than at SGS. The results are mostly plotting above the zero relative percent difference for all grade ranges. The average difference between the two laboratories is 12%, with the results being higher at BV for seventy-seven percent of the case.

Table 11-14: Summary of check assay results for gold and silver

Analyte	# of Pairs	# of Pairs above 10x d.l.	Average RPD	% of Sample Pairs (>10x d.l.) Reporting within			
				$\pm 5\%$	$\pm 10\%$	$\pm 25\%$	$\pm 50\%$
Au	654	542	3%	24%	46%	73%	88%
Ag	631	407	11%	33%	55%	87%	94%

Source: Qualitica, 2025

Figure 11-13: XY and RPD chart for gold and silver in check assays, BV vs SGS



Source: Qualitica, 2025

BV vs ALS

A total of 152 pulps were selected from the samples analysed at BV, Table 11-15. The samples were submitted to ALS for check assaying. The methods from both laboratories are comparable.

Eighty-four percent of the check assay results for gold are within $\pm 25\%$ of the two sets of laboratory results; this is acceptable agreement. In fifty-five percent of cases ALS is higher than BV. The average RPD for gold between ALS and BV is -1%, this indicates that on average the ALS results are higher than BV results by about 1%. Based on the charts in Figure 11-14, the results are well distributed above and below zero percent relative difference.

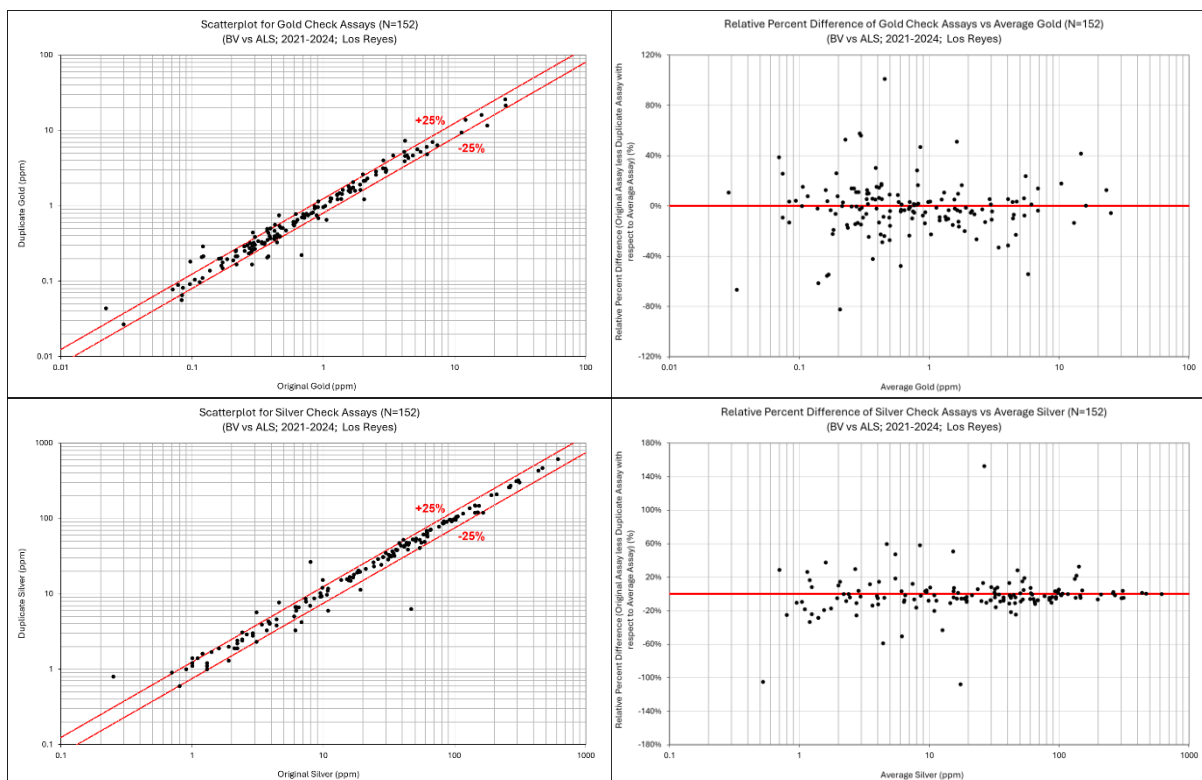
For the silver results, 92% of the duplicate pairs are within $\pm 25\%$. In 63% of cases ALS is higher than BV. The average RPD for silver between ALS and BV is 0.5%, this indicates that on average the ALS results are lower than BV results by about 0.5%. The results are well distributed above and below the zero relative percent difference line for all grade ranges.

Table 11-15: Summary of check assay results for gold and silver

Analyte	# of Pairs	# of Pairs above 10x d.l.	Average RPD	% of Sample Pairs (>10x d.l.) Reporting within			
				$\pm 5\%$	$\pm 10\%$	$\pm 25\%$	$\pm 50\%$
Au	152	150	-1.4%	31%	53%	84%	93%
Ag	152	115	0.5%	43%	76%	92%	96%

Source: Qualitica, 2025

Figure 11-14: XY and RPD chart for gold and silver in check assays, BV vs ALS



Source: Qualitica, 2025

In the opinion of the QP, the sample preparation, security, and analytical procedures for all assay data are adequate to support mineral resource estimation.

12 Data Verification

12.1 Verification and Personal Inspection by John Sims

12.1.1 Drill Hole Database Validation

Spot checks of the 2024 drill hole database (DHDB) revealed only minor errors which were flagged and rectified by the Company. Systematic spot-checking of collar information and assays to confirm the DHDB was suitable for ongoing use through the addition of incoming data was performed. The DHDB contained a total of 744 holes (Table 10-2). Not all of the 493 historical drill holes have complete collar, assay and drill log data.

12.1.2 Data and Spatial Validation

During the site investigation of the Property the Independent QP:

- Located drill collars from the previous and current field campaigns and validated coordinates from the database using a handheld Garmin GPS.
- Reviewed the down hole survey and collar locations spatially in Leapfrog software to ensure that drill collars were on topography and there were no unusual drill hole deviations.
- Reviewed drill logging and procedures to confirm the presence of hydrothermal alteration, banded chalcadonic quartz and heterolithic breccias in drill core.
- Validated the logged geology by visiting various outcrops in the field near drilled areas.
- Compared assays in the Access database against the lab assay certificates as part of the validation process.
- Tracked the logging and assay data from the source Access database to the resource modeling data held in Leapfrog software to ensure the data matched as part of the data transfer validation process.
- The 3D wireframes were reviewed spatially in Leapfrog software to validate them with respect to the logged geology and grade based domaining strategy.
- The exploratory data analysis was reviewed and included: target composite length, data isolation, hard vs. soft boundaries, grade capping, and variography.
- The resource estimation process was reviewed and included: block model set up, various estimation methods, interpolation run strategies, model validation, resource classification strategies, metal loss due to capping and grade restrictions, cut-off grade development, and Mineral Resource reporting.

12.1.3 Limitation to Data Validation by Qualified Person

Limitations to the validation that the Independent QP was able to complete are listed below:

- The Independent QP was not involved in the Property prior to 2022 and therefore cannot validate the field procedures used during drilling and sample collection prior to the involvement by Prime.
- Laboratory inspections were not completed by the Independent QP.
- The BGBG.MX law firm did not have access to all of the original tenure documents.

12.1.4 Opinion of the Independent Qualified Person

The Independent QP site inspection, which was conducted by John Sims, CPG., during November 11th to the 15th of 2022, satisfies the NI 43-101 criteria. It is the opinion of the Independent QP that the field procedures and sampling protocols that were implemented by Prime are reasonable. Also, the quality of the laboratory testing completed during the various stages of the Project is reasonable. The Independent QP is confident that the samples and associated laboratory datasets that are used in this Technical Report are accurate.

Based on this review, Mr. Sims has no reason to question the validity of the exploration and/or drilling conducted and/or the results thereon. That is, the adequacy of the data used in this Technical Report is considered accurate in Mr. Sim's opinion.

12.2 Verification of Analytical Quality Control Data

Chantal Jolette from Qualitica Consulting Inc. analyzed the analytical quality control data produced by Prime from 2021 to 2024 from both ALS and BV commercial laboratories.

Prime provided the external analytical control data containing the assay results for the quality control samples. All data were provided in Microsoft Excel spreadsheets. Ms. Jolette aggregated the assay results of the external analytical control samples for further analysis.

Control samples were charted as follows to highlight their performance:

- Control charts for blank material.
- Control charts for reference materials.
- Scatter plot and relative percent difference (RPD) chart for field duplicates.
- Scatter plot and RPD chart for preparation duplicates.
- Scatter plot and RPD chart for pulp duplicates.
- Scatter plot and RPD chart for check assays.

The charts that were plotted to assess the performance of the primary laboratory ALS Global and Bureau Veritas are provided in Section 11. The performance of the analytical control data is discussed in Section 11.

In the opinion of the QP, the sample preparation, security, and analytical procedures for all assay data are adequate to support mineral resource estimation.

12.3 Verification performed by Rodrigo Capel Pasqua

This data verification was carried out by Rodrigo Capel Pasqua, FAusIMM, of AGP Mining Consultants. The mining engineering team's data verification activities were scoped to confirm that the Mineral Resource block models prepared by the Resource QP (Loury Geoscience Consulting LLC) were fit-for-purpose as inputs to the PEA mine planning workflow. Verification was carried out on the block model set with effective date December 31, 2025, comprising the open-pit models (Guad_LP_5x5x5_Rotated and ZTM_NB_SM_5x5x5_Rotated, both at 5 × 5 × 5 m parent-cell discretization) and the underground models (LP, GE, FRE_NB_SME, ZTM, all at 2.5 m octree discretization).

The verification activities undertaken by the mining engineering team were:

- **Block-model field audit.** Each model variable referenced downstream in the mining planning workflow was checked against the block model handover notes provided by the Resource QP. Field names, units, and data types were confirmed to match the documentation, and the population of each field was verified across all model cells.
- **Null, absent, and negative value checks.** Each field was screened for null, absent, and negative values, with particular focus on grade, density, and classification fields that propagate into NSR, AuEq, and mining cost calculations downstream. No anomalous values were identified that would compromise the use of the models in mining planning.
- **Standard Deswik block-model verification.** The default Deswik.CAD block-model verification routine was run on each model. This routine checks for broken parent-cell hierarchy, mis-aligned sub-cells, missing or duplicate cells, and out-of-bound coordinates against the model's declared origin and extents. All block models passed the verification routine without exception.
- **Fitness for mining planning.** Cell sizes, sub-cell discretization, rotation, and coordinate system were confirmed as appropriate for the mining methods to which the models would be applied — namely $5 \times 5 \times 5$ m cells for open-pit Whittle and NPVS optimization, and 2.5 m octree cells for underground stope optimization in MSO.

The mining engineering team did not undertake independent verification of the underlying drillhole database, geological interpretation, or resource estimation parameters. These activities sit with the Resource QP and are described in Section 12.1 of this Technical Report. The mining engineering team relied on the Resource QP's authorization of the block model set as the basis for accepting the models as inputs to the mine planning workflow.

In the opinion of the QP, the block model data described above are adequate for the mining planning purposes for which they have been used in this PEA.

12.4 Verification of Geotechnical Data by Knight Piésold Ltd.

Ben Peacock, P.Eng., from Knight Piésold Ltd. Reviewed the geotechnical data collected by Prime from 2021 to 2024, including Rock Quality Designation (RQD), estimates of intact rock strength, and discontinuity orientations from drillholes with core orientation. In the opinion of the QP, these data are adequate for the purposes of this PEA.

12.5 Verification of Metallurgical Testwork by Ausenco

Peter Mehrfert, P. Eng., from Ausenco reviewed the metallurgical testwork results from laboratories including KCA, RDI, and Forte from 2023 To 2025 including leach, comminution and flotation testwork. In the opinion of the QP, these data are adequate for the purposes of this PEA.

13 Mineral processing and metallurgical testwork

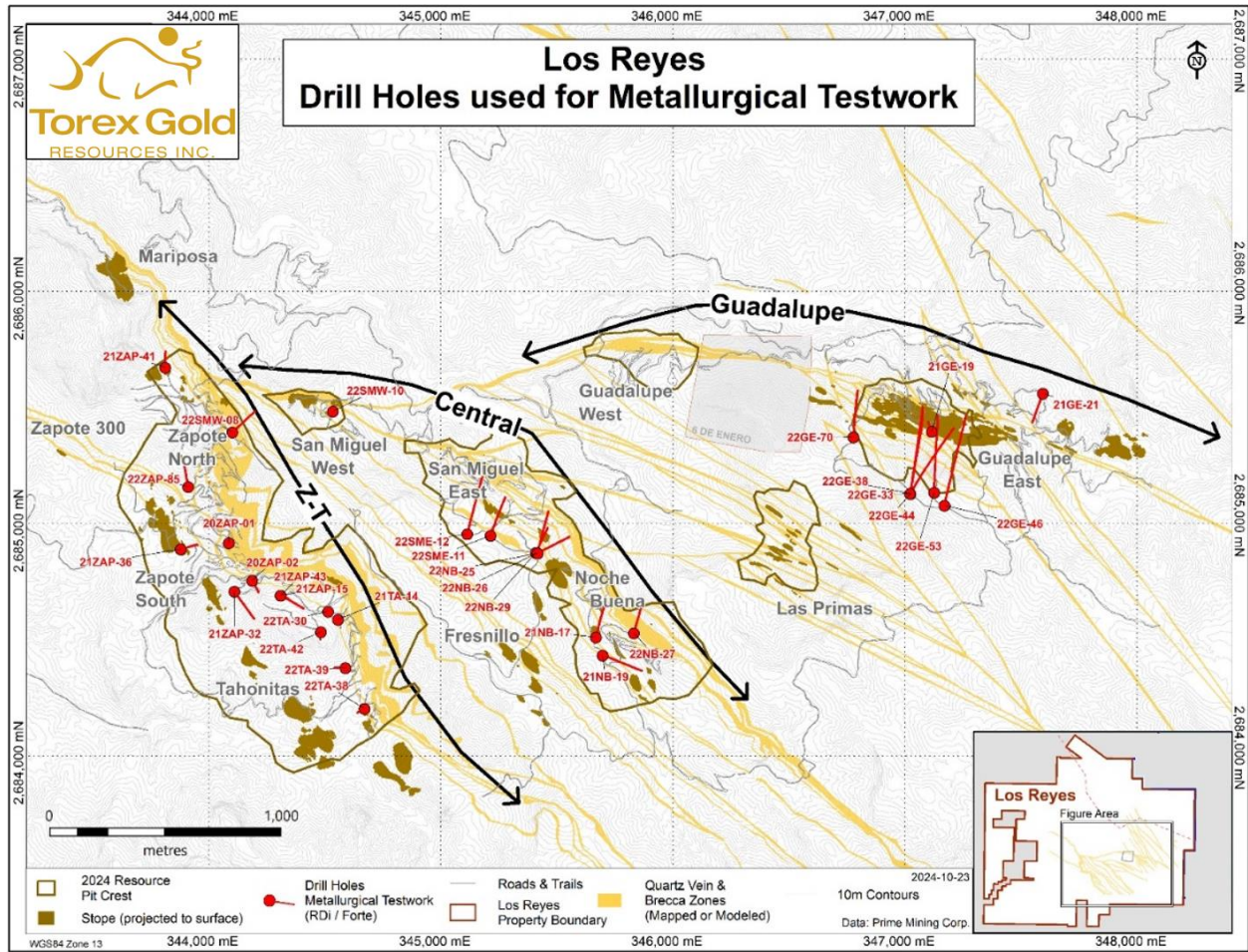
A detailed review of available metallurgical test work was completed by Kappes Cassiday & Associates (KCA) to summarize the metallurgical response of samples from the Los Reyes Project. The metallurgical results were then reviewed and interpreted by Ausenco for the selected process design. A total of five metallurgical test programs have been conducted on samples from the main mineralised areas (Z-T [Zapote & Tahonitas], Central [San Miguel East, San Miguel West & Noche Buena] and Guadalupe [Guadalupe East & Guadalupe West] of the Los Reyes Project by previous owners starting in 1997 including Tenoco (1997), Compañía Minera Campanillas (1998), Vista Gold Corporation (2012) and most recently, between 2020 and 2025 by Prime.

The tests have focused primarily on CIL and direct bottle roll tests with programs completed by Resource Development Inc. (RDi, now Forte Analytical) in 2012, and between 2021 and 2024 and KCA in 2024 with preliminary gravity and flotation test work also being performed by RDi/Forte during this period. A detailed review of available metallurgical test work was completed by KCA to determine preliminary conditions, reagent requirements and metal recoveries for the Los Reyes Project. Drill hole locations for the samples used in the RDi/Forte 2021 to 2023 work are presented Figure 13-1 and drill hole locations used to obtain samples tested by KCA in 2024 are presented in Figure 13-2.

Limited column leach test data are available from a program completed by McClelland Laboratories Mexico (MLM) in 1998 on samples from the Zapote deposit. Additional column leach tests were conducted on four composites by KCA.

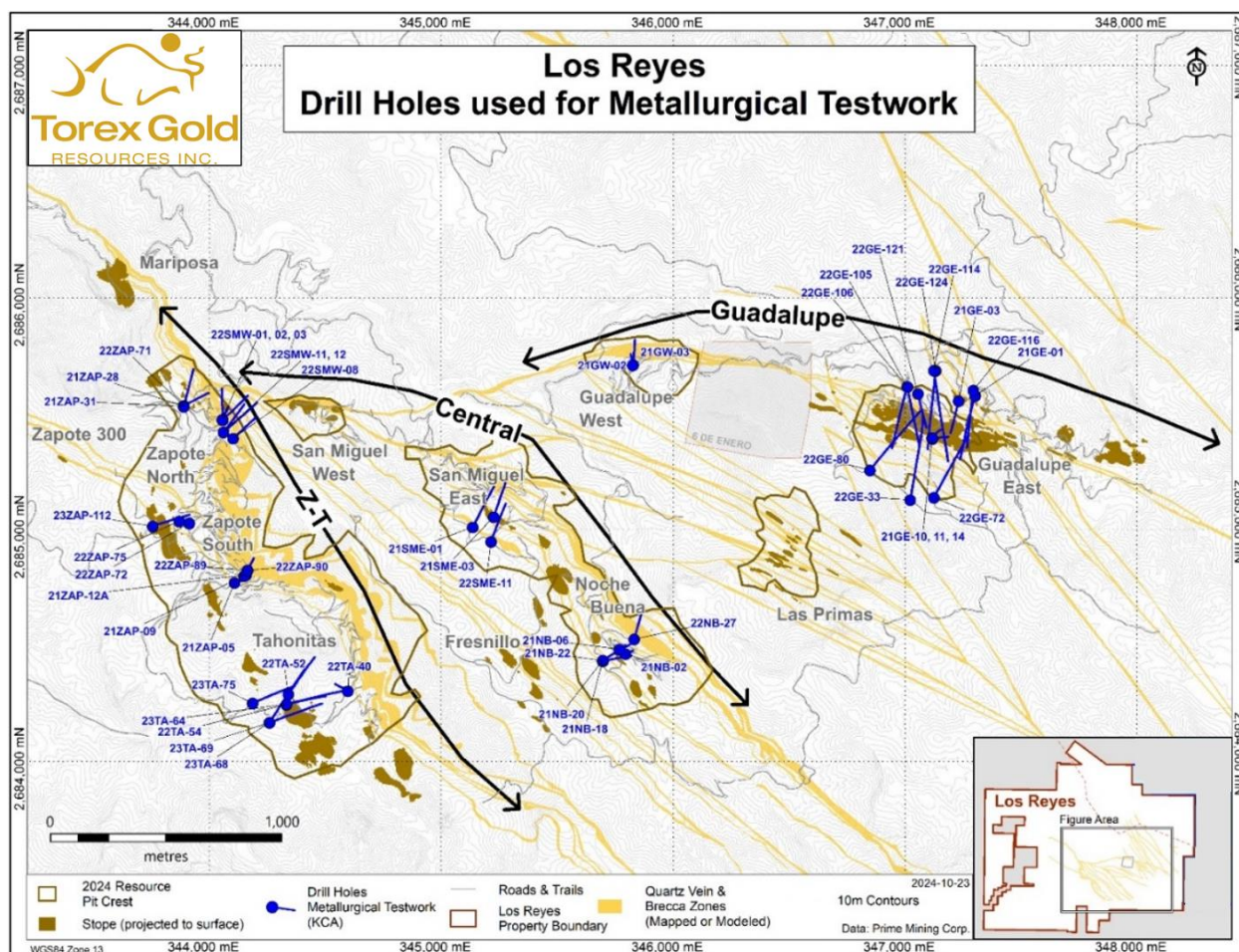
Summarized test work and results relevant to the project are presented in the following sections.

Figure 13-1: Drillhole locations for metallurgical test samples, RDi 2021-2023



Source: Prime, 2024

Figure 13-2: Drillhole locations for metallurgical test samples, KCA 2024



Source: Prime, 2024

13.1 Metallurgical test work results – gravity concentration summary

13.1.1 RDi/Forte gravity test programs 2012, 2021-2023

Gravity concentration testing was completed by RDi/Forte in 2012 and between 2021 and 2023. The 2012 program included two-stage gravity concentration (Knelson followed by Gemini Table) on nine composites generated from drill core rejects. The gravity concentration tests produced gold recoveries ranging between 11.8% and 32.8% and silver recoveries ranging between 1.7% and 24.3% with mass pulls ranging from trace amounts to 0.4% of the initial feed weight. Results from the 2012 RDi gravity program are presented in Table 13-1.

The 2021 to 2023 gravity concentration tests included single stage gravity concentration using a Knelson concentrator on eight composites from drill core rejects. Gemini table cleaning of the Knelson concentrate was also reported; however, no weights were presented and is therefore not considered. Gravity recoveries ranging between 4.3% and 51% for gold and 8% and 23% for silver with mass pulls ranging between 0.2% and 9.1% were achieved. For both the 2012 and 2021-2023 test work the higher recoveries were closely

associated with higher mass pulls. Results from the 2021-2023 RDi/Forte test work are presented in Table 13-2.

Table 13-1: 2012 RDi gravity test work results

Product	Assay, g/t		Distribution %		
	Gold	Silver	Weight	Gold	Silver
Composite No. 1: P₈₀ = 300µm (T-1), GLW					
Gemini Conc.	119.5	6948.6	0.3	17.1	16.9
Gemini Tail	5.07	279.3	12.3	28.3	26.6
Cal. Knelson Conc.	7.92	445.9	12.6	45.4	43.5
Knelson Tail	1.37	83.3	87.4	54.6	56.5
Cal. Feed	2.19	128.9	100.0	100.0	100.0
Composite No. 1: P₈₀ = 150µm (T-2), GLW					
Gemini Conc.	160.4	8927.8	0.3	23.1	20.8
Gemini Tail	4.28	298.2	10.2	20.6	23.2
Cal. Knelson Conc.	8.82	549.2	10.5	43.7	44.0
Knelson Tail	1.34	82.2	89.5	56.3	56.0
Cal. Feed	2.13	131.4	100.0	100.0	100.0
Composite No. 2: P₈₀ = 300µm (T-3), GL Vein					
Gemini Conc.	123.7	5756.6	0.3	13.6	8.5
Gemini Tail	5.11	344.3	11.9	21.2	19.0
Cal. Knelson Conc.	8.19	484.7	12.2	34.8	27.5
Knelson Tail	2.13	177.7	87.8	65.2	72.5
Cal. Feed	2.87	215.2	100.0	100.0	100.0
Composite No. 2: P₈₀ = 150µm (T-4), GL Vein					
Gemini Conc.	125.5	6488.1	0.4	15.2	10.4
Gemini Tail	5.2	415.0	10.8	20.0	19.9
Cal. Knelson Conc.	9.39	610.7	11.1	35.2	30.2
Knelson Tail	2.16	176.1	88.9	64.8	69.8
Cal. Feed	2.96	224.4	100.0	100.0	100.0
Composite No. 3: P₈₀ = 300µm (T-5), Zapote					
Gemini Conc.	450.6	778.4	0.1	13.2	2.5
Gemini Tails	5.45	36.7	12.1	21.9	16.2
Cal. Knelson Conc.	8.67	42.1	12.2	35.1	18.7
Knelson Tails	2.23	25.4	87.8	64.9	81.3
Cal. Feed	3.02	27.4	100.0	100.0	100.0
Composite No. 3: P₈₀ = 150µm (T-6), Zapote					
Gemini Conc.	581.1	586.7	0.1	20.4	1.7
Gemini Tails	6.27	47.7	9.3	20.6	12.9
Cal. Knelson Conc.	12.37	53.4	9.4	41.0	14.6
Knelson Tails	1.85	32.6	90.6	59.0	85.4
Cal. Feed	2.84	34.6	100.0	100.0	100.0
Composite No. 5: P₈₀ = 212µm (T-7), NB					

Product	Assay, g/t		Distribution %		
	Gold	Silver	Weight	Gold	Silver
Gemini Conc.	97.09	875.1	0.2	22.2	24.3
Gemini Tail	1.86	23.8	11.0	23.0	35.7
Cal. Knelson Conc.	3.59	39.3	11.2	45.2	60.0
Knelson Tail	0.55	3.3	88.8	54.8	40.0
Cal. Feed	0.89	7.3	100.0	100.0	100.0
Composite No. 6: P₈₀ = 212µm (T-8), Zapote					
Gemini Conc.	1836.4	2154.4	Trace	15.4	1.7
Gemini Tail	7.20	47.5	11.3	16.8	10.7
Cal. Knelson Conc.	13.78	55.1	11.3	32.2	12.4
Knelson Tail	3.71	49.5	88.7	67.8	87.6
Cal. Feed	4.85	50.1	100.0	100.0	100.0
Composite No. 7: P₈₀ = 212µm (T-9), Zapote					
Gemini Conc.	1111.3	3371.2	Trace	11.8	3.1
Gemini Tail	4.58	56.8	10.8	17.8	18.9
Cal. Knelson Conc.	7.59	65.8	10.8	29.6	22.0
Knelson Tail	2.19	28.3	89.2	70.4	78.0
Cal. Feed	2.77	32.4	100.0	100.0	100.0
Composite No. 9: P₈₀ = 150µm (T-10), SM					
Gemini Conc.	5698.7	16544.9	0.1	32.8	8.3
Gemini Tail	23.25	212.2	11.6	25.5	20.2
Cal. Knelson Conc.	52.89	297.5	1.7	58.3	28.5
Knelson Tail	5.01	98.9	88.3	41.7	71.5
Cal. Feed	10.61	122.1	100.0	100.0	100.0

Table 13-2: 2021-2023 RDi gravity test work results

Test #		GC 1	GC 2	GC 3	GC 4	GC 5	GC 6	GC 7	GC 8
DH #		21GE-19	22GE-38	22GE-38	22GE-38	21TA-14	21TA-14	21TA-14	22GE-70
Composite		11936-11945	22871-22910	222871-22910	222871-22910	19851-19864	19851-19864	19851-19864	50705-50774
Grind, P ₈₀	Tyler	100 mesh	48 mesh	65 mesh	100 mesh	48 mesh	65 mesh	100 mesh	Pre-Screened
	microns	150	300	212	150	300	212	150	48,100,325
Feed	Wt., g	4040	988	992	994	991	992	993	2348
	Wt., %	100	100	100	100	100	100	100	100
	Au g/t	1.71	1.01	0.79	0.64	0.39	0.38	0.40	0.80
	Ag g/t	55	27	28	22	31	50	53	32
Knelson Tail	Wt., g	4022.7	898.8	913.2	933.5	930.7	934.2	946.5	2342.9
	Wt., %	99.6	91.0	92.1	93.9	93.9	94.2	95.3	99.8
	Au g/t	1.17	0.55	0.53	0.34	0.31	0.31	0.30	0.77
	Ag g/t	51.9	23.2	23.8	18.0	26.2	45.0	47.3	31.1
Gravity Conc.	Wt., g	17.30	89.2	78.8	60.5	60.3	57.8	46.5	5.1
	Wt., %	0.4	9.0	7.9	6.1	6.1	5.8	4.7	0.2
	Au g/t	127.83	5.69	3.82	5.38	1.57	1.47	2.30	15.10
	Ag g/t	775	62	76	84	108	124	162	232
% Recovery to Conc.	Au	32.3%	50.8%	38.4%	50.5%	24.9%	22.6%	27.3%	4.3%
	Ag	6.1%	20.9%	21.7%	23.0%	21.1%	14.5%	14.4%	1.7%

The results from the RDi gravity tests show significant variability for both recovery and mass pull.

13.1.2 Forte gravity test program 2024

Single stage Knelson concentrator tests with cyanide leaching of the gravity tails were performed as part of the 2024 Forte test program. The gravity tests were conducted on 12 different composites samples left over from previous test programs. Gravity recoveries ranged from 30.8% to 74.4% for gold and 13% to 39% for silver with mass pulls ranging between 3.1% and 7.4%. Recoveries from the bottle roll leach tests on the gravity tails ranged between 77.1% and 97.4% for gold and 26.6% and 99.9% for silver with overall recoveries ranging from 89.5% to 99.9% for gold and 40.4% to 99.9% for silver. Results for the 2024 gravity test work are presented in Table 13-3 (gravity concentration), Table 13-4 (gravity tailings leach) and Table 13-5 (gravity and leach combined).

Table 13-3: 2024 Forte gravity concentration results

Sample #	Test #	Gravity Concentrate Weight (g)	Mass Fraction in Concentrate	Gold Measured Head (g/t)	Gold Calculated Head (g/t)	Gravity Concentrate Grade Gold (g/t)	% Gold Extracted by Knelson	Initial Assayed Head Grade Silver (g/t)	Silver Calculated Head (g/t)	Gravity Concentrate Grade Silver (g/t)	% Silver Extracted by Knelson
2 - 21GE-21	PBR-18	34.8	0.035	0.28	0.27	3.76	49.5	BD	6	58	0.32
3 - 22GE-46	PBR-22	33.6	0.034	0.37	0.51	6.74	44.1	30	27	151	0.19
4 - 21GE-19	PBR-25	44.2	0.044	1.39	2.57	45.0	77.2	62	61	501	0.36
5 - 22GE-33	PBR-33	67.9	0.068	6.23	5.43	48.6	60.9	143	123	434	0.24
6 - 22GE-53	PBR-26	74.4	0.074	15	14.1	147	77.5	325	302	1577	0.39
7 - 22SME-12	PBR-23	34.1	0.034	0.33	0.72	18.5	87.9	BD	3	25	0.33
8 - 21ZAP-32	PBR-27	31.1	0.031	0.48	3.09	89.4	89.9	24	29	158	0.17
9 - 21ZAP-43	PBR-21	47.3	0.047	1.45	1.32	16.1	57.4	24	24	67	0.13
10 - 21ZAP-36	PBR-20	32.2	0.032	2.65	3.98	83.6	67.6	12	14	79	0.18
12 - 21GE-03	PBR-24	30.8	0.031	11.9	10.1	125	38.3	705	700	5172	0.23
13 - 22TA-39	PBR-19	38.4	0.038	0.39	0.42	2.83	25.9	54	53	270	0.2
14 - 22NB-27	PBR-34	31.5	0.032	4.72	4.47	47.0	33.5	102	96	516	0.17

Table 13-4: 2024 Forte gravity tails leach test results

Sample #	Test #	Gold Recovery in Leach %	Gold Calc. Leach Head Grade (g/t)	Residue Grade Gold (g/t)	Silver Recovery in Leach %	Silver Calc. Leach Head Grade (g/t)	Residue Grade Silver (g/t)	NaCN Consumption (kg/mt)	Lime Consumption (kg/mt)
2 - 21GE-21	PBR-18	79.2	0.14	0.03	77.8	5	BD	2.28	2.28
3 - 22GE-46	PBR-22	93.3	0.30	0.02	91.3	23	BD	1.66	2.36
4 - 21GE-19	PBR-25	95.1	0.61	0.03	97.5	41	BD	1.64	2.11
5 - 22GE-33	PBR-33	97.4	2.28	0.06	99.9	101	BD	2.04	1.08
6 - 22GE-53	PBR-26	97.4	3.40	0.09	92.5	200	15	2.03	1.92
7 - 22SME-12	PBR-23	77.1	0.09	0.02	83.4	2	BD	1.31	2.99
8 - 21ZAP-32	PBR-27	96.8	0.32	BD	28.4	25	18	1.19	1.95
9 - 21ZAP-43	PBR-21	93.2	0.59	0.04	45.8	22	12	1.24	2.72
10 - 21ZAP-36	PBR-20	97.0	1.33	0.04	26.6	12	9	1.85	2.09
12 - 21GE-03	PBR-24	96.0	6.43	0.26	86.9	558	73	1.89	4.44
13 - 22TA-39	PBR-19	90.7	0.32	0.03	65.8	44	15	1.28	2.09
14 - 22NB-27	PBR-34	97.4	3.09	0.08	97.6	83	BD	1.07	0.79

Note: Conditions: Grind (P₈₀): 200 Mesh, Pb(NO₃)₂ 200 kg/t, Leach Time: 48 Hours

Table 13-5: 2024 Forte combined gravity and leach results

Sample #	Test #	Individual Gold Recovery %		Gravity plus Leach Au Recovery %	Individual Silver Recovery %		Gravity plus Leach Ag Recovery %
		Gravity	Leach		Gravity	Leach	
2 - 21GE-21	PBR-18	49.5	40	89.5	32	53	84.8
3 - 22GE-46	PBR-22	44.1	52.2	96.3	19	74	92.9
4 - 21GE-19	PBR-25	77.5	21.9	98.9	36	62	98.4
5 - 22GE-33	PBR-33	60.9	38.1	99.0	24	76	99.9
6 - 22GE-53	PBR-26	77.5	21.9	99.4	39	57	95.4
7 - 22SME-12	PBR-23	87.9	9.3	97.2	33	56	88.9
8 - 21ZAP-32	PBR-27	89.9	9.8	99.7	17	24	40.4
9 - 21ZAP-43	PBR-21	57.4	39.7	97.1	13	40	52.9
10 - 21ZAP-36	PBR-20	67.6	31.4	99.0	18	22	39.6
12 - 21GE-03	PBR-24	38.3	59.2	97.5	23	67	89.9
13 - 22TA-39	PBR-19	25.9	67.2	93.1	20	53	72.5
14 - 22NB-27	PBR-34	33.5	64.8	98.3	17	81	98.0

Notes: Leach recovery calculated based on gravity test calculated heads.

Bottle roll conditions: Grind (P₈₀): 200 mesh, Pb(NO₃)₂ 200 kg/t, Leach time: 48 h.

Although the gravity test results showed significant variability, the gravity tests combined with cyanide leaching of the tails showed consistently high overall recoveries. Further, when compared to the CIL bottle roll leach tests on the same samples presented in Table 13-20, the combined gravity and leach tests resulted in an average increased recovery of 1.6% for gold and 6.8% for silver. Additional gravity and leach testing is recommended for future test programs.

13.2 Metallurgical test work results – flotation

13.2.1 RDi/Forte 2012, 2021-2023 flotation test programs

Preliminary rougher flotation testing was completed by RDi/Forte as part of test programs performed in 2012 and between 2021-2023. The 2012 flotation work was performed on three separate composites (two from Guadelupe West and one from Zapote) from drill core rejects at varying grind sizes and with different collectors. The 2021-2023 flotation tests were performed on two separate composites (one from Guadelupe East and one from Tahonitas) from drill core rejects at varying grind sizes. Results from the flotation test work are presented in Table 13-6 and Table 13-7 for the 2012 and 2021-2023 programs, respectively.

Table 13-6: 2012 RDi flotation test results

Composite #	Composite Description	Grind P ₈₀ mesh	Reagents	Wt. %	Concentrate (9 min's)				Assay, g/t			
					Recovery %		Grade, g/t		Tailings		Cal. Feed	
					Au	Ag	Au	Ag	Au	Ag	Au	Ag
1	GLW	100	PAX, AP404	6.2	85.7	90.9	27.8	1910	0.31	12.7	2.02	131
1	GLW	200	PAX, AP404	9.3	91.5	95.7	21.0	1351	0.20	6.2	2.13	131
1	GLW	200	AP404, AP3477	15.0	90.8	99.4	13.0	800	0.23	<1.7	2.14	121
2	GL Vein	100	PAX, AP404	5.8	88.1	89.4	50.1	3394	0.42	24.8	3.30	220
2	GL Vein	200	PAX, AP404	11.8	89.6	88.2	23.8	1643	0.37	29.3	3.13	219
2	GL Vein	200	AP3477, AP404	7.3	88.4	94.4	41.3	2737	0.43	12.7	3.40	211
3	Zapote	100	PAX, AP404	4.6	77.7	15.4	50.4	130	0.70	34.6	3.00	39
3	Zapote	200	PAX, AP404	8.2	86.7	22.0	31.4	83	0.43	26.3	2.98	31
3	Zapote	200	AP3477, AP404	7.8	86.8	23.5	34.8	97	0.45	26.6	3.13	32

Table 13-7: 2021-2023 RDi/Forte flotation test results

Test #	Drill Hole	Sample #	Grind (P ₈₀)	Conc. Wt%	% Recovery (Au)	Conc. Grade (Au g/t)	Calc. Head Grade (Au g/t)	% Recovery (Ag)	Conc. Grade (Ag g/t)	Calc. Head Grade (Ag g/t)
FT1	22GE-38	22871-22910	100 mesh	5	93.3	13.2	0.7	86.4	437	25
FT2	22GE-38	22871-22910	150 mesh	7	93.5	9.54	0.72	89.5	361	28
FT3	22GE-38	22871-22910	200 mesh	17.6	95.3	4.7	0.87	93	155	29
FT4	21TA-14	19851-19864	100 mesh	12.3	84.2	1.91	0.28	64.4	238	45
FT5	21TA-14	19851-19864	150 mesh	12.5	84.3	1.88	0.28	70	240	43
FT6	21TA-14	19851-19864	200 mesh	12.4	85.4	2.07	0.3	85.4	206	40

The flotation testing shows generally good recoveries for gold and silver with moderate mass pulls for the Guadalupe material, with lower recoveries observed for the Zapote and Tahonitas material. Although the flotation work may be able to be optimized, it is noted that overall higher recoveries were achieved through cyanide leaching (see Table 13-17, Table 13-19, Table 13-20 and Table 13-22). Flotation recoveries for gold and silver do not appear to be significantly influenced by the grind size between P₈₀ 150 µm and 75 µm; however, mass pulls were generally higher at the finer particle sizes. A variety of collectors were tested and all produced similar results.

13.2.2 Forte 2024 flotation test program

Rougher flotation tests were completed on five composite samples from drill core rejects (three samples from Guadalupe East, one sample from San Miguel and one sample from Zapote) as part of the 2024 Forte Analytical test program. The flotation tests were conducted at a grind size of 80% passing 75 µm using reagents Potassium Amyl Xanthate (PAX), Aero Promoter 3477, Aero Promoter 404, MIBC and AeroFroth 65. The flotation tails were subjected to cyanide leaching in CIL bottle roll leach tests and the rougher concentrates were further processed using a Gemini Table. Results for the flotation and flotation concentrate gravity concentration tests are presented in Table 13-8 and flotation tails leach results are presented in Table 13-9.

Table 13-8: 2024 Forte flotation test results

Stage	Product Mass		Assay (g/t)		Distribution %	
	grams	%	Gold	Silver	Gold	Silver
FT-1: Sample 2 - 21GE-21: 28172-28192						
Ro Conc	65.2	6.5	6.5	127	91.9	68.8
Gem Conc	3.1	0.31	7	203	4.6	5.2
Gem Tail	62.1	6.2	6.5	123	87.3	63.6
Ro Tail	936.1	93.5	0.04	4	8.1	31.2
Feed (calculated)	1001.3	100	0.46	12	100	100
FT-2: Sample 3 - 22GE-46: 32712-32717						
Ro Conc	56.2	5.6	4.9	456	39.2	84.4
Gem Conc	2.7	0.27	16	1126	6.2	10.0
Gem Tail	53.5	5.3	4.3	422	33.0	74.4
Ro Tail	945.3	94.4	0.45	5	60.8	15.6
Feed (calculated)	1001.5	100	0.70	30	100	100
FT-3: Sample 4 - 21GE-19: 11936-11945						
Ro Conc	65.6	6.6	22.5	798	92.9	88.9
Gem Conc	2.8	0.28	321	3607	56.7	17.2
Gem Tail	62.8	6.3	9.1	673	36.2	71.8
Ro Tail	931.9	93.4	0.12	7	7.1	11.1
Feed (calculated)	997.5	100	1.59	59	100	100
FT-4: Sample 7 - 22SME-12: 47614-47622						
Ro Conc	40.4	4.0	9.5	62	62.5	46.4
Gem Conc	0.7	0.07	352	416	39.9	5.4
Gem Tail	39.7	3.9	3.5	56	22.6	41.1
Ro Tail	966.4	96.0	0.24	3	37.5	53.6
Feed (calculated)	1006.8	100	0.61	5	100	100
FT-5: Sample 9 - 21ZAP-43: 31100-31114						
Ro Conc	50.4	5.0	17.6	140	50.1	29.3
Gem Conc	0.8	0.08	463	658	20.9	2.2
Gem Tail	49.6	5.0	10.4	132	29.2	27.1
Ro Tail	949.6	95.0	0.93	18	49.9	70.7
Feed (calculated)	1000.0	100	1.77	24	100	100

Table 13-9: 2024 Forte flotation tailings leach test results

Sample #	Test #	Flot. Tailings Product Grade (g/t)		Calc. Leach Head Grade (g/t)		Residue Grade (g/t)		Leach Extraction %		NaCN Consumption
		Gold	Silver	Gold	Silver	Gold	Silver	Gold	Silver	kg/t
2 - 21GE-21	PBR-28	0.04	4	0.05	2.7	0.01	0.8	88.9	70.3	1.13
3 - 22GE-46	PBR-29	0.45	5	0.07	5	0.01	1.2	92.9	75.9	0.83
4 - 21GE-19	PBR-32	0.12	7	0.14	13.1	0.03	6.8	77.9	48.1	1.26
7 - 22SME-12	PBR-30	0.24	3	0.04	1.7	0.01	1	76.7	41.3	1.07
9 - 21ZAP-43	PBR-31	0.93	18	0.25	22.9	0.04	15.8	83.9	30.9	1.13

The rougher flotation showed variable recoveries for gold and silver with gold recoveries ranging between 40% and 93% and silver recoveries ranging between 29% to 89% and mass pulls ranging between 4.0% and 6.6%. Recoveries for the flotation tails leach were generally good averaging 84% for gold and 53% for silver.

A 10 kg simulated complete flowsheet (gravity/flotation/leach) was also completed on one sample from Guadalupe East as part of the 2024 program. The sample was ground to 80% passing 75 µm and processed through a Knelson concentrator, the 72-gram concentrate was then passed over a Gemini table to evaluate upgrading. Rougher flotation was applied to the Knelson tails using previously applied conditions. A bottle roll leach test was conducted on a 1 kg sub-sample of the rougher tailings. Overall circuit results are presented in Table 13-10. Overall flowsheet recoveries were 97.5% for gold and 92.1% for silver, although some uncertainty is associated with the silver leach extraction.

Table 13-10: 2024 Forte 10 kg test - Overall Metallurgical Balance

Stage	Product Mass		Assay (g/t)		Distribution %	
	grams	%	Gold	Silver	Gold	Silver
Gemini Con	20.0	0.2	36	1564	9.7	13.1
Gemini Tailings	51.9	0.5	5	198	3.5	4.3
Flotation Con	586.7	5.8	8.47	273	66.7	66.9
Flotation Tailings	9413.3	93.5	0.16	4	20.2	15.7
CN Leach Solution	-	-	-	-	17.7	7.9
Flot Tails Residue	9413.3	93.5	0.02	2	2.5	7.9
Feed (calculated)	10072	100	0.74	24	100.0	100.0

Flotation test results suggest that flotation by itself is not the most effective processing method. As well, the rougher concentrates generated may be subject to poor smelter payment terms, cleaner upgrading would be required. Additional optimization testing would be required to determine if flotation would provide a net benefit to a circuit that also included leaching with cyanide.

13.3 Metallurgical test work results – cyanide leaching

Cyanidation test work includes column leach tests and coarse and fine direct bottle roll leach tests completed by MLM in 1998 and various direct and CIL bottle leach tests by RDi/Forte in 2012 and 2021 to 2024 and KCA in 2024. The 2012 RDi program evaluated several variability parameters including grind size, cyanide concentration, pulp density, CIL vs. direct leaching, and leaching with or without lead nitrate. Additional variability CIL bottle roll tests were completed for the 2021-2023 RDi/Forte program at different grind sizes and with or without lead nitrate. The 2024 Forte test program included lead nitrate addition optimization testing and CIL bottle roll leach tests and the 2024 KCA program included variability direct bottle roll leach tests at varying grind sizes. Cyanidation leach tests and results are discussed in the following section.

13.3.1 MLM test program 1998

Column leaching test work was completed by MLM in 1998 on samples from the Zapote deposit classified as "Zapote Saddle", "Gaitan Footwall" and "Gaitan Stringer". The program evaluated crush sizes ranging

between 6.3 mm (1/4") and 12.7 mm (1/2") and the leach durations ranged from 47 to 67 days. Results from the column tests are presented in Table 13-11.

Table 13-11: 1998 MLM column leach test results

Sample	P ₁₀₀ (mm)	Leach (days)	Feed Grade (g/t)		Recovery %		NaCN kg/t	Lime kg/t
			Au	Ag	Au	Ag		
Zapote Saddle	12.7	67	3.31	17	64.6	14.3	1.45	1
	9.53	53	3.31	17	62.8	15.2	1.23	1.2
	6.35	53	3.31	17	71.6	17.5	1.17	1.1
Gaitan Footwall	12.7	67	2.8	18	85.8	17.1	1.25	1
	9.53	67	2.8	18	83.9	18.6	1.37	0.9
	6.35	47	2.8	18	91.1	19.5	1.34	1
	6.35	67	2.8	18	83.9	19.8	1.34	1
Gaitan Stringer	12.7	60	1.73	28	83.7	26.9	1.24	1.3
	9.53	60	1.73	28	81.5	29.9	1.19	1.2
	6.35	53	1.73	28	94.3	35.0	1.31	1.2

The column tests showed modest to good recoveries for gold ranging between 63% and 94% with lower recoveries for silver ranging between 14% and 35% and moderate requirements for lime and cyanide, respectively. Gold and silver recoveries generally improved with finer crushing.

Corresponding coarse and fine direct bottle roll leach tests were also conducted with results presented in Table 13-12.

Table 13-12: 1998 MLM direct bottle roll leach test results

Sample	Crush size, mm	% Recovery, Au	% Recovery, Ag	NaCN, kg/t	Lime, kg/t
A-Zapote Saddle	- 12.7	37.5	13.8	0.14	1.3
	- 2	59.6	25.2	0.14	1.5
	- 9.53	40.7	23.6	0.14	1.5
	- 6.35	47.6	16.8	0.23	1.4
C-Gaitan Footwall	- 12.7	57.5	12.6	0.07	1.2
	- 2	73	25.2	0.16	1.2
	- 9.53	59.6	13.1	0.16	1.2
	- 6.35	64.1	17.8	0.16	1.2
D-Gaitan Stringer	- 12.7	57.5	22.2	0.06	1.6
	- 2	72	35.7	0.09	1.8
	- 9.53	62.4	23.4	0.14	1.5
	- 6.35	65	29.4	0.22	1.5
Composite ZPF-01	- 2	60.5	25.3	0.23	2.1
Composite ZPF-02	- 2	64.2	27	0.29	2.7
Composite ZPF-03	- 2	58.2	21.8	0.3	2.6

Sample	Crush size, mm	% Recovery, Au	% Recovery, Ag	NaCN, kg/t	Lime, kg/t
Composite ZPF-04	- 2	47.5	22.9	0.4	2.2
Composite ZPF-05	- 2	57.5	20.1	0.15	2
Composite ZPF-06	- 2	60.5	23.5	0.22	1.9
Composite ZPF-07	- 2	43.0	12.0	0.16	1
Composite ZPF-08	- 2	75.4	27.4	0.37	1.4

Similar to the column leach tests, the bottle roll tests showed improved recoveries with finer crushing with notably lower recoveries and cyanide consumption as compared to the column leach tests at the same crush sizes. KCA believes this discrepancy is likely due to the bottle roll tests being limited to 24-hours.

Overall, the samples tested show that they are amenable to cyanide leaching and that gold recovery by heap leaching could be viable.

13.3.2 RDi test program 2012

Variability bottle roll leach tests were performed by RDi in 2012 on drill core reject samples from Guadalupe West, Guadalupe Vein Zone, and the Zapote deposits.

The variability program initially evaluated material grind size vs. recovery in order to determine the grind size to use for the remaining variability testing. The tests were conducted over 48 hours with 1 g/L NaCN. Based on the results presented in Table 13-13, a grind size of 150 mesh was selected.

Table 13-13: 2012 RDi grind size variability bottle roll leach test results

Test No.	Composite	Composite Description	Grind P ₈₀ mesh	Extraction % (48 hrs.)		Residue, g/t		Calculated Head, g/t		NaCN Consumption kg/t
				Au	Ag	Au	Ag	Au	Ag	
1	1	GLW	65	70.7	34.7	0.69	93	2.30	142	0.169
2	1	GLW	100	57.7	36.2	1.30	89	3.08	140	0.538
3	1	GLW	150	79.8	31.4	0.45	104	2.03	152	1.138
4	1	GLW	200	84.7	38.1	0.31	81	2.02	131	1.259
10	2	GL Vein Zone	65	87.0	48.4	0.41	106	3.13	206	0.597
11	2	GL Vein Zone	100	87.9	43.1	0.37	118	3.05	208	1.258
12	2	GL Vein Zone	150	89.3	44.3	0.34	116	3.16	208	1.380
13	2	GL Vein Zone	200	92.6	47.5	0.24	108	3.25	205	1.318
19	3	El Zapote	65	83.0	28.5	0.52	21	3.04	30	0.109
20	3	El Zapote	100	89.4	34.1	0.33	19	3.11	28	0.170
21	3	El Zapote	150	93.0	37.9	0.20	19	2.80	31	1.139
22	3	El Zapote	200	92.8	38.9	0.22	17	3.02	208	1.198

Additional parameters evaluated during the variability program included cyanide concentration, pulp density, and the effect of direct leaching vs. CIL leaching vs. CIL leaching with lead nitrate addition. Results from these tests are presented in Table 13-14 through Table 13-16, respectively.

Table 13-14: 2012 RDi variable cyanide concentration bottle roll leach test results

Test No.	Composite	Composite Description	NaCN Concentration g/L	Extraction % (48 hrs.)		Residue, g/t		Calculated Head, g/t		NaCN Consumption, kg/t
				Au	Ag	Au	Ag	Au	Ag	
5	1	GLW	0.50	78.3	16.8	0.45	106	2.07	127	0.511
6	1	GLW	0.75	81.3	25.7	0.41	107	2.20	144	0.930
3	1	GLW	1.00	79.8	31.4	0.45	104	2.23	152	1.138
14	2	GL Vein Zone	0.50	78.6	26.4	0.69	148	3.22	201	0.631
15	2	GL Vein Zone	0.75	86.8	34.3	0.44	140	3.34	214	0.990
12	2	GL Vein Zone	1.00	89.3	44.3	0.34	116	3.16	208	1.380
23	3	El Zapote	0.50	88.8	47.2	0.35	12	3.08	22	0.480
24	3	El Zapote	0.75	92.4	38.3	0.24	17	3.19	27	0.993
21	3	El Zapote	1.00	93.0	37.9	0.20	19	2.80	31	1.139

Table 13-15: 2012 RDi variable pulp density bottle roll leach test results

Test No.	Composite	Composite Description	Leach Pulp Density, % Solids	Extraction % (48 hrs.)		Residue, g/t		Calculated Head, g/t		NaCN Consumption, kg/t
				Au	Ag	Au	Ag	Au	Ag	
3	1	GLW	40	79.8	31.4	0.45	104	2.23	152	1.138
7	1	GLW	45	83.1	35.0	0.34	82	2.02	125	0.979
8	1	GLW	50	76.8	27.3	0.55	102	2.37	140	0.756
12	2	GL Vein Zone	40	89.3	44.3	0.34	116	3.16	208	1.380
16	2	GL Vein Zone	45	88.7	41.1	0.38	125	3.38	212	1.175
17	2	GL Vein Zone	50	92.0	39.6	0.23	118	2.93	195	0.960
21	3	El Zapote	40	93.0	37.9	0.20	19	2.80	31	1.139
25	3	El Zapote	45	92.1	41.9	0.29	15	3.00	26	1.029
26	3	El Zapote	50	89.4	38.6	0.33	18	3.08	29	0.839

Table 13-16: 2012 RDi direct vs. CIL vs. CIL with lead nitrate bottle roll test results

Test No.	Composite	Composite Description	Process Parameters	Calculated Head (g/t)		Residue (g/t)		Extraction % (48 hrs)		NaCN Consumption kg/t
				Au	Ag	Au	Ag	Au	Ag	
3	1	GLW	Leach	2.23	152	0.45	104	79.8	31.4	1.14
9			CIL	2.08	147	0.14	46	93.3	68.7	1.05
28			CIL/ Pb(NO ₃) ₂	1.84	118	0.06	15	96.8	87.3	0.57
12	2	GL Vein Zone	Leach	3.16	208	0.34	116	89.3	44.3	1.38
18			CIL	3.32	206	0.13	65	96.0	68.5	1.64
29			CIL/ Pb(NO ₃) ₂	3.06	210	0.09	23	97.1	89.2	0.75
21	3	El Zapote	Leach	2.80	31	0.2	19	93.0	37.9	1.14
27			CIL	3.00	30	0.14	18	95.4	38.9	1.29
30			CIL/ Pb(NO ₃) ₂	2.67	34	0.09	21	96.5	37.8	0.51
31	5	Noche Buena	Leach	0.62	22	0.07	6	88.4	72.0	1.44
32			CIL	0.77	25	0.07	3	91.1	87.7	1.24
33			CIL/ Pb(NO ₃) ₂	0.79	25	0.06	2	93.1	90.4	0.39
34	6	El Zapote	Leach	4.31	52	0.28	38	93.5	27.3	0.89
35			CIL	4.94	51	0.25	35	95.0	32.1	0.82
36			CIL/ Pb(NO ₃) ₂	4.89	52	0.18	36	96.3	31.3	0.52
37	7	El Zapote	Leach	2.60	44	0.15	29	94.2	34.1	1.02
38			CIL	2.90	47	0.12	28	95.8	39.9	0.76
39			CIL/ Pb(NO ₃) ₂	3.24	47	0.12	27	96.3	42.0	0.46
40	8	San Miguel	Leach	4.82	72	0.42	33	91.3	53.7	1.38
41			CIL	4.81	77	0.15	17	96.8	77.5	1.47
42			CIL/ Pb(NO ₃) ₂	4.22	69	0.05	9	98.9	87.0	0.70
43	9	San Miguel	Leach	14.1	127	2.12	68	84.9	46.3	1.20
44			CIL	16.2	137	0.47	32	97.1	76.7	0.99
45			CIL/ Pb(NO ₃) ₂	13.1	122	0.15	20	98.8	83.4	0.81

The following parameters were recommended by RDi to be used for future test programs based on the results from the initial variability testing:

- grind size of 150 mesh or finer,
- cyanide concentration of 1 g/L,
- pulp density of 40% (very little effect on recovery between 40 and 50%), and
- CIL with lead nitrate addition.

It is noted that the recommendations were focused on gold recoveries only and additional variability testing for silver are recommended.

13.3.3 RDi/Forte test program 2021-2023

A total of 59 CIL bottle roll leach tests were conducted by RDi/Forte between 2021 and 2023 on drill hole composites from the Guadalupe East, Noche Buena, San Miguel East, San Miguel West, Zapote and Tahonitas deposits. The test work is an extension of the work completed in 2012 and tested different material grind sizes and lead nitrate additions. Results are presented in Table 13-17.

Table 13-17: 2021-2023 RDi/Forte CIL bottle roll leach test results

Test	Dhole	Grind P ₈₀ mesh	Pb (NO ₃) ₂	Head, calc g/t		Residue g/t		Extraction %		Consumption kg/t	
			Kg/t	Au	Ag	Au	Ag	Au	Ag	NaCN	Lime
BR1	21GE-19	325	0.2	1.62	64	0.01	1.8	99.1	97.2	1.38	5.33
BR2	21GE-19	200	0.2	0.97	30	0.02	1.2	98.4	96	1.20	3.18
BR3	21NB-17	325	0.2	1.26	12	0.01	0.6	99	95	1.02	3.49
BR4	21NB-19	325	0.2	0.24	14	0.02	0.4	93.8	97.1	1.01	3.75
BR5	21ZAP-01	325	0.2	0.16	14	0.01	8.6	96.8	37.1	0.83	3.56
BR6	22SME-11	325	0.2	2.57	144	0.03	17.4	98.9	87.9	0.92	6.48
BR7	21ZAP-02	325	0.2	0.89	30	0.04	13.8	95.5	53.2	1.02	3.60
BR8	22GE-38	400	0.2	0.78	29	0.01	1.6	98.2	94.4	1.06	4.12
BR9	22TA-30	400	0.2	0.36	20	0.02	2.2	94.4	89.1	0.89	3.72
BR10	22SME-12	400	0.2	0.38	4	0.01	1.1	98.2	72.2	0.83	4.15
BR11	22NB-26	400	0.2	0.64	75	0.02	7.6	97.3	89.8	0.88	5.09
BR12	22SMW-10	400	0.2	1.31	127	0.03	6.2	97.9	95.1	1.19	4.93
BR13	22SMW-10	6	none	0.27	4	0.11	2.0	96.8	38.1	0.51	6.53
BR14	22GE-38	200	0.2	0.98	30	0.01	1.6	98.6	94.7	0.84	3.98
BR15	22GE-38	270	0.2	0.74	28	0.02	1.7	97.2	94	1.19	3.98
BR16	21TA-14	6	none	0.86	40	0.63	19.3	23.6	48.9	0.86	3.59
BR17	21TA-14	400	0.2	0.41	45	0.04	4.9	90.8	89	1.19	3.56
BR18	21ZAP-43	6	none	0.95	22	0.36	17.6	52.7	24.4	0.36	3.57
BR19	21ZAP-43	400	0.2	0.96	24	0.04	10.6	96	56.5	1.02	3.50
BR20	21TA-14	200	0.2	0.39	40	0.05	7.1	87.4	82.3	1.01	3.20
BR21	21TA-14	270	0.2	0.46	34	0.04	3.7	91	89.1	1.24	3.01
BR22	21ZAP-36	400	0.2	2.11	11	0.03	7.9	98.6	29.2	1.18	3.07
BR23	21ZAP-36	6	none	2.13	12	0.65	10.9	51.2	24.9	0.53	3.34
BR24	22GE-70	400	0.2	0.8	19	0.02	0.2	97.1	99.2	1.26	2.98
BR25	21ZAP-32	6	none	0.32	47	0.09	32.3	66.5	5.8	0.27	3.28
BR26	21ZAP-32	400	0.2	0.31	34	0.03	26.6	93.6	22	0.96	3.47
BR27	21ZAP-32	6	none	1.01	38	0.64	35.6	30.6	6.7	0.38	3.33

Test	Dhole	Grind P ₈₀ mesh	Pb (NO ₃) ₂	Head, calc g/t		Residue g/t		Extraction %		Consumption kg/t	
			Kg/t	Au	Ag	Au	Ag	Au	Ag	NaCN	Lime
BR28	21ZAP-32	400	0.2	0.66	36	0.03	25.3	97	29.2	1.07	3.24
BR29	22NB-27	400	0.2	1.41	34	0.03	10.2	98.6	70	1.97	3.48
BR30	21ZAP-41	400	0.2	0.32	3	0.02	1.2	93.7	61.7	0.90	6.52
BR31	21ZAP-41	6	none	0.26	7	0.07	6.1	67.2	12.6	0.38	6.12
BR32	22ZAP-85	400	0.2	0.93	12	0.02	6.3	97.8	48	0.96	6.78
BR33	22ZAP-85	400	0.2	0.4	14	0.02	10.0	95.1	27.5	0.90	7.23
BR34	22ZAP-85	400	0.2	0.24	9	0.01	5.4	95.9	39.3	0.96	6.34
BR35	22ZAP-85	400	0.2	1.23	10	0.03	3.8	97.6	63.6	0.90	6.74
BR36	22GE-33	400	0.2	1.27	32	0.03	2.8	97.6	91.3	1.09	5.52
BR37	22GE-33	400	0.2	1.02	35	0.04	1.5	96.1	95.6	1.44	5.73
BR38	22GE-33	400	0.2	4.98	119	0.04	3.4	99.2	97.1	1.20	6.38
BR39	22NB-25	400	0.2	0.32	25	0.03	1.5	90.6	93.8	1.38	4.14
BR40	22NB-25	6	0.2	0.5	1	0.02	0.3	72	136.2	0.50	3.53
BR41	22NB-29	400	0.2	0.28	20	0.02	5.5	92.8	72.1	1.36	6.36
BR42	22NB-29	400	0.2	0.31	67	0.02	2.7	96.7	96	1.02	4.19
BR43	22GE-44	400	0.2	0.2	14	0.02	0.3	90.2	97.8	1.97	2.97
BR44	22GE-44	400	0.2	0.18	9	0.02	0.3	94.5	98.4	2.38	3.42
BR45	22GE-46	400	0.2	0.57	41	0.02	0.3	96.5	99.6	1.61	4.12
BR46	22GE-46	400	0.2	0.2	14	0.02	1.7	90.1	87.7	1.55	4.63
BR47	22GE-33	6	0.2	0.22	8	0.02	3.7	34.3	60.7	0.41	3.92
BR48	22GE-33	400	0.2	2.19	154	0.14	2.1	99.1	98.7	1.02	4.14
BR49	22GE-53	400	0.2	0.84	31	0.02	0.3	97.6	99.5	1.20	4.16
BR50	22GE-53	400	0.2	0.44	30	0.02	1.9	95.5	93.5	2.20	8.13
BR51	22GE-53	400	0.2	23.78	451	0.07	48.5	99.7	89.3	1.79	4.75
BR54	21GE-21	6	0.2	0.3	5	0.09	2.6	71.8	65.8	0.33	3.51
BR55	22TA-38	6	0.2	0.15	39	0.02	2.4	62.9	60	0.44	3.35
BR56	22TA-38	6	0.2	0.08	21	0.03	10.7	65	61.5	0.56	3.26
BR57	22TA-38	400	0.2	0.31	54	0.02	6.6	96.8	87.7	1.32	5.10
BR58	22TA-38	6	0.2	0.19	40	0.06	19.6	79.2	51.8	0.62	3.37
BR59	22TA-38	270	0.2	11.23	24	0.02	1.6	99.9	93.6	1.08	4.88
BR60	22TA-38	6	0.2	0.13	37	0.03	24.2	68.4	36.5	0.56	4.14
BR61	22TA-38	270	0.2	0.15	39	0.02	13.6	93.4	65.5	1.02	4.69

Results from the program are largely consistent with previous results. Some of the samples showed improved recoveries with finer grinding to 37 µm with silver recoveries benefitting more from the finer grinding. Overall incremental recovery improvements for gold on average were very minor between the 53 µm and 37 µm grind sizes.

Head grades for gold for the test work were generally low, with 44 of the 59 tests having a gold grade of less than 1 g/t and approximately half of the tests having a grade of less than 0.5 g/t. There does not appear to be a strong correlation between gold head grade and gold recovery; however, it is recommended that future test work be performed on more grade-representative material.

13.3.4 Forte 2024 test program

Cyanide leach tests were completed on drill core reject samples left over from previous test programs. In total, twelve samples were evaluated with head analyses for gold and silver presented in Table 13-18.

Table 13-18: 2024 Forte composite sample head analyses

Sample Name	Gold g/t	Silver g/t
1 - 22GE-38: 22867-22913	0.58	22
2 - 21GE-21: 28172-28192	0.28	6
3 - 22GE-46: 32712-32717	0.37	30
4 - 21GE-19: 11936-11945	1.39	62
5 - 22GE-33: 22757-27764	6.23	143
6 - 22GE-53: 42728-42737	15.02	325
7 - 22SME-12: 47614-47622	0.33	3
8 - 21ZAP-32: 24206-24231	0.48	24
9 - 21ZAP-43: 31100-31114	1.45	24
9 - 21ZAP-43: 31100-31114 dup	1.14	27
10 - 21ZAP-36: 24719-24720	2.65	12
11 - 22SMW-10: 62101	7.07	478
12 - 21GE-03: 4338-4340	11.87	705
13 - 22TA-39: 45645-45666	0.39	54
14 - 22NB-27: 48773-48775	4.72	102

Head assays for gold on the 14 samples varied from 0.28 to 15 g/t Au, while silver assays varied from 6.2 to 705 g/t Ag.

The cyanide leach testing was developed in two phases. The first phase of the test work evaluated different lead nitrate ($\text{Pb}(\text{NO}_3)_2$) additions ranging from 0 g/t to 500 g/t on a sample from Guadalupe East with results presented in Table 13-19. Samples were ground to 80% passing 200 mesh and leached for 48 hours with carbon added after 24 hours of leaching. The results showed the optimal lead nitrate addition at 200 g/t which was used for the second phase of testing.

The second phase of the leach test work included CIL bottle roll leach tests on each of the 12 composite samples. Each sample was ground to 80% passing 200 mesh and leached at 2 g/L NaCN and 200 mg/kg of lead nitrate. Results from the CIL bottle roll leach tests are presented in Table 13-19.

Table 13-19: 2024 Forte lead nitrate optimization test results

Test #	Pb(NO ₃) ₂ Addition (g/t)	Assayed Head Grade (g/t)		Calc. Head Grade (g/t)		Residue Grade (g/t)		Recovery (%)		Consumption (kg/t)	
		Gold	Silver	Gold	Silver	Gold	Silver	Gold	Silver	NaCN	Lime
PBR-1	0	0.58	22	0.72	22	0.08	BD	89.4	87.6	1.38	2.03
PBR-2	100	0.58	22	0.68	24	0.04	BD	94.1	88.9	1.32	2.00
PBR-3	200	0.58	22	0.66	20	0.03	BD	95.4	96.4	1.17	3.28
PBR-4	500	0.58	22	0.64	21	0.03	BD	95.3	96.2	1.45	2.19

Table 13-20: 2024 Forte bottle roll leach tests

Sample #	Test #	Assayed Head Grade (g/t)		Calc. Head Grade (g/t)		Residue Grade (g/t)		Recovery (%)		Consumption (kg/t)	
		Gold	Silver	Gold	Silver	Gold	Silver	Gold	Silver	NaCN	Lime
2 - 21GE-21	PBR-16	0.28	BD	0.34	<7	0.03	BD	91.1	84.3	1.66	2.52
3 - 22GE-46	PBR-15	0.37	30	0.39	25	0.03	BD	92.4	96.1	1.60	2.43
4 - 21GE-19	PBR-17	1.39	62	1.81	56	0.04	BD	97.8	94.6	1.48	2.35
5 - 22GE-33	PBR-12	6.23	143	6.32	125	0.09	BD	98.6	95.2	1.57	2.52
6 - 22GE-53	PBR-10	15	325	14.7	303	0.17	31	98.8	89.8	2.19	1.21
7 - 22SME-12	PBR-9	0.33	BD	0.33	<7	0.03	BD	93.9	56.9	1.17	2.86
8 - 21ZAP-32	PBR-13	0.48	24	0.64	24	0.03	17	95.3	28.4	1.12	1.72
9 - 21ZAP-43	PBR-14	1.45	24	1.19	23	0.06	12	95	47.1	1.30	1.92
10 - 21ZAP-36	PBR-11	2.65	12	5.24	16	0.06	10	98.9	38.7	1.66	1.22
11 - 22SMW-10	PBR-7	7.07	478	6.67	451	0.32	94	95.2	79.2	1.97	4.25
12 - 21GE-03	PBR-8	11.9	705	11.9	660	0.41	174	96.6	73.7	1.97	5.07
13 - 22TA-39	PBR-6	0.39	54	0.42	50	0.04	15	90.6	70.2	1.18	3.03
14 - 22NB-27	PBR-5	4.72	102	4.38	97	0.12	BD	97.3	96.9	1.88	5.14

Notes: Conditions: Grind (P80): 200 Mesh, Pb(NO₃)₂ 200 kg/t, Leach Time: 48 h.

The results from the CIL bottle roll leach tests were consistent with previous test work with high recoveries for gold, ranging from 90.6% to 98.9% and variable recoveries for silver ranging from 28.4% to 96.9%. Overall recoveries averaged 95.5% for gold and 73% for silver. NaCN consumptions averaged 1.6 kg/t and lime averaged 2.8 kg/t.

13.3.5 KCA test program 2024

Variability bottle roll leach tests were completed by KCA in 2024 on 16 composite samples generated from drill core. Head assays for gold and silver for each composite sample are presented in Table 13-21.

Table 13-21: 2024 KCA composite sample head analyses

KCA Sample No.	Description	Gold Assays (g/t)			
		Assay 1	Assay 2	Assay 3	Average
99401 A	ZS_HG-VHG	7.20	6.99	7.54	7.25
99402 A	ZN_HG-VHG_2	5.09	5.01	5.33	5.14
99403 A	TA_HG-VHG	1.73	1.59	1.79	1.71
99404 A	NB_HG-VHG	1.23	1.52	1.40	1.38
99405 A	SME_HG-VHG	1.50	1.67	1.49	1.55
99406 A	ZN_HG-VHG_1	0.71	0.82	0.78	0.77
99407 A	ZN_LG-MG	0.51	0.61	0.51	0.55
99408 A	NB_LG-MG	0.21	0.17	0.18	0.19
99409 A	TA_LG-MG	0.23	0.22	0.23	0.23
99410 A	GE_HG-VHG_1	8.04	8.67	8.81	8.51
99411 A	GE_HG-VHG_2	5.07	5.62	5.55	5.42
99412 A	GE_HG-VHG_4	2.73	2.76	3.10	2.86
99413 A	GE_HG-VHG_3	1.78	2.01	2.35	2.05
99414 A	GE_LG-MG_2	0.39	0.27	0.33	0.33
99415 A	GW_LG-MG_1	0.50	0.51	0.52	0.51
99416 A	GE_LG-MG_1	0.42	0.39	0.42	0.41

KCA Sample No.	Description	Silver Assays (g/t)			
		Assay 1	Assay 2	Assay 3	Average
99401 A	ZS_HG-VHG	59	58	57	58
99402 A	ZN_HG-VHG_2	25	26	26	26
99403 A	TA_HG-VHG	104	101	111	105
99404 A	NB_HG-VHG	59	72	79	70
99405 A	SME_HG-VHG	70	74	78	74
99406 A	ZN_HG-VHG_1	18	19	23	20
99407 A	ZN_LG-MG	14	16	16	15
99408 A	NB_LG-MG	5	6	6	6
99409 A	TA_LG-MG	20	19	17	19
99410 A	GE_HG-VHG_1	724	648	726	699
99411 A	GE_HG-VHG_2	176	159	160	165
99412 A	GE_HG-VHG_4	392	335	462	396
99413 A	GE_HG-VHG_3	79	59	48	62
99414 A	GE_LG-MG_2	20	18	22	20
99415 A	GW_LG-MG_1	19	19	19	19
99416 A	GE_LG-MG_1	30	19	27	25

Notes:

The detection limit for silver with FAAS finish is 0.5 g/t.

For the purpose of calculation, a value of 1/2 the detection limit is utilized for assays less than the detection limit.

Direct bottle roll tests were conducted on each composite sample at varying material sizes ranging from 80% passing 1.7 mm to 80% passing 0.075 mm (200 mesh) using a NaCN dosage of 2 g/L. Lead nitrate was

added to each sample at 200 g/t based on results from the Forte Analytical lead nitrate optimization test work. Coarse bottle roll tests (0.3 mm and 1.7 mm) were leached for 96 hours to ensure the samples were completely leached vs. 24 hours for the milled samples. The goal of the variability test work was to confirm the material grind size requirements and to determine whether a direct agitated leach circuit would be viable instead of the CIL process. Results are presented in Table 13-22.

Table 13-22: 2024 KCA direct bottle roll leach test

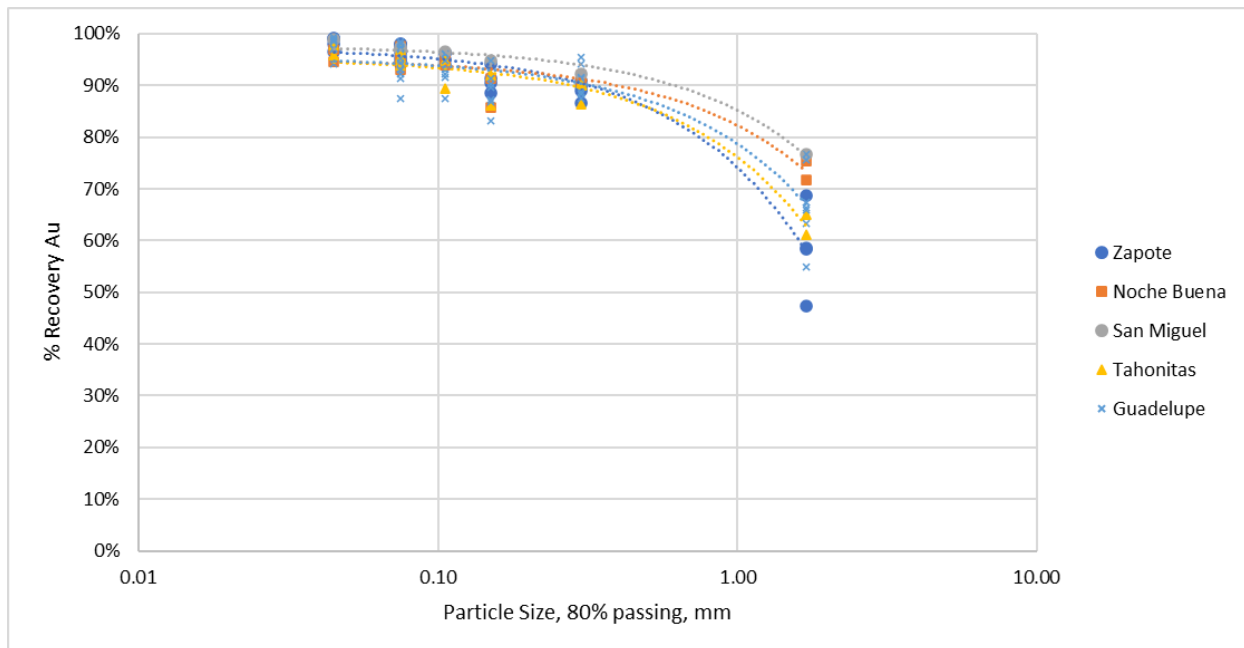
KCA Sample No.	Descript.	Target P ₈₀ Size (mm)	Leach Time (hr)	Calculated Head (g/t)		Residue (g/t)		% Extracted		Consumption kg/t	
				Gold	Silver	Gold	Silver	Gold	Silver	NaCN	Ca(OH) ₂
99401 B	ZS_HG-VHG	1.7	96	7.16	55	4.19	12	59	22	0.11	0.50
99401 B	ZS_HG-VHG	0.3	96	6.88	62	5.96	21	87	33	0.03	0.75
99401 B	ZS_HG-VHG	0.15	24	6.07	63	5.72	24	94	38	0.07	0.50
99401 B	ZS_HG-VHG	0.106	24	6.34	61	6.12	24	96	40	0.15	0.50
99401 B	ZS_HG-VHG	0.075	24	6.78	66	6.63	32	98	48	0.23	0.50
99402 B	ZN_HG-VHG_2	1.7	96	5.26	25	3.61	5	69	19	0.11	1.25
99402 B	ZN_HG-VHG_2	0.3	96	5.21	25	4.68	8	90	30	0.12	1.00
99402 B	ZN_HG-VHG_2	0.15	24	4.55	25	4.16	8	91	33	0.07	1.00
99402 B	ZN_HG-VHG_2	0.106	24	4.68	25	4.51	10	96	38	0.31	0.75
99402 B	ZN_HG-VHG_2	0.075	24	5.09	26	4.96	11	97	42	0.53	0.75
99403 B	TA_HG-VHG	1.7	96	1.81	113	1.10	40	61	35	0.18	0.50
99403 B	TA_HG-VHG	0.3	96	1.93	117	1.74	83	91	71	0.26	0.75
99403 B	TA_HG-VHG	0.15	24	1.67	113	1.53	70	92	62	0.15	0.50
99403 B	TA_HG-VHG	0.106	24	1.62	110	1.53	74	95	67	0.38	0.50
99403 B	TA_HG-VHG	0.075	24	1.75	119	1.67	78	95	66	0.38	0.50
99404 B	NB_HG-VHG	1.7	96	1.24	93	0.89	61	72	66	0.49	0.50
99404 B	NB_HG-VHG	0.3	96	1.46	93	1.33	77	91	84	0.34	0.75
99404 B	NB_HG-VHG	0.15	24	1.22	76	1.12	66	91	87	0.22	0.50
99404 B	NB_HG-VHG	0.106	24	1.24	70	1.18	60	95	86	0.45	0.50
99404 B	NB_HG-VHG	0.075	24	1.28	68	1.23	60	95	89	0.38	0.50
99405 B	SME_HG-VHG	1.7	96	1.57	90	1.20	42	77	46	0.26	1.50
99405 B	SME_HG-VHG	0.3	96	1.98	82	1.82	67	92	82	0.27	1.50
99405 B	SME_HG-VHG	0.15	24	1.62	83	1.53	60	95	72	0.31	1.25
99405 B	SME_HG-VHG	0.106	24	1.64	81	1.58	63	96	78	0.38	1.25
99405 B	SME_HG-VHG	0.075	24	1.52	93	1.47	71	97	76	0.61	0.75
99406 B	ZN_HG-VHG_1	1.7	96	0.87	19	0.41	3	47	14	0.03	0.50
99406 B	ZN_HG-VHG_1	0.3	96	0.73	25	0.67	5	91	19	0.03	0.50
99406 B	ZN_HG-VHG_1	0.15	24	0.78	21	0.69	6	89	31	0.15	0.50
99406 B	ZN_HG-VHG_1	0.106	24	0.75	20	0.71	7	94	35	0.30	0.50
99406 B	ZN_HG-VHG_1	0.075	24	0.85	21	0.81	8	95	41	0.37	0.50
99407 B	ZN_LG-MG	1.7	96	0.58	14	0.34	2	58	12	0.34	1.75
99407 B	ZN_LG-MG	0.3	96	0.59	16	0.52	4	89	21	0.19	1.50

KCA Sample No.	Descript.	Target P ₈₀ Size (mm)	Leach Time (hr)	Calculated Head (g/t)		Residue (g/t)		% Extracted		Consumption kg/t	
				Gold	Silver	Gold	Silver	Gold	Silver	NaCN	Ca(OH) ₂
99407 B	ZN_LG-MG	0.15	24	0.54	17	0.49	5	91	27	0.30	1.00
99407 B	ZN_LG-MG	0.106	24	0.52	17	0.49	5	95	31	0.38	1.25
99407 B	ZN_LG-MG	0.075	24	0.57	16	0.54	6	94	35	0.76	0.75
99408 B	NB_LG-MG	1.7	96	0.23	6	0.17	4	75	61	0.26	0.75
99408 B	NB_LG-MG	0.3	96	0.27	7	0.25	6	92	83	0.11	0.75
99408 B	NB_LG-MG	0.15	24	0.25	7	0.22	6	86	86	0.07	0.75
99408 B	NB_LG-MG	0.106	24	0.20	7	0.18	6	94	87	0.30	0.50
99408 B	NB_LG-MG	0.075	24	0.18	8	0.17	7	93	89	0.38	0.50
99409 B	TA_LG-MG	1.7	96	0.27	20	0.18	3	65	13	0.03	0.50
99409 B	TA_LG-MG	0.3	96	0.29	22	0.25	5	86	23	0.03	0.75
99409 B	TA_LG-MG	0.15	24	0.23	22	0.20	6	86	29	0.15	0.50
99409 B	TA_LG-MG	0.106	24	0.24	22	0.21	7	90	33	0.22	0.50
99409 B	TA_LG-MG	0.075	24	0.26	22	0.25	8	95	39	0.38	0.50
99410 B	GE_HG-VHG_1	1.7	96	8.75	693	5.77	374	66	54	0.74	0.75
99410 B	GE_HG-VHG_1	0.3	96	7.76	684	6.79	449	87	66	0.86	0.75
99410 B	GE_HG-VHG_1	0.15	24	7.60	669	6.32	242	83	36	0.60	0.75
99410 B	GE_HG-VHG_1	0.106	24	7.61	641	6.65	242	87	38	1.13	0.75
99410 B	GE_HG-VHG_1	0.075	24	8.12	711	7.10	319	87	45	1.51	0.50
99411 B	GE_HG-VHG_2	1.7	96	5.73	221	3.85	139	67	63	0.26	0.50
99411 B	GE_HG-VHG_2	0.3	96	5.15	198	4.71	156	91	79	0.42	0.50
99411 B	GE_HG-VHG_2	0.15	24	4.45	177	3.86	111	87	63	0.30	0.50
99411 B	GE_HG-VHG_2	0.106	24	5.08	186	4.82	122	95	65	0.53	0.50
99411 B	GE_HG-VHG_2	0.075	24	5.28	204	5.12	140	97	68	0.68	0.50
99412 B	GE_HG-VHG_4	1.7	96	2.54	395	1.92	215	76	54	0.49	0.50
99412 B	GE_HG-VHG_4	0.3	96	2.80	419	2.67	305	96	73	0.64	0.50
99412 B	GE_HG-VHG_4	0.15	24	2.58	390	2.45	180	95	46	0.45	0.50
99412 B	GE_HG-VHG_4	0.106	24	2.65	369	2.55	166	96	45	0.75	0.50
99412 B	GE_HG-VHG_4	0.075	24	2.79	390	2.71	214	97	55	0.98	0.50
99413 B	GE_HG-VHG_3	1.7	96	1.98	67	1.31	45	66	66	0.19	0.50
99413 B	GE_HG-VHG_3	0.3	96	2.02	72	1.85	61	92	85	0.19	0.50
99413 B	GE_HG-VHG_3	0.15	24	1.98	69	1.79	53	90	78	0.15	0.50
99413 B	GE_HG-VHG_3	0.106	24	2.04	66	1.96	55	96	83	0.37	0.50
99413 B	GE_HG-VHG_3	0.075	24	2.10	72	2.04	56	97	78	0.30	0.50
99414 B	GE_LG-MG_2	1.7	96	0.33	21	0.25	13	77	61	0.19	0.50
99414 B	GE_LG-MG_2	0.3	96	0.44	25	0.41	22	94	87	0.26	0.50
99414 B	GE_LG-MG_2	0.15	24	0.37	24	0.33	20	90	84	0.30	0.50
99414 B	GE_LG-MG_2	0.106	24	0.36	22	0.33	19	92	85	0.75	0.50
99414 B	GE_LG-MG_2	0.075	24	0.44	24	0.41	21	93	88	0.83	0.50
99415 B	GW_LG-MG_1	1.7	96	0.46	18	0.25	9	55	49	0.46	1.50

KCA Sample No.	Descript.	Target P ₈₀ Size (mm)	Leach Time (hr)	Calculated Head (g/t)		Residue (g/t)		% Extracted		Consumption kg/t	
				Gold	Silver	Gold	Silver	Gold	Silver	NaCN	Ca(OH) ₂
99415 B	GW_LG-MG_1	0.3	96	0.52	20	0.46	13	89	66	0.27	1.50
99415 B	GW_LG-MG_1	0.15	24	0.51	19	0.44	13	87	68	0.31	1.25
99415 B	GW_LG-MG_1	0.106	24	0.51	19	0.48	13	93	67	0.38	1.25
99415 B	GW_LG-MG_1	0.075	24	0.52	20	0.49	15	94	72	0.53	0.75
99416 B	GE_LG-MG_1	1.7	96	0.40	21	0.25	13	63	59	0.14	0.50
99416 B	GE_LG-MG_1	0.3	96	0.45	24	0.40	20	88	84	0.26	0.50
99416 B	GE_LG-MG_1	0.15	24	0.40	23	0.37	19	91	83	0.30	0.50
99416 B	GE_LG-MG_1	0.106	24	0.38	21	0.35	18	92	83	0.53	0.50
99416 B	GE_LG-MG_1	0.075	24	0.38	24	0.35	21	93	87	0.68	0.50

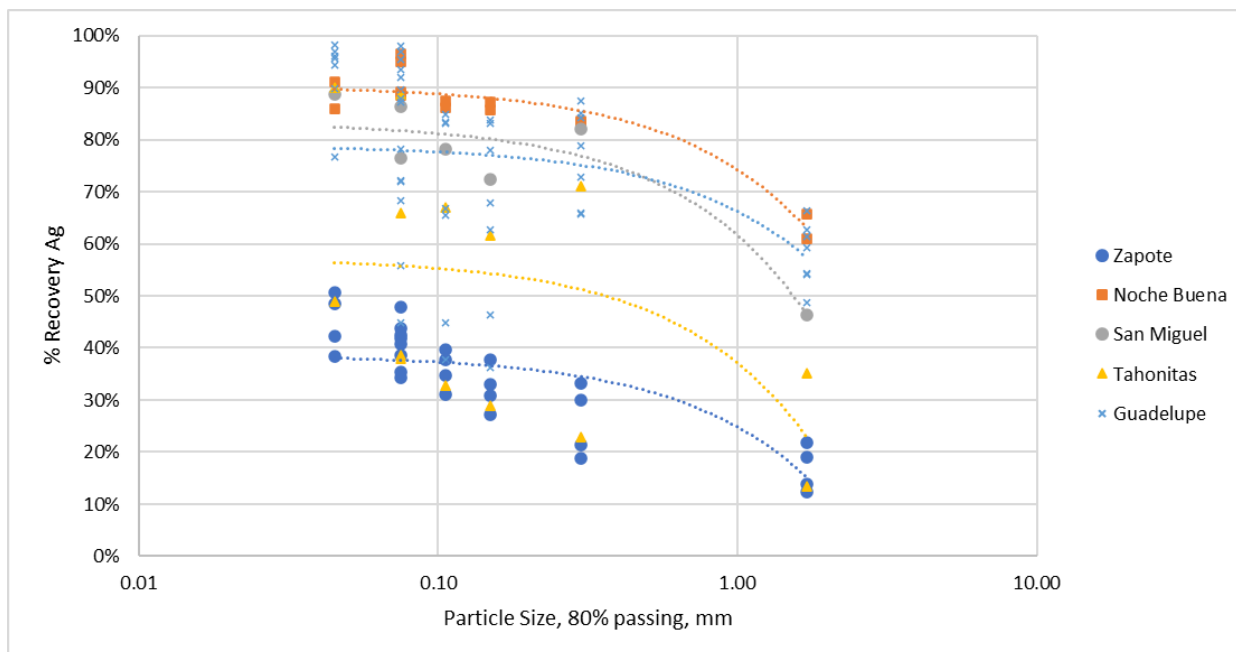
The bottle roll tests showed that recoveries for gold and silver were strongly dependent on grind size, as shown in Figure 13-3 for gold and Figure 13-4 for silver.

Figure 13-3: KCA 2024 Particle Size vs. Recovery, Gold



Source: KCA, 2024

Figure 13-4: KCA 2024 Particle Size vs. Recovery, Silver



Source: KCA, 2024

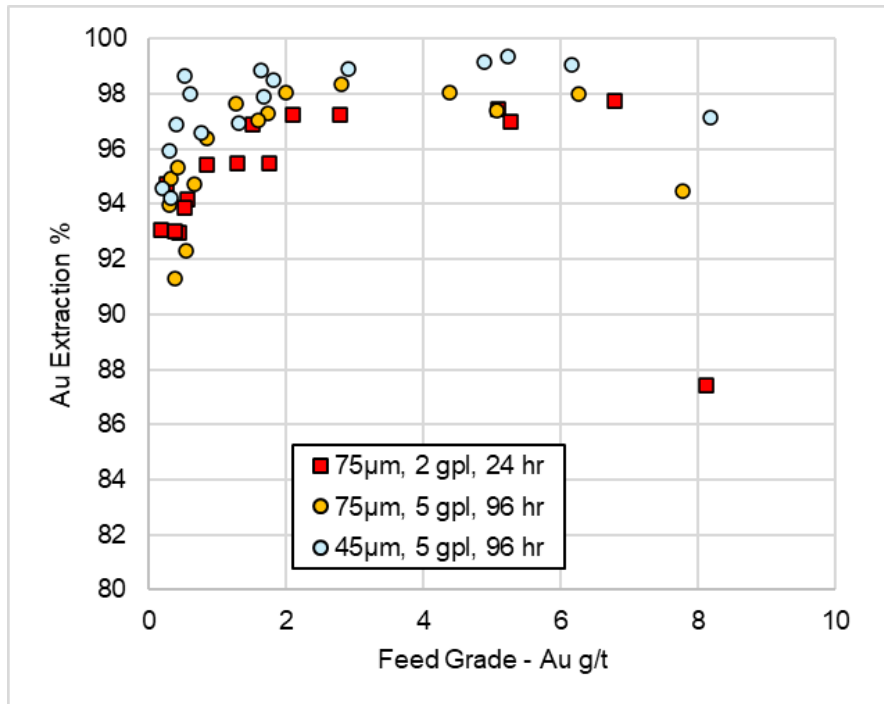
Gold recoveries at 80% passing 150 and 200 mesh for all material types were generally high ranging from 87% to 98% with an overall average of 94%. Silver recoveries varied significantly by material type ranging from 31% to 89% with an overall average of 62%. The results indicate that direct leaching is viable achieving similar recoveries compared to CIL bottle roll leach tests from previous test work programs, with the exception of silver recoveries for the Tahonitas samples which were significantly lower than previous work.

Each of the 16 samples was also tested using 5 g/L NaCN dosages and 96-hour leach durations, one test at 75 μm P₈₀ and another at 45 μm P₈₀. Results are compared graphically in Figure 13-5 and Figure 13-4 to the 24-hour extractions with 2 g/L NaCN and 75 μm P₈₀ primary grind size. Average results of each data set are presented in Table 13-23.

Table 13-23: Average KCA leach results at alternate conditions

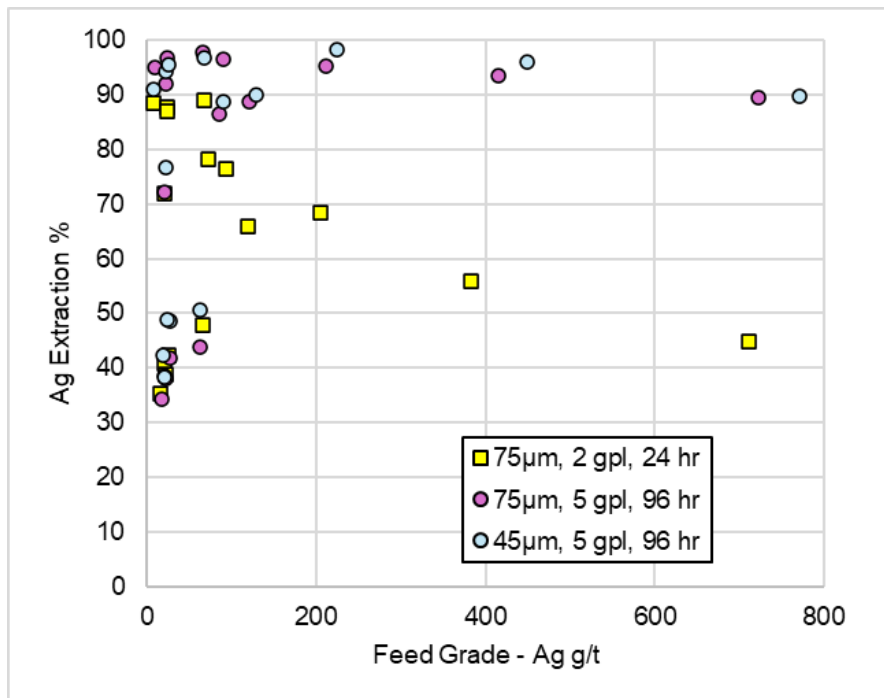
Leach Conditions	Grind Size	Leach Time	NaCN Dosage	Extraction %		Reagent Cons. kg/t	
	μm P ₈₀	hours	g/L	Au	Ag	NaCN	Lime
Standard	75	24	2	94.9	63.7	0.60	0.56
Alternate 1	75	96	5	96.0	75.0	1.56	0.56
Alternate 2	45	96	5	97.6	76.4	2.62	0.50

Figure 13-5: Comparison of KCA gold extractions at alternate conditions



Source: Ausenco, 2026

Figure 13-6: Comparison of KCA silver extractions at alternate conditions



Source: Ausenco, 2026

The higher NaCN dosages and longer leach time only marginally affected gold extractions, aside from one high grade sample (GE_HG-VHG_1). The silver extractions however improved considerably under these more aggressive leach conditions, particularly for samples with feed grades greater than 100 g/t silver.

Three of the composites, along with a blend of two of the composites, were selected for column leach testing. The four composites were tested at crush sizes of -50mm and -12.5mm. Column leaching was conducted over 163 days, following a pre-treatment with lead nitrate solution. Initial leach solutions containing 1 g/L NaCN were applied, subsequent on-flow solution was maintained at 0.5 g/L. Results are presented in Table 13-24.

Table 13-24: KCA column leach test results

KCA Test No.	Description	Crush Top Size	Residue Size	Calculated Head (g/t)		Extracted %		NaCN Consumption
		mm	P80 mm	Au	Ag	Au	Ag	kg/t
99433.00	ZN_HG-VHG_1	50.0	37.3	0.88	19.7	25.9	2.9	2.89
99436.00	ZN_HG-VHG_1	12.5	10.1	0.87	18.6	41.5	8.2	4.07
99439.00	ZN_LG-MG	50.0	33.8	0.63	14.6	46.9	4.3	3.75
99442.00	ZN_LG-MG	12.5	9.1	0.73	17.1	56.3	8.1	7.07
99445.00	TA_LG-MG	50.0	37.2	0.24	19.9	54.7	3.1	2.63
99448.00	TA_LG-MG	12.5	10.4	0.30	21.9	60.8	8.0	3.99
99451.00	SME+NB	50.0	37.6	0.88	41.4	76.7	15.9	3.25
99454.00	SME+NB	12.5	9.8	0.75	41.9	76.0	34.8	5.13

Final extractions of the -12.5mm crushed feeds averaged 59% for gold and 15% for silver. Sodium cyanide consumptions for this crush size averaged 5 kg/t, however in KCA's experience, cyanide consumption in production heaps would be 25-33% of laboratory column test consumptions. Heap leaching would not likely provide favourable economics compared to fine grinding and tank leaching, based on these results.

13.4 Comminution testing

A limited number of comminution tests were completed over the series of metallurgical programs. A few Bond ball mill work index tests were conducted in the earlier programs, the bulk of the data comes from a sub-set of composites tested in the 2024 KCA program. Results are summarized in Table 13-25.

Table 13-25: Comminution test results

Sample ID	Description	Composite ID	Abrasion Index (g)	Bond Ball Mill Wi		SMC
				Closing μm	[kWh/t]	Axb
99403 A	Zapote	TA_HG-VHG	0.46	150	18.6	39.3
99405 A	San Miguel	SME_HG-VHG	0.30	150	18.4	35.8
99409 A	Tahonitas	TA_LG-MG	0.59	150	19.2	37.7
99415 A	Guadalupe W	GW_LG-MG	0.36	150	17.8	40.1
Composite 1	Guadalupe W	GLW	0.26	100	16.9	-
Composite 2	Guadalupe	Vein Zone	0.34	100	17.3	-
Composite 3	Zapote	El Zapote	0.46	100	17.4	-
21TA-14	Tahonitas	19851-19864	-	150	15.8	-

The results suggest that the samples are moderately competent with respect to breakage in a SAG mill, and moderately hard to hard with respect to breakage in a ball mill. The Abrasion index values range from moderately abrasive to very abrasive.

13.5 Metallurgical test work interpretation

Ausenco reviewed and interpreted the metallurgical test work results, and recommends the design parameters discussed in the following sections:

- Heap leaching is not considered for this study as the bulk of the resource has sufficient value to warrant milling to a fine grind size and achieving high tank leaching extractions. There is possibly economic justification to heap leach some low-grade materials, however this processing path was not considered for the study due to the limited amount of test data on low-grade samples.
- Mill Parameters:
 - Target grind size of 80% passing 75 μm .
 - A leach residence time of 72 hours to allow for increased silver extractions.

13.5.1 Recovery Analysis

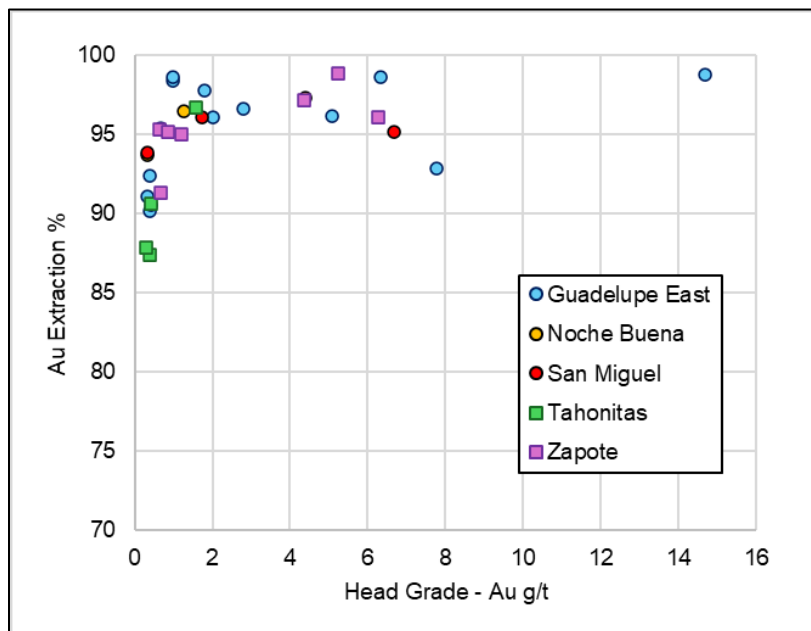
Test results from the 2012 through to 2025 test programs that met the grind size criteria were selected to evaluate laboratory extraction as a function of feed grade for each deposit. Results from a total of 33 unique samples across the deposits formed the extraction model data set. Results are presented graphically in Figure 13-7 and Figure 13-8. In some cases, 48-hour extraction results were used for samples that were only tested at this leach duration in order to include the greatest number of variability samples in the analysis.

Extraction curves as function of feed grade were fit to each deposit for both gold and silver. In most cases a logarithmic curve provided the best fit, in a few cases a linear equation was selected. Guadalupe West had only a single sample, so a constant recovery was applied. There are two deposits in the mine plan for which metallurgical testing has not yet been completed: Mariposa and Fresnillo. Metallurgical performance calculations for deposits in a similar trend or vicinity were treated using the same equations, for instance, Zapote and Noche Buena.

Laboratory extractions for each deposit are summarized in Table 14-26 at their respective LOM feed grades. The total LOM extraction for the project is calculated based on the weighting of gold and silver mill feed

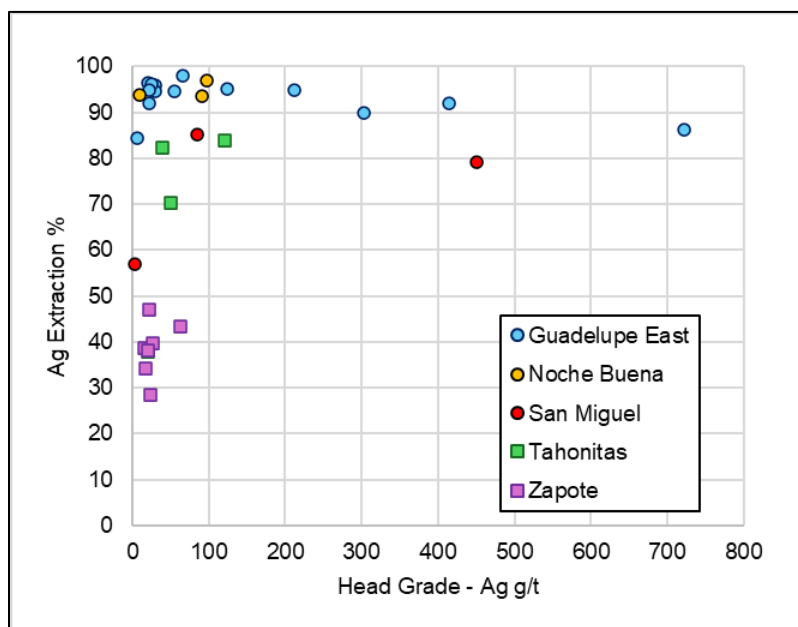
ounces in each deposit. The resulting LOM extractions are 95.8% for gold and 82.1% for silver. A 1% deduction in recovery will also be applied to the financial model for both gold and silver to account for solution losses.

Figure 13-7: Gold Recovery Data Set



Source: Ausenco 2026

Figure 13-8: Silver Recovery Data Set



Source: Ausenco 2026

Table 13-26: Leach Extractions for Deposits at LOM Feed Grades

Material Source	LOM Grade (g/t)		Recovery - 72 hours	
	Gold	Silver	Gold	Silver
Guadalupe East	2.31	125	95.9	93.4
Guadalupe West	0.82	25	92.3	72.1
Noche Buena	1.04	26	95.6	94.5
San Miguel	1.14	60	94.9	74.8
Tahonitas	1.81	91	97.5	84.0
Zapote	1.66	25	95.4	38.6
Mariposa	2.77	19	96.3	37.3
Fresnillo	1.61	22	96.2	94.4
LOM Recovery	1.68	63	95.8	82.1

13.5.2 Reagent consumptions

Reagent consumptions for the agitated leach have been estimated based on material type from all available test work. Sodium cyanide consumptions varied considerably due to both leach residence time and reagent concentration. Earlier testing maintained NaCN dosages of 2 g/L while a significant portion of the KCA tests was completed with 5 g/L. It is estimated that a starting dosage of 2.5 g/L will be maintained for the first 24 hours of the operation to promote favourable silver extraction kinetics but will be allowed to drift for the remainder of the residence time. Although the testing does not specifically demonstrate this condition, the laboratory consumptions suggest that an end of leach concentration of 1.5 g/L is a reasonable expected value and likely includes a slight overestimate of cyanide consumption. Considering the portion of the CCD underflow that reports to cyanide detoxification circuit, the estimated NaCN consumption for the circuit on this basis is 2.16 kg/t.

Lime requirements for the agitated leach have been taken at a 1:1 ratio with the average lab results.

13.6 Conclusions

A comprehensive collection of metallurgical testing has been completed on the Los Reyes project, more than is typically required for this level of study. The material responds well to leaching with sodium cyanide at a primary grind size of 75 μm P₈₀. The target precious metals can be concentrated by froth flotation, but recoveries to a concentrate are not superior to leaching at the same primary grind size. Gravity concentration did not appear to enhance overall circuit recoveries. The metallurgical performance was quite consistent across the deposits for gold, but silver recoveries varied and were lower in the Zapote materials.

Samples, required for PEA study, were provided and evaluated to assess comminution properties. The samples were of moderate hardness with respect to SAG milling and would be considered moderately hard with respect to attrition grinding in a ball mill.

To the extent known, there are no processing factors or deleterious elements that could have a significant effect on potential economic extraction from materials represented by these samples and composites.

14 Mineral resource estimates

14.1 Mineral resource statement

The Mineral Resource statement for Los Reyes includes estimates for three principal mineralized areas on the property - (1) Z-T, which includes the Tahonitas, Zapote North, and Zapote South deposits. (2) Central, which includes the Noche Buena, San Miguel East, and San Miguel West deposits. (3) Guadalupe, which includes the Guadalupe East and Guadalupe West deposits (Figures 10-3, 10-7, and 10-11). Three additional generative exploration targets adjacent to the principal trends, Fresnillo, Mariposa, and Las Primas, are also included in the Resource statement under 'Generative Areas'. All stated Resource estimates are contained ounces.

The Mineral Resources are stated in accordance with CIM guidelines and definitions in National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101) and were estimated using drillholes completed on or before December 31, 2024. The Resource update presented here was prepared to support continued exploration and project work on the Los Reyes property.

Mineral Resources were reported below the most recent LiDAR topographic surface and are contained within economically constrained pit shells generated using the Hochbaum Pseudoflow algorithm implemented in Datamine's Studio NPVS or underground stope shapes generated using Datamine's Mineable Shape Optimizer (MSO). Open pit Resources are estimated using a 0.17 (g/t) gold-only cutoff grade. The Mineral Resources are classified as Indicated or Inferred based on drill spacing and geological continuity, Measured Resources are not classified. Table 14-1 through Table 14-3 show the classified Mineral Resources for the Los Reyes property.

Table 14-1: Mineral Resource Statement

Mining Method and Process	Class	Tonnage (kt)	Au (g/t)	Au (koz)	Ag (g/t)	Ag (koz)	AuEq (g/t)	AuEq (koz)
Open pit plant	Measured	-	-	-	-	-	-	-
	Indicated	26,448	1.13	961	30.8	26,192	1.47	1,251
	Measured and Indicated	26,448	1.13	961	30.8	26,192	1.47	1,251
	Inferred	8,056	0.93	240	34.4	8,917	1.31	339
Underground plant	Measured	-	-	-	-	-	-	-
	Indicated	5,361	2.48	428	129.0	22,231	3.91	674
	Measured and Indicated	5,361	2.48	428	129.0	22,231	3.91	674
	Inferred	5,430	1.84	322	72.7	12,685	2.65	463
Total Plant	Measured	-	-	-	-	-	-	-
	Indicated	31,809	1.36	1,388	47.3	48,423	1.88	1,925
	Measured and Indicated	31,809	1.36	1,388	47.3	48,423	1.88	1,925
	Inferred	13,487	1.30	562	49.8	21,602	1.85	801
Open Pit Heap Leach	Measured	-	-	-	-	-	-	-
	Indicated	19,441	0.25	154	11.9	7,437	0.30	187
	Measured and Indicated	19,441	0.25	154	11.9	7,437	0.30	187
	Inferred	7,413	0.24	57	12.6	3,013	0.30	71
Total	Measured	-	-	-	-	-	-	-
	Indicated	51,250	0.94	1,542	33.9	55,860	1.28	2,111
	Measured and Indicated	51,250	0.94	1,542	33.9	55,860	1.28	2,111
	Inferred	20,899	0.92	620	36.6	24,615	1.30	872

Notes:

1. Mineral resources were prepared in accordance with the CIM Definition Standards (2014) and Estimation of Mineral Resource and Mineral Reserve Best Practice guidelines (2019).
2. Mineral resources are inclusive of mineral reserves. Mineral resources that are not mineral reserves do not have demonstrated economic viability.
3. Numbers may not add due to rounding.
4. Mineral resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.
5. The effective date of the estimate is April 25, 2026
6. Mineral resources for Los Reyes are based on underlying metal prices of \$1,950/oz Au and \$25.24/oz Ag, unless otherwise noted.
7. The preparation of the mineral resource estimate was supervised by John Sims, President of Sims Resources LLC, an independent contractor and QP as a CPG member with AIPG.
8. Los Reyes – Open Pit (Plant and Heap Leach):
 - a. Open Pit Resource estimates are based on economically constrained revenue factor one (RF1) open pits generated using the Hochbaum Pseudoflow algorithm in Datamine's Studio NPVS and the following optimization parameters:
 - b. Assumes plant recoveries of 94.8% for Au and 81.1% for Ag and heap leach recoveries of 73% Au and 25% Ag.
 - c. Pit slopes by area ranging from 42-47 degrees overall slope angle.
 - d. 5% mineralized material loss and 5% dilution factor applied to the 5 m x 5 m x 5 m open pit resource block models.

- e. Mining costs of \$2.00 per tonne of waste mined and \$2.50 per tonne of mineralized material mined. Processing costs of \$16.81 per tonne processed. Heap leach costs of \$5.53 per tonne processed. G&A cost of \$2.00 per tonne of material processed.
 - f. A 0.17 g/t gold only cutoff was applied to ex-pit processed material (which is above the heap-leaching NSR cutoff).
 - g. $\text{AuEq (Open Pit Plant)} = \text{Au (g/t)} + (\text{Ag (g/t)} * 0.0111)$ and $\text{AuEq (Open Pit Heap Leach)} = \text{Au (g/t)} + (\text{Ag (g/t)} * 0.0044)$.
9. Los Reyes – Underground (Plant):
- a. Underground Resource estimates are based on economically constrained stopes generated using Datamine's Mineable Shape Optimizer (MSO) algorithm and the following optimization parameters:
 - b. Diluted to a minimum 3 m stope width with a 95% mining recovery.
 - c. Assumes plant recoveries of 94.8% for Au and 81.1% for Ag.
 - d. Longhole Open Stope mining with a \$50.00 per tonne cost. Processing costs of \$17.24 per tonne processed. G&A cost of \$4.50 per tonne of material processed.
 - e. $\text{AuEq (Underground Plant)} = \text{Au (g/t)} + (\text{Ag (g/t)} * 0.0111)$.

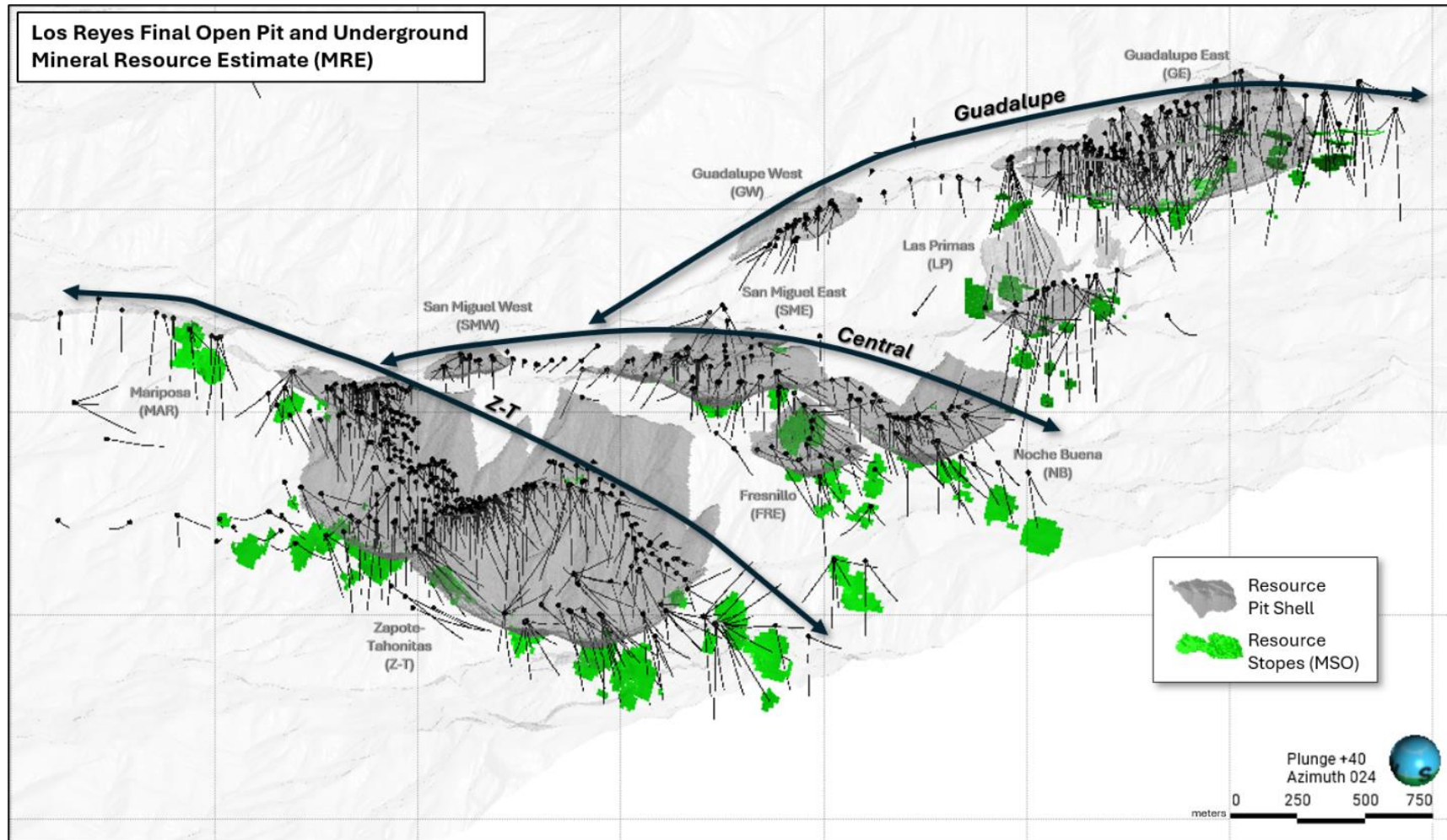
Table 14-2: Mineral Resource Statement by mining method and area

Area	Mining Method	Class	Tonnage (kt)	Au (g/t)	Au (koz)	Ag (g/t)	Ag (koz)	AuEq (g/t)	AuEq (koz)
Z-T Trend	Open pit	Indicated	30,418	0.77	757	21.4	20,897	0.98	954
		Inferred	9,428	0.64	195	29.5	8,942	0.92	279
	Underground	Indicated	5	1.25	0	13.4	2	1.40	0
		Inferred	2,263	1.76	128	65.5	4,766	2.48	181
Guadalupe Trend	Open pit	Indicated	1,944	0.67	42	21.3	1,333	0.87	54
		Inferred	228	0.32	2	18.6	136	0.46	3
	Underground	Indicated	5,247	2.49	420	130.8	22,060	3.94	664
		Inferred	1,231	2.16	86	128.1	5,072	3.58	142
Central Trend	Open pit	Indicated	11,798	0.75	285	27.9	10,568	1.03	390
		Inferred	3,348	0.49	52	19.4	2,091	0.68	73
	Underground	Indicated	58	2.76	5	46.6	87	3.27	6
		Inferred	722	1.19	28	69.5	1,614	1.96	46
Generative Areas	Open pit	Indicated	1,729	0.56	31	14.9	831	0.71	39
		Inferred	2,464	0.60	48	9.6	761	0.69	55
	Underground	Indicated	51	1.50	2	49.9	82	2.05	3
		Inferred	1,213	2.08	81	31.6	1,232	2.43	95
Total	Open pit	Indicated	45,890	0.76	1,115	22.8	33,629	0.97	1,437
		Inferred	15,469	0.60	298	24.0	11,930	0.82	410
	Underground	Indicated	5,361	2.48	428	129.0	22,231	3.91	674
		Inferred	5,430	1.84	322	72.7	12,685	2.65	463
	Total	Indicated	51,250	0.94	1,542	33.9	55,860	1.28	2,111
		Inferred	20,899	0.92	620	36.6	24,615	1.30	872

Table 14-3: Description of Resource Components for Tables 14-1 and 14-2, and Figure 14-1

Area	Deposit	Resource Description
Z-T Trend	Zapote-Tahonitas	Revenue Factor 1.0 Open Pits @0.17g/T Au only cutoff plus remaining economic stopes
Guadalupe Trend	Guadalupe East	Economic underground stopes
	Guadalupe West	Revenue Factor 1.0 Open Pits @0.17g/T Au only cutoff
Central Trend	Noche Buena	Revenue Factor 1.0 Open Pits @0.17g/T Au only cutoff plus remaining economic stopes
	San Miguel East	Revenue Factor 1.0 Open Pits @0.17g/T Au only cutoff plus remaining economic stopes
	San Miguel West	Revenue Factor 1.0 Open Pits @0.17g/T Au only cutoff
Generative Areas	Mariposa	Economic underground stopes
	Las Primas	Revenue Factor 1.0 Open Pits @0.17g/T Au only cutoff plus remaining economic stopes
	Fresnillo	Revenue Factor 1.0 Open Pits @0.17g/T Au only cutoff plus remaining economic stopes

Figure 14-1: Oblique view showing pit shells and stopes Los Reyes Mineral Resource Estimate (MRE)



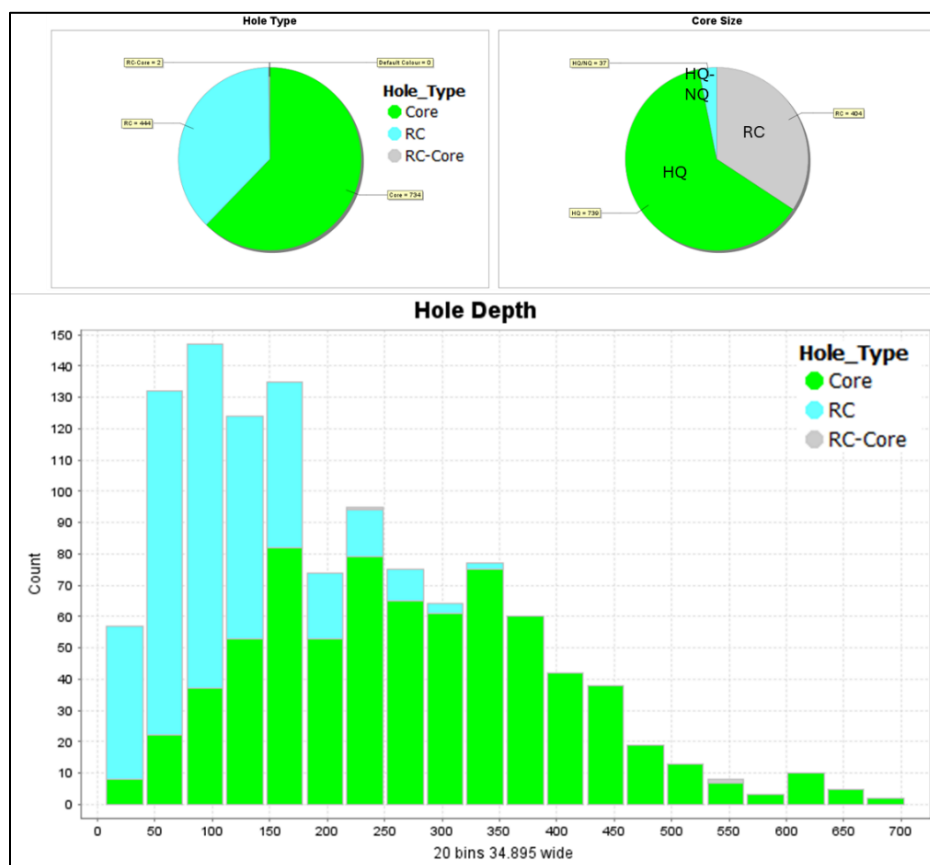
Source: Torex, 2026

14.2 Database

The Los Reyes database is maintained in a Microsoft SQL database management system, which contains collar locations, downhole survey data, qualitative logging information, and assay and multielement geochemical data, among other items. Data for geologic modeling and resource estimation purposes were exported as .csv files and then imported into Leapfrog Geo v.2024.1.1 and ioGAS v.8.2 for analysis. The database used for this report includes all drillholes on the property completed on or before December 31, 2024.

The full database export contains 1,240 drillholes, of which 60.6% are diamond drill holes, 39.2% are RC holes, and 0.2% are RC pre-collars with core tails. Fifty-nine holes were excluded from the Resource estimate, either due to uncertainty in assay, survey, or collar information for historical holes, due to abandonment prior to achieving targeted depth, or due to the drillhole location falling outside the resource model extents. The resulting final estimation dataset contains 373 drillholes completed by Northern Crown Mines between 1993 and 1997, 5 completed by Meridian in 2001, 48 completed by Vista in 2011 and 2012, 41 completed by Great Panther in 2015, and 714 completed by Prime between 2021 and December 31, 2024, for a total of 260,347 m in 1,181 holes. Drilling statistics for the final estimation dataset are presented in Figure 14-2 and Table 14-4.

Figure 14-2: Hole type, core size, and hole depth statistics for the Los Reyes final estimation dataset1



Notes: 1. Statistics do not consider information from drillholes excluded from the estimation.

Source: Torex, 2026

Table 14-4: Final resource estimation dataset breakdown by area, company, and year

Company		NCM					Meridian	Vista			Great Panther	Prime					Total
Year		1993	1994	1996	1997	Sub-total	2001	2011	2012	sub-total	2015	2021 ¹	2022 ²	2023	2024 ³	sub-total	
Guadalupe	drill holes			49	30	79		10	8	18	9	42	105	23	16	186	292
	metres			5,333	5,216	10,549		1,470	1,482	2,952	1,494	11,297	36,973	8,031	5,516	61,817	76,812
Z-T	drill holes	28	38	55	113	234	5		15	15	11	58	94	80	67	299	564
	metres	2,337	3,376	4,249	8,025	17,986	829		1,886	1,886	1,156	11,726	21,692	27,234	22,820	83,417	105,276
Central	drill holes		4	9	46	59			15	15	21	31	50	29	24	134	229
	metres		247	1,017	5,003	6,267			2,582	2,582	2,855	6,769	11,759	8,751	9,023	36,301	48,004
Other	drill holes				1	1				0		10	11	39	35	95	96
	metres				166	166				0		1,901	3,033	11,642	13,514	30,090	30,256
Total	drill holes	28	42	113	190	373	5	10	38	48	41	141	260	171	142	714	1,181
	metres	2,337	3,623	10,599	18,410	34,968	829	1,470	5,950	7,420	5,505	31,692	73,457	55,658	50,874	211,625	260,347

Notes:

1. Two drill holes were started in December 2020, one of which was completed in 2020.
2. Only holes completed prior to December 31, 2024, are included in the current Resource Estimate.

14.3 Mineral Resource Estimate

The Los Reyes resource estimate was prepared under the supervision of Sims Resources LLC (John Sims, Independent QP). Geologic and estimation domains were constructed using Leapfrog Geo v.2024.1.1, including input from geochemical analyses completed in ioGAS v.8.2. Geostatistical evaluations and Exploratory Data Analysis (EDA), including topcut selection, declustering, variography, and Sequential Gaussian Simulation (SGS) were completed using Snowden Supervisor v.9.0. Resource estimation was prepared using Leapfrog EDGE v.2024.1.1.

Two sets of block models were prepared for this resource estimate - (1) 5x5x5m block models were generated for use in Open Pit optimization, and (2) 2.5x2.5x2.5m block models were generated for use in underground Mineable Shape Optimizer (MSO). The 5x5x5m and 2.5x2.5x2.5m models are estimated independently, and employ distinct compositing, domaining, and estimation strategies which are suitable for their respective mining methods and Selective Mining Units (SMU). Parameters for each set of block models are described in detail below.

14.3.1 Data Preparation

Drillhole data used in the Los Reyes resource estimate were checked for overlapping sample intervals, negative or invalid values, and irregular downhole survey deviation in Leapfrog Geo. All errors were assessed and corrected prior to completing statistical analysis and estimation.

Drillhole collars were also visually checked against the most current topographic surface. Most collars are set to the topographic surface, with minor deviations (<1 metre) in some collars attributed to local variations in the topography.

Gold and silver assay values less than the detection limit were assigned a value equal to half of the detection limit value, which, depending on the analysis date and laboratory, was 0.001ppm, 0.005ppm, or 0.03ppm for Au and 0.1ppm, 0.5ppm, or 0.7ppm for Ag. Null values for Au and Ag were assigned for all intervals with no recovery or where historical mine workings, voids, or backfill material were encountered.

14.3.2 Wireframes

14.3.2.1 Topography

Topography data for the Los Reyes project was gathered by Pioneer Exploration Consultants Ltd. in March 2021. No significant disturbance (mining) has occurred in the project area since. Data were collected by airborne LiDAR survey and were processed using Green Valley LiDAR 360 software, and the resulting 1 m-resolution bare earth surface was used to define the topographic limits for the Los Reyes geologic model and resource estimation block models discussed below.

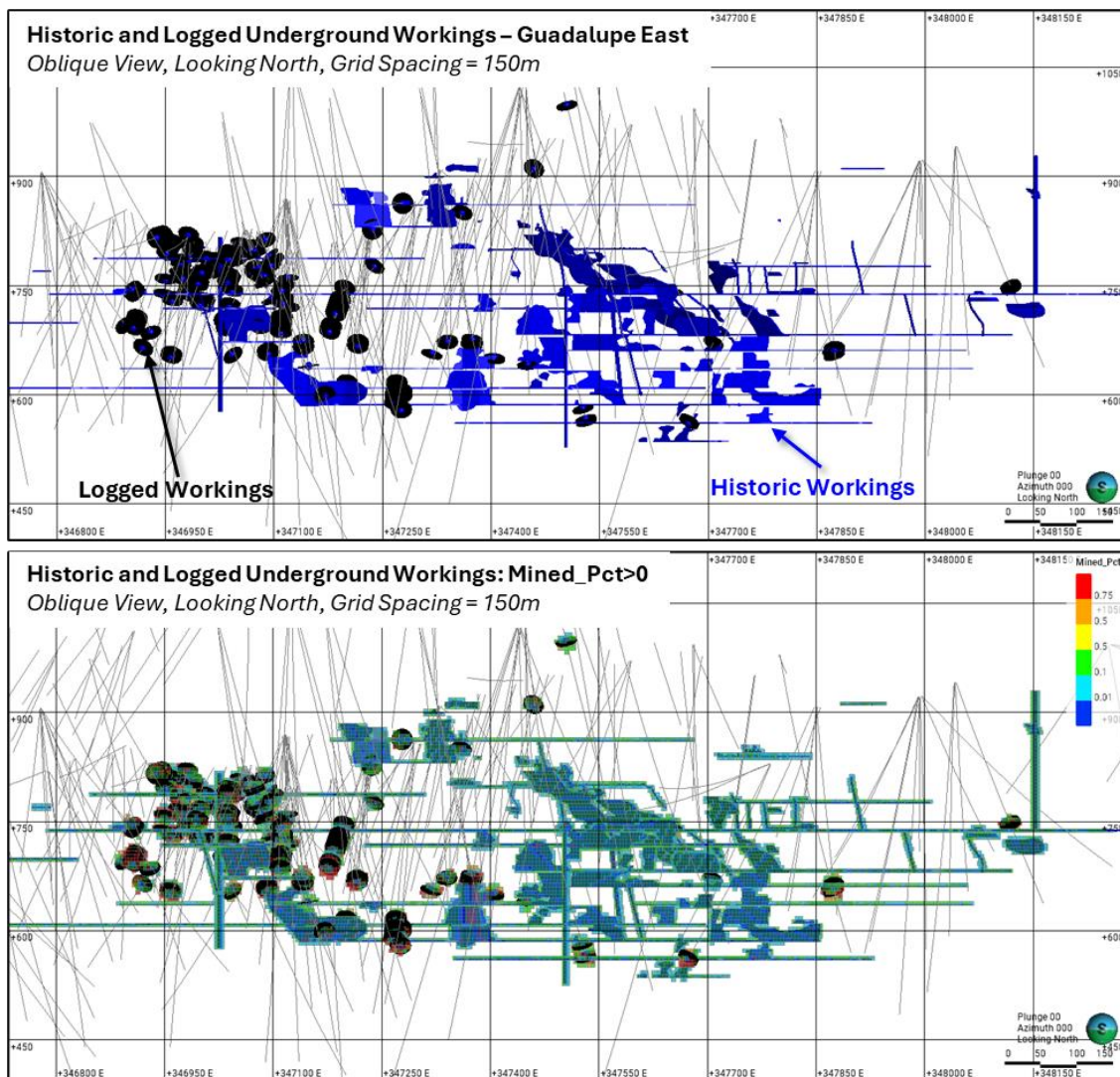
14.3.2.2 Historical Underground Workings

Historical mine workings solids were constructed in AutoCAD using a combination of underground LiDAR scans, digitized plan maps and long sections, and logged working intervals intercepted in drilling (Lindstrom, 2023). LiDAR scans were collected and processed by Unmanned Aerial Services Incorporated in April 2022 for accessible underground workings on the property. The output wireframes from this work were then used in conjunction with intercepted workings intervals from surveyed drillholes to georeferenced historical map data, particularly in areas where collapse, backfill, or other conditions prevented full access to scan the workings. Additional data were gathered by Unmanned Aerial Services in 2024, and were used to supplement the 2023 dataset.

The proportion of a block which lies within the historical workings wireframes was calculated for model depletion and reporting purposes. Final block density was then determined by considering the weighted average of the in-situ portion of the block, at its original density according to lithology and estimation domain, and the historically mined portion of the block, at an assigned density of 0.0 g/cm³. Logged workings intervals encountered in drilling outside the wireframes were also depleted in the model using an indicator estimate, in which logged workings were assigned a value of 1 and all other lithology codes were assigned a value of 0. Blocks with estimated indicator values greater than 0.5 were then flagged to the model as 100% mined (Mined_Pct = 1, Figure 14-3 and Figure 14-4).

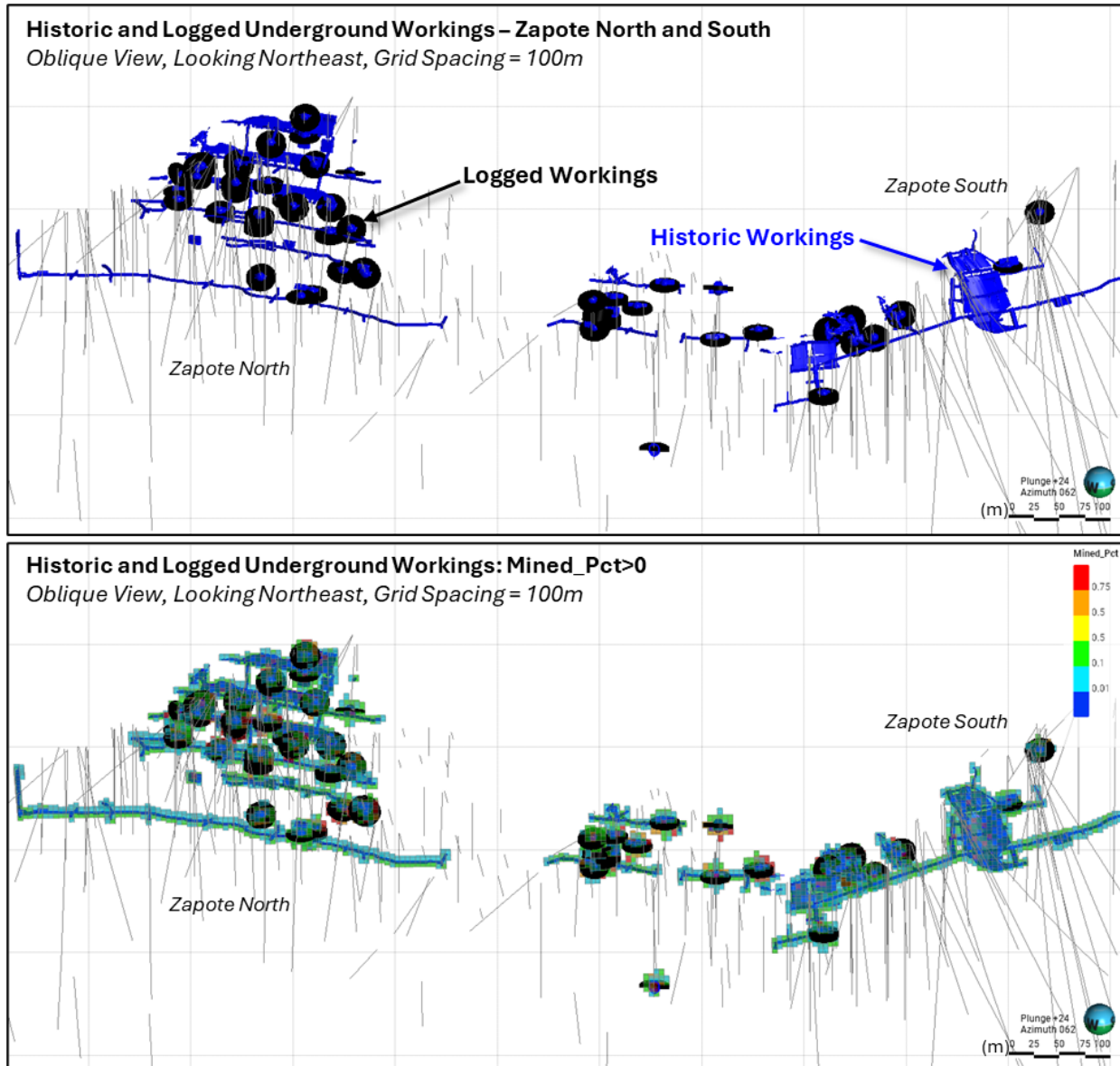
Sensitivity work assuming a mined-out halo around mapped or inferred workings was also performed. The overall resource was not materially sensitive to these estimates.

Figure 14-3: Upper – Underground workings wireframes and logged workings intervals in the Guadalupe East area. Lower – Model filtered for blocks with mined proportion greater than 0 (Mined_Pct >0).



Source: Torex, 2026

Figure 14-4: Upper – Underground workings wireframes and logged workings intervals in the Zapote North and South areas. Lower – Model filtered for blocks with mined proportion greater than 0 (Mined_Pct >0).



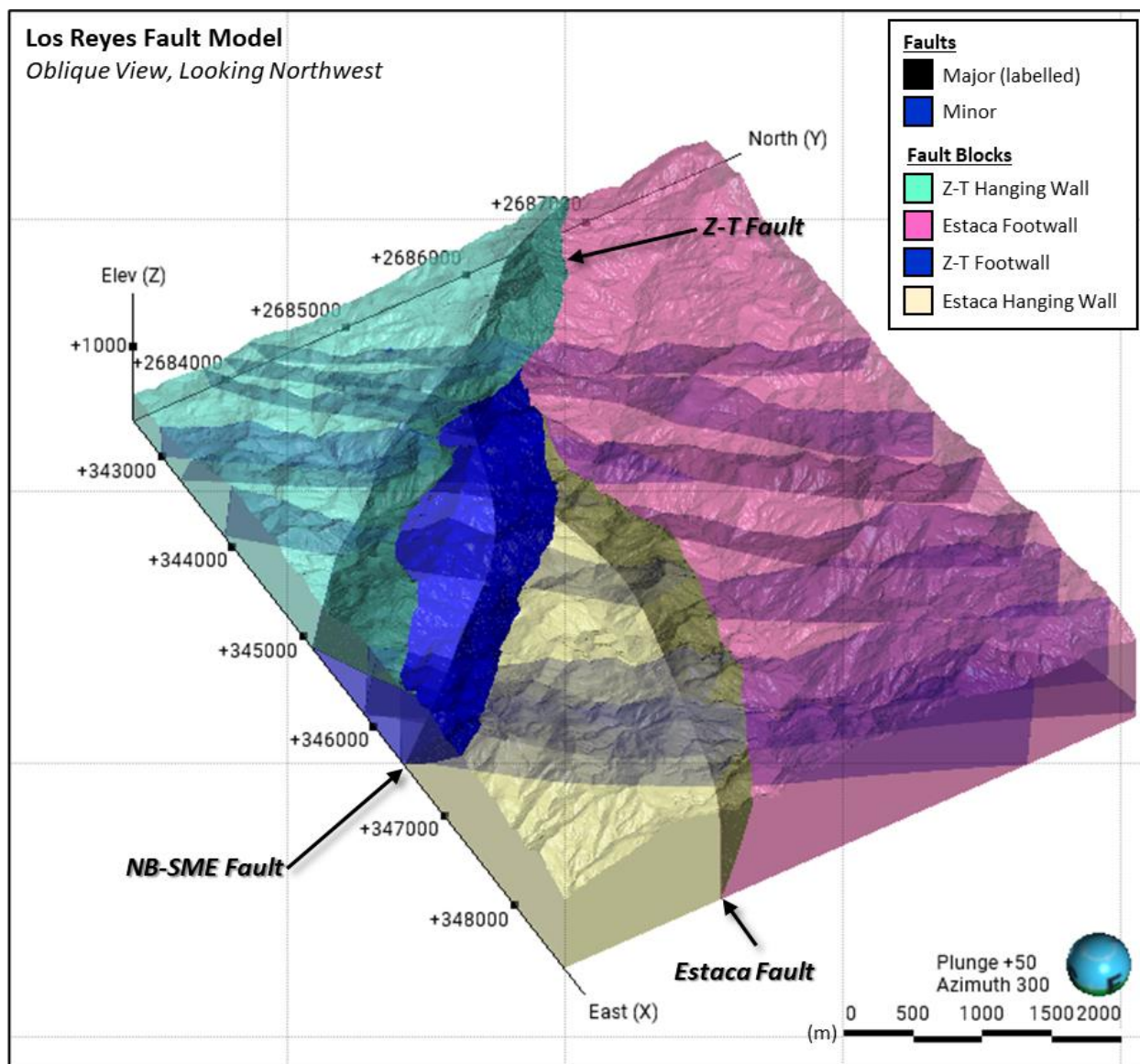
Source: Torex, 2026

14.3.2.3 Lithology and Faults

Lithology and fault wireframes were generated in Leapfrog Geo using interval selection based on a merged table containing qualitative logging data (lithology, alteration, structure), Au and Ag assays, multi-element geochemistry, and geotechnical data (RQD, recovery). Structural measurements from oriented core were used to influence modeled surfaces where available. GIS data from 1:5,000 and 1:2,000 scale geologic mapping, including lithologic contacts and structural point data, were also applied as direct inputs to the modeled surfaces. Three faults were activated in the geologic model where major offsets in lithology are

apparent; (1) the Z-T fault places rhyolitic intrusives in the hanging wall against andesite and granodiorite in the footwall, (2) the NB-SME fault places dacitic intrusives in the hanging wall against andesites in the footwall, and (3) the Estaca fault places rhyolitic intrusives, dacitic intrusives, and andesites in the hanging wall against granodiorite in the footwall (Figure 14-5 and Figure 14-6). These three fault zones are pre-syn mineral and are key district-scale controlling structures for quartz veining and Au-Ag mineralization. Several minor, generally northeast-striking post-mineral faults were also built but were not activated in the model because they do not show appreciable offset. All fault and lithology wireframes were snapped to drillhole data and were checked for closure and consistency prior to resource estimation. Six major lithologies were modeled and are outlined in Table 14-5.

Figure 14-5: Los Reyes Fault Model

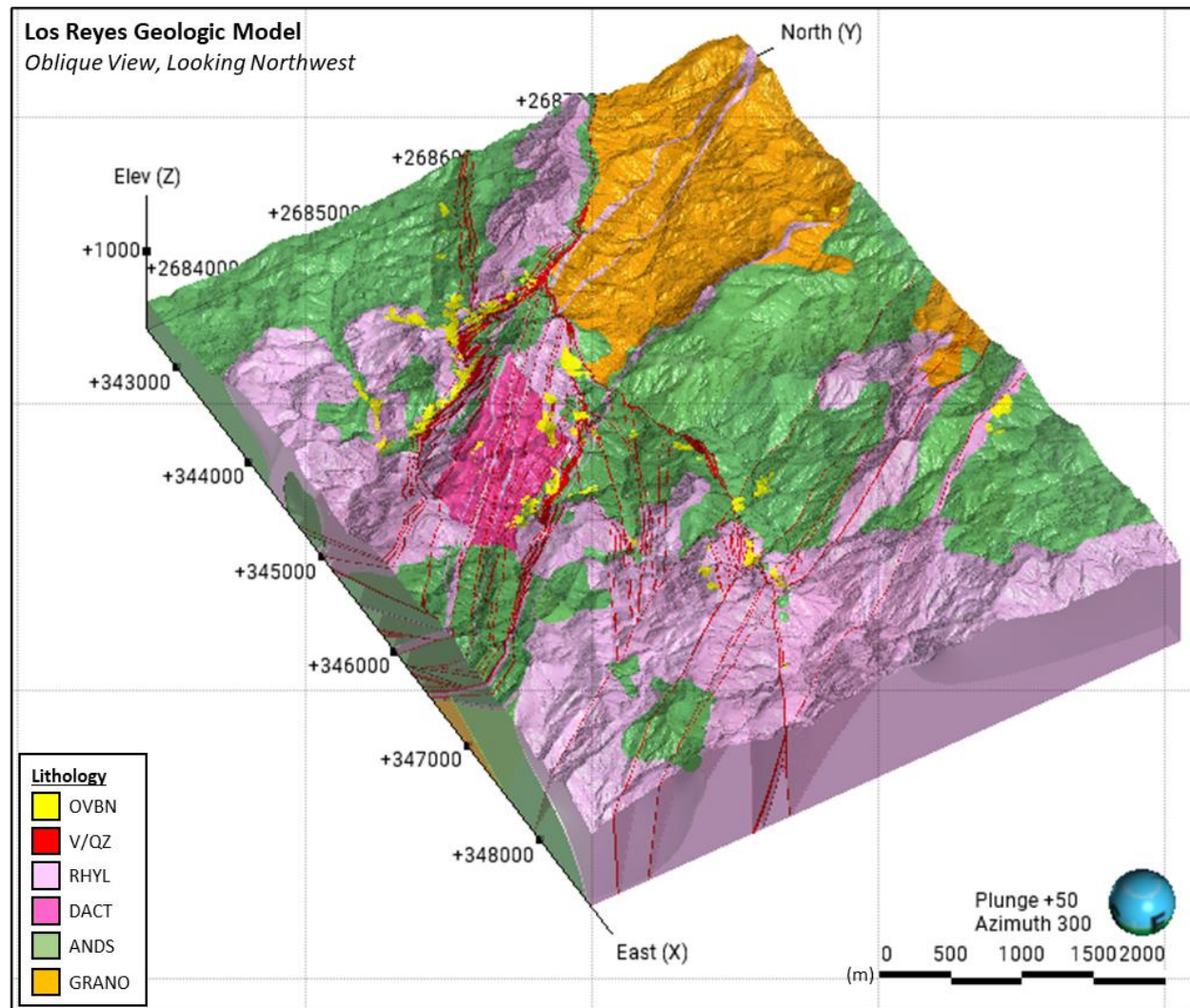


Source: Torex, 2026

Table 14-5: Lithology Codes

Code	Lithology	Description
1	V/QZ	Quartz vein/stockwork and silicified breccias
2	RHYL	Rhyolitic intrusive domes and dikes
3	DACT	Dacitic intrusive domes and dikes
4	ANDS	Andesites and andesitic tuffs
5	GRANO	Granodiorite
6	OVBN	Overburden

Figure 14-6: Los Reyes Lithology Model

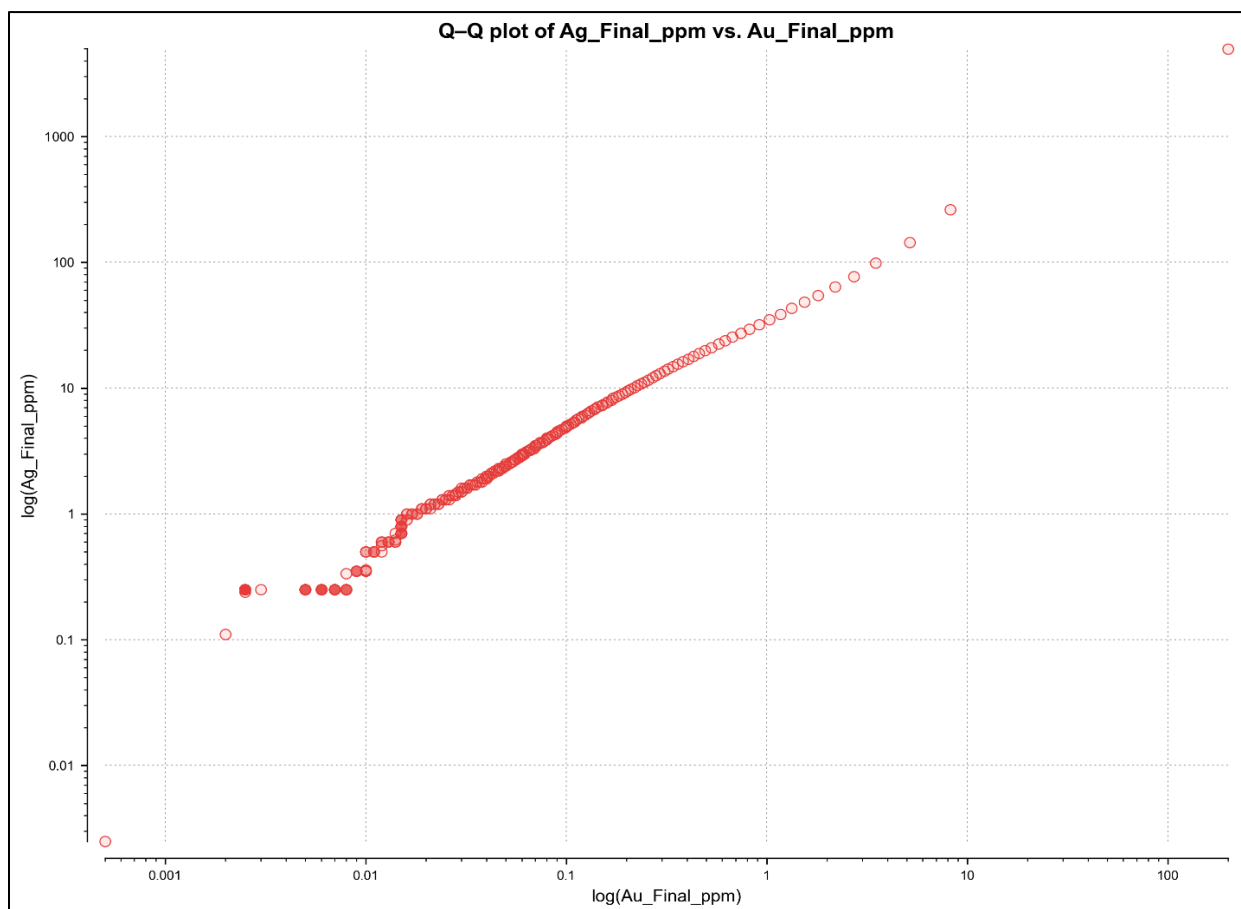


Source: Torex, 2026

14.3.2.4 Mineralization

Gold and silver mineralization are spatially and genetically associated with low-sulphidation epithermal quartz veins at Los Reyes, with a strong positive correlation observed between concentrations of the two metals across all deposits (Figure 14-7). Given this relationship between Au and Ag, mineralized domain wireframes were constructed based on gold-equivalent (AuEq) grades, using the formula: $\text{AuEq} = \text{Au (g/t)} + (\text{Ag (g/t)} \times (\$25.24/\$1,950))$, where \$25.24/oz and \$1,950/oz are the assumed metals prices for silver and gold, respectively. This calculation does not consider process recoveries and as such is only used for the purposes of estimation domain construction, was not used for resource reporting, and was kept identical to that used in domain construction for the previous, October, 2024 Mineral Resource Estimate. A cutoff grade of 0.2 g/t AuEq was selected for mineralized domains in the 5x5x5m set of block models, and a 1.0 g/t AuEq cutoff grade was selected for mineralized domains in the 2.5x2.5x2.5m set of block models described below. Grade shells were generated for both cutoff grades using the Indicator Interpolant tool and spherical interpolant function in Leapfrog Geo, with geometry and continuity controlled by a structural trend generated from vein midpoint surfaces in the quartz vein and breccia model (V/QZ). Grade shell volumes were restricted to the V/QZ solid and were subsequently divided based on changes in orientation to generate the final mineralized domains for estimation (Figure 14-8 and Figure 14-9; Table 14-6 and Table 14-7).

Figure 14-7: Q-Q plot showing Au vs. Ag in assays at Los Reyes

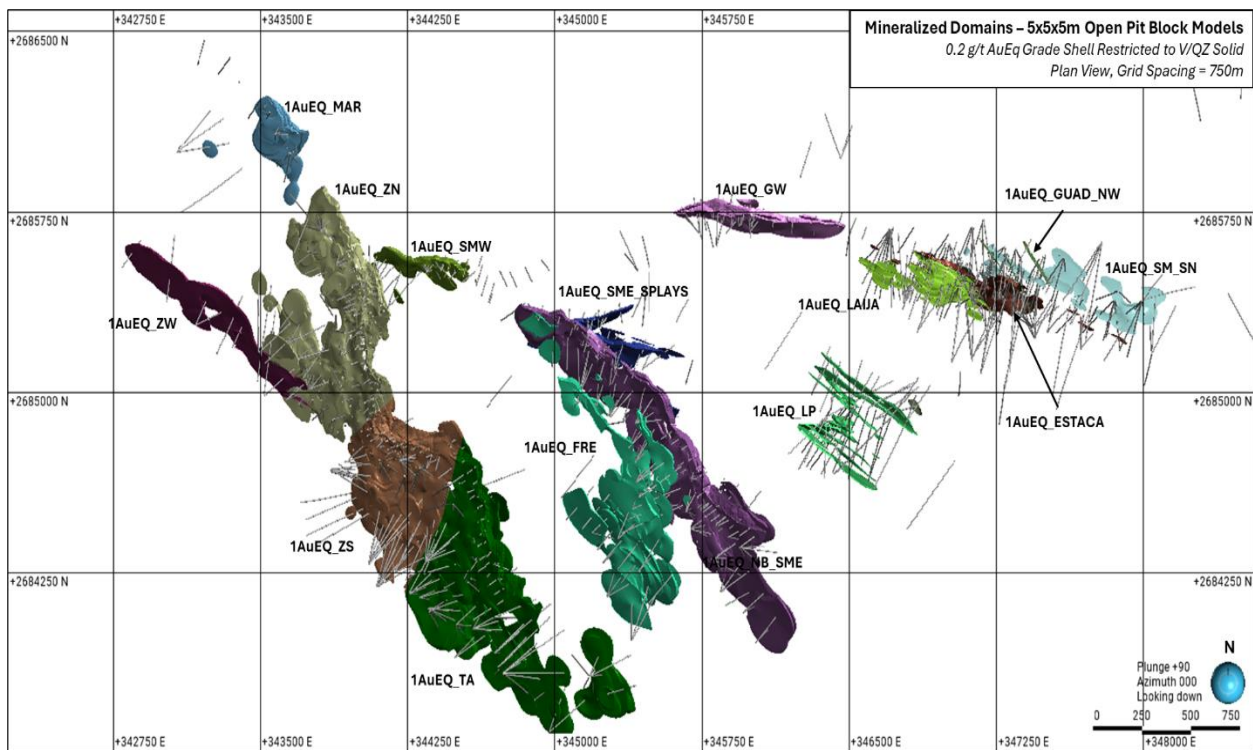


Source: Torex, 2026

14.3.2.5 5x5x5m Block Models

The domain strategy for the 5x5x5m block models used in Open Pit optimization follows a nested approach, with three major domain groups – (1) Mineralized domains, with the prefix '02AuEQ', represent volumes inside the V/QZ solid that are also inside the grade shells generated at a 0.2 g/t AuEq cutoff grade (Figure 14-8). (2) Quartz vein and breccia envelope domains with the prefix QV_BX, represent volumes which are inside the V/QZ solid, but fall outside the 0.2 g/t AuEq grade shells. (3) Background domains with the prefix BACKGROUND, represent volumes outside the grade shells and outside the V/QZ solid. Table 14-6 lists all domains estimated in the 5x5x5m block models.

Figure 14-8: Mineralized domains for Au and Ag grade estimation in 5x5x5m block models used for open pit optimization



Source: Torex, 2026

Table 14-6: Los Reyes estimation domains for Au and Ag grade estimation in the 5x5x5m block models used for open pit optimization

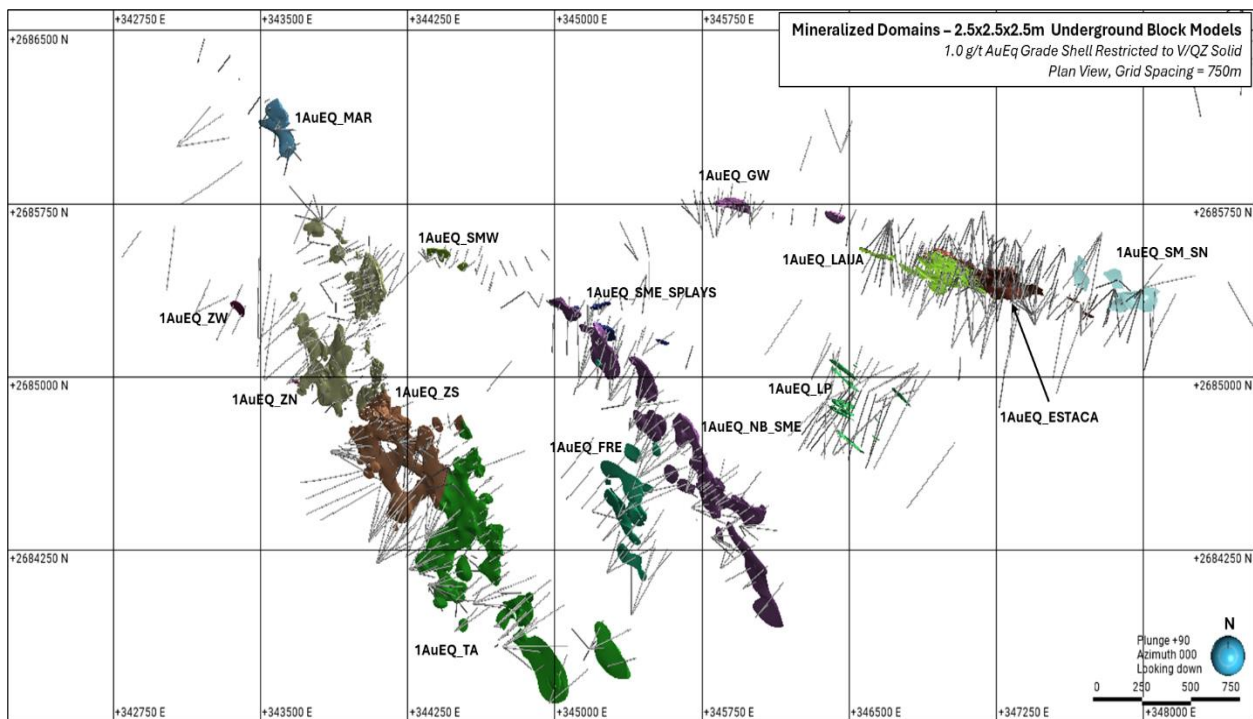
Domain Group	Block Model	Code	Domain	Description
QV_BX mineralized	ZTM_NB_SM_5x5x5_Rotated	100	02AuEQ_MAR	Mariposa mineralized quartz vein/bx (inside 0.2 g/t Au grade shell)
		101	02AuEQ_ZN	Zapote North mineralized quartz vein/bx (inside 0.2 g/t Au grade shell)
		102	02AuEQ_ZS	Zapote South mineralized quartz vein/bx (inside 0.2 g/t Au grade shell)
		103	02AuEQ_TA	Tahonitas mineralized quartz vein/bx (inside 0.2 g/t Au grade shell)
		104	02AuEQ_SMW	San Miguel West mineralized quartz vein/bx (inside 0.2 g/t Au grade shell)
		105	02AuEQ_NB_SME	Noche Buena/San Miguel East mineralized quartz vein/bx (inside 0.2 g/t Au grade shell)
		106	02AuEQ_SME_SPLAYS	San Miguel East Splays mineralized quartz vein/bx (inside 0.2 g/t Au grade shell)
	GUAD_LP_5x5x5_Rotated	114	02AuEQ_ZW	Zapote West mineralized quartz vein/bx (inside 0.2 g/t Au grade shell)
		107	02AuEQ_GW	Guadalupe West mineralized quartz vein/bx (inside 0.2 g/t Au grade shell)
		108	02AuEQ_LAIJA	Laija mineralized quartz vein/bx (inside 0.2 g/t Au grade shell)
		109	02AuEQ_ESTACA	Estaca mineralized quartz vein/bx (inside 0.2 g/t Au grade shell)
		110	02AuEQ_SM_SN	San Manuel and San Nicolas quartz vein/bx (inside 0.2 g/t Au grade shell)
		111	02AuEQ_LP	Las Primas mineralized quartz vein/bx (inside 0.2 g/t Au grade shell)
		112	02AuEQ_FRE	Fresnillo mineralized quartz vein/bx (inside 0.2 g/t Au grade shell)
		113	02AuEQ_GUAD_NW	NW-striking Guadalupe mineralized quartz vein/bx (inside 0.2g/t Au grade shell)
QV_BX non-mineralized	ZTM_NB_SM_5x5x5_Rotated	200	QV_BX_ZTM	ZTM structure non-mineralized quartz vein/bx (outside 0.2g/t Au grade shell)
		201	QV_BX_FRE	Fresnillo non-mineralized quartz vein/bx (outside 0.2g/t Au grade shell)
		202	QV_BX_NB_SME	Noche Buena/San Miguel East non-mineralized quartz vein/bx (outside 0.2g/t Au grade shell)
		203	QV_BX_SME_SPLAYS	San Miguel East Splays non-mineralized quartz vein/bx (outside 0.2g/t Au grade shell)
		210	QV_BX_ZW	Zapote West non-mineralized quartz vein/bx (outside 0.2g/t Au grade shell)
	GUAD_LP_5x5x5_Rotated	204	QV_BX_LP	Las Primas non-mineralized quartz vein/bx (outside 0.2g/t Au grade shell)

Domain Group	Block Model	Code	Domain	Description
		205	QV_BX_GW	Guadalupe West non-mineralized quartz vein/bx (outside 0.2g/t Au grade shell)
		206	QV_BX_LAIJA	Laija non-mineralized quartz vein/bx (outside 0.2g/t Au grade shell)
		207	QV_BX_ESTACA	Estaca non-mineralized quartz vein/bx (outside 0.2g/t Au grade shell)
		208	QV_BX_SM_SN	San Manuel and San Nicolas non-mineralized quartz vein/bx (outside 0.2g/t Au grade shell)
		209	QV_BX_GUAD_NW	NW-striking Guadalupe structures non-mineralized quartz vein/bx (outside 0.2 g/t Au grade shell)
Background	ZTM_NB_SM_5x5x5_Rotated	301	BACKGROUND_ZTM_NB_SM	ZTM_NB_SM_5x5x5 model background domain (outside vein/bx solid)
	GUAD_LP_5x5x5_Rotated	302	BACKGROUND_GUAD_LP	GUAD_LP_5x5x5 model background domain (outside vein/bx solid)

14.3.2.6 2.5x2.5x2.5m Block Models

The domain strategy for the 2.5x2.5x2.5m block models used in underground Mineable Shape Optimization (MSO) follows a nested approach, with three major domain groups – (1) Mineralized domains, with the prefix '1AuEQ', represent volumes inside the V/QZ solid that are also inside the grade shells generated at a 1.0 g/t AuEq cutoff grade (Figure 14-9). (2) Quartz vein and breccia envelope domains with the prefix QV_BX, represent volumes which are inside the V/QZ solid, but fall outside the 1.0 g/t AuEq grade shells. (3) Background domains with the prefix BACKGROUND, represent volumes outside the grade shells and outside the V/QZ solid. Table 14-7 lists all domains estimated in the 2.5x2.5x2.5m block models.

Figure 14-9: Mineralized domains for Au and Ag grade estimation in 2.5x2.5x2.5m block models used for Mineable Shape Optimization (MSO)



Source: Torex, 2026

Table 14-7: Los Reyes estimation domains for Au and Ag grade estimation in the 2.5x2.5x2.5m block models used for underground optimization

Domain Group	Model	Domain Code	Domain	Description
QV_BX mineralized	ZTM_NB_SM_5x5x5_Rotated	100	1AuEQ_MAR	Mariposa mineralized quartz vein/bx (inside 1.0 g/t Au grade shell)
		101	1AuEQ_ZN	Zapote North mineralized quartz vein/bx (inside 1.0 g/t Au grade shell)
		102	1AuEQ_ZS	Zapote South mineralized quartz vein/bx (inside 1.0 g/t Au grade shell)
		103	1AuEQ_TA	Tahonitas mineralized quartz vein/bx (inside 1.0 g/t Au grade shell)
		104	1AuEQ_SMW	San Miguel West mineralized quartz vein/bx (inside 1.0 g/t Au grade shell)
		105	1AuEQ_NB_SME	Noche Buena/San Miguel East mineralized quartz vein/bx (inside 1.0 g/t Au grade shell)
		106	1AuEQ_SME_SPLAYS	San Miguel East Splays mineralized quartz vein/bx (inside 1.0 g/t Au grade shell)
		114	1AuEQ_ZW	Zapote West mineralized quartz vein/bx (inside 1.0 g/t Au grade shell)
	GUAD_LP_5x5x5_Rotated	107	1AuEQ_GW	Guadalupe West mineralized quartz vein/bx (inside 1.0 g/t Au grade shell)
		108	1AuEQ_LAIJA	Laija mineralized quartz vein/bx (inside 1.0 g/t Au grade shell)
		109	1AuEQ_ESTACA	Estaca mineralized quartz vein/bx (inside 1.0 g/t Au grade shell)
		110	1AuEQ_SM_SN	San Manuel and San Nicolas quartz vein/bx (inside 1.0 g/t Au grade shell)
		111	1AuEQ_LP	Las Primas mineralized quartz vein/bx (inside 1.0 g/t Au grade shell)
		112	1AuEQ_FRE	Fresnillo mineralized quartz vein/bx (inside 1.0 g/t Au grade shell)
		113	1AuEQ_GUAD_NW	NW-striking Guadalupe mineralized quartz vein/bx (inside 1.0g/t Au grade shell)

Domain Group	Model	Domain Code	Domain	Description
QV_BX non-mineralized	ZTM_NB_SM_5x5x5_Rotated	200	QV_BX_ZTM	ZTM structure non-mineralized quartz vein/bx (outside 1.0g/t Au grade shell)
		201	QV_BX_FRE	Fresnillo non-mineralized quartz vein/bx (outside 1.0g/t Au grade shell)
		202	QV_BX_NB_SME	Noche Buena/San Miguel East non-mineralized quartz vein/bx (outside 1.0g/t Au grade shell)
		203	QV_BX_SME_SPLAYS	San Miguel East Splays non-mineralized quartz vein/bx (outside 1.0g/t Au grade shell)
		210	QV_BX_ZW	Zapote West non-mineralized quartz vein/bx (outside 1.0g/t Au grade shell)
	GUAD_LP_5x5x5_Rotated	204	QV_BX_LP	Las Primas non-mineralized quartz vein/bx (outside 1.0g/t Au grade shell)
		205	QV_BX_GW	Guadalupe West non-mineralized quartz vein/bx (outside 1.0g/t Au grade shell)
		206	QV_BX_LAIJA	Laija non-mineralized quartz vein/bx (outside 1.0g/t Au grade shell)
		207	QV_BX_ESTACA	Estaca non-mineralized quartz vein/bx (outside 1.0g/t Au grade shell)
		208	QV_BX_SM_SN	San Manuel and San Nicolas non-mineralized quartz vein/bx (outside 1.0g/t Au grade shell)
		209	QV_BX_GUAD_NW	NW-striking Guadalupe structures non-mineralized quartz vein/bx (outside 1.0g/t Au grade shell)
Background	ZTM_NB_SM_5x5x5_Rotated	301	BACKGROUND_ZTM_NB_SM	ZTM_NB_SM_5x5x5 model background domain (outside vein/bx solid)
	GUAD_LP_5x5x5_Rotated	302	BACKGROUND_GUAD_LP	GUAD_LP_5x5x5 model background domain (outside vein/bx solid)

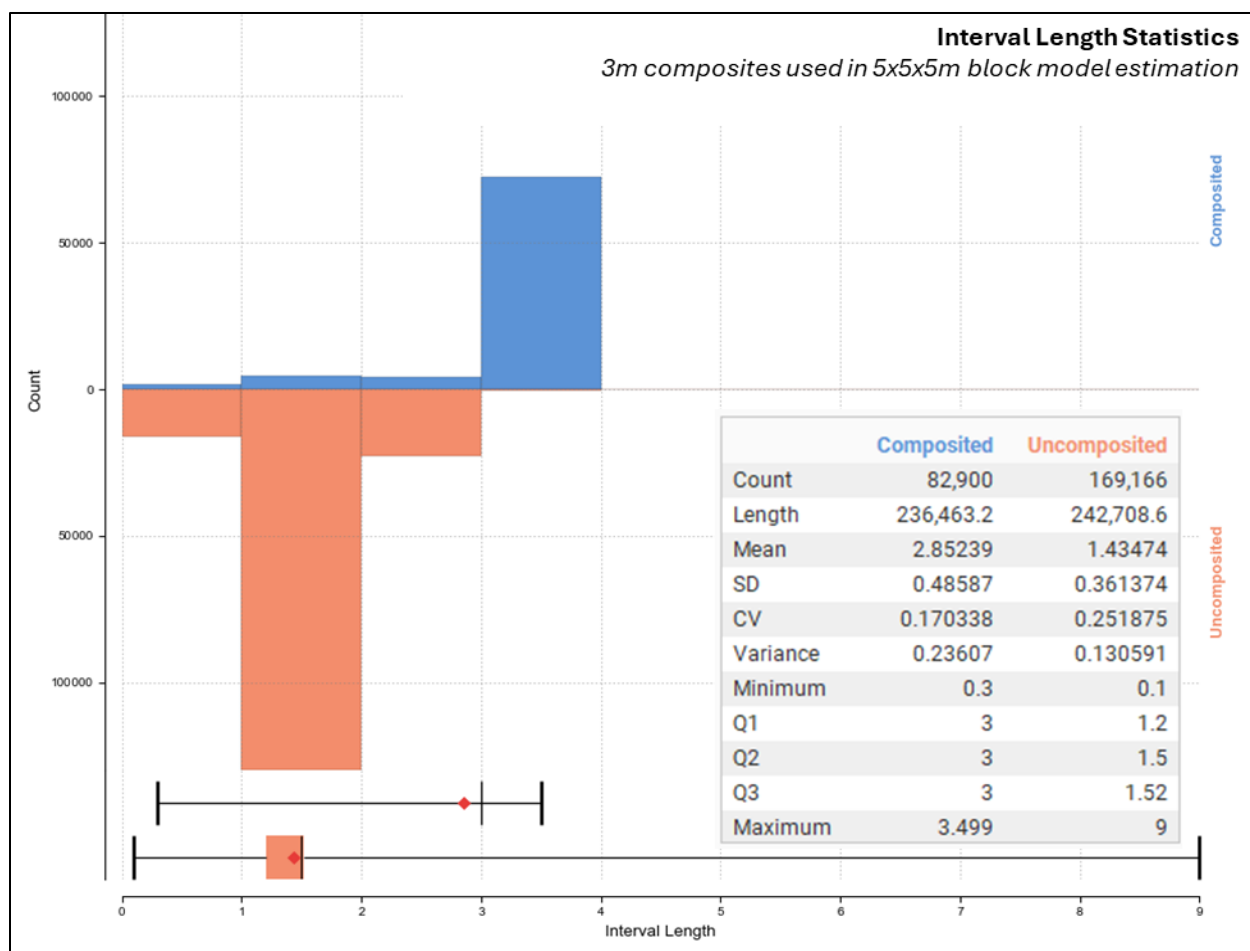
14.3.3 Composites

The most frequent sample interval in the assay data table is 1.5 metre, with sample intervals 1.5 ± 0.05 metre in length representing 51.7% of the total dataset. Compositing strategies for the 5x5x5m and 2.5x2.5x2.5m block models are distinct, given the difference in Selective Mining Unit (SMU), and are outlined individually below.

14.3.3.1 5x5x5m Block Models

In order to reduce variability and ensure the same support for estimation, a 3 metre compositing interval was selected for gold and silver estimation in the 5x5x5m block models. The 3 metre compositing length was selected because it is a multiple of the most frequent, 1.5 metre sample length in the assay table, is more appropriate for the 5x5x5m SMU, and because it maintains the same composite interval to SMU ratio as applied for the 2.5x2.5x2.5m block models (0.6). Composites were generated to respect the domain boundaries presented in Table 14-6. Figure 14-10 shows a global comparison between raw assay interval lengths and composited data used for estimation in the 5x5x5m block models.

Figure 14-10: Global compositing interval length statistics for 3m composites used for estimation in 5x5x5m block models

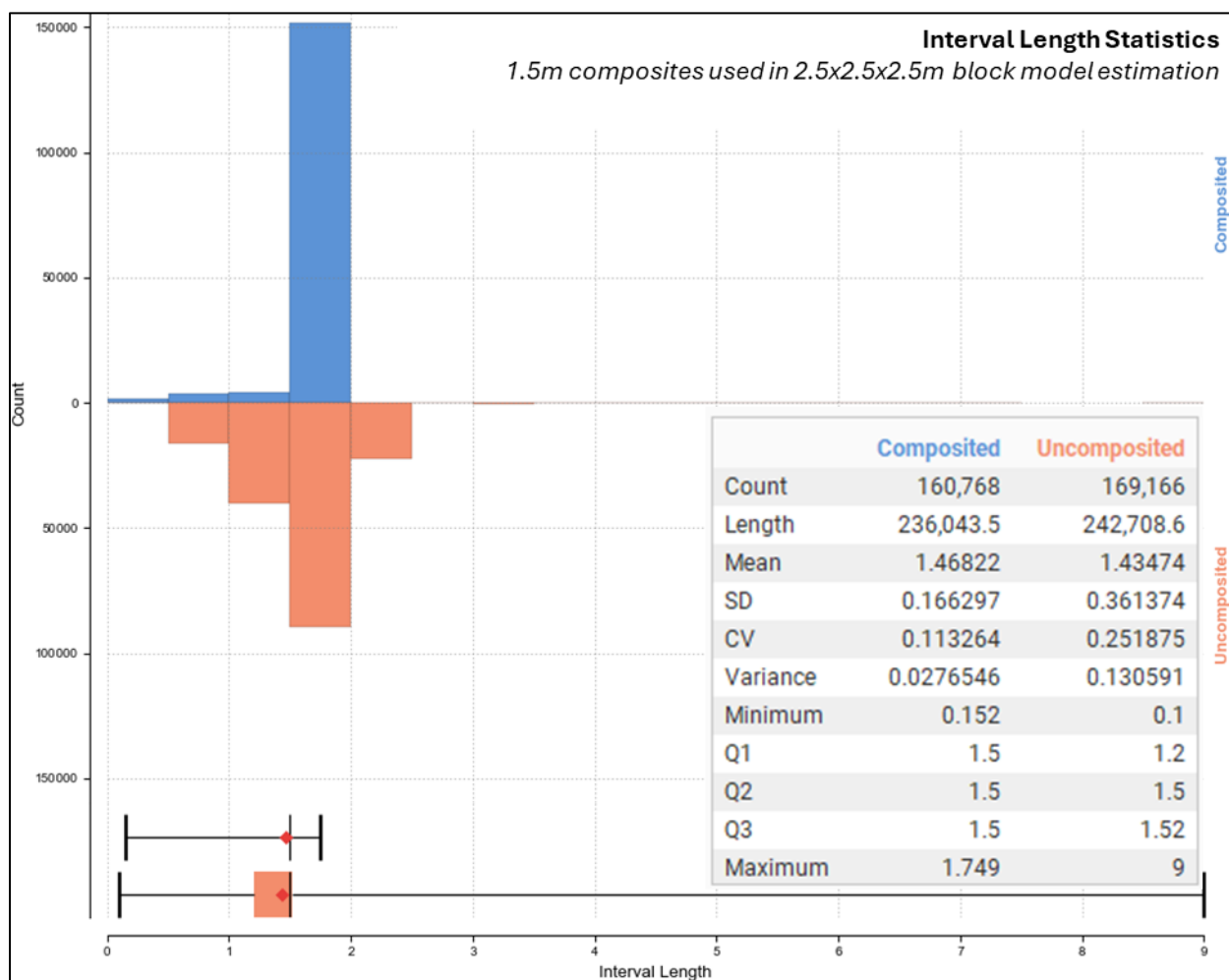


Source: Torex, 2026

14.3.3.2 2.5x2.5x2.5m Block Models

Gold and silver assays were composited using a 1.5 metre composite length for estimation in the 2.5x2.5x2.5m SMU block models, consistent with the most frequent sample interval in the assay data table. Composites were generated to respect the domain boundaries presented in Table 14-7. Figure 14-11 shows a global comparison between raw assay interval lengths and composited data used for estimation in the 2.5x2.5x2.5m block models.

Figure 14-11: Global compositing interval length statistics for 1.5m composites used for estimation in 2.5x2.5x2.5m block models



Source: Torex, 2026

14.3.4 Exploratory Data Analysis

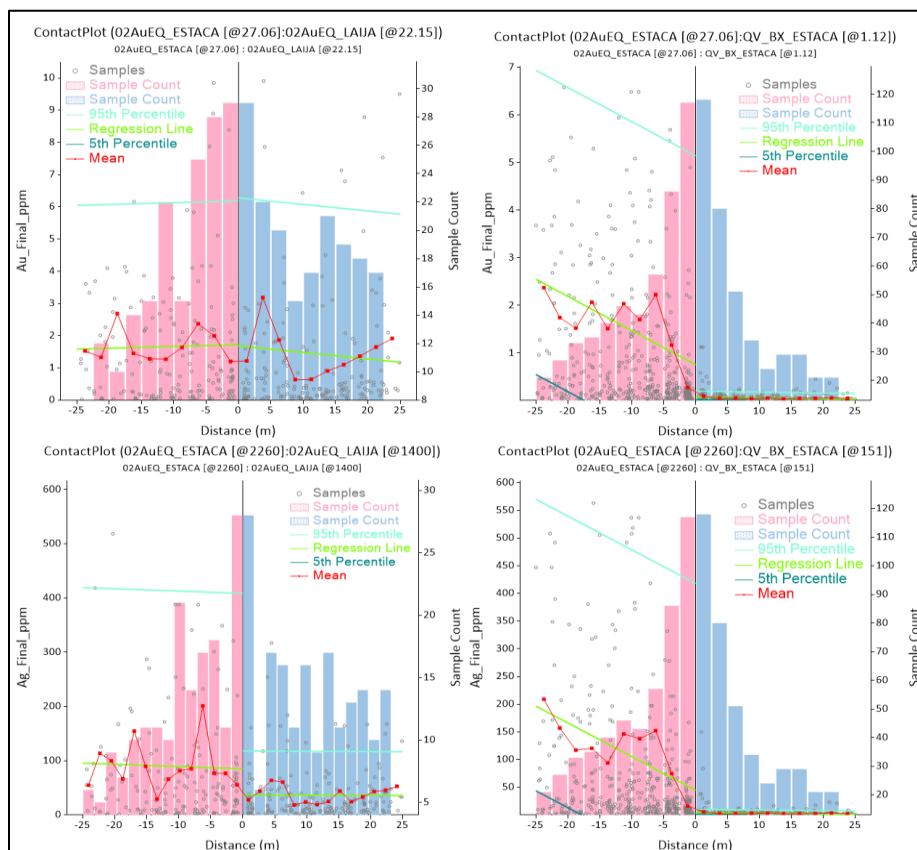
14.3.4.1 Contact Analysis

Contact profiles were generated for Au and Ag across all estimation domains to assess grade interpolation limits between adjacent domains. Methodology for both sets of block models is described below.

14.3.4.1.1 5x5x5m Block Models

Based on the analysis of 3 metre Au and Ag composited data across contacting domains in the 5x5x5m models (Table 14-6), all contacts between mineralized domains (02AuEQ prefix) and other domain groups (QV_BX and BACKGROUND prefixes) were treated as hard, and a hard boundary was similarly applied between the QV_BX and BACKGROUND domain groups. Soft boundaries were applied between contacting mineralized domains of different structural orientations (ex: 02AuEQ_Estaca and 02AuEQ_LAIJA in Figure 14-12), with a maximum soft boundary search distance set to half the first pass search dimensions of the primary domain. No boundary was applied between domains belonging to the main Zapote-Tahonitas (Z-T) structure because mineralization is continuous between the Tahonitas (TA), Zapote South (ZS), and Zapote North (ZN) areas. These domains were separated solely to isolate changes in orientation of the Z-T structure for variography. Contact analysis summaries for domains estimated in the 5x5x5m block models are shown in Table 14-8.

Figure 14-12: Contact plot examples from the 02AuEQ_ESTACA domain, showing soft boundaries (left) and hard boundaries (right) for Au (upper) and Ag (lower)



Source: Torex, 2026

Table 14-8: Contact analysis summary for domains in 5x5x5m block models for open pit optimization

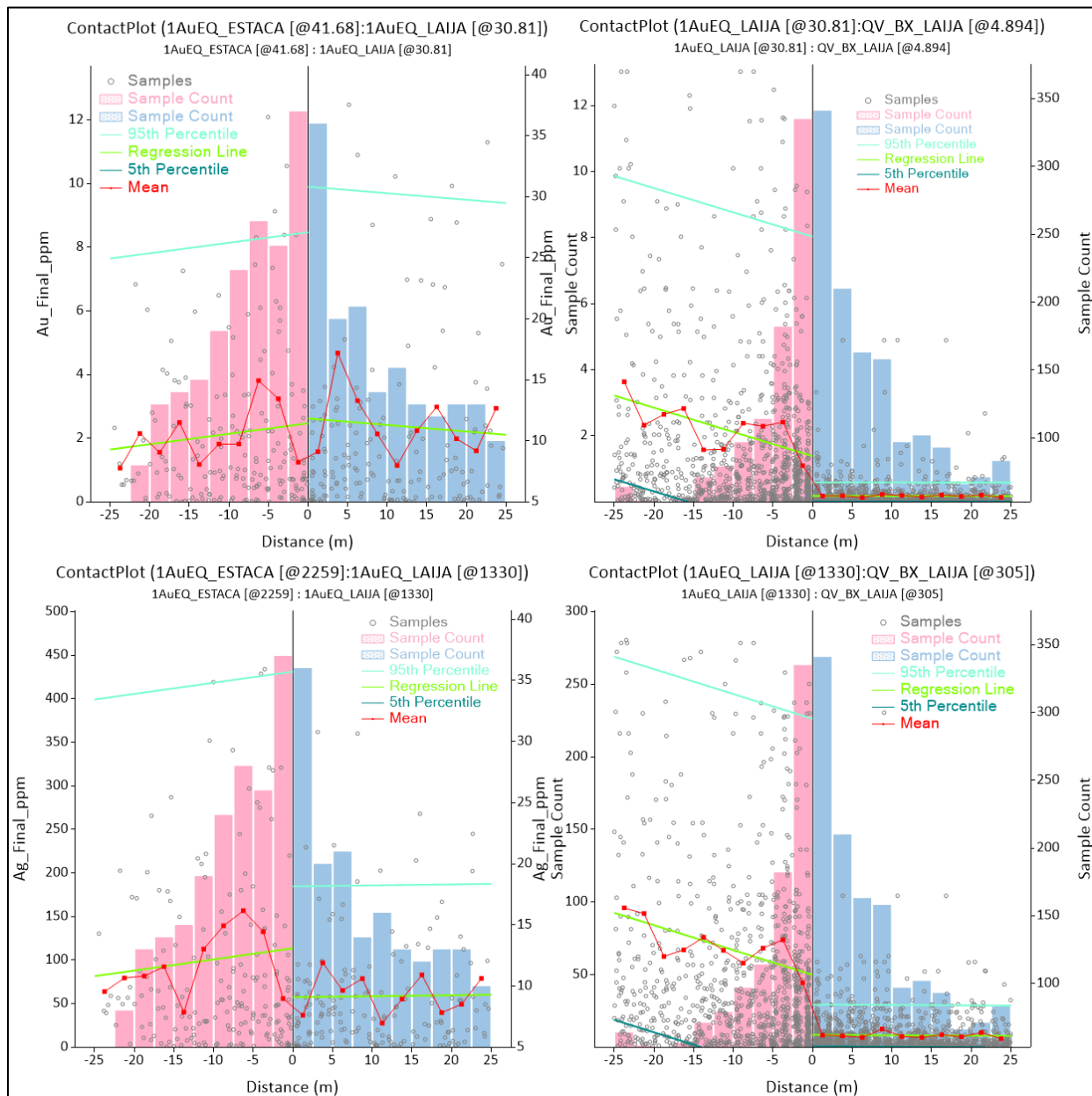
Domain Group	Au_Domain	Soft Boundary	Soft Boundary search max. (m)	Comment
QV_BX Mineralized	02AuEQ_ESTACA	02AuEQ_LAIJA, 02AuEQ_SM_SN	30	Laija and SM/SN are antithetic to Estaca but part of the same vein system. Half pass 1 search dir1 distance used.
	02AuEQ_FRE	-	N/A	Hard boundary
	02AuEQ_GUAD_NW	-	N/A	Hard boundary
	02AuEQ_GW	-	N/A	Hard boundary
	02AuEQ_LAIJA	02AuEQ_Estaca	30	Laija veins are antithetic to Estaca but part of same system. Half pass 1 search dir1 used.
	02AuEQ_MAR	-	N/A	Hard boundary
	02AuEQ_NB_SME	02AuEQ_SME_SPLAYS	30	SME footwall splay off main NB_SME structure. Half pass 1 search dir1 distance used.
	02AuEQ_SM_SN	02AuEQ_Estaca	30	SM/SN veins are antithetic to Estaca but part of same system. Half pass 1 search dir1 distance used.
	02AuEQ_SME_SPLAYS	02AuEQ_NB_SME	30	SME footwall splay off main NB_SME structure. Half pass 1 search dir1 distance used.
	02AuEQ_SMW	-	N/A	Hard boundary
	02AuEQ_TA	02AuEQ_ZS	120	Full search pass 2 search direction 1 distance used (no boundary). Very few samples available for contact analysis but TA, ZS, and ZN are part of the same structure and are only separated for estimation due to changes in the strike/dip of the ZT structure.
	02AuEQ_ZN	02AuEQ_ZS	120	
	02AuEQ_ZS	02AuEQ_TA, 02AuEQ_ZS	120	

Domain Group	Au_Domain	Soft Boundary	Soft Boundary search max. (m)	Comment
QV_BX Non-Mineralized	QV_BX_ESTACA	-	N/A	Hard boundary
	QV_BX_FRE	-	N/A	Hard boundary
	QV_BX_GUAD_NW	-	N/A	Hard boundary
	QV_BX_GW	-	N/A	Hard boundary
	QV_BX_LAIJA	-	N/A	Hard boundary
	QV_BX_LP	-	N/A	Hard boundary
	QV_BX_NB_SME	-	N/A	Hard boundary
	QV_BX_SM_SN	-	N/A	Hard boundary
	QV_BX_SME_SPLAYS	-	N/A	Hard boundary
	QV_BX_ZTM	-	N/A	Hard boundary
Background	BACKGROUND_GUAD_LP	-	N/A	Hard boundary
	BACKGROUND_ZTM_NB_SM	-	N/A	Hard boundary

14.3.4.1.2 2.5x2.5x2.5m Block Models

Based on the analysis of 1.5 metre Au and Ag composited data across contacting domains in the 2.5x2.5x2.5m models (Table 14-7), all contacts between mineralized domains (1AuEQ prefix) and other domain groups (QV_BX and BACKGROUND prefixes) were treated as hard, and a hard boundary was similarly applied between the QV_BX and BACKGROUND domain groups. Soft boundaries were applied between contacting mineralized domains of different structural orientations (ex: 1AuEQ_Estaca and 1AuEQ_LAIJA in Figure 14-13), with a maximum soft boundary search distance set to half the first pass search dimensions of the primary domain. No boundary was applied between domains belonging to the main Zapote-Tahonitas (Z-T) structure because mineralization is continuous between the Tahonitas (TA), Zapote South (ZS), and Zapote North (ZN) areas. These domains were separated solely to isolate changes in orientation of the Z-T structure for variography. Contact analysis summaries for domains estimated in the 2.5x2.5x2.5m block models are shown in Table 14-9.

Figure 14-13: Contact plot examples from the 1AuEQ_LAIJA domain, showing soft boundaries (left) and hard boundaries (right) for Au (upper) and Ag (lower)



Source: Torex, 2026

Table 14-9: Contact analysis summary for domains in 2.5x2.5x2.5m block models for underground optimization

Domain Group	Au_Domain	Soft Boundary	Soft Boundary search max. (m)	Comment
QV_BX Mineralized	1AuEQ_ESTACA	1AuEQ_LAIJA, 1AuEQ_SM_SN	30	Laija and SM/SN are antithetic to Estaca but part of the same vein system. Half pass 1 search dir1 distance used.
	1AuEQ_FRE	-	N/A	Hard boundary
	1AuEQ_GUAD_NW	-	N/A	Hard boundary
	1AuEQ_GW	-	N/A	Hard boundary
	1AuEQ_LAIJA	1AuEQ_Estaca	30	Laija veins are antithetic to Estaca but part of same system. Half pass 1 search dir1 distance used.
	1AuEQ_MAR	-	N/A	Hard boundary
	1AuEQ_NB_SME	1AuEQ_SME_SPLAYS	30	SME footwall splays off main NB_SME structure. Half pass 1 search dir1 distance used.
	1AuEQ_SM_SN	1AuEQ_Estaca	30	SM/SN veins are antithetic to Estaca but part of same system. Half pass 1 search dir1 distance used.
	1AuEQ_SME_SPLAYS	1AuEQ_NB_SME	30	SME footwall splays off main NB_SME structure. Half pass 1 search dir1 distance used.
	1AuEQ_SMW	-	N/A	Hard boundary
	1AuEQ_TA	1AuEQ_ZS	120	Full search pass 2 search direction 1 distance used (no boundary). Very few samples available for contact analysis but TA, ZS, and ZN are part of the same structure and are only separated for estimation due to changes in the strike/dip of the ZT structure.
	1AuEQ_ZN	1AuEQ_ZS	120	
	1AuEQ_ZS	1AuEQ_TA, 1AuEQ_ZS	120	
QV_BX Non-Mineralized	QV_BX_ESTACA	-	N/A	Hard boundary
	QV_BX_FRE	-	N/A	Hard boundary
	QV_BX_GUAD_NW	-	N/A	Hard boundary
	QV_BX_GW	-	N/A	Hard boundary
	QV_BX_LAIJA	-	N/A	Hard boundary
	QV_BX_LP	-	N/A	Hard boundary
	QV_BX_NB_SME	-	N/A	Hard boundary

Domain Group	Au_Domain	Soft Boundary	Soft Boundary search max. (m)	Comment
Background	QV_BX_SM_SN	-	N/A	Hard boundary
	QV_BX_SME_SPLAYS	-	N/A	Hard boundary
	QV_BX_ZTM	-	N/A	Hard boundary
	BACKGROUND_GUAD_LP	-	N/A	Hard boundary
	BACKGROUND_ZTM_NB_SM	-	N/A	Hard boundary

14.3.4.2 Outlier Management and Topcut Strategy

Capping analysis was completed for Au and Ag across all estimation domains in the 5x5x5m and 2.5x2.5x2.5m block models, using log-probability plots, histograms, mean-variance plots, cumulative metal plots, and disintegration analysis. Specific methodology for each block model set is described below.

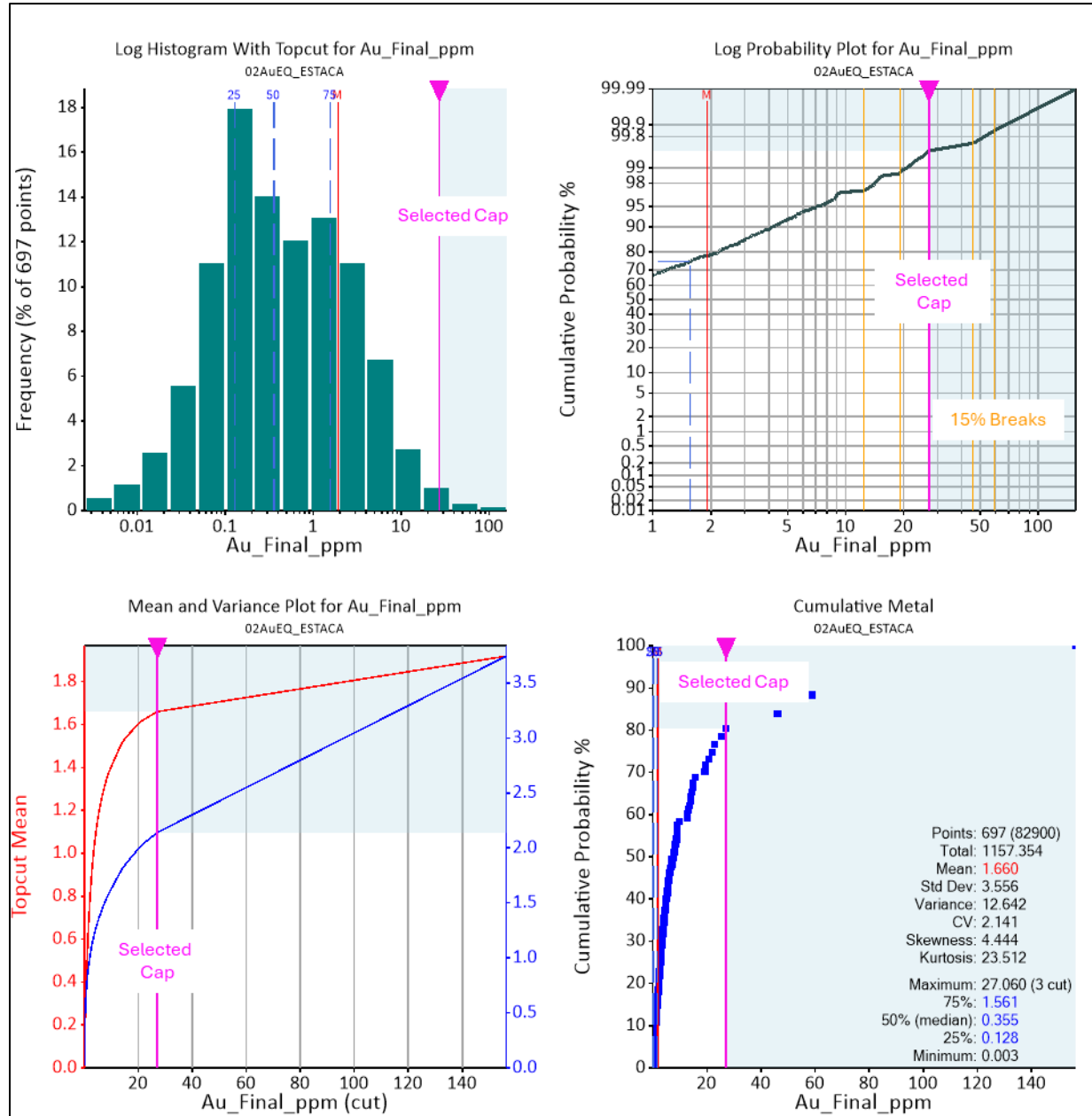
14.3.4.2.1 5x5x5m Block Models

Capping analysis was completed on 3 metre composited data for each estimation domain in the 5x5x5m block models, using the methods described above (Figure 14-14 and Figure 14-15). Capped samples were then evaluated in 3D within each domain to ensure that the samples were not clustered and represented true outliers. Inverse Distance Cubed (ID3) estimates were also completed within each mineralized domain (02AuEQ prefix), using both the capped and uncapped datasets to assess the impact to average grade and contained metal (Table 14-10 and Table 14-11).

14.3.4.2.2 2.5x2.5x2.5m Block Models

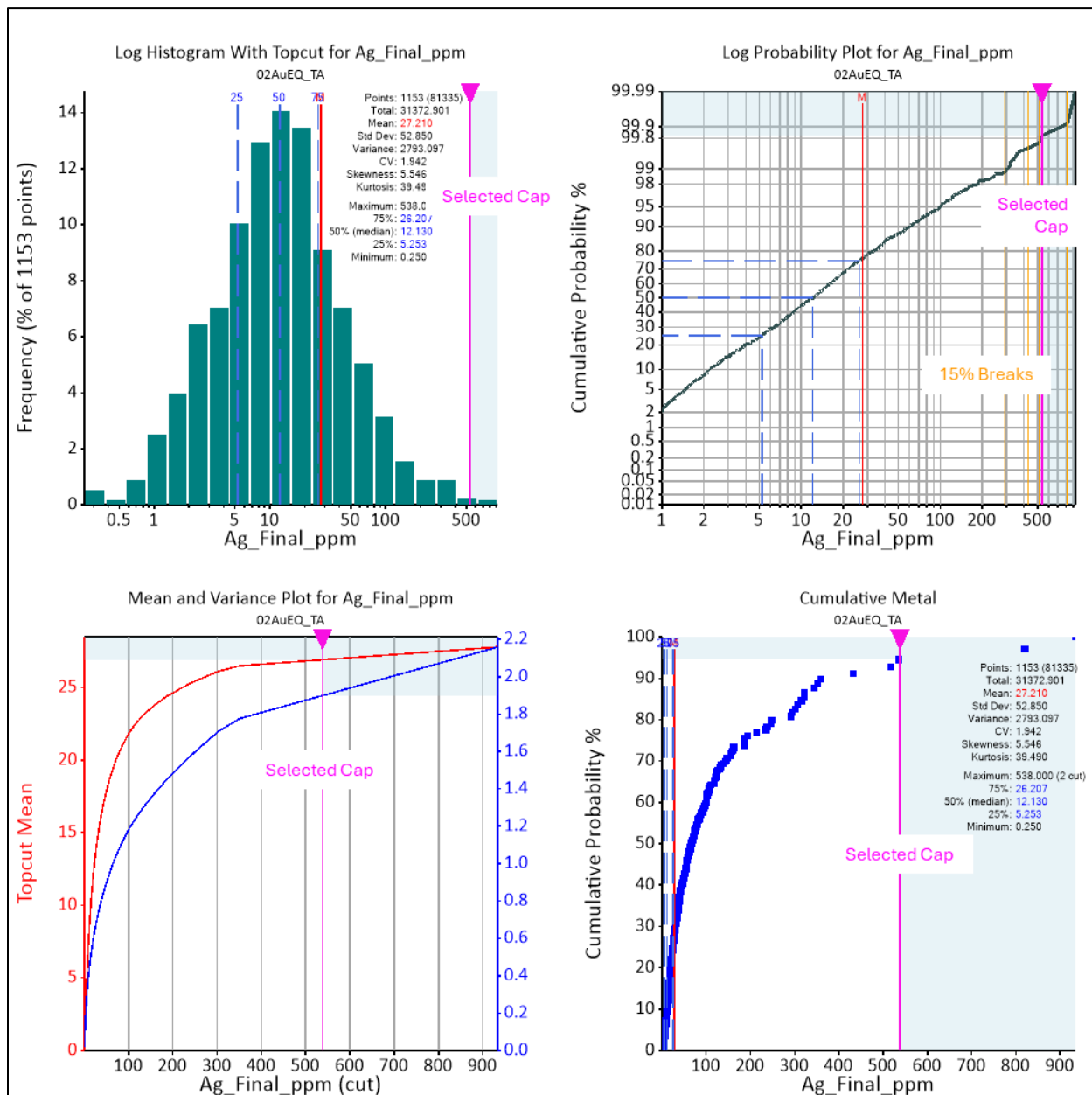
Capping analysis was completed on 1.5 metre composited data for each estimation domain in the 2.5x2.5x2.5m block models, using the same methods as for the 5x5x5m models (Figure 14-16 and Figure 14-17). Capped samples were then evaluated in 3D within each domain to ensure that the samples were not clustered and represented true outliers. Inverse Distance Cubed (ID3) estimates were also completed within each mineralized domain (1AuEQ prefix), using both the capped and uncapped datasets to assess the impact to average grade and contained metal (Table 14-12 and Table 14-13).

Figure 14-14: Gold topcut analysis for the 02AuEQ_ESTACA estimation domain



Source: Torex, 2026

Figure 14-15: Silver topcut analysis for the 02AuEQ_TA estimation domain



Source: Torex, 2026

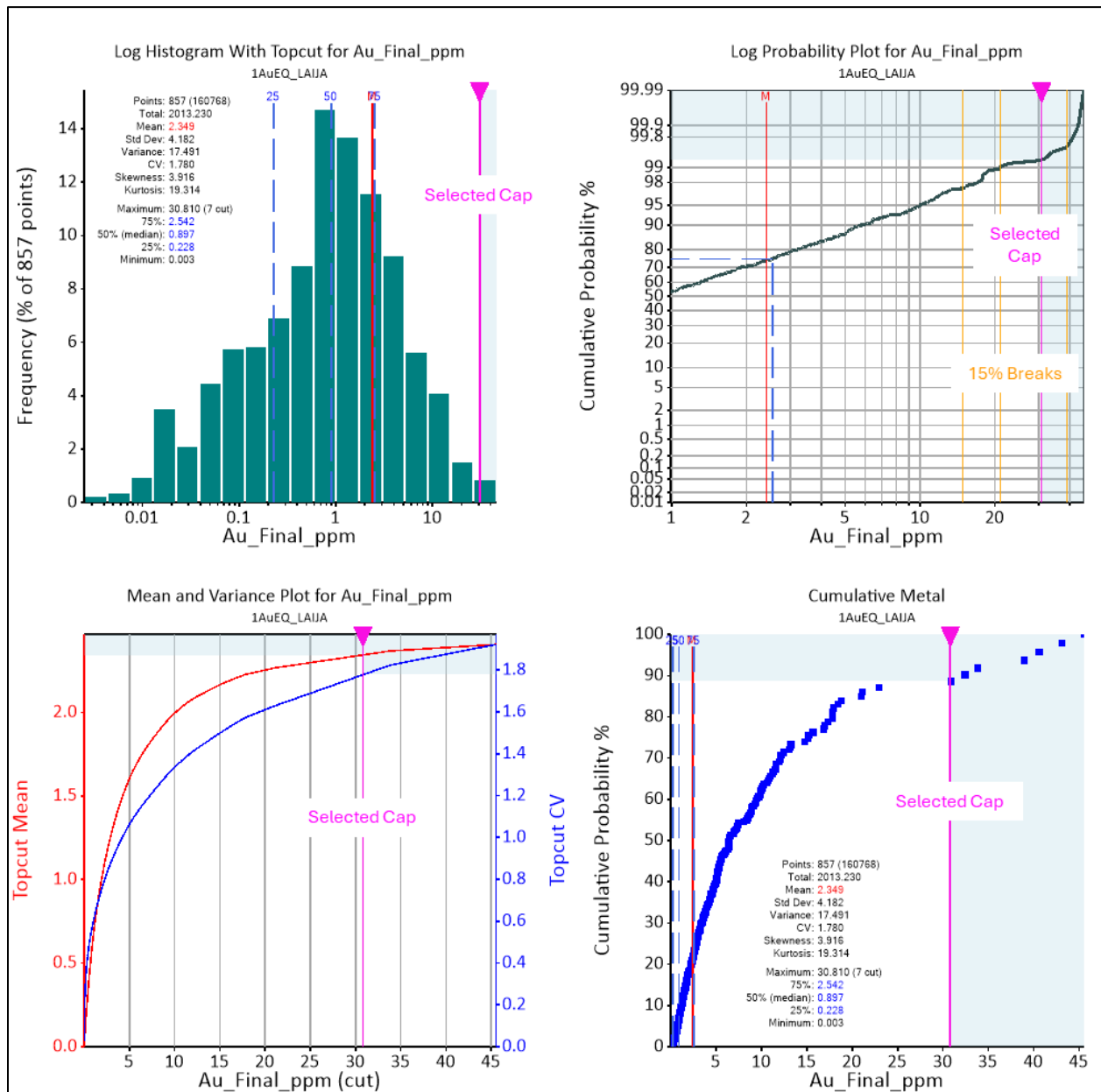
Table 14-10: Gold topcut statistics for mineralized domains used for estimation in 5x5x5m block models

Domain	3m Composite Statistics: Au (g/t)									Au (g/t) ID3 Estimate Mean Comparison (Block Model)			
	Max uncapped (g/t)	Cap (g/t)	Percentile	Total samples	Capped samples	Mean Uncapped	Mean Capped	CV Uncapped	CV Capped	Capped (g/t)	Uncapped (g/t)	Lost (%)	Total Domain Tonnes
02AuEQ_ESTACA	156.29	27.06	99.6%	697	3	1.92	1.66	3.75	2.14	1.712	1.949	-2.2%	5,911,484
02AuEQ_FRE	6.67	5.29	99.8%	493	1	0.41	0.40	1.48	1.42	0.416	0.417	-0.2%	18,109,021
02AuEQ_GUAD_NW	12.97	10.76	98.8%	86	1	0.59	0.56	3.14	3.00	0.28	0.28	0.0%	907,648
02AuEQ_GW	10.69	10.69	100%	435	0	0.53	0.53	1.62	1.62	0.38	0.38	0.0%	6,658,637
02AuEQ_LAIJA	44.31	22.15	99.9%	838	1	1.05	1.04	2.33	2.18	1.17	1.16	-0.8%	4,747,290
02AuEQ_LP	22.81	5.00	99.7%	317	1	0.42	0.42	0.68	0.68	0.38	0.39	-0.5%	9,478,731
02AuEQ_MAR	16.22	14.63	99.5%	205	1	0.64	0.63	2.72	2.65	0.61	0.62	-1.5%	5,419,960
02AuEQ_NB_SME	32.61	15.65	99.9%	1636	2	0.44	0.44	2.22	2.10	0.45	0.45	-0.4%	32,643,711
02AuEQ_SM_SN	14.25	7.41	98.4%	188	3	0.61	0.57	2.43	2.06	0.81	0.85	-4.2%	2,805,628
02AuEQ_SME_SPLAYS	7.21	5.00	99.2%	240	2	0.41	0.40	2.19	2.09	0.40	0.39	-3.8%	3,707,223
02AuEQ_SMW	44.95	20.63	97.2%	147	4	1.19	0.99	4.08	3.43	0.67	0.76	-12.6	2,309,141
02AuEQ_TA	19.85	14.57	99.8%	1153	2	0.43	0.42	2.34	2.27	0.46	0.46	-1.3%	34,943,241
02AuEQ_ZN	57.58	29.06	99.9%	2782	2	0.52	0.51	2.87	2.38	0.52	0.53	-1.7%	34,948,569
02AuEQ_ZS	17.41	15.77	99.9%	1695	1	0.55	0.55	2.87	2.38	0.61	0.61	0.0%	23,894,733
02AuEQ_ZW	4.42	4.42	100.0%	36	0	0.40	0.40	1.76	1.76	0.37	0.37	0.0%	3,367,568

Table 14-11: Silver topcut statistics for mineralized domains used for estimation in 5x5x5m block models

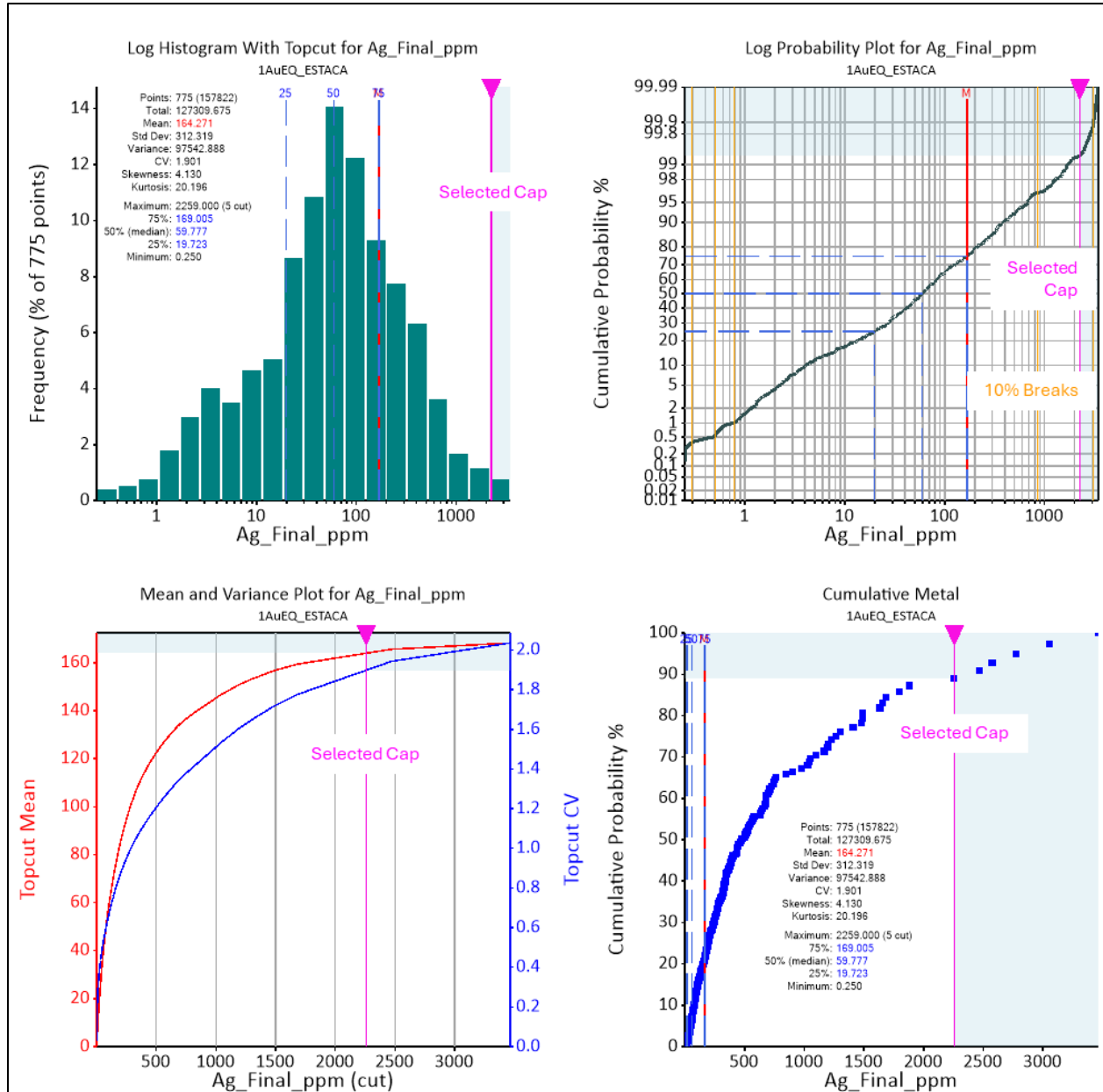
Domain	3m Composite Statistics: Ag (g/t)									Ag (g/t) ID3 Estimate Mean (Block Model) Comparison			
	Max Uncapped (g/t)	Cap (g/t)	Percentile	Total samples	Capped samples	Mean Uncapped	Mean Capped	CV Uncapped	CV Capped	Capped (g/t)	Uncapped (g/t)	Lost (%)	Total Domain Tonnes
02AuEQ_ESTACA	3017.00	2260.00	99.9%	697	1	106.74	105.65	2.37	2.29	109.50	110.71	-1.1%	5,911,484
02AuEQ_FRE	168.5	67.40	99.6%	493	2	7.16	6.92	1.70	1.43	7.47	7.70	-2.9%	18,109,021
02AuEQ_GUAD_NW	304.95	261.80	98.90%	87	1	21.38	20.88	2.27	2.19	14.36	14.39	-0.2%	907,648
02AuEQ_GW	405.70	405.70	100.0%	435	0	19.80	19.80	1.48	1.48	19.07	19.07	0.0%	6,658,637
02AuEQ_LAIJA	2411.10	1400.00	99.8%	838	2	47.57	46.16	2.83	2.47	46.9	47.34	-0.9%	4,747,290
02AuEQ_LP	892.00	615.00	99.4%	317	2	19.89	18.71	3.36	2.83	15.34	15.63	-1.8%	9,478,731
02AuEQ_MAR	142.60	142.60	100.0%	205	0	10.39	10.39	1.08	1.08	10.47	10.47	0.0%	5,419,960
02AuEQ_NB_SME	673.37	478.00	99.9%	1609	1	22.76	22.64	1.86	1.80	20.36	20.40	-0.2%	32,643,711
02AuEQ_SM_SN	2150.00	388.50	98.4%	188	3	45.88	34.65	3.84	2.29	50.02	60.29	-17.0	2,805,628
02AuEQ_SME_SPLAYS	584.08	523.00	99.6%	240	1	33.26	33.00	2.11	2.07	29.17	29.55	-1.4%	3,707,223
02AuEQ_SMW	431.90	376.00	99.3%	147	1	21.19	20.81	2.57	2.49	15.10	14.87	-1.5%	2,309,141
02AuEQ_TA	934.2	538.00	99.8%	1153	2	27.80	27.21	2.57	2.49	24.74	25.66	-3.6%	34,943,241
02AuEQ_ZN	336.2	332.9	99.9%	2465	1	13.28	13.28	0.92	0.92	11.64	11.67	-0.3%	34,948,569
02AuEQ_ZS	281.20	281.20	100.0%	1489	0	20.65	20.65	1.12	1.12	18.71	18.71	0.0%	23,894,733
02AuEQ_ZW	41.88	41.88	100.0%	36	0	6.23	6.23	1.22	1.22	5.16	5.16	0.0%	3,364,568

Figure 14-16: Gold topcut analysis for the 1AuEQ_LAIJA estimation domain



Source: Torex, 2026

Figure 14-17: Silver topcut analysis for the 1AuEQ_ESTACA estimation domain



Source: Torex, 2026

Table 14-12: Gold topcut statistics for mineralized domains used for estimation in 2.5x2.5x2.5m block models

Domain	1.5m Composite Statistics: Au (g/t)									Au (g/t) ID3 Estimate Mean Comparison (Block Model)			
	Max Uncapped (g/t)	Cap (g/t)	Percentile	Total samples	Capped samples	Mean Uncapped	Mean Capped	CV Uncapped	CV Capped	Capped (g/t)	Uncapped (g/t)	Lost (%)	Total Domain Tonnes
1AuEQ_ESTACA	156.29	41.68	99.6%	775	3	2.94	2.67	2.88	1.89	2.66	2.96	-10.2%	3,971,994
1AuEQ_GW	29.48	15.18	99.1%	214	2	1.44	1.37	1.74	1.36	1.24	1.29	-3.9%	678,317
1AuEQ_LAIJA	45.53	30.81	99.3%	857	7	2.41	2.35	1.92	1.78	2.15	2.24	-4.0%	3,099,890
1AuEQ_GUAD_NW	2.21	2.21	100.0%	6	0	0.43	0.43	1.86	1.86	0.38	0.38	0.0%	91,790
1AuEQ_MAR	29.41	20.28	98.7%	78	1	2.28	2.17	1.99	1.81	1.91	2.02	-5.4%	1,146,402
1AuEQ_NB_SME	41.81	28.16	99.9%	878	1	1.41	1.40	1.96	1.85	1.21	1.21	-0.1%	6,684,983
1AuEQ_LP	10.06	10.06	100%	139	0	0.90	0.90	1.72	1.72	0.94	0.94	-0.3%	1,957,599
1AuEQ_FRE	7.47	5.79	99.9	109	1	1.20	1.17	1.26	1.19	1.17	1.23	-4.8%	1,731,023
1AuEQ_SM_SN	14.56	11.56	94.1%	51	3	2.16	1.99	1.70	1.58	1.75	1.80	-2.9%	780,593
1AuEQ_SME_SPLAYS	13.03	8.42	99.0%	96	1	1.39	1.34	1.45	1.32	1.54	1.64	-6.2%	363,506
1AuEQ_SMW	69.57	31.56	97.1%	34	1	8.15	7.03	1.73	1.44	8.18	9.42	-13.1%	113,437
1AuEQ_TA	38.18	18.61	99.7%	577	2	1.24	1.20	2.14	1.88	1.00	1.01	-1.3%	8,223,171
1AuEQ_ZN	77.56	37.03	99.8%	1333	3	2.19	2.16	1.90	1.71	1.80	1.82	-1.1%	5,466,571
1AuEQ_ZS	30.93	21.33	99.6%	1080	4	2.04	2.03	1.49	1.46	1.74	1.75	-0.3%	5,835,479
1AuEQ_ZW	6.68	6.68	100.0%	5	0	2.13	2.13	1.11	1.11	1.68	1.68	0.0%	54,534

Table 14-13: Silver topcut statistics for mineralized domains used for estimation in 2.5x2.5x2.5m block models

Domain	1.5m Composite Statistics: Ag (g/t)									Ag (g/t) ID3 Estimate Mean Comparison (Block Model)			
	Max Uncapped (g/t)	Cap (g/t)	Percentile	Total samples	Capped samples	Mean Uncapped	Mean Capped	CV Uncapped	CV Capped	Capped (g/t)	Uncapped (g/t)	Lost (%)	Total Domain Tonnes
1AuEQ_ESTACA	3464.49	2259.00	99.3%	775	5	168.20	164.27	2.04	1.90	164.16	167.04	-1.7%	3,971,994
1AuEQ_GW	210.52	169.20	99.9%	109	1	23.63	23.25	1.23	1.16	21.09	21.22	-0.6%	678,317
1AuEQ_LAIJA	3121.28	1330.00	99.3%	857	6	80.35	75.53	2.55	2.02	83.69	87.32	-4.2%	3,099,890
1AuEQ_GUAD_NW	37.20	37.20	100.0%	6	0	37.20	37.20	1.00	1.00	20.83	20.83	0.0%	91,790
1AuEQ_MAR	295.88	267.50	98.7%	78	1	21.12	20.76	1.55	1.43	21.59	22.04	-2.0%	1,957,599
1AuEQ_NB_SME	1210.96	575.00	99.7%	866	3	57.23	56.46	1.44	1.32	52.58	53.46	-1.6%	1,731,023
1AuEQ_LP	1180.32	868.00	99.3%	139	1	43.16	40.91	2.53	2.12	30.59	30.76	-0.6%	6,684,983
1AuEQ_FRE	210.52	169.20	99.9%	109	1	23.63	23.25	1.23	1.16	21.09	21.22	-0.6%	678,317
1AuEQ_SM_SN	2150.00	399.00	97.7%	51	5	190.47	117.47	1.97	1.27	78.53	104.24	-24.7%	780,593
1AuEQ_SME_SPLAYS	1055.71	561.00	99.0%	96	1	106.90	101.74	1.42	1.24	112.91	123.34	-8.5%	363,506
1AuEQ_SMW	571.21	386.00	97.2%	34	2	95.10	88.86	1.32	1.18	97.61	105.63	-7.6%	113,437
1AuEQ_TA	116.32	730.00	99.3%	577	3	74.78	73.38	1.54	1.42	68.31	70.02	-2.4%	8,223,171
1AuEQ_ZN	175.99	174.20	99.9%	1206	1	23.11	23.11	0.67	0.67	23.80	23.80	0.0%	5,466,571
1AuEQ_ZS	375.41	375.41	100.0%	910	0	40.15	40.15	0.88	0.88	42.52	42.52	0.0%	5,835,479
1AuEQ_ZW	67.00	67.00	100.0%	5	0	21.76	21.76	1.05	1.05	17.18	17.18	0.0%	54,534

14.3.5 Variography

Variography was completed for Au and Ag within mineralized estimation domains using Snowden Supervisor v.8.15 and was completed independently for domains estimated in the 5x5x5m and 2.5x2.5x2.5m block models. Variogram modeling was completed on normal scores-transformed data and variograms were modeled using as few structures as possible, with a nugget obtained from down hole variograms and generally 2 spherical structures used. The back-transformation of normal scores variograms to original units was then completed for Au and Ag variograms in each domain using 90 Hermite polynomials, and the orientation of the variograms were checked against the mineralization orientation for each domain in 3D prior to use in estimation. Search orientations determined from variography were used in both the ordinary kriging (OK) estimates and in the final inverse distance cubed (ID3) estimates used for resource reporting.

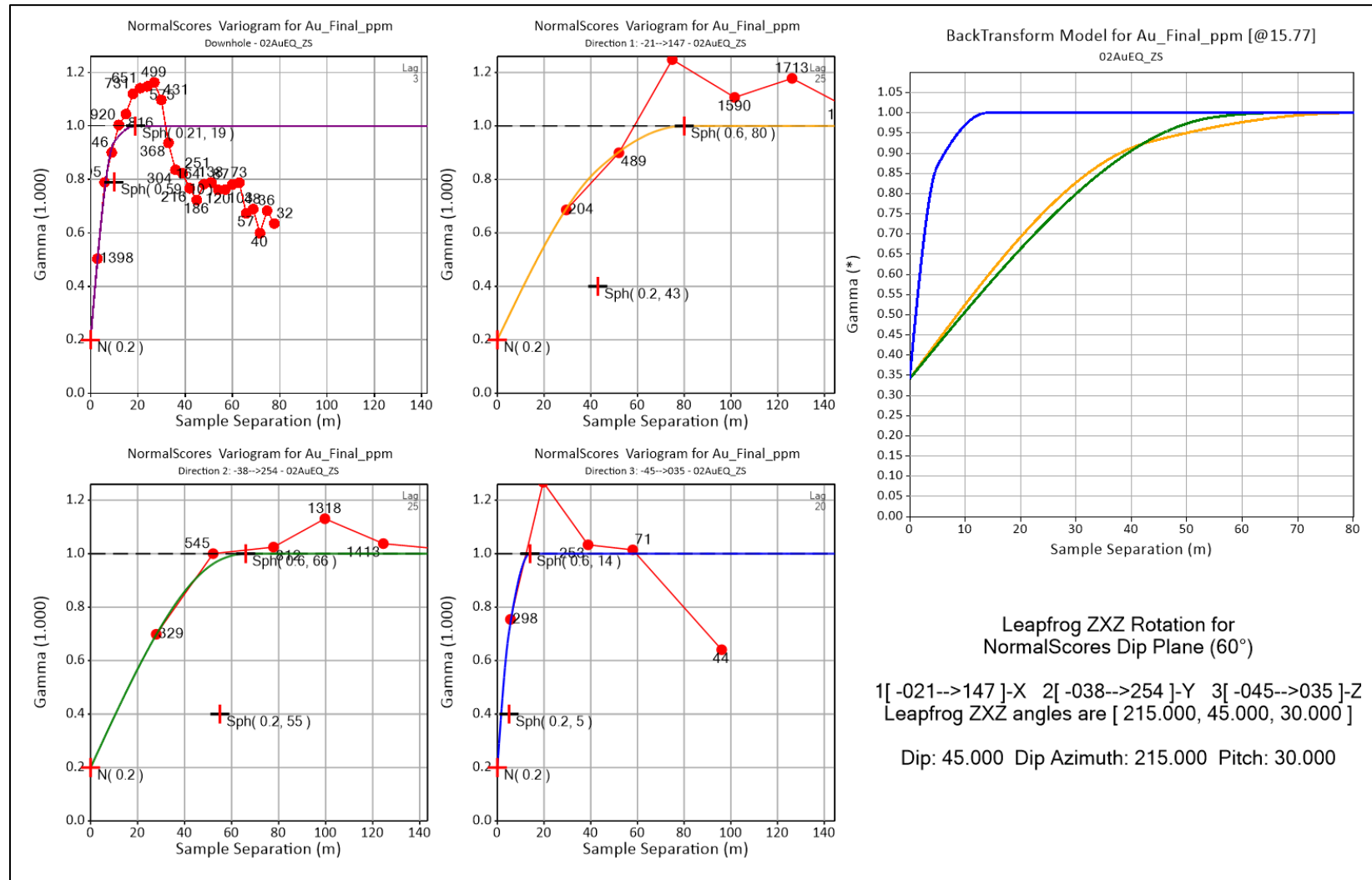
14.3.5.1 5x5x5m Block Models

Experimental variograms were calculated from capped 3 metre composites for mineralized domains (02AuEQ prefix) estimated in the 5x5x5m block models. Examples from several domains are shown in Figure 14-18 and Figure 14-19, with results for all domains presented in Table 14-12 through Table 14-13.

14.3.5.2 2.5x2.5x2.5m Block Models

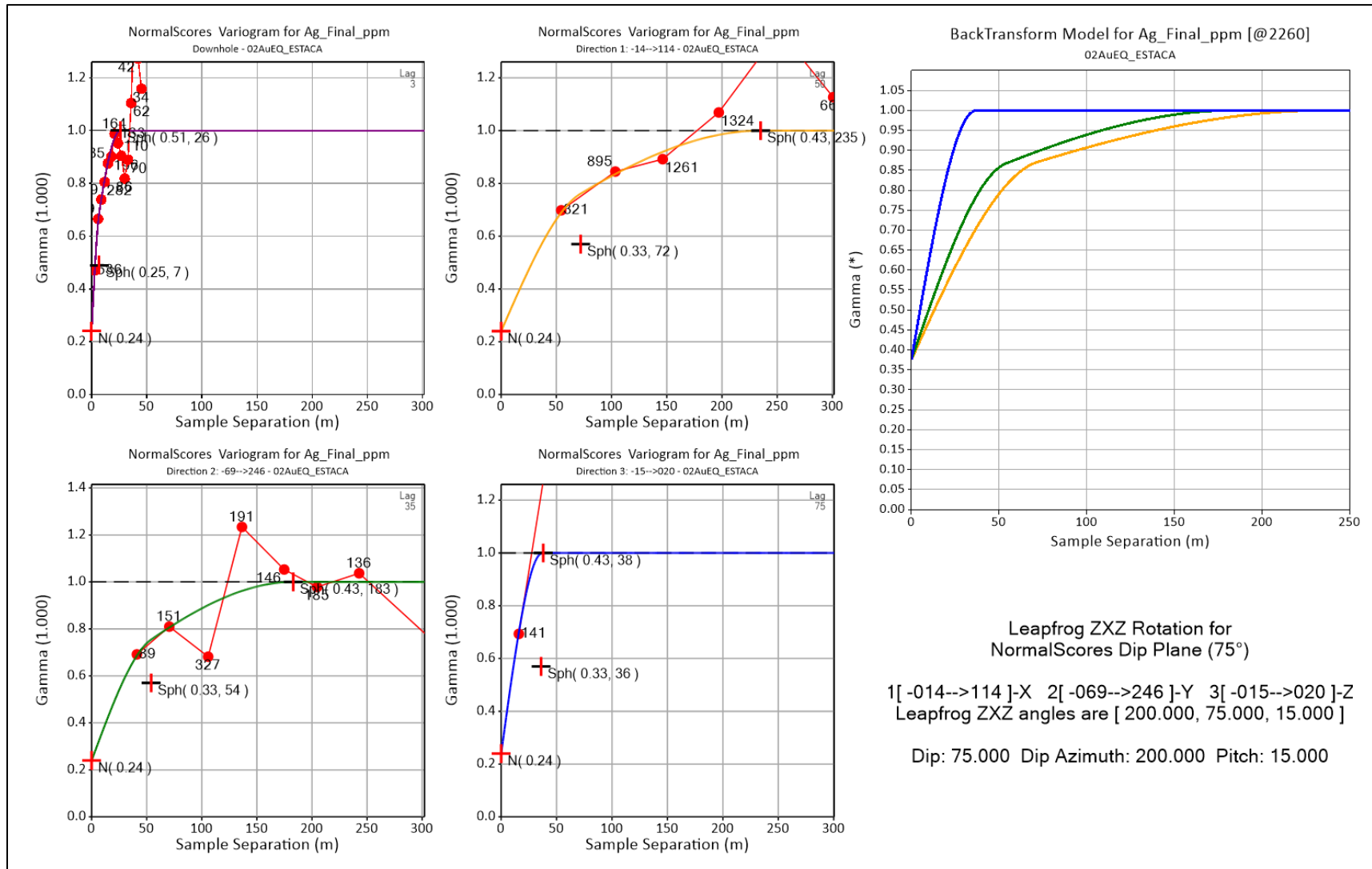
Experimental variograms were calculated from capped 1.5 metre composites for Au and Ag in mineralized domains (1AuEQ prefix) estimated in the 2.5x2.5x2.5m block models. Examples from several domains are shown in Figure 14-20 through Figure 14-21, with results for all domains presented in Table 14-13 through Table 14-14.

Figure 14-18: Normal scores variography and back transform model for gold estimation in the 02AuEQ_ZS domain



Source: Torex, 2026

Figure 14-19: Normal scores variography and backtransform model for silver estimation in the 02AuEQ_ESTACA domain



Source: Torex, 2026

Table 14-14: Gold variogram parameters for mineralized domains used for estimation in 2.5x2.5x2.5m block models

Domain	Rotation – Snowden Supervisor				Structure 1					Structure 2				
	Horizontal	Across Strike	Dip Plane	Nugget	Type	Normalized Sill	Major (m)	Semi-Major (m)	Minor (m)	Type	Normalized Sill	Major (m)	Semi-Major (m)	Minor (m)
1AuEQ_ESTACA	110	195	75	0.28	Spherical	0.464	52	60	27	Spherical	0.257	111	65	28
1AuEQ_FRE	135	215	70	0.24	Spherical	0.76	183	183	10	-	-	-	-	-
1AuEQ_GW	100	30	100	0.265	Spherical	0.735	79	52	17	-	-	-	-	-
1AuEQ_LAIJA	300	200	310	0.295	Spherical	0.534	34	42	19	Spherical	0.171	59	43	20
1AuEQ_GUAD_NW	325	180	90	0.3	Spherical	0.7	40	40	10	-	-	-	-	-
1AuEQ_LP	310	190	150	0.349	Spherical	0.651	140	140	15	-	-	-	-	-
1AuEQ_MAR	310	135	315	0.326	Spherical	0.674	80	80	15	-	-	-	-	-
1AuEQ_NB_SME	145	210	50	0.181	Spherical	0.472	78	78	2	Spherical	0.347	88	85	9
1AuEQ_SM_SN	305	225	270	0.3	Spherical	0.7	40	40	10	-	-	-	-	-
1AuEQ_SME_SPLAYS	285	145	105	0.236	Spherical	0.764	50	50	5	-	-	-	-	-
1AuEQ_SMW	100	210	70	0.3	Spherical	0.7	40	40	10	-	-	-	-	-
1AuEQ_TA	150	220	55	0.267	Spherical	0.452	61	61	6	Spherical	0.281	81	102	9
1AuEQ_ZN	165	220	55	0.254	Spherical	0.468	43	25	3	Spherical	0.277	47	29	7
1AuEQ_ZS	125	225	60	0.198	Spherical	0.533	34	18	4	Spherical	0.269	80	40	10
1AuEQ_ZW	120	210	90	0.411	Spherical	0.589	40	40	10	-	-	-	-	-

Notes: Nugget and normalized sill values from back-transformed normal scores variograms.

Table 14-15: Silver variogram parameters for mineralized domains used for estimation in 2.5x2.5x2.5m block models

Domain	Rotation – Snowden Supervisor			Nugget	Structure 1					Structure 2				
	Horizontal	Across Strike	Dip Plane		Type	Normalized Sill	Major (m)	Semi-Major (m)	Minor (m)	Type	Normalized Sill	Major (m)	Semi-Major (m)	Minor (m)
1AuEQ_ESTACA	110	195	75	0.23	Spherical	0.545	54	54	12	Spherical	0.225	194	66	14
1AuEQ_FRE	135	215	70	0.246	Spherical	0.754	170	170	11	-	-	-	-	-
1AuEQ_GW	100	30	100	0.216	Spherical	0.409	46	63	7	Spherical	0.375	96	65	25
1AuEQ_LAIJA	300	200	310	0.369	Spherical	0.551	25	35	25	Spherical	0.0797	61	60	26
1AuEQ_GUAD_NW	325	180	90	0.3	Spherical	0.7	40	40	10	-	-	-	-	-
1AuEQ_LP	310	190	150	0.276	Spherical	0.724	97	97	15	-	-	-	-	-
1AuEQ_MAR	310	135	315	0.359	Spherical	0.641	127	127	20	-	-	-	-	-
1AuEQ_NB_SME	145	210	50	0.197	Spherical	0.423	72	28	8	Spherical	0.38	80	77	20
1AuEQ_SM_SN	305	225	270	0.3	Spherical	0.7	40	40	10	-	-	-	-	-
1AuEQ_SME_SPLAYS	285	145	105	0.139	Spherical	0.861	72	55	10	-	-	-	-	-
1AuEQ_SMW	100	210	70	0.19	Spherical	0.81	116	116	15	-	-	-	-	-
1AuEQ_TA	150	220	55	0.181	Spherical	0.381	71	110	3	Spherical	0.438	141	123	11
1AuEQ_ZN	165	220	55	0.142	Spherical	0.341	22	41	5	Spherical	0.518	59	53	17
1AuEQ_ZS	125	225	60	0.125	Spherical	0.373	33	26	4	Spherical	0.501	119	131	9
1AuEQ_ZW	120	210	90	0.351	Spherical	0.649	40	40	20	-	-	-	-	-

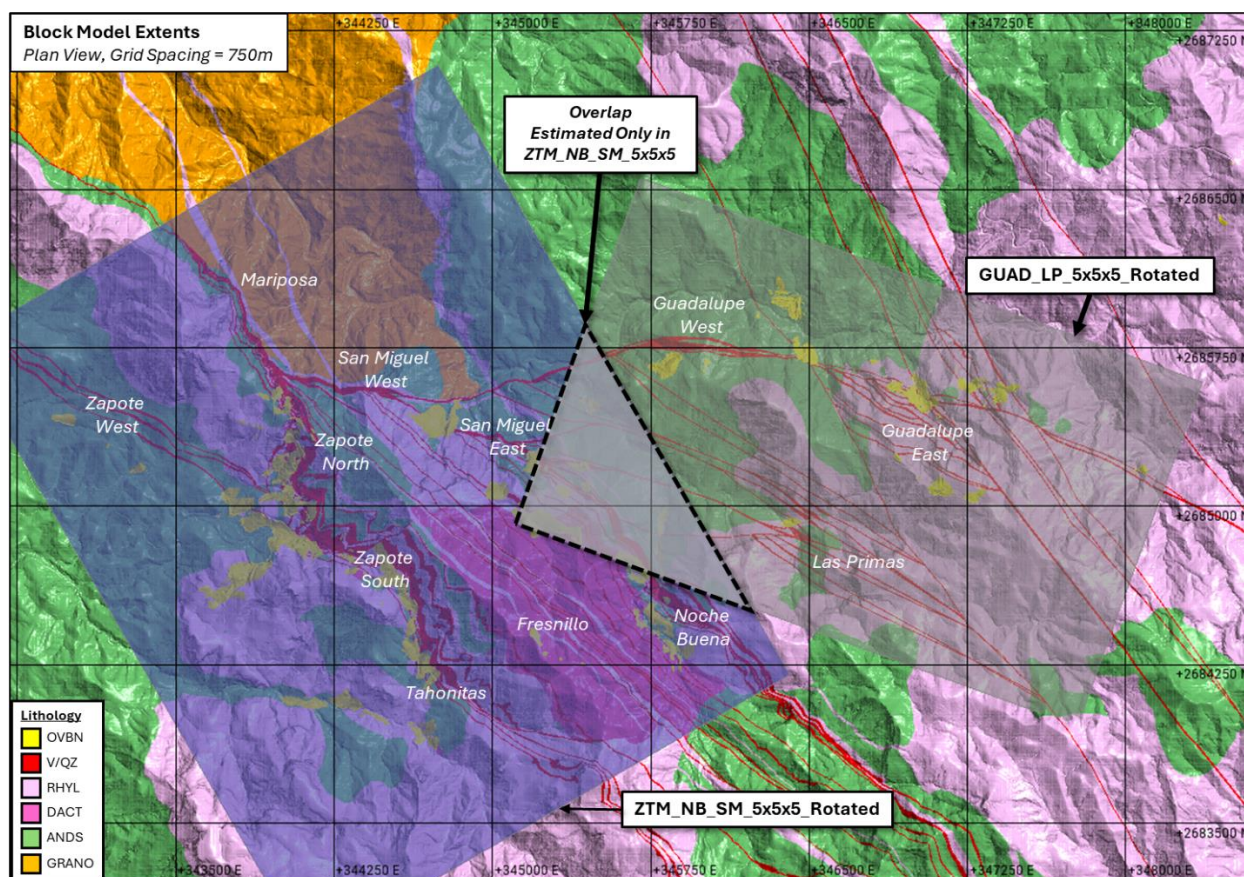
Notes: Nugget and normalized sill values from back-transformed normal scores variograms.

14.3.6 Block Model Set Up

14.3.6.1 5x5x5m Block Models

Two 5x5x5m block models were constructed in Leapfrog EDGE v.2024.1.1, both in the WGS84/UTM Zone 13 N coordinate system (Figure 14-22). The ZTM_NB_SM_5x5x5_Rotated model includes the Fresnillo, Mariposa, Noche Buena, San Miguel East, San Miguel West, Tahonitas, Zapote North, Zapote South, and Zapote West areas, and is rotated 30 degrees counterclockwise to match the dominant strike of mineralized structures in the western project area. The GUAD_LP_5x5x5_Rotated model includes the Guadalupe East, Guadalupe West, and Las Primas areas, and is rotated 20 degrees clockwise to match the dominant strike of mineralized structures in the eastern project area. Both models have a 5x5x5m regular block size. Table 14-16 shows block model parameters for both models.

Figure 14-20: Model extents for 5x5x5m block models



Source: Torex, 2026

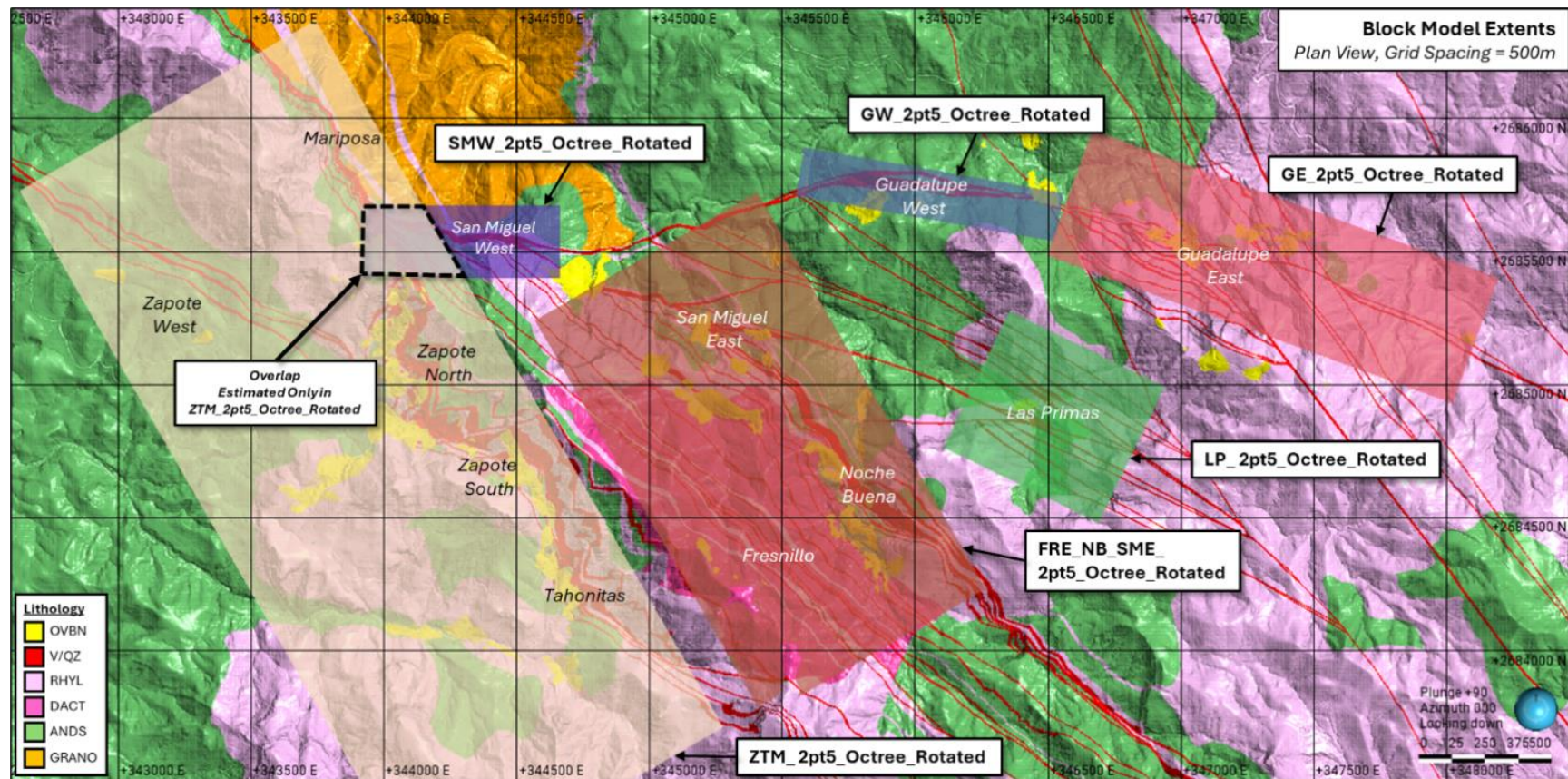
Table 14-16: Block model parameters for 5x5x5m block models

Model Build:	Leapfrog EDGE v.2024.1.1					
Coordinate System:	WGS84 / UTM Zone 13 N					
Model:	GUAD_LP_5x5x5_Rotated			ZTM_NB_SM_5x5x5_Rotated		
Rotation (azi/dip/pitch):	020/0/0			330/0/0		
Coordinate:	Easting (X)	Northing (Y)	Elevation (Z)	Easting (X)	Northing (Y)	Elevation (Z)
Block Size (m)	5	5	5	5	5	5
Min. Corner (m)	345,110	2,684,925	280	344,125	2,682,885	200
Min. Centroid (m)	348,359.5	2,685,586.0	282.5	344,741.1	2,687,082.9	202.5
Number of Blocks	566	347	174	527	666	160

14.3.6.2 2.5x2.5x2.5m Block Models

Six 2.5x2.5x2.5m block models were constructed in Leapfrog EDGE v.2024.1.1, all in the WGS84/UTM Zone 13 N coordinate system (Figure 14-23). All models have a 2.5x2.5x2.5m parent block size and are sub-blocked to 1.25x1.25x1.25m along mineralized domain contacts (1AuEQ prefix) and along the contacts of the V/QZ solid from the Los Reyes lithology model. The FRE_NB_SME_2pt5_Octree_Rotated model includes the Fresnillo, Noche Buena, and San Miguel East areas, and is rotated 30 degrees counterclockwise to match the dominant strike of mineralized structures along the Central trend. The GE_2pt5_Octree_Rotated model includes the Guadalupe East area and is rotated 20 degrees clockwise to match the dominant strike of the Estaca and Laija veinsets. The GW_2pt5_Octree_Rotated model includes the Guadalupe West area and is rotated 10 degrees clockwise to match the dominant strike of the western extension of the Estaca vein. The LP_2pt5_Octree_Rotated model includes the Las Primas area and is rotated 26 degrees clockwise to match the dominant strike of the Las Primas veinset. The SMW_2pt5_Octree_Rotated model includes the San Miguel West area and is not rotated due to the generally east-west strike of the Central structure at San Miguel West. The ZTM_2pt5_Octree_Rotated model includes the Tahonitas, Mariposa, Zapote North, Zapote South, and Zapote West areas, and is rotated 30 degrees counterclockwise to match the dominant strike of mineralized structures along the Z-T trend. Figure 14-17 shows block model parameters for all 2.5x2.5x2.5m models.

Figure 14-21: Model extents for 2.5x2.5x2.5m block models



Source: Torex, 2026

Table 14-17: Model parameters for 2.5x2.5x2.5m block models

Model Build:	Leapfrog EDGE v.2024.1.1								
Coordinate System:	WGS84/UTM Zone 13 N								
Model:	FRE_NB_SME_2pt5_Octree_Rotated			GE_2pt5_Octree_Rotated			GW_2pt5_Octree_Rotated		
Rotation (azi/dip/pitch):	330/0/0			020/0/0			010/0/0		
Coordinate:	East (X)	North (Y)	Elev. (Z)	East (X)	North (Y)	Elev. (Z)	East (X)	North (Y)	Elev. (Z)
Parent Block Size (m)	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5
Sub-blocks (Octree)	2	2	2	2	2	2	2	2	2
Min. Corner (m)	345,430	2,683,750	340	346,495	2,685,485	410	345,550	2,685,710	485
Min. Centroid (m)	345,430.45	2,683,751.70	341.25	346,496.60	2,685,485.70	411.25	345,551.40	2,685,711.00	486.25
Number of Blocks	386	689	226	650	201	272	396	74	166
Model Build:	Leapfrog EDGE v.2024.1.1								
Coordinate System:	WGS84/UTM Zone 13 N								
Model:	LP_2pt5_Octree_Rotated			SMW_2pt5_Octree_Rotated			ZTM_2pt5_Octree_Rotated		
Rotation (azi/dip/pitch):	026/0/0			000/0/0			330/0/0		
Coordinate:	East (X)	North (Y)	Elev. (Z)	East (X)	North (Y)	Elev. (Z)	East (X)	North (Y)	Elev. (Z)
Parent Block Size (m)	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5
Sub-blocks (Octree)	2	2	2	2	2	2	2	2	2
Min. Corner (m)	346,105	2,684,765	410	343,920	2,685,400	500	344,165	2,683,070	175
Min. Centroid (m)	346,106.67	2,684,765.57	411.25	343,921.13	2,685,401.10	501.25	344,165.45	2,683,071.70	176.25
Number of Blocks	260	288	248	298	110	108	514	1228	292

14.3.7 Grade Interpolation

Gold and silver grades were estimated by inverse distance cubed (ID3), ordinary kriging (OK), and nearest neighbor (NN) in all mineralized domains for both the 5x5x5m and 2.5x2.5x2.5m block models. Search ellipse orientation and radii were selected based on variogram models for each individual estimation domain, with variable search orientation (VO) applied according to the nearest vein midpoint surface in the quartz vein and breccia model (V/QZ). Initial search parameters for each domain were selected using Kriging Neighborhood Analysis (KNA) and were then refined based on results from preliminary model validation checks.

14.3.7.1 5x5x5m Block Models

Gold and silver grades in the 5x5x5m block models were estimated by ID3 and NN, using 3 metre composited data within the domains listed in Table 14-8. Ordinary Kriging (OK) was also performed for mineralized domains (02AuEQ prefix). A two-pass search strategy was applied for all domains, with search ellipse distances doubled in the second estimation pass. Estimation parameters for Au and Ag for domains estimated in the 5x5x5m models are summarized in Table 14-18 and Table 14-19.

Inverse distance cubed (ID3) was selected as the final estimation method for Au and Ag in the 5x5x5m block models because it reconciles well with the nearest neighbor estimates and generally falls within grade-tonnage envelopes generated from Sequential Gaussian Simulation (SGS; see 'Model Validation' section below). The OK estimate was used for comparison purposes but was not selected as the final estimation method because it tends to show a higher degree of smoothing relative to the NN estimate in Swath plots for most domains, in addition to generally higher tonnes and lower grade than limits defined by SGS grade-tonnage envelopes.

14.3.7.2 2.5x2.5x2.5m Block Models

Gold and silver grades in the 2.5x2.5x2.5m block models were estimated by ID3 and NN, using 1.5 metre composited data within the domains listed in Table 14-9. Ordinary Kriging (OK) was also performed for mineralized domains (1AuEQ prefix). A two-pass search strategy was applied for all domains, with search ellipse distances doubled in the second estimation pass. Estimation parameters for Au and Ag for all domains estimated in the 2.5x2.5x2.5m models are summarized in Table 14-20 and Table 14-21.

Inverse distance cubed (ID3) was selected as the final estimation method for Au and Ag in the 2.5x2.5x2.5m block models because it reconciles well with the nearest neighbor estimates and shows a lesser degree of grade smoothing when compared to Ordinary Kriging (OK) in most domains.

Table 14-18: Gold estimation parameters used for estimation in 5x5x5m block models

Variable: Au (g/t) Composites: 3m	Leapfrog1				Pass 1 Data Search						Pass 2 Data Search					
Domain	Dip	Dip Azi.	Pitch	Discretization (x/y/z)	Major (m)	Semi-Major (m)	Minor (m)	Min. Sample	Max. Sample	Max. Sample/Hole	Major (m)	Semi-Major (m)	Minor (m)	Min. Sample	Max. Sample	Max. Sample/Hole
02AuEQ_ESTACA	75	200	15	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
02AuEQ_FRE	55	225	20	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
02AuEQ_GW	60	190	10	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
02AuEQ_LAIJA	70	35	140	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
02AuEQ_GUAD_NW	90	55	0	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
02AuEQ_LP	80	40	120	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
02AuEQ_MAR	45	220	45	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
02AuEQ_NB_SME	60	235	40	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
02AuEQ_SM_SN	45	35	0	2x2x2	40	40	10	4	10	2	80	80	20	1	4	2
02AuEQ_SME_SPLAYS	55	195	15	2x2x2	40	40	10	4	10	2	80	80	20	1	4	2
02AuEQ_SMW	60	190	20	2x2x2	40	40	7.5	4	10	2	80	80	15	1	4	2
02AuEQ_TA	50	240	35	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
02AuEQ_ZN	55	250	35	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
02AuEQ_ZS	45	215	30	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
02AuEQ_ZW	60	210	0	2x2x2	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_ESTACA	75	200	15	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_FRE	55	220	30	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_GUAD_NW	90	55	0	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_GW	60	190	10	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_LAIJA	70	30	140	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_LP	80	205	0	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_NB_SME	50	220	50	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_SM_SN	45	35	0	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_SME_SPLAYS	55	195	15	-	60	60	10	4	10	2	120	120	20	1	4	2

Variable: Au (g/t) Composites: 3m	Leapfrog1				Pass 1 Data Search						Pass 2 Data Search					
Domain	Dip	Dip Azi.	Pitch	Discretization (x/y/z)	Major (m)	Semi-Major (m)	Minor (m)	Min. Sample	Max. Sample	Max. Sample/Hole	Major (m)	Semi-Major (m)	Minor (m)	Min. Sample	Max. Sample	Max. Sample/Hole
QV_BX_ZTM	40	250	20	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_ZW	60	210	0	-	60	60	10	4	10	2	120	120	20	1	4	2
BACKGROUND_GUAD_LP	0	0	90	-	60	60	20	1	4	2	-	-	-	-	-	-
BACKGROUND_ZTM_NB_SM	0	0	90	-	60	60	20	1	4	2	-	-	-	-	-	-

Notes: The search ellipse orientations shown above are the global plunge direction for each domain. Local search orientation is determined from variable orientation models.

Table 14-19: Silver estimation parameters used for estimation in 5x5x5m block models

Variable: Ag (g/t) Composites: 3m	Leapfrog1				Pass 1 Data Search						Pass 2 Data Search					
Domain	Dip	Dip Azi.	Pitch	Discretization (x/y/z)	Major (m)	Semi-Major (m)	Minor (m)	Min. Sample	Max. Sample	Max. Sample/Hole	Major (m)	Semi-Major (m)	Minor (m)	Min. Sample	Max. Sample	Max. Sample/Hole
02AuEQ_ESTACA	75	200	15	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
02AuEQ_FRE	55	220	60	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
02AuEQ_GW	60	190	10	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
02AuEQ_LAIJA	70	35	140	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
02AuEQ_GUAD_NW	90	55	0	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
02AuEQ_LP	80	40	120	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
02AuEQ_MAR	45	220	45	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
02AuEQ_NB_SME	60	235	220	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
02AuEQ_SM_SN	45	35	0	2x2x2	40	40	10	4	10	2	80	80	20	1	4	2
02AuEQ_SME_SPLAYS	55	195	15	2x2x2	40	40	10	4	10	2	80	80	20	1	4	2
02AuEQ_SMW	60	190	20	2x2x2	40	40	7.5	4	10	2	80	80	15	1	4	2
02AuEQ_TA	50	240	35	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
02AuEQ_ZN	55	250	35	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
02AuEQ_ZS	45	220	50	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
02AuEQ_ZW	60	210	0	2x2x2	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_ESTACA	75	200	35	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_FRE	55	220	30	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_GUAD_NW	90	55	0	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_GW	60	190	10	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_LAIJA	70	30	140	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_LP	80	205	0	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_NB_SME	50	220	50	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_SM_SN	45	35	0	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_SME_SPLAYS	55	195	15	-	60	60	10	4	10	2	120	120	20	1	4	2

Variable: Ag (g/t) Composites: 3m	Leapfrog1				Pass 1 Data Search						Pass 2 Data Search					
Domain	Dip	Dip Azi.	Pitch	Discretization (x/y/z)	Major (m)	Semi-Major (m)	Minor (m)	Min. Sample	Max. Sample	Max. Sample/Hole	Major (m)	Semi-Major (m)	Minor (m)	Min. Sample	Max. Sample	Max. Sample/Hole
QV_BX_ZTM	40	250	20	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_ZW	60	210	0	-	60	60	10	4	10	2	120	120	20	1	4	2
BACKGROUND_GUAD_LP	0	0	90	-	60	60	20	1	4	2	-	-	-	-	-	-
BACKGROUND_ZTM_NB_SM	0	0	90	-	60	60	20	1	4	2	-	-	-	-	-	-

Notes: The search ellipse orientations shown above are the global plunge direction for each domain. Local search orientation is determined from variable orientation models.

Table 14-20: Gold estimation parameters used for estimation in 2.5x2.5x2.5m block models

Variable: Au (g/t) Composites: 1.5m	Leapfrog1				Pass 1 Data Search						Pass 2 Data Search					
Domain	Dip	Dip Azi.	Pitch	Discretization (x/y/z)	Major (m)	Semi-Major (m)	Minor (m)	Min. Sample	Max. Sample	Max. Sample/Hole	Major (m)	Semi-Major (m)	Minor (m)	Min. Sample	Max. Sample	Max. Sample/Hole
1AuEQ_ESTACA	75	200	15	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
1AuEQ_FRE	55	225	20	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
1AuEQ_GW	60	190	10	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
1AuEQ_LAIJA	70	30	140	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
1AuEQ_GUAD_NW	90	55	0	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
1AuEQ_LP	80	40	120	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
1AuEQ_MAR	45	220	45	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
1AuEQ_NB_SME	60	235	40	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
1AuEQ_SM_SN	45	35	0	2x2x2	40	40	10	4	10	2	80	80	20	1	4	2
1AuEQ_SME_SPLAYS	55	195	15	2x2x2	40	40	10	4	10	2	80	80	20	1	4	2
1AuEQ_SMW	60	190	20	2x2x2	40	40	7.5	4	10	2	80	80	15	1	4	2
1AuEQ_TA	50	240	35	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
1AuEQ_ZN	55	250	35	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
1AuEQ_ZS	45	215	30	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
1AuEQ_ZW	60	210	0	2x2x2	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_ESTACA	75	200	15	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_FRE	55	220	30	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_GUAD_NW	90	55	0	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_GW	60	190	10	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_LAIJA	70	30	140	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_LP	80	205	0	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_NB_SME	50	220	50	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_SM_SN	45	35	0	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_SME_SPLAYS	55	195	15	-	60	60	10	4	10	2	120	120	20	1	4	2

Variable: Au (g/t) Composites: 1.5m	Leapfrog1				Pass 1 Data Search						Pass 2 Data Search					
Domain	Dip	Dip Azi.	Pitch	Discretization (x/y/z)	Major (m)	Semi-Major (m)	Minor (m)	Min. Sample	Max. Sample	Max. Sample/Hole	Major (m)	Semi-Major (m)	Minor (m)	Min. Sample	Max. Sample	Max. Sample/Hole
QV_BX_ZTM	40	250	20	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_ZW	60	210	0	-	60	60	10	4	10	2	120	120	20	1	4	2
BACKGROUND_GUAD_LP	0	0	90	-	60	60	20	1	4	2	-	-	-	-	-	-
BACKGROUND_ZTM_NB_SM	0	0	90	-	60	60	20	1	4	2	-	-	-	-	-	-

Notes: The search ellipse orientations shown above are the global plunge direction for each domain. Local search orientation is determined from variable orientation models.

Table 14-21: Silver estimation parameters used for estimation in 2.5x2.5x2.5m block models

Variable: Ag (g/t) Composites: 1.5m	Leapfrog1			Pass 1 Data Search							Pass 2 Data Search					
Domain	Dip	Dip Azi.	Pitch	Discretization (x/y/z)	Major (m)	Semi-Major (m)	Minor (m)	Min. Sample	Max. Sample	Max. Sample/Hole	Major (m)	Semi-Major (m)	Minor (m)	Min. Sample	Max. Sample	Max. Sample/Hole
1AuEQ_ESTACA	75	200	15	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
1AuEQ_FRE	55	225	20	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
1AuEQ_GW	60	190	10	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
1AuEQ_LAIJA	70	30	140	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
1AuEQ_GUAD_NW	90	55	0	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
1AuEQ_LP	80	40	120	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
1AuEQ_MAR	45	220	45	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
1AuEQ_NB_SME	60	235	40	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
1AuEQ_SM_SN	45	35	0	2x2x2	40	40	10	4	10	2	80	80	20	1	4	2
1AuEQ_SME_SPLAYS	55	195	15	2x2x2	40	40	10	4	10	2	80	80	20	1	4	2
1AuEQ_SMW	60	190	20	2x2x2	40	40	7.5	4	10	2	80	80	15	1	4	2
1AuEQ_TA	50	240	35	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
1AuEQ_ZN	55	250	35	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
1AuEQ_ZS	45	215	30	2x2x2	60	45	10	4	10	2	120	90	20	1	4	2
1AuEQ_ZW	60	210	0	2x2x2	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_ESTACA	75	200	35	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_FRE	55	220	30	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_GUAD_NW	90	55	0	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_GW	60	190	10	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_LAIJA	70	30	140	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_LP	80	205	0	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_NB_SME	50	220	50	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_SM_SN	45	35	0	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_SME_SPLAYS	55	195	15	-	60	60	10	4	10	2	120	120	20	1	4	2

Variable: Ag (g/t) Composites: 1.5m		Leapfrog1			Pass 1 Data Search						Pass 2 Data Search					
Domain	Dip	Dip Azi.	Pitch	Discretization (x/y/z)	Major (m)	Semi-Major (m)	Minor (m)	Min. Sample	Max. Sample	Max. Sample/Hole	Major (m)	Semi-Major (m)	Minor (m)	Min. Sample	Max. Sample	Max. Sample/Hole
QV_BX_ZTM	40	250	20	-	60	60	10	4	10	2	120	120	20	1	4	2
QV_BX_ZW	60	210	0	-	60	60	10	4	10	2	120	120	20	1	4	2
BACKGROUND_GUAD_LP	0	0	90	-	60	60	20	1	4	2	-	-	-	-	-	-
BACKGROUND_ZTM_NB_SM	0	0	90	-	60	60	20	1	4	2	-	-	-	-	-	-

Notes: The search ellipse orientations shown above are the global plunge direction for each domain. Local search orientation is determined from variable orientation models.

14.3.8 Bulk Density Modeling

A total of 4,776 density values from drill core were assessed according to logged lithology and alteration, modeled lithology, and by estimation domain to ensure the most representative possible values were assigned to the block models. Density was assigned to the mineralized domains (02AuEQ and 1AuEQ prefixes in the 5x5x5m and 2.5x2.5x2.5m block models, respectively) and QV_BX domains according to the average values within each domain group, and to all other modeled lithologies according to the average values within each solid. A value of 0.0 g/cm³ is assigned to historical workings. Density data collection is ongoing as new drilling is completed, and additional measurements will be incorporated into future resource estimates. Table 14-22 shows the current density values assigned to both the 5x5x5m and 2.5x2.5x2.5m block models.

Table 14-22: Density Values Assigned to Models

Domain/Lithology	Density (g/cm ³)	Source
Mineralized Domains (02AuEQ and 1AuEQ prefixes)	2.543	Mean value from Mineralized domains
QV_BX Domains	2.496	Mean value from QV_BX domains
RHYL	2.425	Mean value from RHYL solid in lithology model
DAC	2.444	Mean value from DAC solid in lithology model
ANDS	2.513	Mean value from ANDS solid in lithology model
GRANO	2.592	Mean value from GRANO solid in lithology model
OVBN	2.463	Mean value from OVBN solid in lithology model
Historical Workings	0.000	Historical workings wireframe model

Notes: To determine final block density, the original density value flagged according to domain and lithology is corrected by the proportion of the block which has been mined, by the formula: $\text{Density}[\text{Final}] = \text{Density}[\text{Original}] \times (1 - \text{Mined_Pct})$, where 'Mined_Pct' is the historically mined portion of the block.

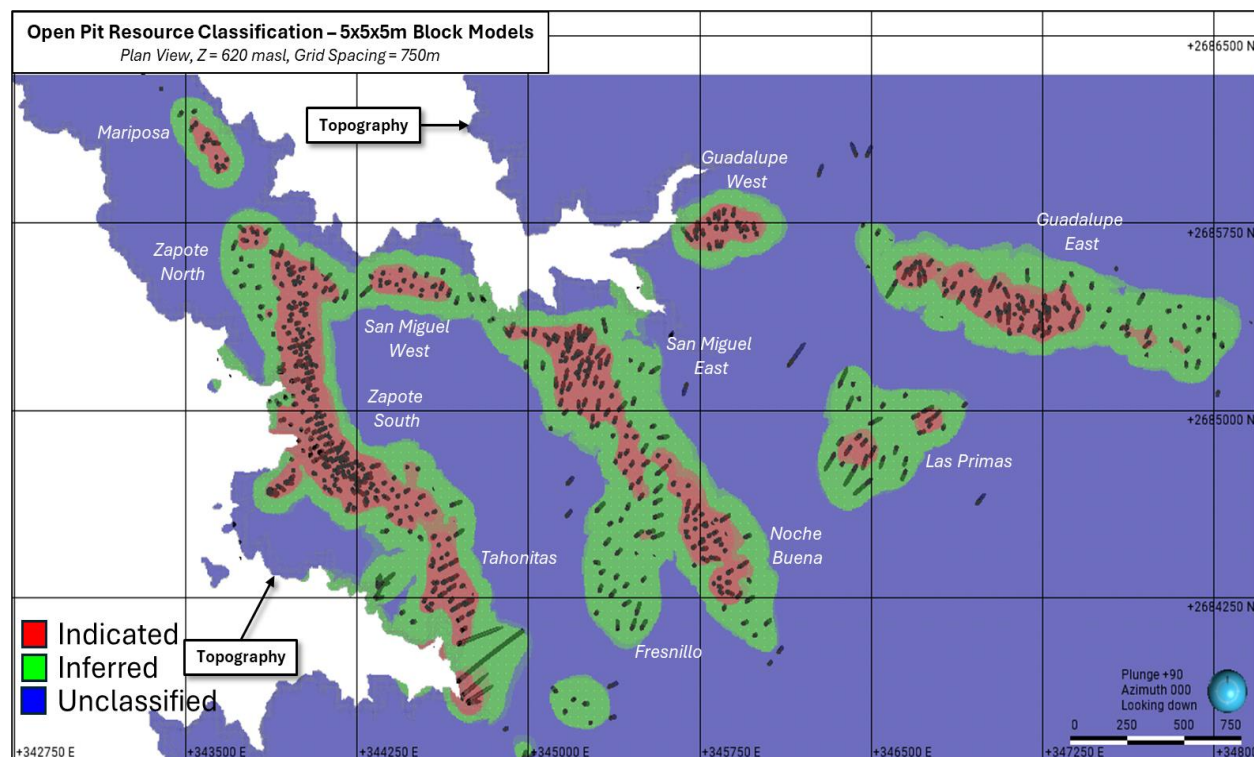
14.3.9 Mineral Resource Classification

Mineral Resources for the Los Reyes project are classified under the categories of Indicated and Inferred, in accordance with CIM Definition Standards. The Measured resource category was not used in either set of models because no modern mining has been undertaken at Los Reyes and it is therefore not possible to reconcile the models against production or tightly spaced data such as grade control drilling.

14.3.9.1 Open Pit Resource Classification (5x5x5m Block Models)

Data spacing sufficient for Open Pit Indicated Resources was determined by calculating the weighted average distance at which the direction 1 variogram models reach 80% of the normalized sill ($\text{Gamma} = 0.8$), determined graphically from back transformed variograms for the mineralized domains in the 5x5x5m block models (02AuEQ prefix). Inferred resource data spacing was determined by calculating the weighted average distance at which the direction 1 variogram models reach 95% of the normalized sill ($\text{Gamma} = 0.95$). Weights for each domain were assigned according to their total Au Oz and Ag Oz inventory, reported from the 5x5x5m block models. As a result of this analysis, Open Pit Indicated Resources were categorized based on a drill spacing of 40 metres or less and Inferred Resources were categorized based on a drill spacing of 40-80 metres. Figure 14-24 shows the classification codes for the 5x5x5m block models.

Figure 14-22: Los Reyes resource classification for 5x5x5m block models



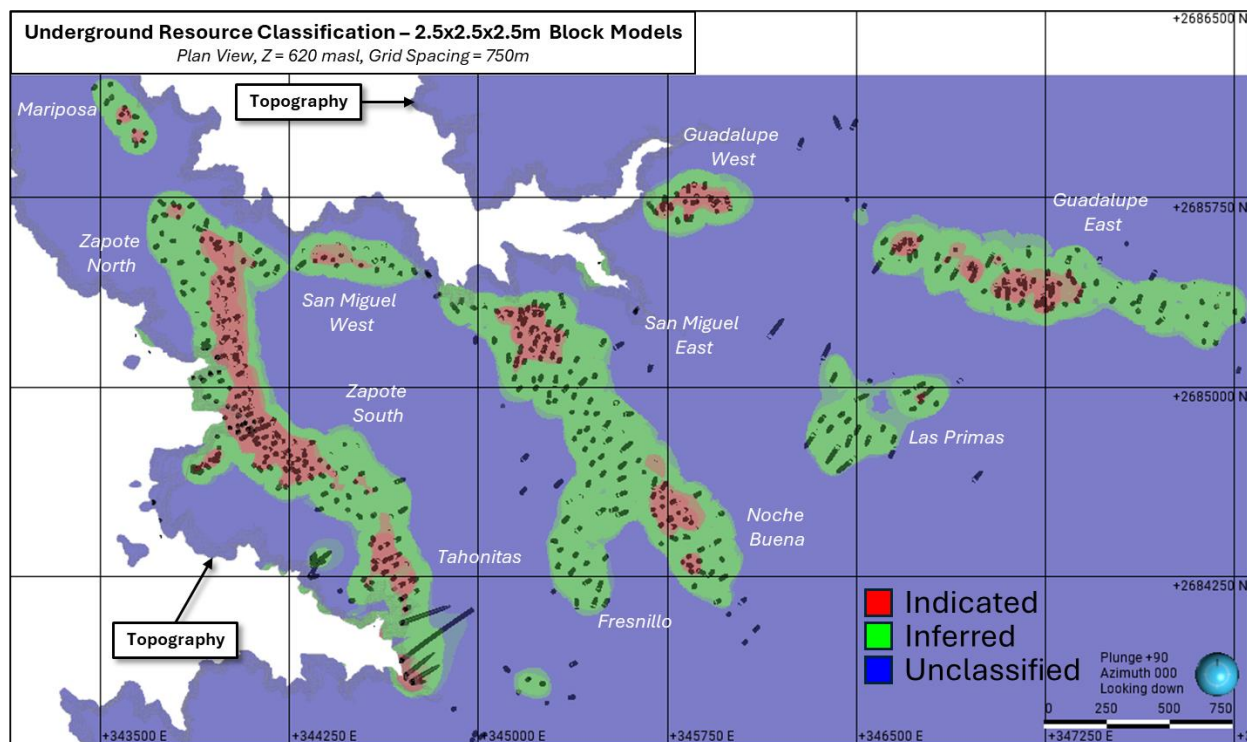
Source: Torex, 2026

Note: Plan view, looking down, at 620 m elevation.

14.3.9.2 Underground Resource Classification (2.5x2.5x2.5m Block Models)

Data spacing sufficient for Underground Indicated Resources was determined by calculating the weighted average distance at which the direction 1 variogram models reach 80% of the normalized sill (Gamma = 0.8), determined graphically from back transformed variograms for the mineralized domains in the 2.5x2.5x2.5m block models (1AuEQ prefix). Inferred resource data spacing was determined by calculating the weighted average distance at which the direction 1 variogram models reach 95% of the normalized sill (Gamma = 0.95). Weights for each domain were assigned according to their total Au Oz and Ag Oz inventory, reported from the 2.5x2.5x2.5m block models. As a result of this analysis, Underground Indicated Resources were categorized based on a drill spacing of 30 metres or less and Inferred Resources were categorized based on a drill spacing of 30-60 metres. Figure 14-25 shows the classification codes for the 2.5x2.5x2.5m block models.

Figure 14-23: Los Reyes resource classification for 2.5x2.5x2.5m block models



Source: Torex, 2026

Note: Plan view, looking down, at 620m elevation.

14.3.10 Model Validation

Validation checks for both block models are focused on the mineralized domains (02AuEq and 1AuEq prefixes), which contain >97% of the reported Au inventory and >98% of the reported Ag inventory. The models were validated using the following methods:

- Statistical comparison (ID3 vs. Uncapped ID3, NN, and OK)
- Sectional validation – visual comparison between block grades and composite grades
- Swath plots
- Comparison to grade-tonnage envelopes from Sequential Gaussian Simulation (SGS)

14.3.10.1 Estimate Comparison (ID3 vs Uncapped ID3, NN, and OK)

Statistics for the final Inverse Distance Cubed (ID3) estimates for both Au and Ag were compared to the Nearest Neighbor (NN) and Ordinary Kriging (OK) estimates, globally and domain by domain. The final ID3 estimates were also compared with an uncapped ID3 estimate to evaluate metal loss.

14.3.10.1.1 5x5x5m Block Models

The ID3 estimates selected for final resource reporting in the 5x5x5m block models were compared with NN and OK estimates for each mineralized domain (02AuEQ prefix), considering Indicated and Inferred blocks only, with no pit constraint applied. The difference in average estimated grade between the ID3 and NN estimates is less than 5% for all mineralized domains aside from 02AuEQ_GUAD_NW, which is sparsely

drilled and contains 0% of the total reported pit-constrained Au and Ag inventory. The difference in average estimated grade between the ID3 and OK estimates is less than 5% for Au and Ag in all mineralized domains.

The final ID3 estimates for Au and Ag were also compared against estimates prepared using the uncapped 3 metre composite dataset (ID3 Uncapped), to evaluate metal loss. The search parameters for the uncapped estimate were otherwise kept identical to the final ID3 estimates. Metal loss due to capping is less than 10% for all domains aside from Au_ID3 in the 02AuEQ_ESTACA and 02AuEQ_SMW domains and Ag_ID3 in the 02AUEQ_SM_SN domain. The large difference between the capped and uncapped estimates in these three cases is driven by extreme outliers in the uncapped dataset (for example, a single 156 g/t Au composite in the 02AuEQ_ESTACA domain). Table 14-23 and 14-24 show the comparison between the various estimation methods for Au and Ag, domain by domain.

Table 14-23: Estimate mean comparison between Au_ID3, Au_ID3 Uncapped, Au_NN, and Au_OK for mineralized domains in the 5x5x5m block models

Domain	3 metre Composite Mean (g/t)	Au_ID3 (g/t)	Au_ID3 Uncapped (g/t)	ID3 vs. ID3 Uncapped	Au_NN (g/t)	ID3 vs. NN	Au_OK (g/t)	ID3 vs. OK
02AuEQ_ESTACA	1.660	1.712	1.949	-12.2%	1.746	-1.9%	1.679	2.0%
02AuEQ_FRE	0.403	0.416	1.417	-0.2%	0.418	-0.5%	0.413	0.7%
02AuEQ_GUAD_NW	0.528	0.269	0.270	-0.4%	0.249	8.0%	0.266	1.1%
02AuEQ_GW	0.564	0.380	0.380	0.0%	0.371	2.4%	0.385	-2.3%
02AuEQ_LAIJA	1.373	1.160	1.169	-0.8%	1.192	-2.7%	1.218	-4.8%
02AuEQ_LP	0.423	0.384	0.386	-0.5%	0.381	0.8%	0.378	1.6%
02AuEQ_MAR	0.632	0.610	0.619	-1.5%	0.610	0.0%	0.378	2.2%
02AuEQ_NB_SME	0.534	0.450	0.452	-0.4%	0.445	1.1%	0.448	0.4%
02AuEQ_SM_SN	0.569	0.812	0.848	-4.2%	0.804	1.0%	0.806	0.7%
02AuEQ_SME_SPLAYS	0.406	0.385	0.400	-3.7%	0.382	0.8%	0.395	-2.5%
02AuEQ_SMW	0.987	0.665	0.761	-12.6%	0.662	0.5%	0.675	-2.5%
02AuEQ_TA	0.468	0.458	0.464	-2.3%	0.455	0.7%	0.459	-0.2%
02AuEQ_ZN	0.740	0.511	0.522	-2.1%	0.518	0.8%	0.518	-1.4%
02AuEQ_ZS	0.816	0.370	0.370	0.0%	0.364	1.2%	0.622	-2.3%

Notes:

1. Au_ID3, Au_NN, and Au_OK values are estimated using capped 3 metre composite dataset
2. Au_ID3 Uncapped is estimated using uncapped 3 metre composites, with the same search parameters as for Au_ID3
3. Indicated and Inferred resource categories only, no pit constraint applied

Table 14-24: Estimate mean comparison between Ag_ID3, Ag_ID3 Uncapped, Ag_NN, and Ag_OK for mineralized domains in the 5x5x5m block models

Domain	3 metre Composite Mean (g/t)	Ag_ID3 (g/t)	Ag_ID3 Uncapped (g/t)	ID3 vs. ID3 Uncapped	Ag_NN (g/t)	ID3 vs. NN	Ag_OK (g/t)	ID3 vs. OK
02AuEQ_ESTACA	105.65	109.499	110.710	-1.1%	114.52	-4.4%	105.28	4.0%
02AuEQ_FRE	6.922	7.474	7.700	-2.9%	7.538	-0.8%	7.425	0.7%
02AuEQ_GUAD_NW	20.881	14.364	14.39	-0.2%	7.538	9.6%	14.589	-2.5%
02AuEQ_GW	19.803	19.068	19.068	0.0%	18.990	0.4%	19.051	0.1%
02AuEQ_LAIJA	46.163	46.929	47.340	-0.9%	46.030	2.0%	47.869	-2.0%
02AuEQ_LP	18.714	15.342	15.630	-1.8%	15.258	0.6%	15.296	0.3%
02AuEQ_MAR	10.390	10.465	10.465	0.0%	10.508	-0.4%	10.410	0.5%
02AuEQ_NB_SME	22.636	20.355	20.400	-0.2%	20.141	1.1%	20.241	0.6%
02AuEQ_SM_SN	34.649	50.021	60.29	-17.0%	49.381	1.3%	49.268	1.5%
02AuEQ_SME_SPLAYS	33.003	29.169	29.550	-2.3%	29.303	-0.5%	29.558	-2.3%
02AuEQ_SMW	20.807	14.874	15.100	-2.5%	14.612	1.8%	14.786	-0.4%
02AuEQ_TA	27.210	24.738	25.660	-3.4%	24.564	0.7%	24.841	-0.4%
02AuEQ_ZN	13.281	11.635	11.670	-0.3%	11.607	0.2%	11.558	0.7%
02AuEQ_ZS	20.651	18.711	18.711	0.0%	18.691	0.1%	18.650	0.3%

Notes:

1. **Ag_ID3, Ag_NN, and Ag_OK values are estimated using the capped 3 metre composite dataset**
2. **Ag_ID3 Uncapped is estimated using uncapped 3 metre composites, with the same search parameters as for Ag_ID3**
3. **Indicated and Inferred resource categories only, no pit constraint applied**

14.3.10.1.2 2.5x2.5x2.5m Block Models

The ID3 estimates selected for final resource reporting in the 2.5x2.5x2.5m block models were compared with NN and OK estimates for each mineralized domain (1AuEQ prefix), considering Indicated and Inferred blocks only, with no MSO constraint applied. The difference in average estimated grade between the ID3 and NN estimates is less than 5% for all mineralized domains aside from 1AuEQ_LP, 1AuEQ_SM_SN, and 1AuEQ_SMW, which collectively contain <3% of the total reported Au and Ag inventory.

The final ID3 estimates for Au and Ag were also compared against estimates prepared using the uncapped 3 metre composite dataset (ID3 Uncapped), to evaluate metal loss. The search parameters for the uncapped estimate were otherwise kept identical to the final ID3 estimates. Metal loss due to capping is less than 10% for all domains aside from Au_ID3 in the 1AuEQ_ESTACA and 1AuEQ_SMW domains and Ag_ID3 in the 1AuEQ_SM_SN domain. The large difference between the capped and uncapped estimates in these cases is driven by extreme outliers in the uncapped dataset (for example, a single 156 g/t Au composite in the 1AuEQ_ESTACA domain). Table 14-25 and Table 14-26 show the comparison between the various estimation methods for Au and Ag, domain by domain.

Table 14-25: Estimate mean comparison between Au_ID3, Au_ID3 Uncapped, Au_NN, and Au_OK for mineralized domains in the 2.5x2.5x2.5m block models

Domain	1.5 metre Composite Mean (g/t)	Au_ID3 (g/t)	Au_ID3 Uncapped (g/t)	ID3 vs. ID3 Uncapped	Au_NN (g/t)	ID3 vs. NN	Au_OK (g/t)	ID3 vs. OK
1AuEQ_ESTACA	2.669	2.658	2.960	-10.2%	2.747	-3.2%	2.437	9.1%
1AuEQ_FRE	1.168	1.171	1.230	-4.8%	1.156	1.3%	1.192	-1.8%
1AuEQ_GW	1.373	1.240	1.290	-3.9%	1.204	3.0%	1.231	0.7%
1AuEQ_LAIJA	2.349	2.150	2.240	-4.0%	2.130	0.9%	2.036	5.6%
1AuEQ_LP	0.897	0.937	0.940	-0.3%	0.895	4.7%	0.917	2.2%
1AuEQ_MAR	2.167	1.910	2.020	-5.4%	1.854	3.0%	1.889	1.1%
1AuEQ_NB_SME	1.396	1.205	1.210	-0.4%	1.189	1.3%	1.182	1.9%
1AuEQ_SM_SN	1.995	1.748	1.800	-2.9%	1.923	-9.1%	1.679	4.1%
1AuEQ_SME_SPLAYS	1.344	1.539	1.640	-6.2%	1.558	-1.2%	1.457	5.6%
1AuEQ_SMW	7.028	8.182	9.420	-13.1%	7.720	6.0%	7.955	2.9%
1AuEQ_TA	1.202	0.995	1.008	-1.3%	0.995	0.0%	0.982	1.3%
1AuEQ_ZN	2.156	1.800	1.820	-1.1%	1.765	2.0%	1.762	2.2%
1AuEQ_ZS	2.027	1.744	1.750	-0.3%	1.741	0.2%	1.732	0.7%

Notes:

1. Au_ID3, Au_NN, and Au_OK values are estimated using capped 1.5 metre composite dataset.
2. Au_ID3 Uncapped is estimated using uncapped 1.5 metre composites, with the same search parameters as for Au_ID3.
3. Indicated and Inferred resource categories only, no pit constraint applied.

Table 14-26: Estimate mean comparison between Ag_ID3, Ag_ID3 Uncapped, Ag_NN, and Ag_OK for mineralized domains in the 2.5x2.5x2.5m block models

Domain	1.5 metre Composite Mean (g/t)	Ag_ID3 (g/t)	Ag_ID3 Uncapped (g/t)	ID3 vs. ID3 Uncapped	Ag_NN (g/t)	ID3 vs. NN	Ag_OK (g/t)	ID3 vs. OK
1AuEQ_ESTACA	164.271	164.159	167.04	-1.7%	171.561	-4.3%	155.938	5.3%
1AuEQ_FRE	23.249	21.087	21.22	-0.6%	20.898	0.9%	21.306	-1.0%
1AuEQ_GW	35.207	46.077	48.11	-4.2%	46.097	-0.0%	46.172	-0.2%
1AuEQ_LAIJA	75.526	83.692	87.320	-4.2%	83.635	0.1%	78.920	6.0%
1AuEQ_LP	40.913	30.586	30.760	-0.6%	30.053	1.8%	26.24	16.6%
1AuEQ_MAR	20.761	21.592	22.040	-2.0%	21.943	-1.6%	21.097	2.3%
1AuEQ_NB_SME	56.455	52.578	53.460	-1.6%	52.343	0.4%	51.636	1.8%
1AuEQ_SM_SN	107.859	78.532	104.24	-24.7%	91.768	-14.4%	71.918	9.2%
1AuEQ_SME_SPLAYS	101.742	112.914	123.34	-9.5%	116.238	-2.9%	107.599	4.9%
1AuEQ_SMW	88.856	97.610	105.630	-7.6%	98.54	-0.9%	90.965	7.3%
1AuEQ_TA	73.377	68.307	70.02	-2.4%	68.339	0.0%	67.047	1.9%
1AuEQ_ZN	23.106	23.798	23.800	0.0%	23.838	-0.2%	23.107	3.0%
1AuEQ_ZS	40.147	42.525	42.525	0.0%	42.808	-0.7%	41.779	1.8%
1AuEQ_ZW	21.642	17.181	17.181	0.0%	17.694	-2.9%	14.711	16.8%

Notes:

1. Ag_ID3, Ag_NN, and Ag_OK values are estimated using the capped 1.5 metre composite dataset.
2. Ag_ID3 Uncapped is estimated using uncapped 1.5 metre composites, with the same search parameters as for Ag_ID3.
3. Indicated and Inferred resource categories only, no pit constraint applied.

14.3.10.2 Sectional Validation – Blocks versus Composites

Estimated gold and silver block grades, resource classification, lithology model and underground workings wireframe assignment to blocks, and drill hole composite data were compared visually in plan and cross section for all deposit areas and in both sets of block models, discussed below.

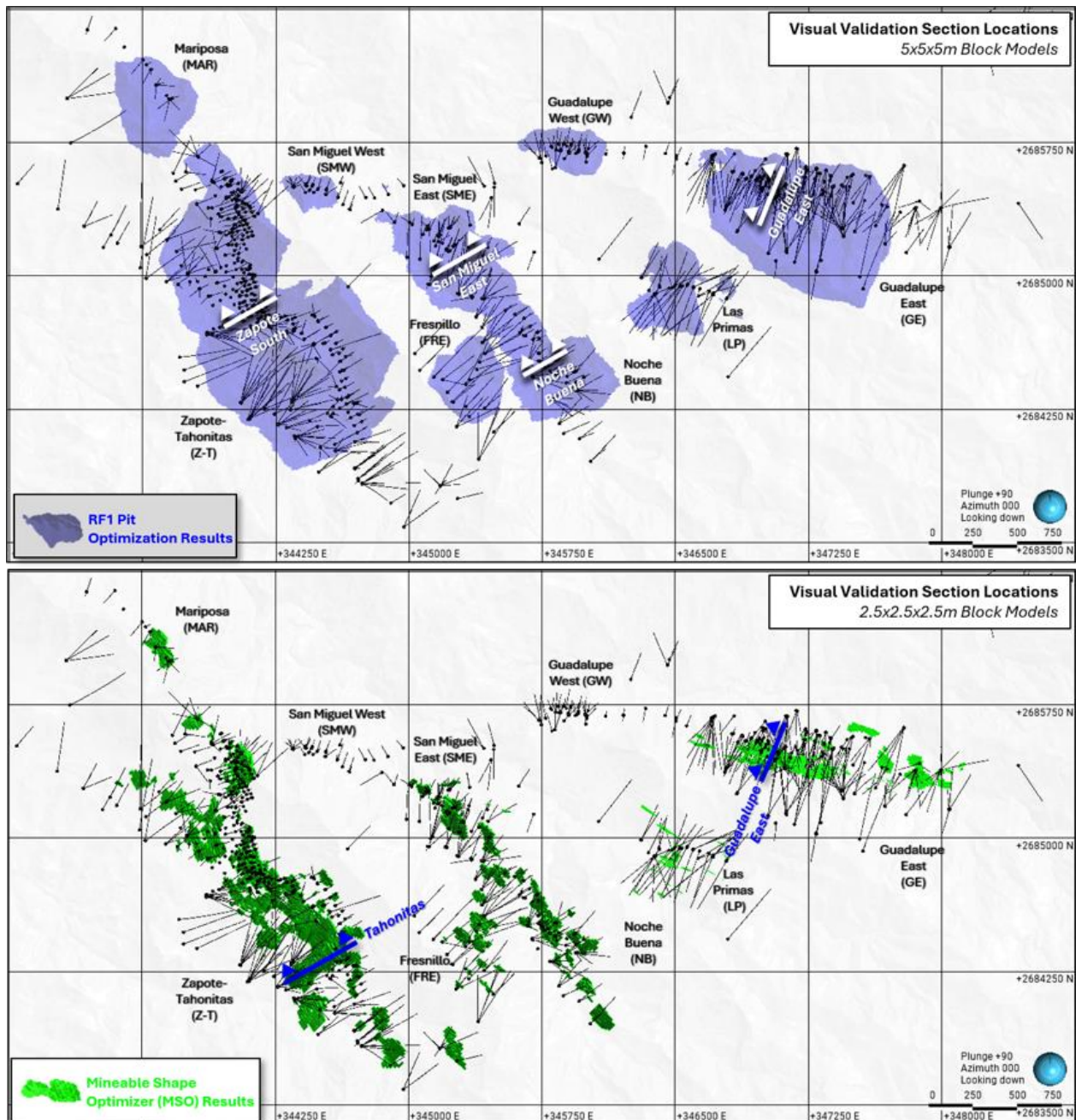
14.3.10.2.1 5x5x5m Block Models

Visual validation demonstrates that ID3-estimated 5x5x5m block grades reproduce the 3 metre composite grades well. Figure 14-27 through 14-30 show several examples comparing estimated Au and Ag grades to the composited dataset within pit shells generated using the Hochbaum Pseudoflow algorithm (see the section titled ‘Assessment of Reasonable Prospects for Eventual Economic Extraction’ below).

14.3.10.2.2 2.5x2.5x2.5m Block Models

Visual validation demonstrates that ID3-estimated 2.5x2.5x2.5m block grades reproduce the 1.5 metre composite grades well. Figure 14-31 through Figure 14-34 show several examples comparing estimated Au and Ag grades to the composited dataset within stopes generated using Datamine’s Mineable Stope Optimizer (MSO, see the section titled ‘Assessment of Reasonable Prospects for Eventual Economic Extraction’ below).

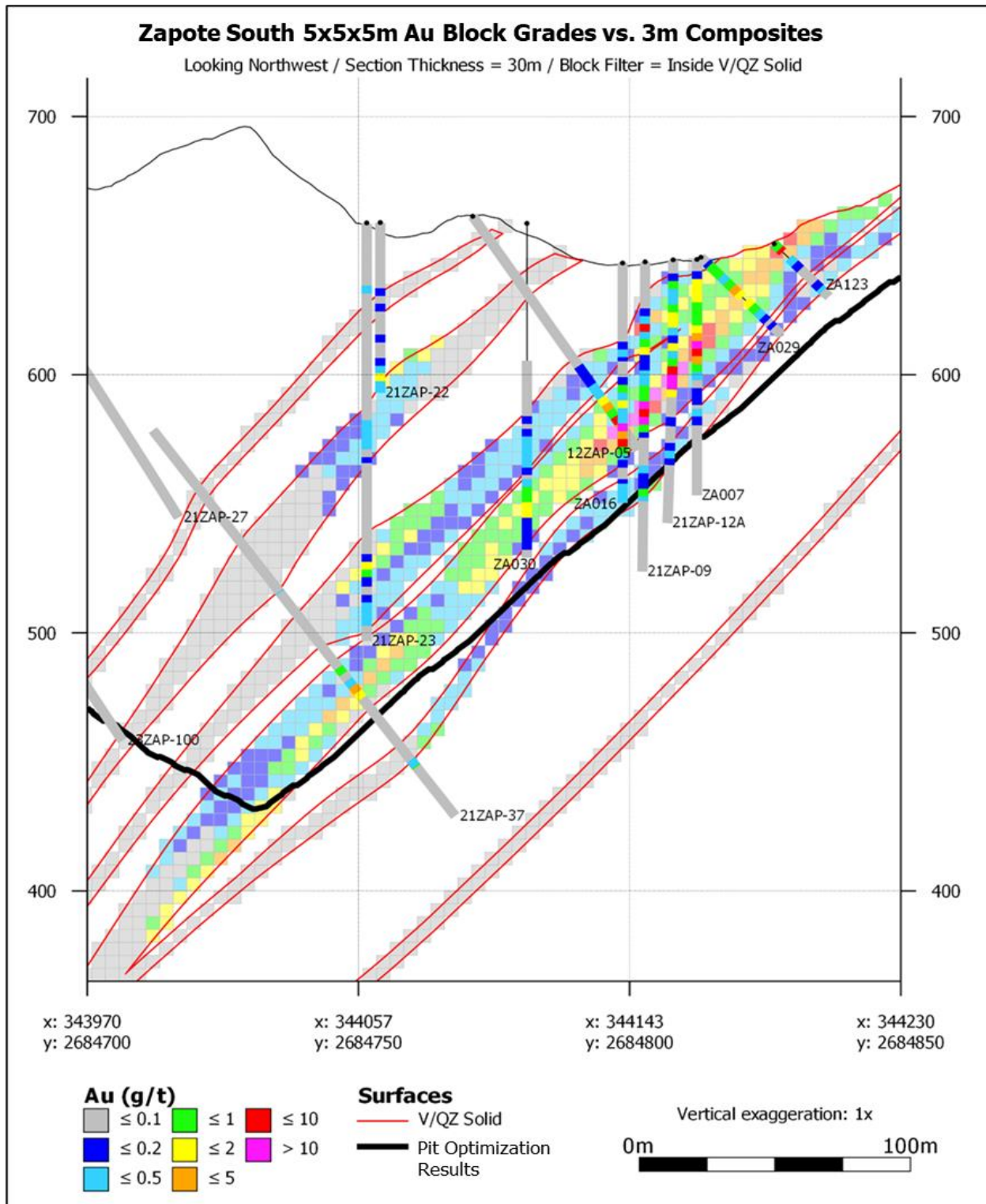
Figure 14-24: Section locations for Figures 14-27 through 14-30 (upper) and Figures 14-31 through 14-34 (lower)



Source: Torex, 2026

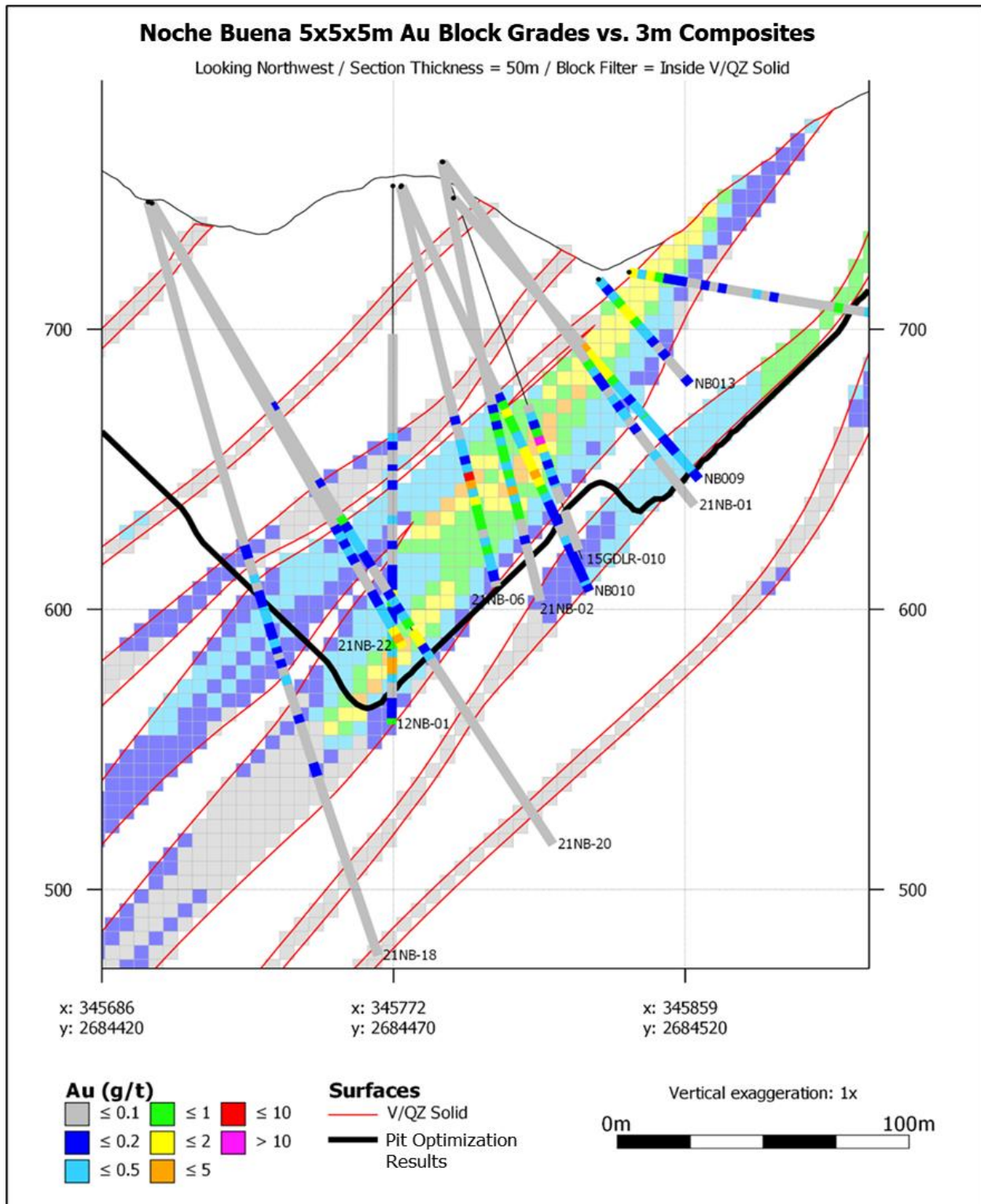
Notes: Pit optimization and Mineable Shape Optimization (MSO) results are determined independently using the 5x5x5m block models and 2.5x2.5x2.5m block models, respectively (optimization parameters are listed in the section titled 'Assessment of Reasonable Prospects for Eventual Economic Extraction' below). These shapes are used for model validation and sensitivity analysis only and do not represent the final combined set of open pit and underground Resource-constraining wireframes (Figure 14-46).

Figure 14-25: Zapote South Au Visual Validation – 5x5x5m Block Model



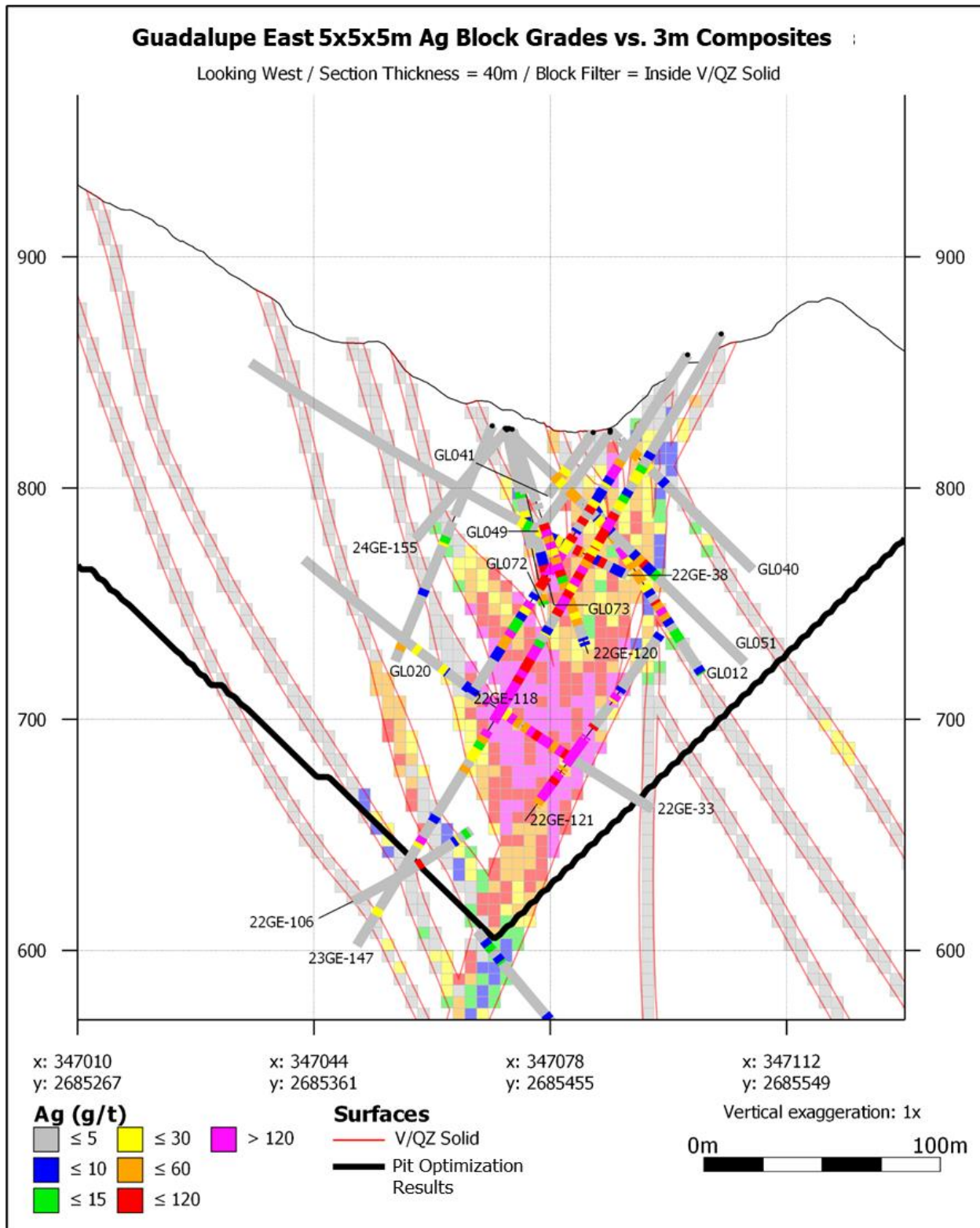
Source: Torex, 2026

Figure 14-26: Noche Buena Au Visual Validation – 5x5x5m Block Model



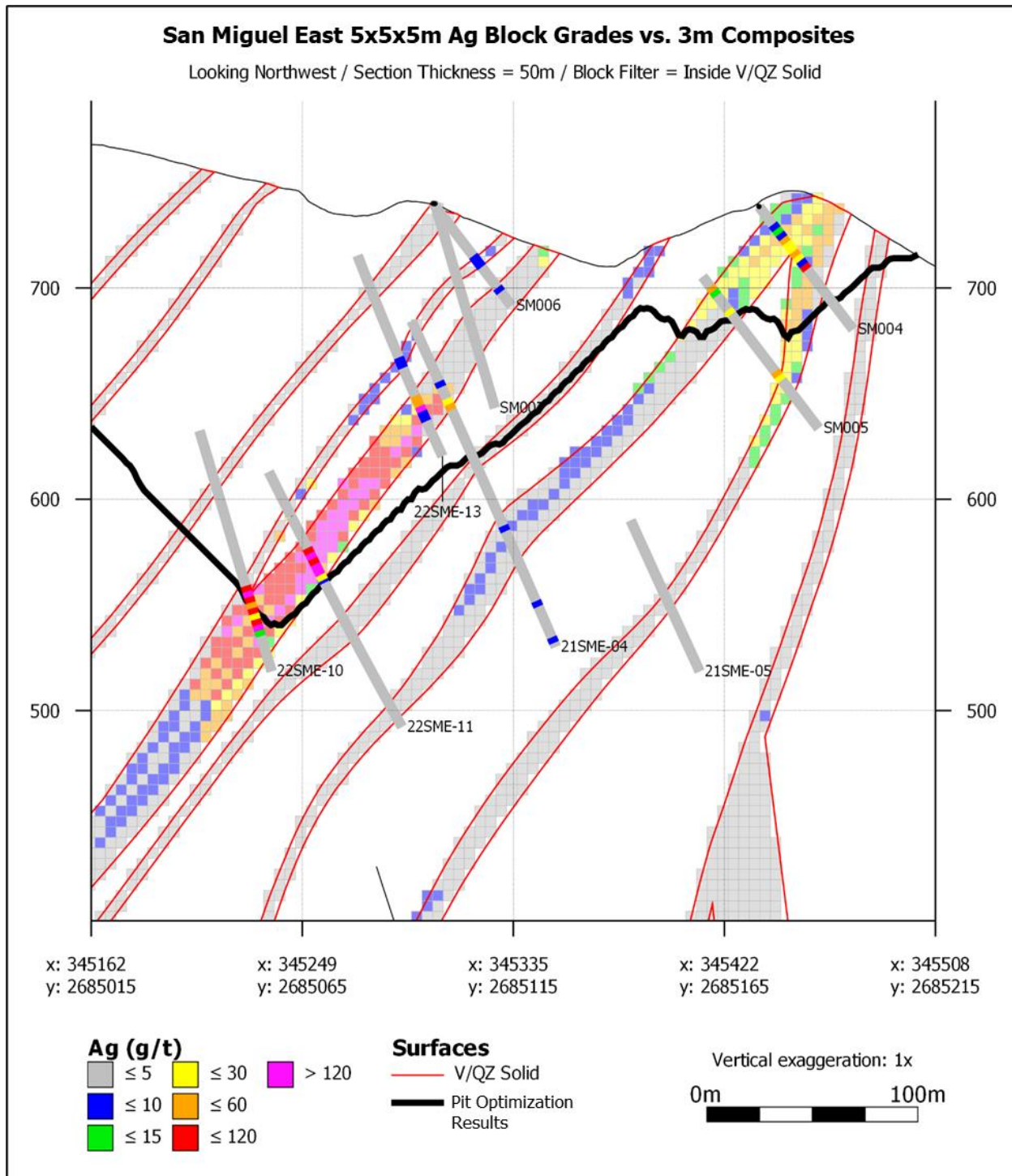
Source: Torex, 2026

Figure 14-27: Guadalupe East Ag Visual Validation – 5x5x5m Block Model



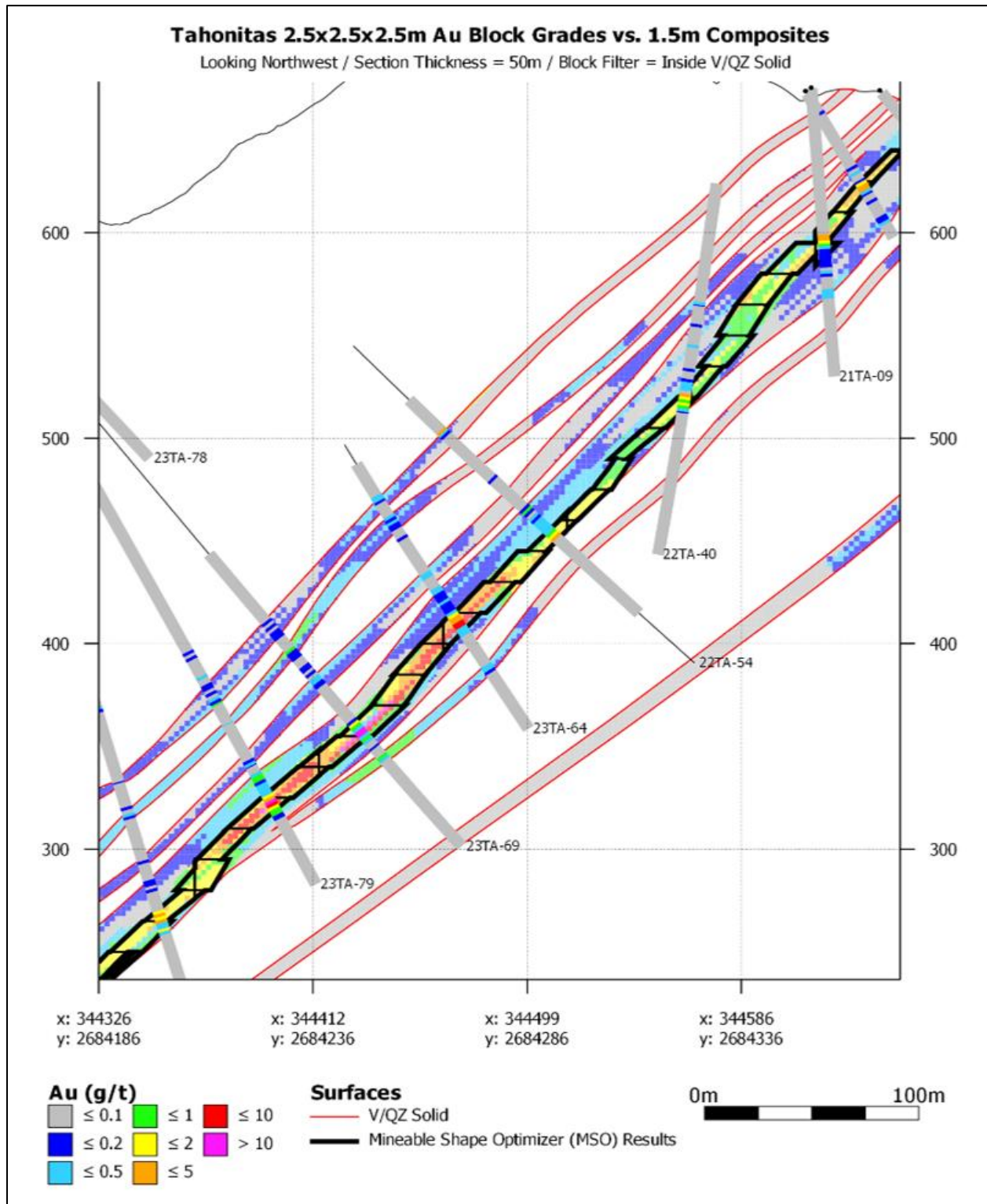
Source: Torex, 2026

Figure 14-28: San Miguel East Ag Visual Validation – 5x5x5m Block Model



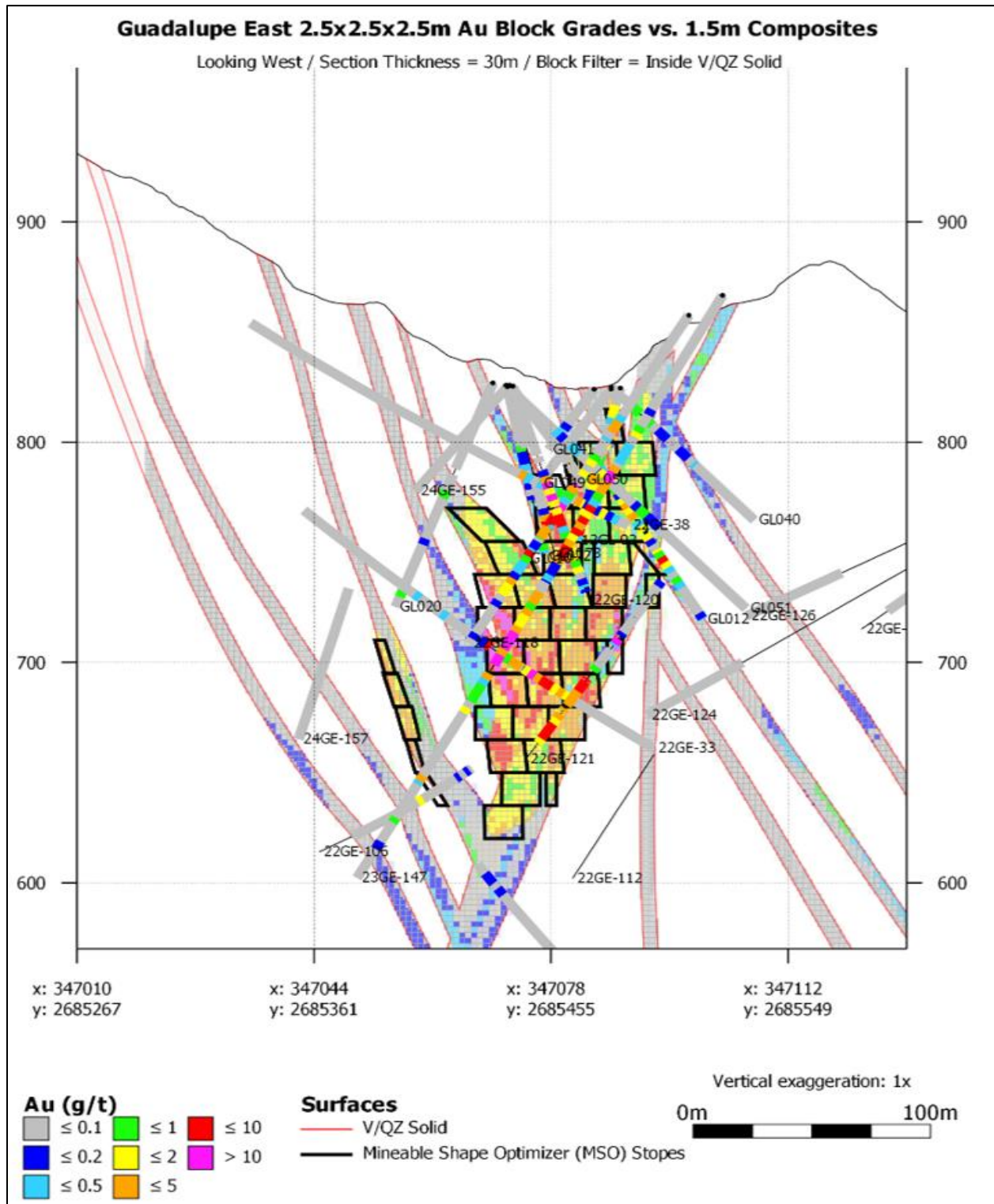
Source: Torex, 2026

Figure 14-29: Tahonitas Au Visual Validation – 2.5x2.5x2.5m Block Model



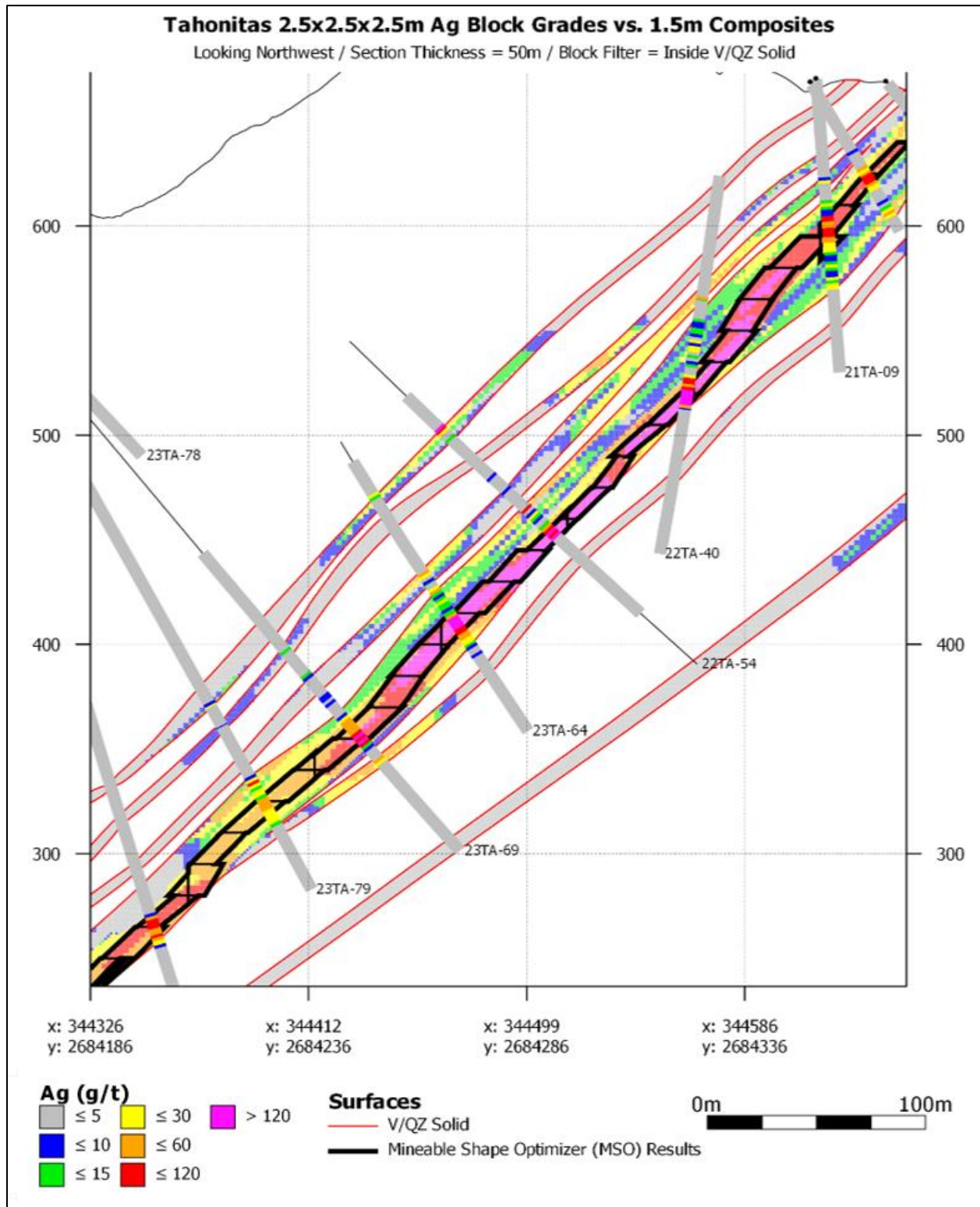
Source: Torex, 2026

Figure 14-30: Guadalupe East Au Visual Validation – 2.5x2.5x2.5m Block Model



Source: Torex, 2026

Figure 14-31: Tahonitas Ag Visual Validation – 2.5x2.5x2.5m Block Model



Source: Torex, 2026

14.3.10.3 Swath Plots

Swath plots were generated for each mineralized estimation domain to compare the ID3, NN, and OK estimates against one another and against composite grades. Results from each set of block models is described below.

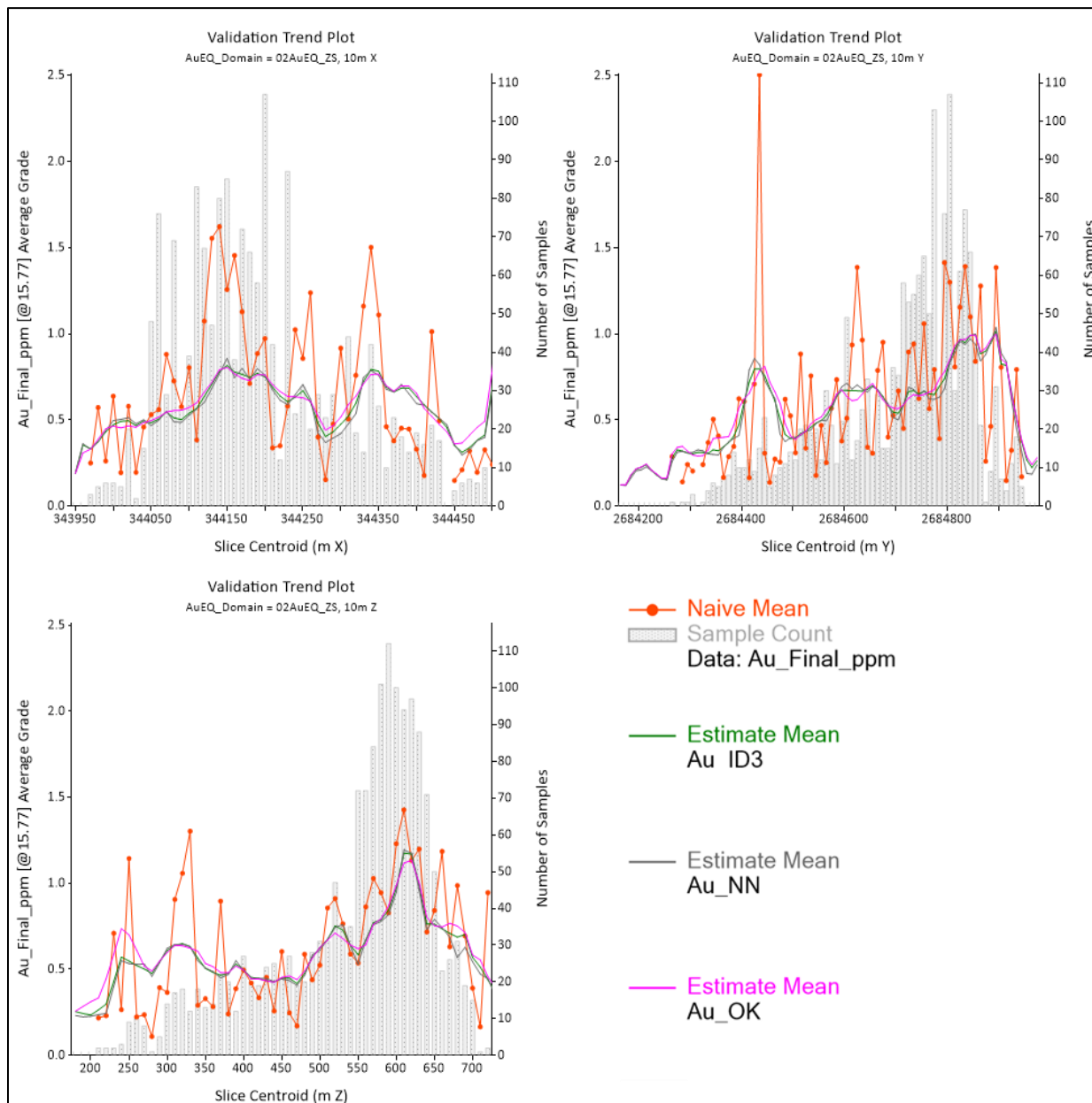
14.3.10.3.1 5x5x5m Block Models

Results demonstrate that the ID3 estimates for Au and Ag in mineralized domains (02AuEQ prefix) in the 5x5x5m block models do not show a systematic high or low bias against the NN estimate or 3 metre composites, and that the estimated grades for all three methods match the composite grades well in easting, northing, and elevation. The OK estimates tend to show a higher degree of smoothing relative to ID3, hence the selection of ID3 as the final estimation method. Figure 14-35 through Figure 14-38 show examples from several mineralized domains.

14.3.10.3.2 2.5x2.5x2.5m Block Models

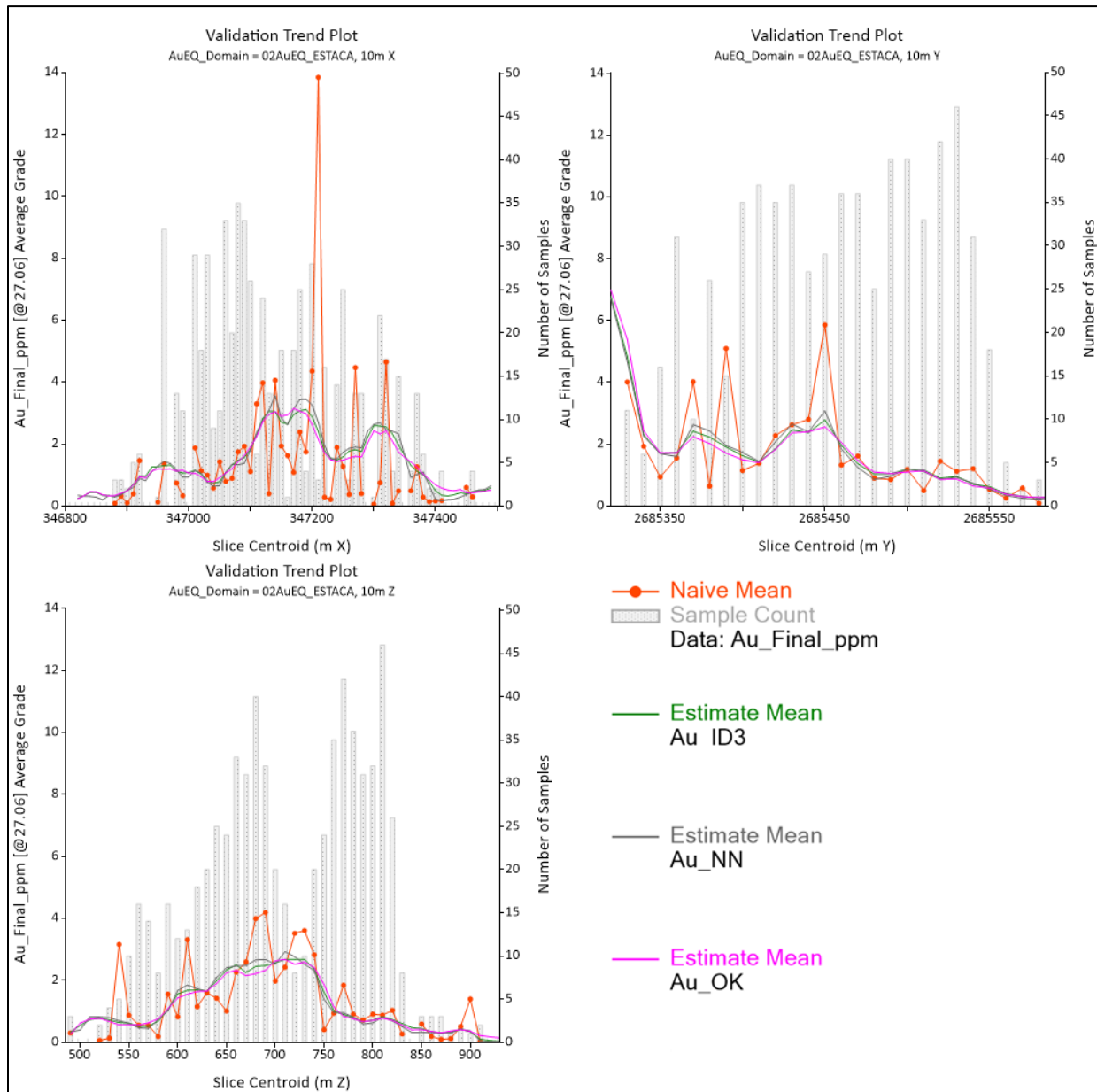
Results demonstrate that the ID3 estimates for Au and Ag in mineralized domains (1AuEQ prefix) in the 2.5x2.5x2.5m block models do not show a systematic high or low bias against the NN estimate or 1.5 metre composites, and that the estimated grades for all three methods match the composite grades well in easting, northing, and elevation. The OK estimates tend to show a higher degree of smoothing relative to ID3, hence the selection of ID3 as the final estimation method. Figure 14-39 through Figure 14-42 show examples from several mineralized domains.

Figure 14-33: Au Swath plots from the 02AuEQ_ZS estimation domain (5x5x5m block model)



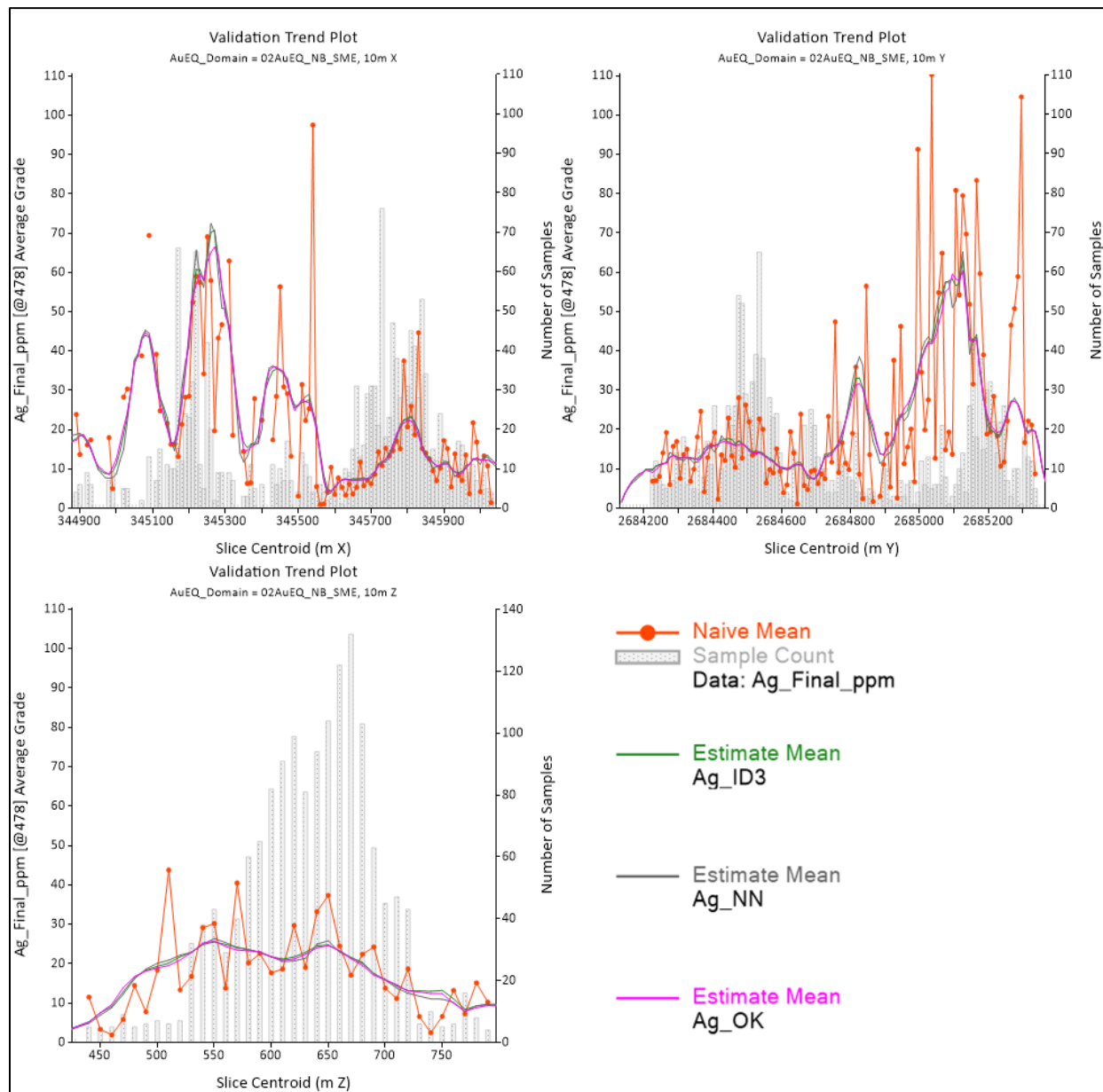
Source: Torex, 2026

Figure 14-34: Au Swath plots from the 02AuEQ_ESTACA estimation domain (5x5x5m block model)



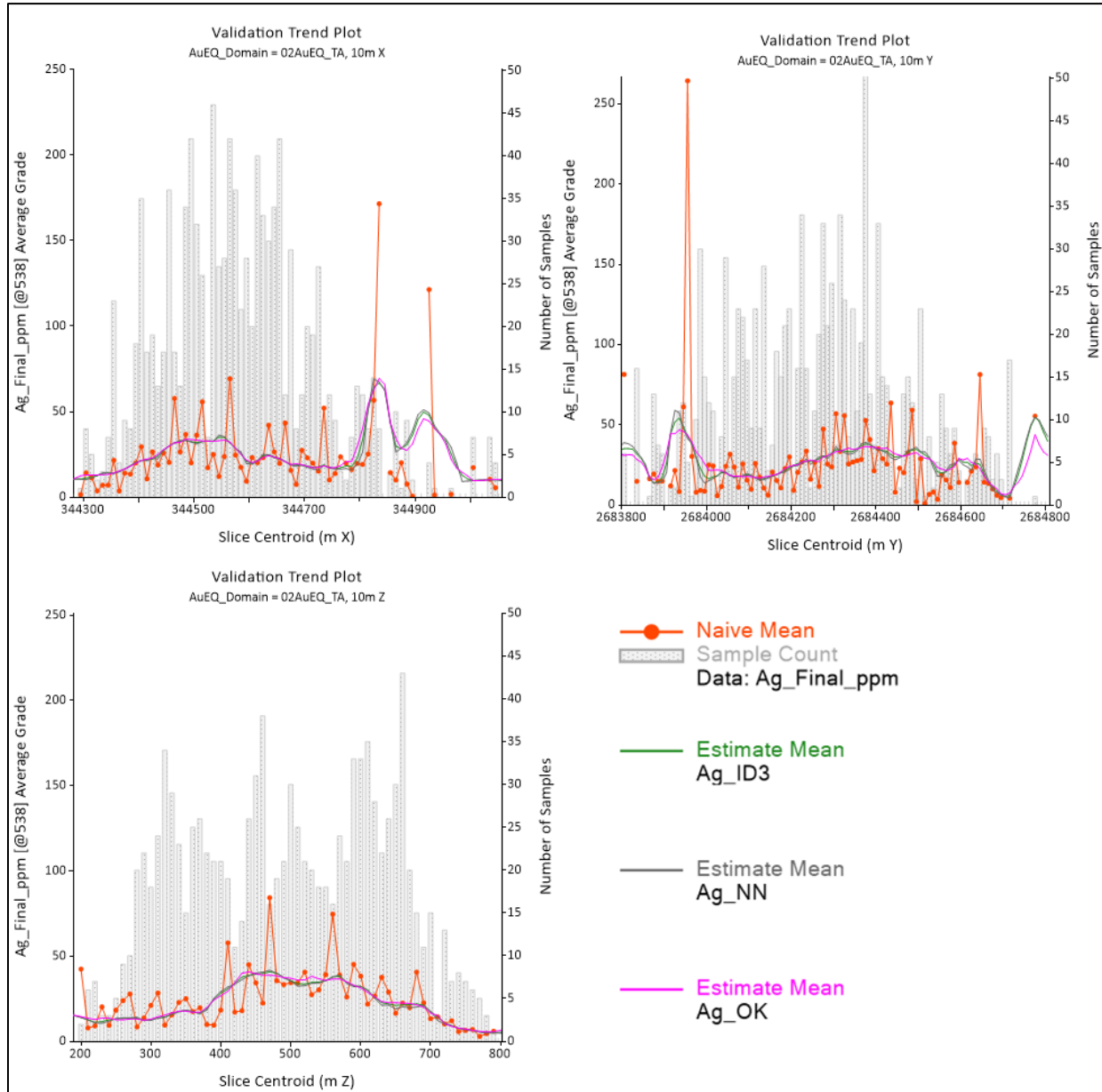
Source: Torex, 2026

Figure 14-35: Ag Swath plots from the 02AuEQ_NB_SME estimation domain (5x5x5m block model)



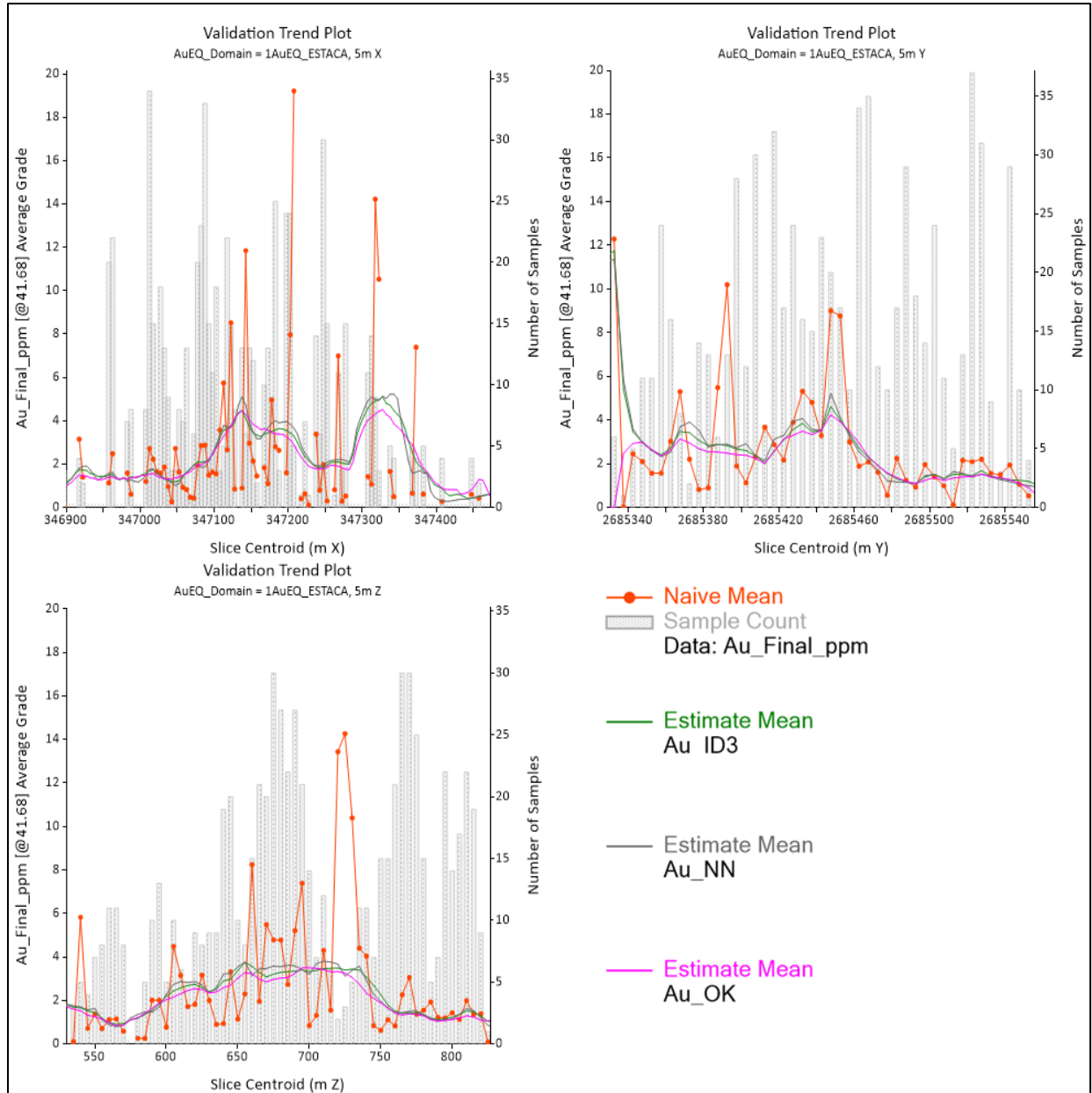
Source: Torex, 2026

Figure 14-36: Ag Swath plots from the 02AuEQ_TA estimation domain (5x5x5m block model)



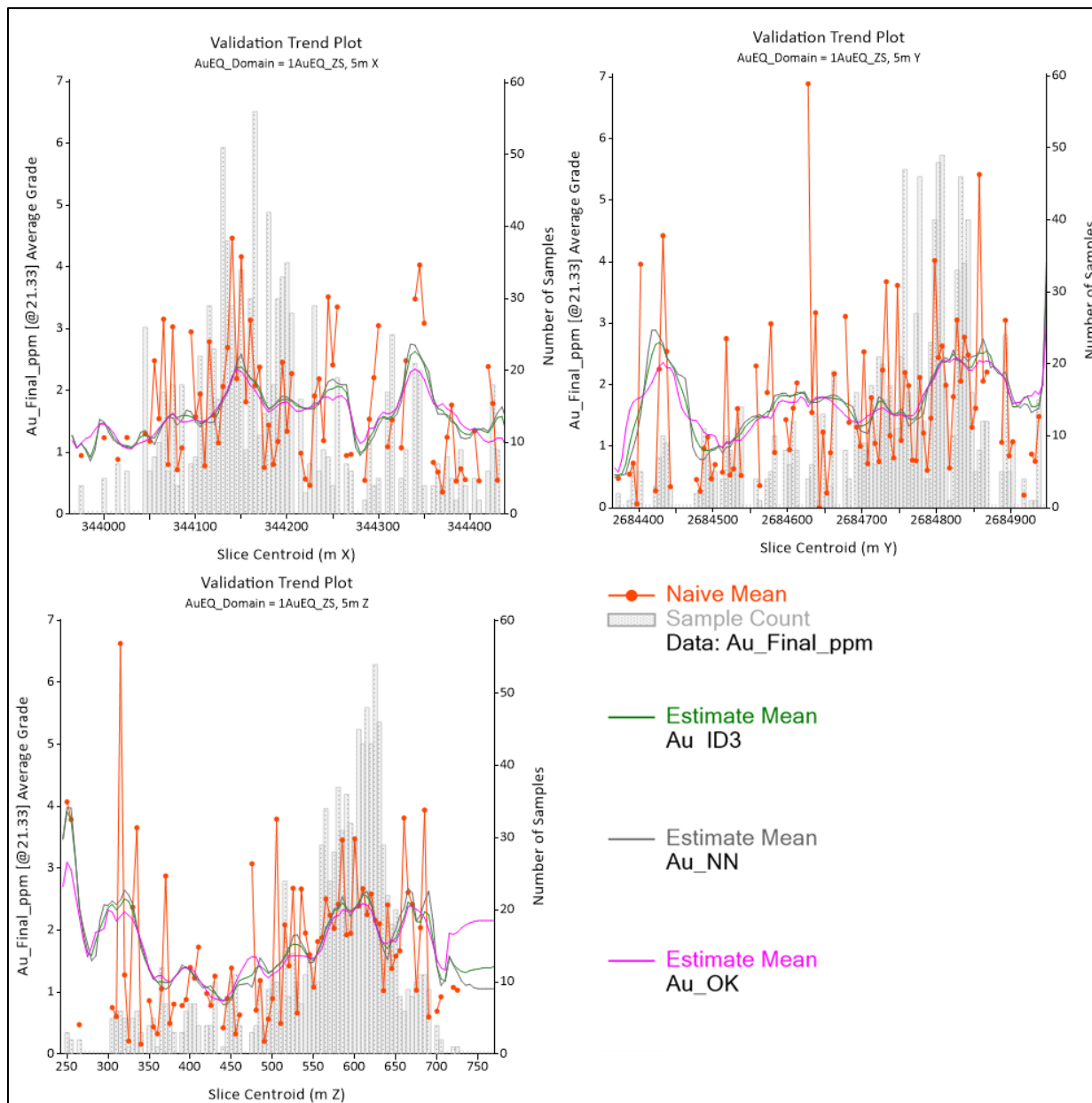
Source: Torex, 2026

Figure 14-37: Au Swath plots from the 1AuEQ_ESTACA estimation domain (2.5x2.5x2.5m block model)



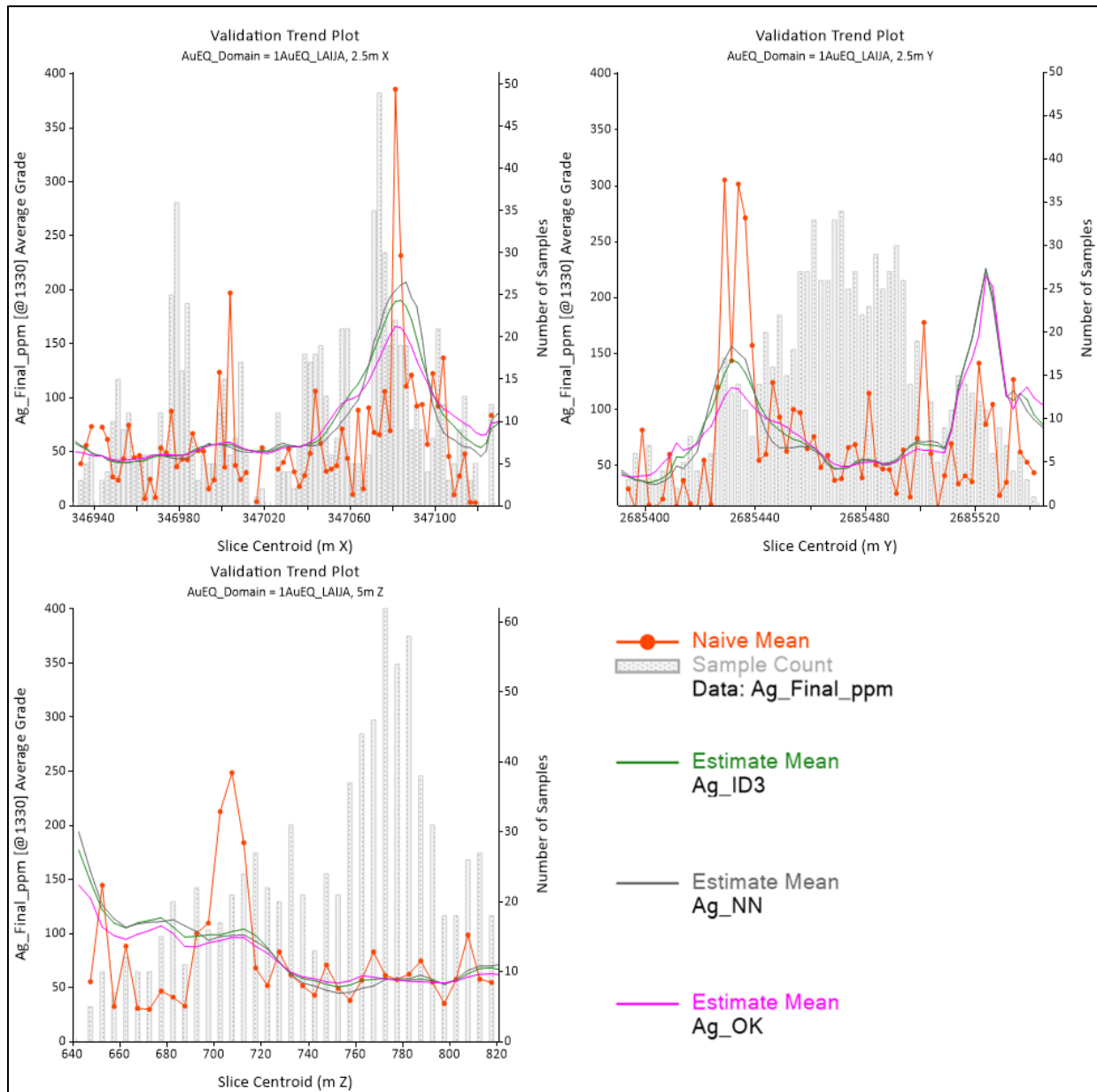
Source: Torex, 2026

Figure 14-38: Au Swath plots from the 1AuEQ_ZS estimation domain (2.5x2.5x2.5m block model)



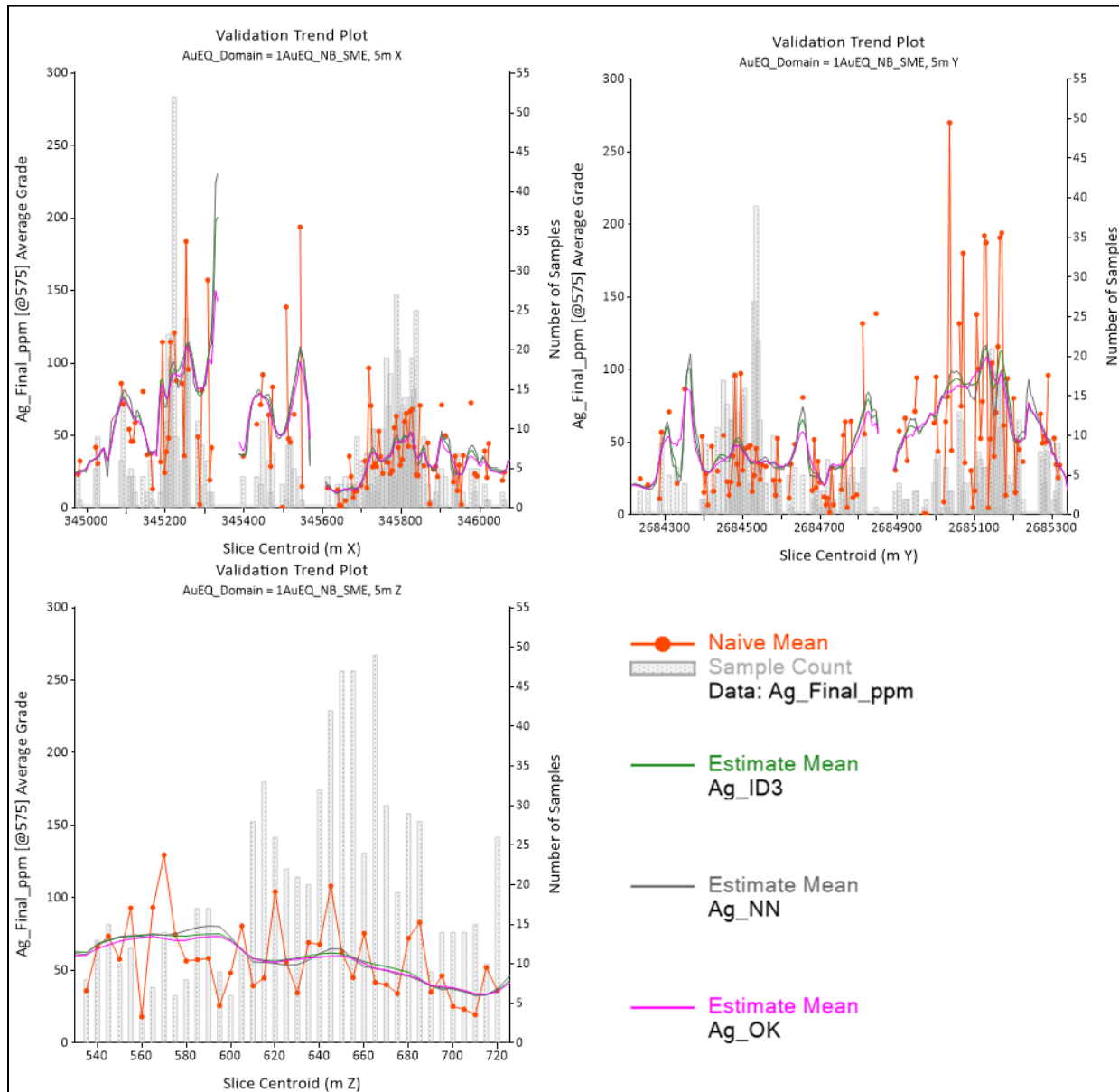
Source: Torex, 2026

Figure 14-39: Ag Swath plots from the 1AuEQ_LAIJA estimation domain (2.5x2.5x2.5m block model)



Source: Torex, 2026

Figure 14-40: Ag Swath plots from the 1AuEQ_NB_SME estimation domain (2.5x2.5x2.5m block model)

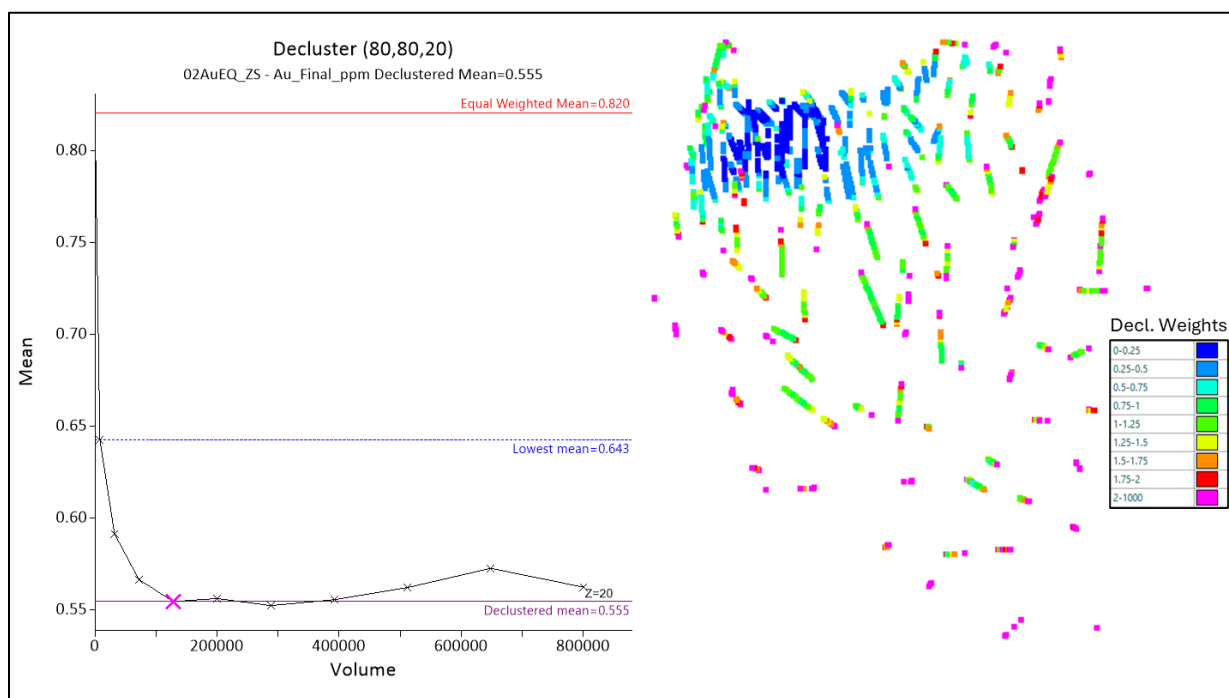


Source: Torex, 2026

14.3.10.4 Sequential Gaussian Simulation

Sequential Gaussian Simulation (SGS) was completed for mineralized estimation domains in the 5x5x5m set of block models to provide a range of possible grade-tonnage scenarios, to refine OK and ID3 estimation parameters, and to aid in selecting the final estimation method for resource reporting. The simulations were completed using declustered, normal scores transformed data and normal scores variograms for each domain, with simple kriging selected as the estimator. An example showing cell size selection for declustering is shown in Figure 14-43, and the full set of simulation parameters for Au and Ag are presented in Table 14-27 and Table 14-28. Simulation results were checked to ensure a mean estimated normal score value close to 0 and a variance close to 1 were achieved in each domain prior to use in validation. Simulation was not completed for domains in the 2.5x2.5x2.5m block models due to software limitations related to models with very large numbers of cells.

Figure 14-41: Cell declustering and weights for the 02AuEQ_ZS estimation domain (5x5x5m block model)



Source: Torex, 2026

Comparison of the ID3 and OK estimates to SGS grade-tonnage envelopes demonstrates that the ID3 estimate tends to fall between the 5th and 95th ranked simulations (p5-p95) for both grade and tonnes (Figure 14-28 and Figure 14-29). The OK estimates, however, tend to show lower grades than the p5 simulations and higher tonnes than the p95 simulations. This suggests that OK produces an over-smoothed result, and ID3 was therefore selected as the grade variable for final reporting purposes for both Au and Ag.

Table 14-27: Sequential Gaussian Simulation parameters for Au in mineralized estimation domains (5x5x5m block models)

Domain2	Kriging Type	Number of Simulations	Block Size			Points Per Block			Search (m)1			Assign Data to Node?	Min. Samples	Max. Samples	Max. previously simulated nodes
			X	Y	Z	X	Y	Z	Major	Semi-Major	Minor				
1AuEQ_ESTACA	Simple	100	5	5	5	2	2	2	120	90	20	N	1	40	20
1AuEQ_FRE	Simple	100	5	5	5	2	2	2	120	90	20	N	1	40	20
1AuEQ_GW	Simple	100	5	5	5	2	2	2	120	90	20	N	1	40	20
1AuEQ_LAIJA	Simple	100	5	5	5	2	2	2	120	90	20	N	1	40	20
1AuEQ_LP	Simple	100	5	5	5	2	2	2	120	90	20	N	1	40	20
1AuEQ_MAR	Simple	100	5	5	5	2	2	2	120	90	20	N	1	40	20
1AuEQ_NB_SME	Simple	100	5	5	5	2	2	2	120	90	20	N	1	40	20
1AuEQ_SM_SN	Simple	100	5	5	5	2	2	2	80	80	20	N	1	40	20
1AuEQ_SME_SPLAYS	Simple	100	5	5	5	2	2	2	80	80	20	N	1	40	20
1AuEQ_SMW	Simple	100	5	5	5	2	2	2	80	80	15	N	1	40	20
1AuEQ_TA	Simple	100	5	5	5	2	2	2	120	90	20	N	1	40	20
1AuEQ_ZN	Simple	100	5	5	5	2	2	2	120	90	20	N	1	40	20
1AuEQ_ZS	Simple	100	5	5	5	2	2	2	120	90	20	N	1	40	20

Notes:

1. Search directions for each domain are taken from Table 14-18, with search dimensions set equal to that of the second search pass used in estimation
2. Simulations were completed considering Indicated and Inferred resource categories only

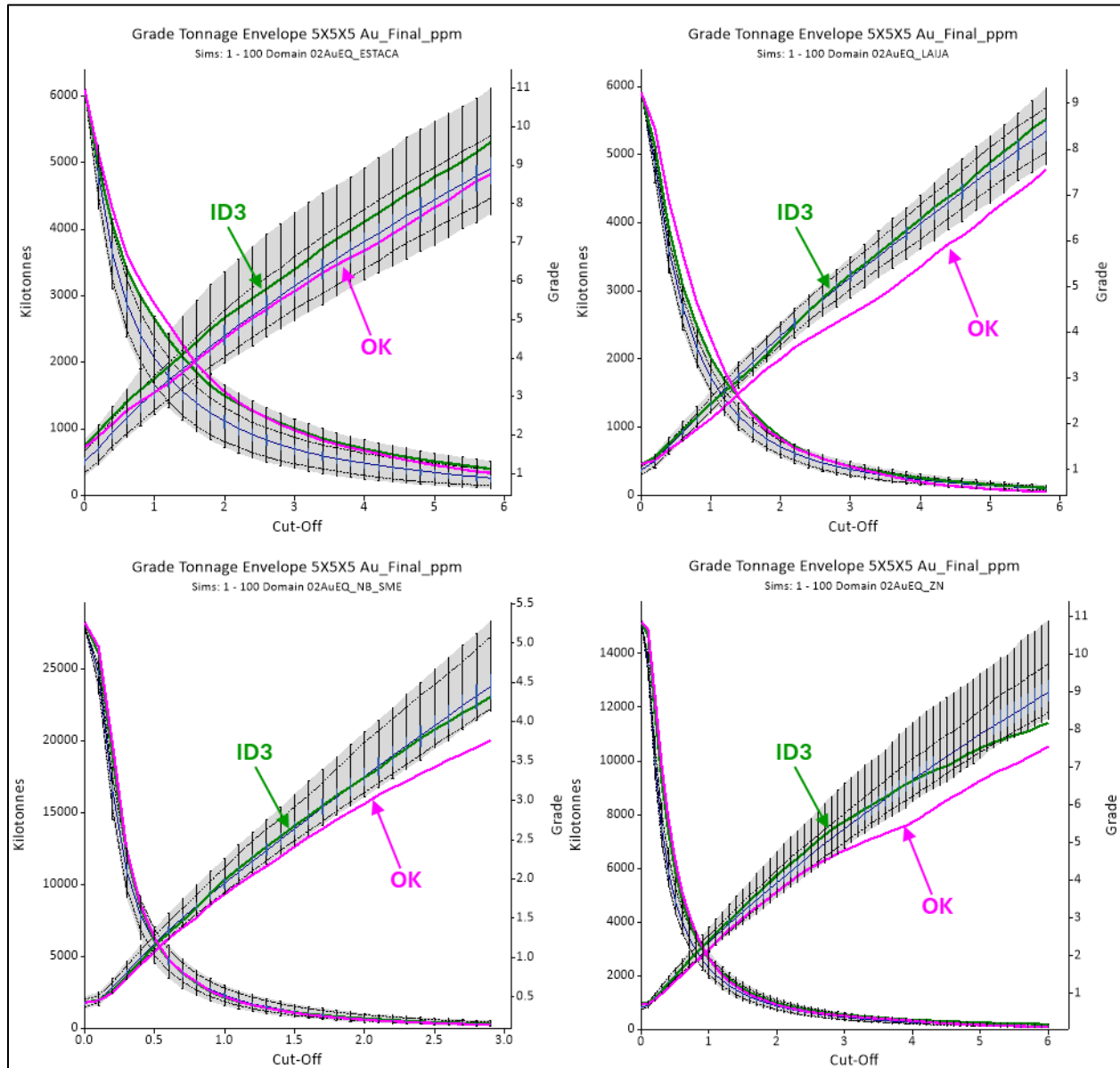
Table 14-28: Sequential Gaussian Simulation parameters for Ag in mineralized estimation domains (5x5x5m block models)

Domain2	Kriging Type	Number of Simulations	Block Size			Points Per Block			Search (m)1			Assign Data to Node?	Min. Samples	Max. Samples	Max. previously simulated nodes
			X	Y	Z	X	Y	Z	Major	Semi-Major	Minor				
1AuEQ_ESTACA	Simple	100	5	5	5	2	2	2	120	90	20	N	1	40	20
1AuEQ_FRE	Simple	100	5	5	5	2	2	2	120	90	20	N	1	40	20
1AuEQ_GW	Simple	100	5	5	5	2	2	2	120	90	20	N	1	40	20
1AuEQ_LAIJA	Simple	100	5	5	5	2	2	2	120	90	20	N	1	40	20
1AuEQ_LP	Simple	100	5	5	5	2	2	2	120	90	20	N	1	40	20
1AuEQ_MAR	Simple	100	5	5	5	2	2	2	120	90	20	N	1	40	20
1AuEQ_NB_SME	Simple	100	5	5	5	2	2	2	120	90	20	N	1	40	20
1AuEQ_SM_SN	Simple	100	5	5	5	2	2	2	80	80	20	N	1	40	20
1AuEQ_SME_SPLAYS	Simple	100	5	5	5	2	2	2	80	80	20	N	1	40	20
1AuEQ_SMW	Simple	100	5	5	5	2	2	2	80	80	15	N	1	40	20
1AuEQ_TA	Simple	100	5	5	5	2	2	2	120	90	20	N	1	40	20
1AuEQ_ZN	Simple	100	5	5	5	2	2	2	120	90	20	N	1	40	20
1AuEQ_ZS	Simple	100	5	5	5	2	2	2	120	90	20	N	1	40	20

Notes:

1. Search directions for each domain are taken from Table 14-19, with search dimensions set equal to that of the second search pass used in estimation
2. Simulations were completed considering Indicated and Inferred resource categories only

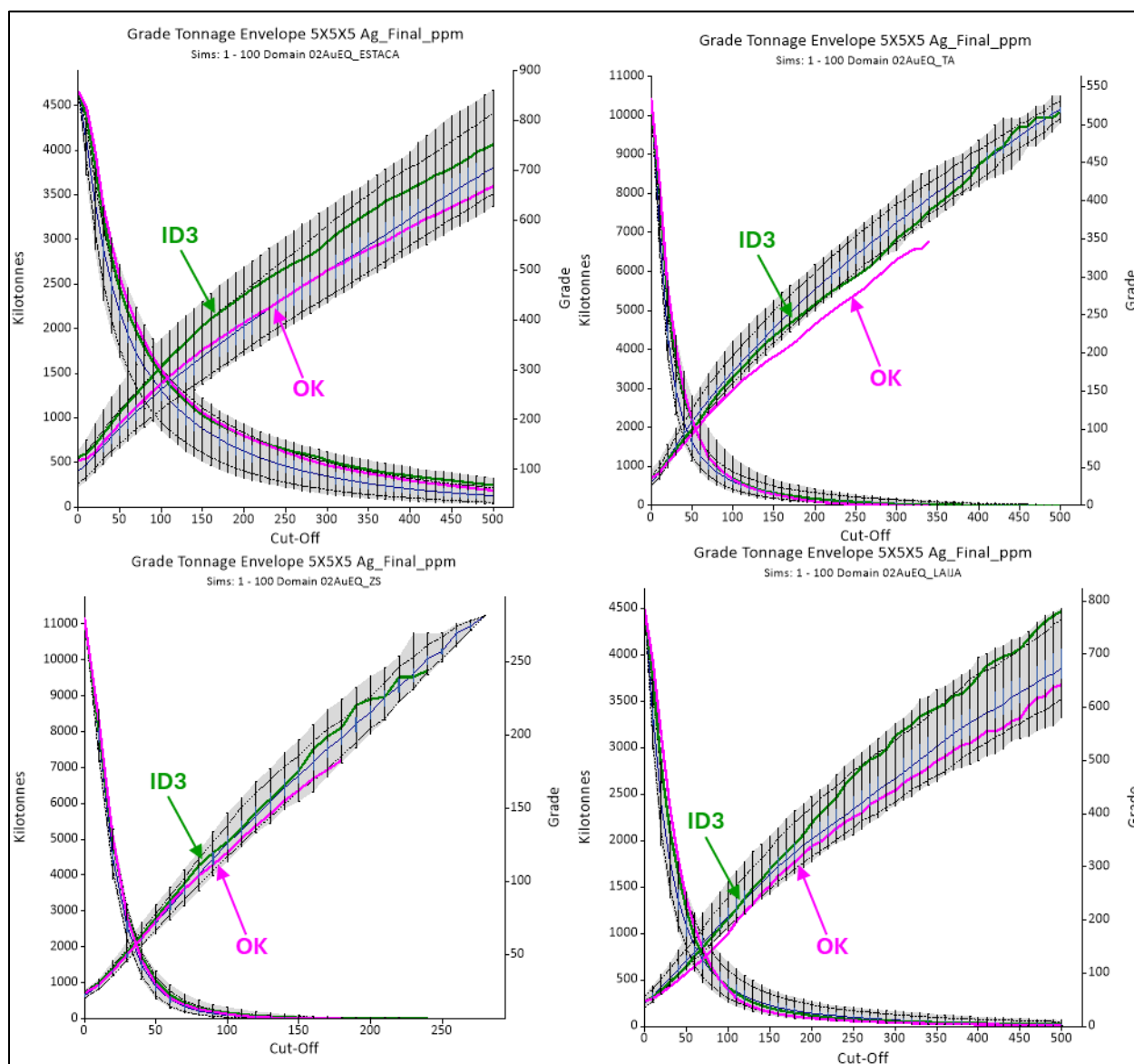
Figure 14-42: Grade-tonnage curve comparison between ID3, OK, and SGS for Au in select mineralized domains



Source: Torex, 2026

Notes: The median (p50) simulation is represented by the solid blue line in the centre of the grade-tonnage envelope. Dashed lines represent the p5 and p95 simulations

Figure 14-43: Grade-tonnage curve comparison between ID3, OK, and SGS for Ag in select mineralized domains



Notes: The median (p50) simulation is represented by the solid blue line in the centre of the grade-tonnage envelope. Dashed lines represent the p5 and p95 simulations. Source: Torex, 2026

14.3.11 Assessment of Reasonable Prospects for Eventual Economic Extraction

Two sets of optimization were completed to support the assessment of reasonable prospects for eventual economic extraction of open pit and underground Resources – (1) Pit optimization using the 5x5x5m set of block models (Table 14-16), (2) Underground stope optimization with Mineable Shape Optimizer (MSO) using the 2.5x2.5x2.5m set of block models (Table 14-17). Each is described in detail below.

14.3.11.1 Open Pit Optimization – 5x5x5m Block Models

Open pit optimization was prepared using Datamine Studio NPVS, a strategic mine planning software package that generates an optimized pit shell based on economic input parameters and overall slope angles using the Hochbaum Pseudoflow algorithm. The optimization considers blocks of Indicated and Inferred assurance categories only. The density of the historically mined portion blocks was assumed to be 0.0 g/cm³. Selected pits were computed using the Net Smelter Return (NSR) cutoff, which were subsequently filtered to include blocks with grades above the 0.17 g/t gold-only cutoff as stated (or other cutoff grade sensitivities using the same methodology as shown in Table 14-29).

Table 14-29: Pit optimization parameters

	Parameter	Unit	Value
Mining	Waste mining cost	\$/tonne	2.00
	Ore mining cost ¹	\$/tonne	2.50
	Mining loss	%	5
	Mining dilution	%	5
	Slope angle	degrees	42-47 (by zone)
Processing	Heap leach recoveries	%	73% Au, 25% Ag
	Mill recoveries	%	94.8% Au, 81.1% Ag
	Heap leach cost	\$/tonne (ore)	5.53
	Mill cost	\$/tonne (ore)	16.81
	General and administration (G&A)	\$/tonne (ore)	2.00
Selling	Price ²	\$/oz	1,950 Au, 25.24 Ag
	Selling cost	%	1 (on both Au and Ag)
	Royalty	%	0.00

Notes:

- Ore mining cost allows for additional haulage distance**
- Base selling prices of gold and silver are based on three-year trailing averages**
- All costs in United States Dollars (USD)**
- Unless otherwise stated, open pit Resources declared in this report are reported at a gold-only cutoff grade of greater than or equal to 0.17 g/t**

14.3.11.2 Underground Optimization – 2.5x2.5x2.5m Block Models

Underground optimization was prepared using Datamine Studio's Mineable Shape Optimizer (MSO), a strategic mine planning software package used to generate the optimal shape and location of stopes for underground mine design using an input of block model grades or values, economic input assumptions, and stope design parameters. The optimization considers blocks of Indicated and Inferred assurance categories only and assumes a longhole open stope mining method. Blocks which touch or are within historically mined workings were not considered by MSO. Optimization parameters are listed in Table 14-30.

Table 14-30: Mineable Stope Optimization (MSO) parameters

	Parameter	Unit	Value
Longhole Open Stope Mining	Ore mining cost ¹	\$/tonne	50.00
	Minimum stope width	metres	3.0
	Stope height	metres	15.0
	Stope length	metres	15.0
	Dilution	-	In design
	Mining Recovery	%	95.0
Processing	Heap leach recoveries	-	N/A
	Mill recoveries	%	94.8% Au, 81.1% Ag
	Heap leach cost	-	N/A
	Mill cost	\$/tonne (ore)	16.81
	General and administration (G&A)	\$/tonne (ore)	4.50
Selling	Price ²	\$/oz	1,950 Au, 25.24 Ag
	Selling cost	%	1 (on both Au and Ag)
	Royalty	%	0.00

Notes:

1. Assumes mechanized longhole open stope mining
2. Base selling prices of gold and silver are based on three-year trailing averages
3. All costs in United States Dollars (USD)

14.3.11.3 Open Pit and Underground Indicated and Inferred Resources

Indicated and Inferred Resources for Los Reyes are reported using a combination of economically constrained open pits, underground stopes, and residual open pits generated using the methods described above. Resources for each individual deposit are either sourced from - (1) Pit-constrained Resources only, (2) Underground MSO-constrained Resources only, or (3) Pit-constrained Resources and underground MSO-constrained Resources contained in stopes outside the pit volumes. The final reported Indicated and Inferred Resources and constituent components for each individual deposit are shown in Table 14-31 and Table 14-32 and Figure 14-47 through Figure 14-48.

Table 14-31: Mineral Resource Statement by mining method and process stream

Mining Method and Process	Class	Tonnage (kt)	Au (g/t)	Au (koz)	Ag (g/t)	Ag (koz)	AuEq (g/t)	AuEq (koz)
Open Pit Plant	Measured	-	-	-	-	-	-	-
	Indicated	26,448	1.13	961	30.8	26,192	1.47	1,251
	Measured and Indicated	26,448	1.13	961	30.8	26,192	1.47	1,251
	Inferred	8,056	0.93	240	34.4	8,917	1.31	339
Underground	Measured	-	-	-	-	-	-	-
Plant	Indicated	5,361	2.48	428	129.0	22,231	3.91	674
	Measured and Indicated	5,361	2.48	428	129.0	22,231	3.91	674
	Inferred	5,430	1.84	322	72.7	12,685	2.65	463
Total Plant	Measured	-	-	-	-	-	-	-
	Indicated	31,809	1.36	1,388	47.3	48,423	1.88	1,925
	Measured and Indicated	31,809	1.36	1,388	47.3	48,423	1.88	1,925
	Inferred	13,487	1.30	562	49.8	21,602	1.85	801
Open Pit -	Measured	-	-	-	-	-	-	-
Heap Leach	Indicated	19,441	0.25	154	11.9	7,437	0.30	187
	Measured and Indicated	19,441	0.25	154	11.9	7,437	0.30	187
	Inferred	7,413	0.24	57	12.6	3,013	0.30	71
Total	Measured	-	-	-	-	-	-	-
	Indicated	51,250	0.94	1,542	33.9	55,860	1.28	2,111
	Measured and Indicated	51,250	0.94	1,542	33.9	55,860	1.28	2,111
	Inferred	20,899	0.92	620	36.6	24,615	1.30	872

Notes:

1. Mineral resources were prepared in accordance with the CIM Definition Standards (2014) and Estimation of Mineral Resource and Mineral Reserve Best Practice guidelines (2019).
2. Mineral resources are inclusive of mineral reserves. Mineral resources that are not mineral reserves do not have demonstrated economic viability.
3. Numbers may not add due to rounding.
4. Mineral resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.
5. The effective date of the estimate is April 25, 2026.
6. Mineral resources for Los Reyes are based on underlying metal prices of \$1,950/oz Au and \$25.24/oz Ag, unless otherwise noted.
7. The preparation of the mineral resource estimate was supervised by John Sims, President of Sims Resources LLC, an independent contractor and QP as a CPG member with AIPG.
8. Los Reyes – Open Pit (Plant and Heap Leach):
 - a. Open Pit Resource estimates are based on economically constrained revenue factor one (RF1) open pits generated using the Hochbaum Pseudoflow algorithm in Datamine's Studio NPVS and the following optimization parameters:
 - b. Assumes plant recoveries of 94.8% for Au and 81.1% for Ag and heap leach recoveries of 73% Au and 25% Ag.
 - c. Pit slopes by area ranging from 42-47 degrees overall slope angle.

- d. 5% mineralized material loss and 5% dilution factor applied to the 5 m x 5 m x 5 m open pit resource block models.
 - e. Mining costs of \$2.00 per tonne of waste mined and \$2.50 per tonne of mined material. Processing costs of \$16.81 per tonne processed. Heap leach costs of \$5.53 per tonne processed. G&A cost of \$2.00 per tonne of material processed.
 - f. A 0.17 g/t gold only cutoff was applied to ex-pit processed material (which is above the heap-leaching NSR cutoff).
 - g. $\text{AuEq (Open Pit Plant)} = \text{Au (g/t)} + (\text{Ag (g/t)} * 0.0111)$ and $\text{AuEq (Open Pit Heap Leach)} = \text{Au (g/t)} + (\text{Ag (g/t)} * 0.0044)$.
9. Los Reyes – Undergr/ound (Plant):
- a. Underground Resource estimates are based on economically constrained stopes generated using Datamine's Mineable Shape Optimizer (MSO) algorithm and the following optimization parameters:
 - b. Diluted to a minimum 3 m stope width with a 95% mining recovery.
 - c. Assumes plant recoveries of 94.8% for Au and 81.1% for Ag.
 - d. Longhole Open Stope mining with a \$50.00 per tonne cost. Processing costs of \$17.24 per tonne processed. G&A cost of \$4.50 per tonne of material processed.
 - e. $\text{AuEq (Underground Plant)} = \text{Au (g/t)} + (\text{Ag (g/t)} * 0.0111)$.

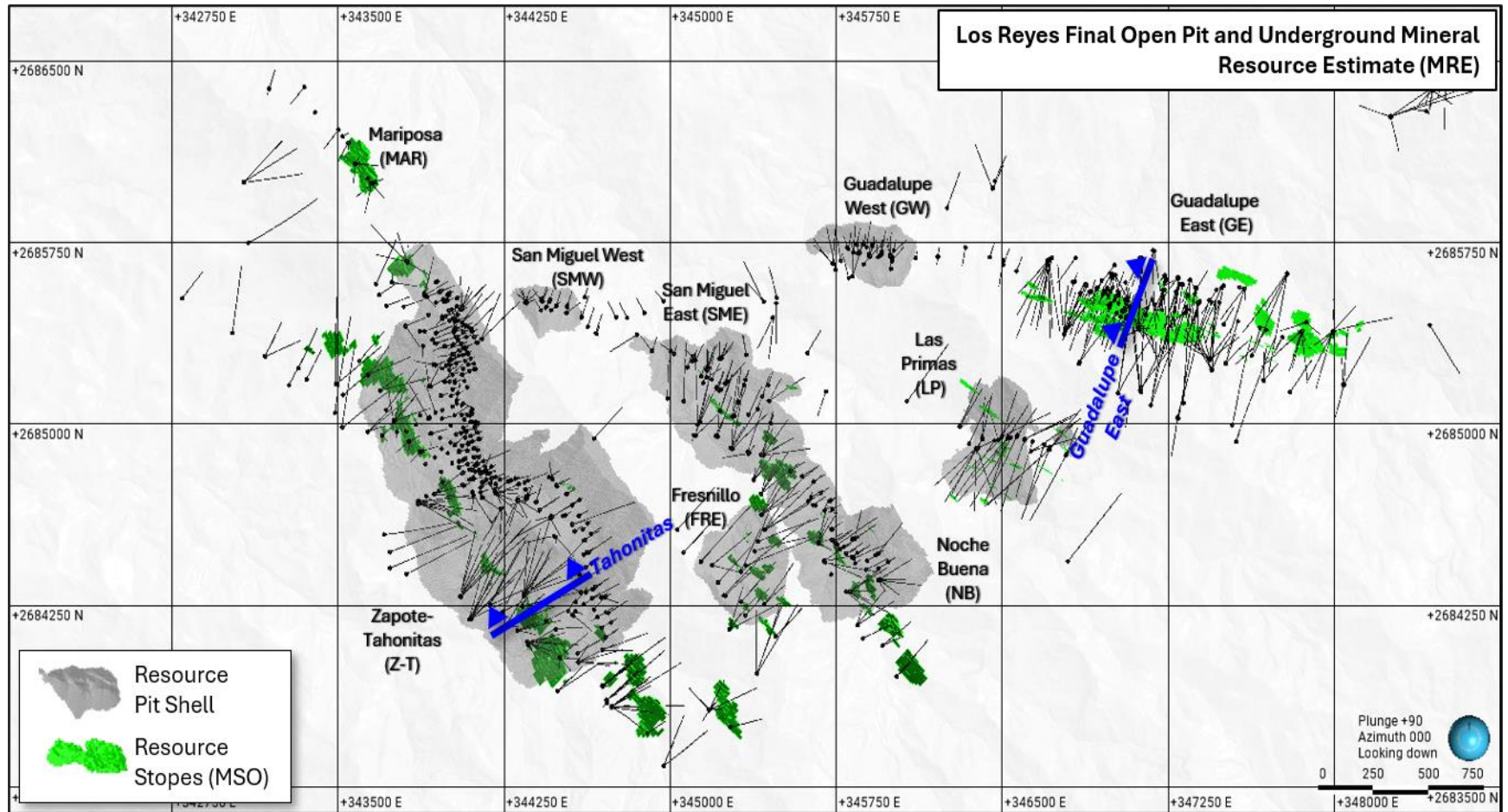
Table 14-32: Mineral Resource Statement by Mining Method and Area

Area	Mining Method	Class	Tonnage (kt)	Au (g/t)	Au (koz)	Ag (g/t)	Ag (koz)	AuEq (g/t)	AuEq (koz)
ZT Trend	Open Pit	Indicated	30,418	0.77	757	21.4	20,897	0.98	954
		Inferred	9,428	0.64	195	29.5	8,942	0.92	279
	Underground	Indicated	5	1.25	0	13.4	2	1.40	0
		Inferred	2,263	1.76	128	65.5	4,766	2.48	181
Guadalupe Trend	Open Pit	Indicated	1,944	0.67	42	21.3	1,333	0.87	54
		Inferred	228	0.32	2	18.6	136	0.46	3
	Underground	Indicated	5,247	2.49	420	130.8	22,060	3.94	664
		Inferred	1,231	2.16	86	128.1	5,072	3.58	142
Central Trend	Open Pit	Indicated	11,798	0.75	285	27.9	10,568	1.03	390
		Inferred	3,348	0.49	52	19.4	2,091	0.68	73
	Underground	Indicated	58	2.76	5	46.6	87	3.27	6
		Inferred	722	1.19	28	69.5	1,614	1.96	46
Generative Areas	Open Pit	Indicated	1,729	0.56	31	14.9	831	0.71	39
		Inferred	2,464	0.60	48	9.6	761	0.69	55
	Underground	Indicated	51	1.50	2	49.9	82	2.05	3
		Inferred	1,213	2.08	81	31.6	1,232	2.43	95
Total	Open Pit	Indicated	45,890	0.76	1,115	22.8	33,629	0.97	1,437
		Inferred	15,469	0.60	298	24.0	11,930	0.82	410
	Underground	Indicated	5,361	2.48	428	129.0	22,231	3.91	674
		Inferred	5,430	1.84	322	72.7	12,685	2.65	463
	Total	Indicated	51,250	0.94	1,542	33.9	55,860	1.28	2,111
		Inferred	20,899	0.92	620	36.6	24,615	1.30	872

Table 14-33: Description of Resource Components for Tables 14-31 and 14-32, and Figure 14-46

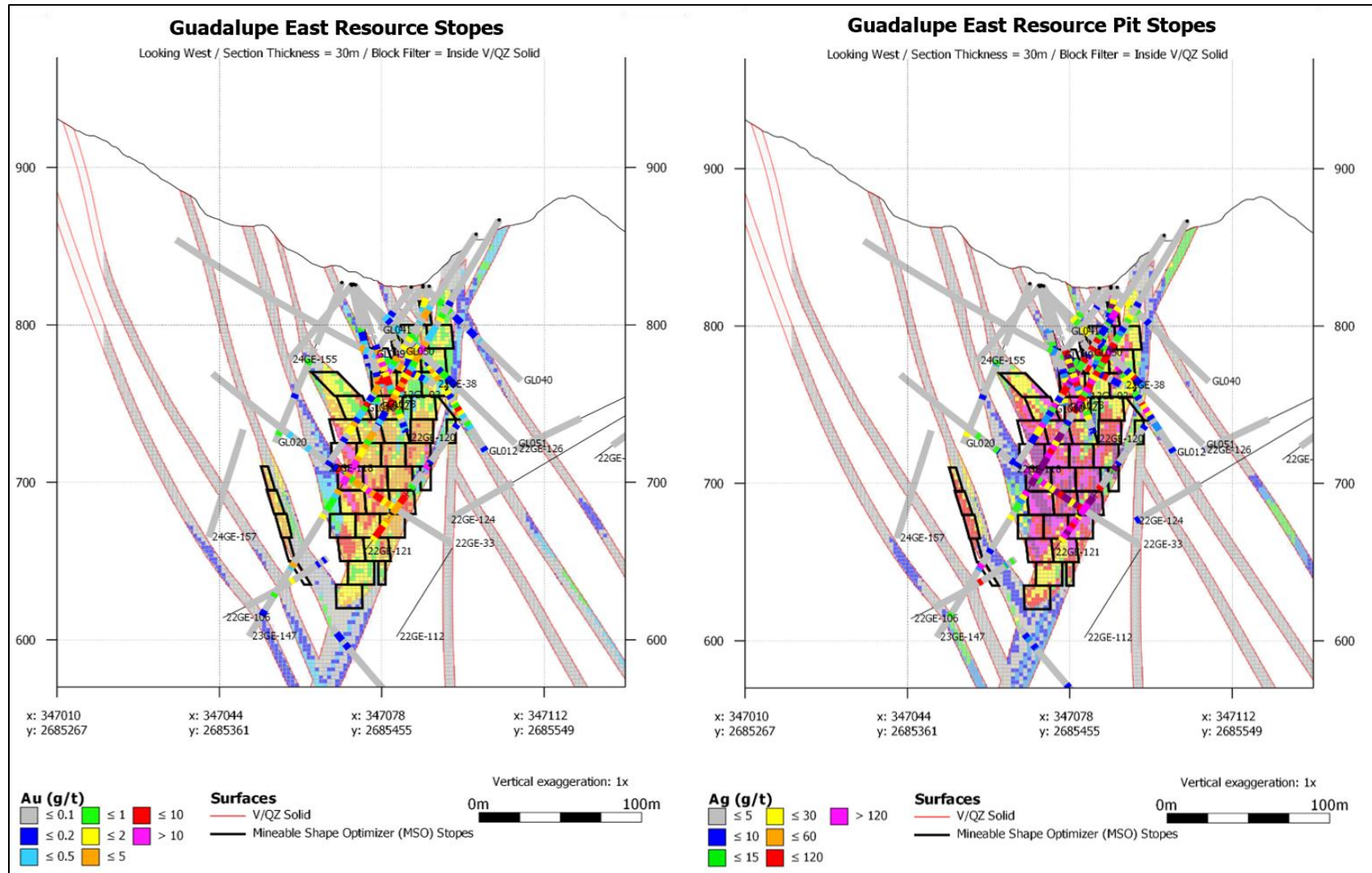
Area	Deposit	Resource Description
Z-T Trend	Zapote-Tahonitas	Revenue Factor 1.0 Open Pits @0.17g/T Au only cutoff plus remaining economic stopes
Guadalupe Trend	Guadalupe East	Economic underground stopes
	Guadalupe West	Revenue Factor 1.0 Open Pits @0.17g/T Au only cutoff
Central Trend	Noche Buena	Revenue Factor 1.0 Open Pits @0.17g/T Au only cutoff plus remaining economic stopes
	San Miguel East	Revenue Factor 1.0 Open Pits @0.17g/T Au only cutoff plus remaining economic stopes
	San Miguel West	Revenue Factor 1.0 Open Pits @0.17g/T Au only cutoff
Generative Areas	Mariposa	Economic underground stopes
	Las Primas	Revenue Factor 1.0 Open Pits @0.17g/T Au only cutoff plus remaining economic stopes
	Fresnillo	Revenue Factor 1.0 Open Pits @0.17g/T Au only cutoff plus remaining economic stopes

Figure 14-44: Plan view demonstrating final open pit and underground Resource solids for Los Reyes. Section Lines for Figures 14-47 and 14-48 also shown



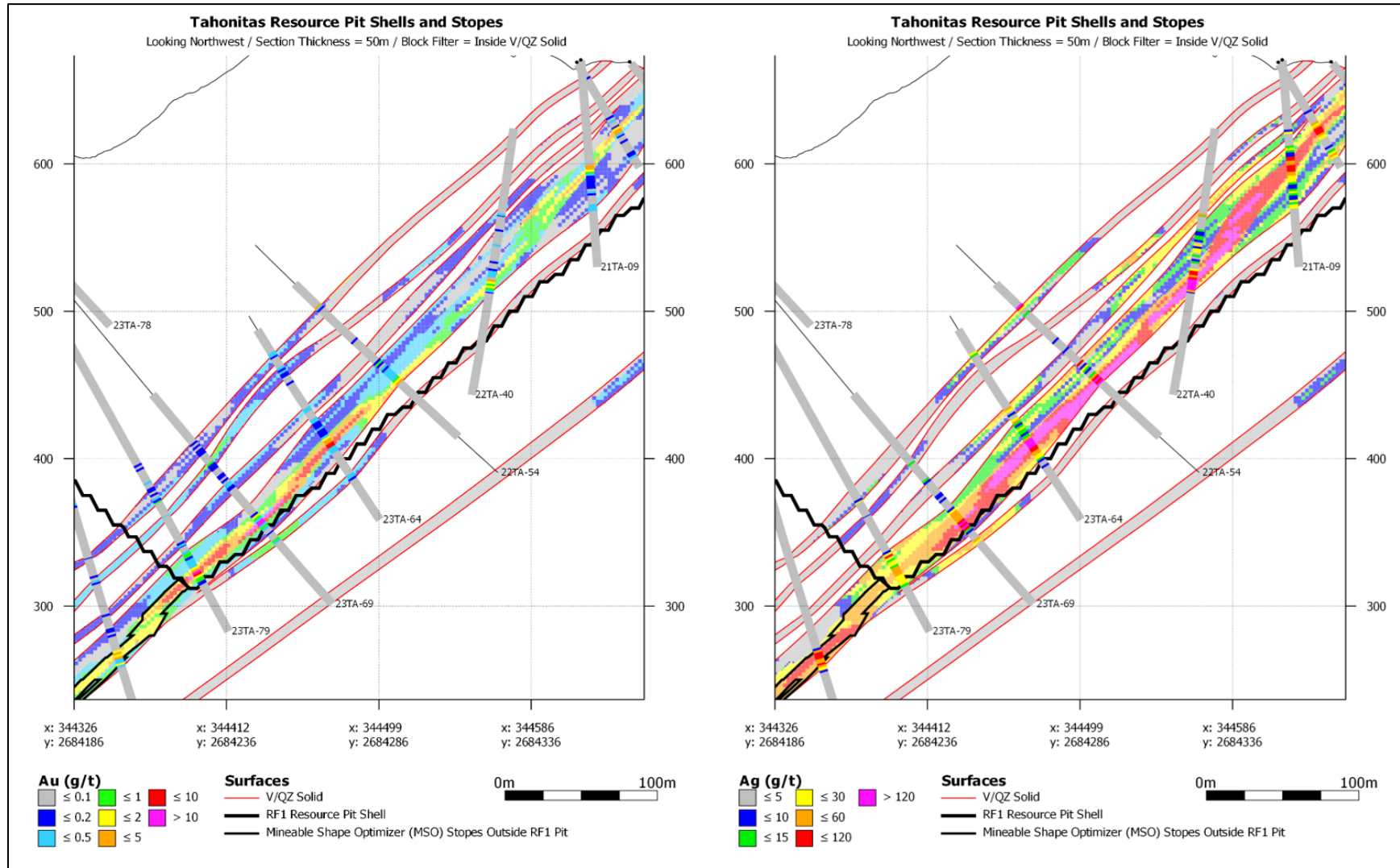
Source: Torex, 2026

Figure 14-45: Cross section showing underground and open pit gold (left) and silver (right) Resources at Guadalupe East



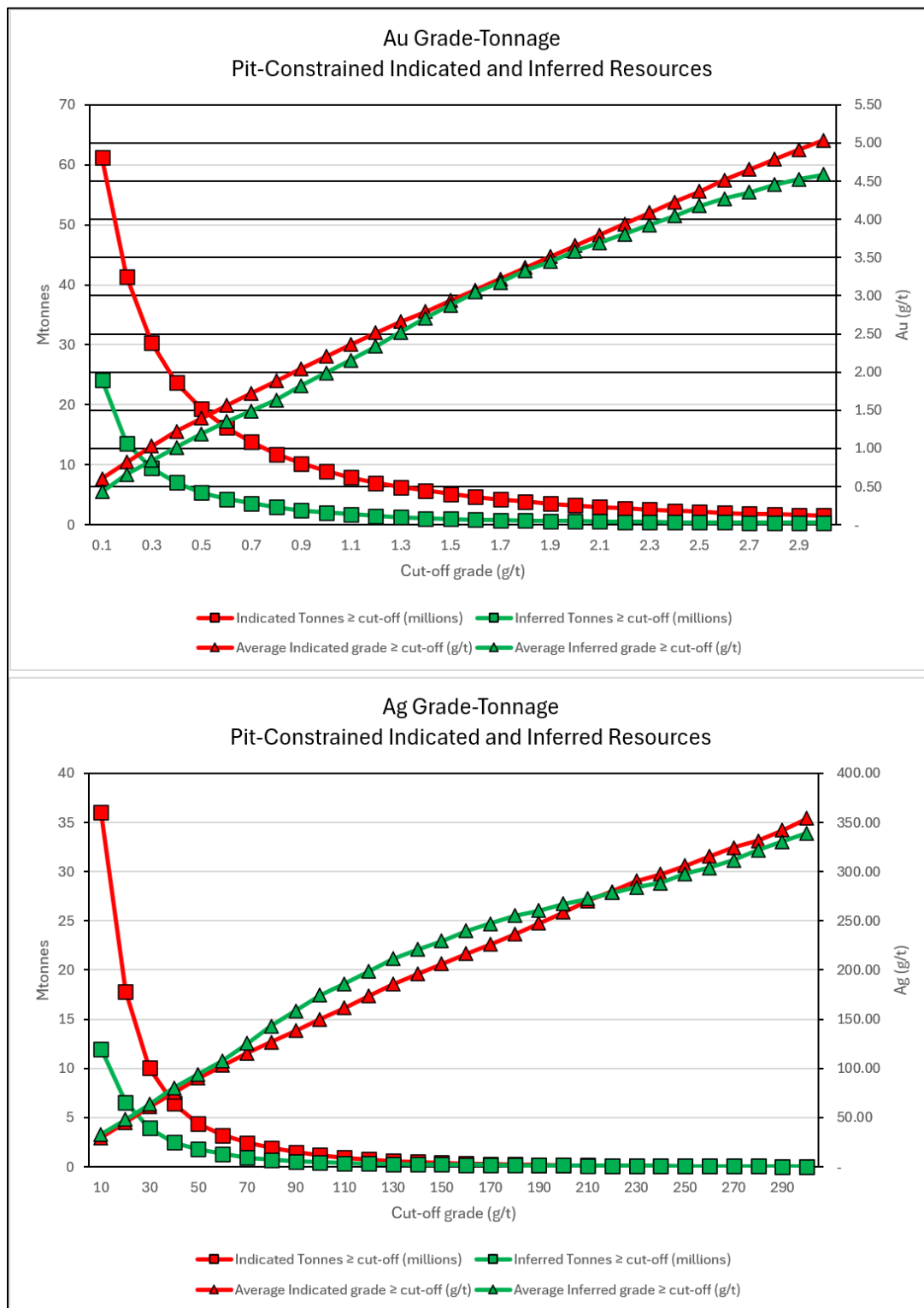
Source: Torex, 2026

Figure 14-46: Cross section showing open pit and underground gold (left) and silver (right) Resources at Tahonitas



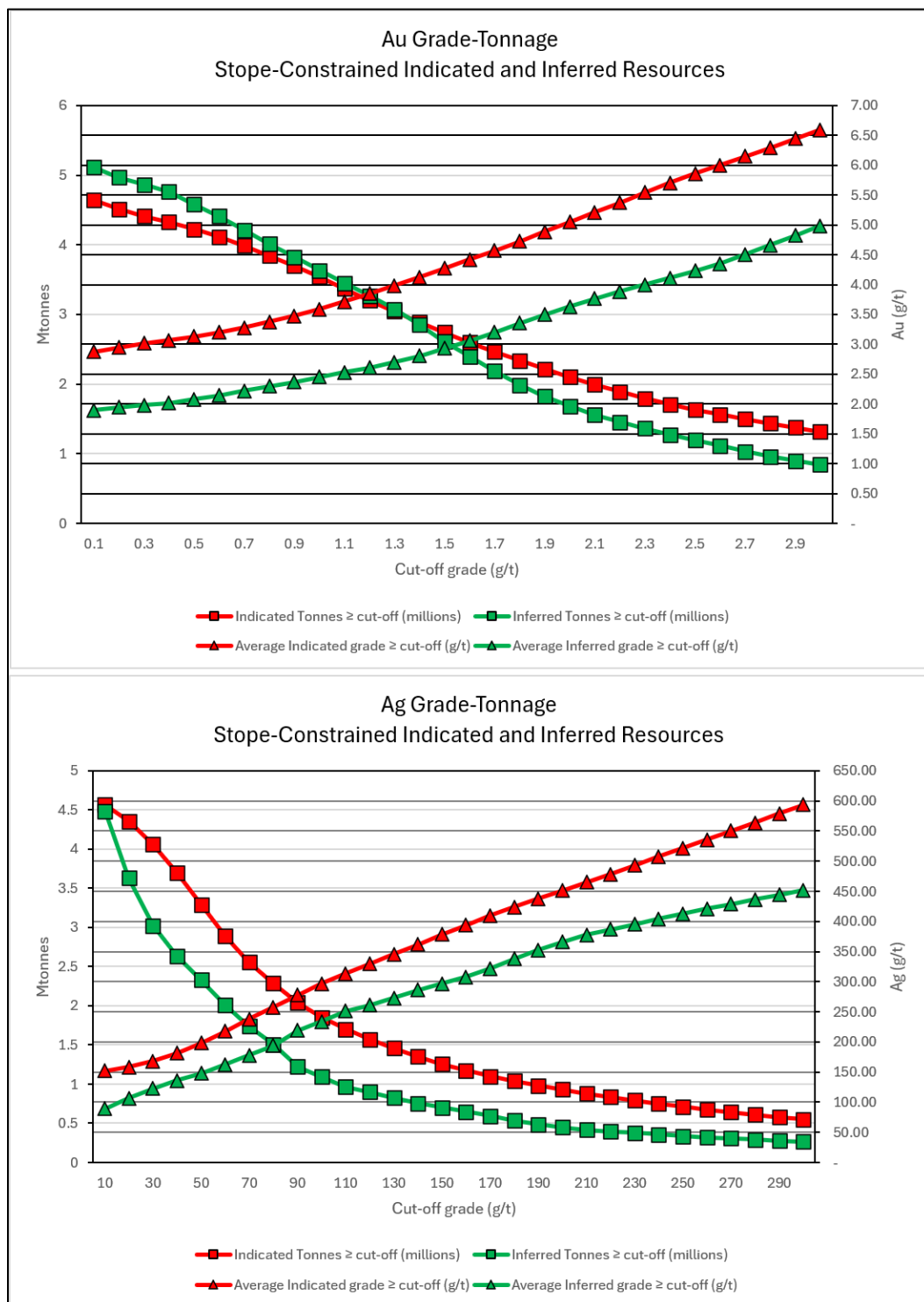
Source: Torex, 2026

Figure 14-47: Grade-tonnage curves for Los Reyes pit-constrained Indicated and Inferred Resources at various gold and silver cutoff grades



Source: Torex, 2026

Figure 14-48: Grade-tonnage curves for Los Reyes stope-constrained Indicated and Inferred Resources at various gold and silver cutoff grades



Source: Torex, 2026

14.3.12 Comparison to Previous Resource Estimates

The Mineral Resource Estimate (MRE) detailed in this report employs a similar estimation strategy to that used in the previous MRE disclosed by Torex in the Year-End 2025 Reserves & Resources Report for the Los Reyes Project. Minor updates to the geological model and estimation domains were completed based on an additional 20,175m drilled in 60 core holes between the previous MRE drilling cutoff date of July 17, 2024, and the current MRE drilling cutoff of December 31, 2024. Estimation strategy, including topcut selection, final interpolation methodology, density modeling, and Resource classification drillhole spacing thresholds are otherwise unchanged from the previous estimation. Key changes relative to the Year-End 2025 Mineral Resource Estimate are outlined below:

1. Dataset – The dataset used for this resource estimate includes an additional 20,175m drilled in 60 core holes relative to the October 2024 MRE. Targets drilled include Tahonitas, Noche Buena, Fresnillo, Mariposa, and Guadalupe East.
2. Change in assumed underground mining method – Underground Resources in the October 2024 estimate were reported from stopes generated assuming mechanized cut and fill mining. In this Mineral Resource Estimate, underground Resources are reported from stopes generated assuming longhole open stoping. The change to a less-selective and lower-cost mining method, along with the additional drilling completed since the 2024 MRE, has resulted in the following changes in Underground Resources relative to the previous resource estimate - (1) An increase of 26 Koz Au (+6.4%) and 1,989 Koz Ag (+9.8%), a decrease in Au grade of 0.54 g/t (-18.0%), and a decrease in Ag grade of 23.38 g/t (-15.3%) in the Indicated Resource category, (2) An increase of 49 Koz Au (+17.8%) and 2,438 Koz Ag (+23.8%), a decrease in Au grade of 0.25 g/t (-12.1%), and a decrease in Ag grade of 5.95 g/t (-7.6%) in the Inferred Resource category.
3. Removal of 'residual' pit shells – The October 2024 Mineral Resource Estimate was reported using a combination of Revenue Factor 1 (RF1) pit shells, cut and fill stopes, and 'residual' pits. These residual pits assumed that any underground stopes were backfilled with zero-grade material at two-thirds of the original rock density, and economically constrained open pits were then generated with this mined-out, backfilled material flagged to the open pit SMU (5x5x5m) block models. Pit optimization for the residual pits otherwise assumed the same optimization parameters as used for RF1 pit optimization. The current MRE considers a combination of RF1 pit shells and underground stopes (Table 14-3) but does not include residual pit shells. This change in methodology, along with the additional drilling completed since the 2024 MRE, has resulted in the following changes in Open Pit Resources relative to the October, 2024 MRE - (1) An increase of 25 Koz Au (+2.3%) and an increase in Au grade of 0.02 g/t (+2.3%), a decrease of 124 Koz Ag (-0.4%), and a decrease in Ag grade of 0.09 g/t (-0.4%) in the Indicated Resource category, (2) An increase of 33 Koz Au (+12.4%) and 616 Koz Ag (+5.4%), an increase in Au grade of 0.08 g/t (+12.4%), and an increase in Ag grade of 1.46 g/t (+5.4%) in the Inferred Resource category.
4. The final resource estimate issued in this report, considering the updates and changes in methodology noted above, represents an increase of 51 Koz Au (+3.4%) and 1,864 Koz Ag (+3.4%) in the Indicated Resource category and an increase of 81 Koz Au (+15.1%) and 3,054 Koz Ag (14.1%) in the Inferred Resource category relative to the October, 2024 Mineral Resource Estimate for the Los Reyes Project.

14.3.13 Comment on Mineral Resources

John Sims, AIPG Certified Professional Geologist, has certified that, to the best of his professional judgment as a Qualified Person (as defined under NI 43-101), the Mineral Resource estimates have been prepared in accordance with NI 43-101, including the CIM Definition Standards incorporated by reference, and conform to generally accepted mining industry best practices. Mineral resources are not mineral reserves and there is no assurance that mineral resources will ultimately be classified as Proven or Probable reserves.

The mineral resource presented here should be accepted with the understanding that additional data and analysis available after the date of the estimates may necessitate revision. Potential risks that may impact the accuracy of the resource estimate include the following:

1. The geologic interpretation, modeling of geologic attributes such as lithology, faults, and mineralization controls, and the resulting resource estimates were prepared using the most accurate information available at the time this report was completed. However, additional drilling, data collection, and analysis may require revisions to wireframes, interpolation methodologies, density modeling, or other attributes which may impact future resource estimates.
2. Commodity price changes and capital and operating cost estimates could impact revenue and cost inputs used in the resource estimate, and overall economic interpretation of the viability of project study and development.
3. The accuracy of historical mine workings cannot be verified in cases where collapse or other unsafe conditions result in limited access to the excavations. Construction of the current depletion solids therefore relied on a combination of LiDAR-surveyed workings where available, voids encountered in surveyed drillholes, and historical production maps. While the work was completed to the highest level of accuracy possible with the current dataset, the position of some underground workings may be inaccurate. Other historical mine workings may also be present which have not been documented.
4. Future technical studies, including geotechnical and metallurgical, could result in revisions to pit slope angles, underground stope dimensions, and process recovery assumptions.

15 Mineral reserve estimates

This Technical Report is a Preliminary Economic Assessment (PEA) prepared in accordance with Canadian Securities Administrators' National Instrument 43-101 — Standards of Disclosure for Mineral Projects. No Mineral Reserves, as defined in NI 43-101 and the CIM Definition Standards on Mineral Resources and Mineral Reserves, are estimated or reported for the Project. The mine plan and economic analysis presented in this Technical Report are based on a mining inventory derived from Mineral Resources. The mining inventory includes Inferred Mineral Resources, which are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as Mineral Reserves. The reader is cautioned that the PEA is preliminary in nature, and that there is no certainty that the results of the PEA will be realised.

16 Mining methods

16.1 Introduction and design basis

16.1.1 Purpose and scope

This section presents the mining methods developed for the Los Reyes Gold-Silver Project at PEA level. It covers the concurrent underground (UG) and open-pit (OP) operations that together feed a single processing plant at a nominal 5,000 t/d. Topics include the mining-inventory design basis, production schedule, mobile fleet, operating workforce, lateral and vertical development, primary and secondary ventilation, underground services, and paste backfill. Capital and operating cost build-ups for the mining content are presented in Section 21 and not repeated here.

The Project is a brownfield redevelopment of the historical Los Reyes mining district in Sinaloa, Mexico. As a PEA prepared under NI 43-101, the Technical Report does not estimate Mineral Reserves (see Section 15); the mine plan is based on a mining inventory derived from Mineral Resources and includes Inferred Mineral Resources, which are considered too geologically speculative to support Mineral Reserve categorisation under NI 43-101. The reader is cautioned that the PEA is preliminary in nature and that there is no certainty that the results of the PEA will be realised.

Cross-references to other sections appear where relevant — in particular to Section 14 (Mineral Resource Estimate) for the resource model that underpins the mining inventory, Section 17 (Recovery Methods) for processing recoveries that sit outside this section's scope, Section 18 (Project Infrastructure) for the paste plant and surface facilities, and Section 22 (Economic Analysis) for consumption of the production schedule.

16.1.2 Qualified person

This section has been prepared by Rodrigo Capel Pasqua, BEng (Mining), FAusIMM, GAICD, of AGP Mining Consultants Inc., who is a Qualified Person as defined by NI 43-101 for the purposes of Section 16. The QP has the experience relevant to the subject of the section and to the style of mineralisation, type of deposit and mining methods under consideration, as required under NI 43-101, and has taken responsibility for the content of the section.

The QP visited the Project site from May 31, 2025 to June 2, 2025, spending June 1, 2025 on site and a half-day on June 2, 2025 reviewing core and geological data at the Cosalá office. Elements inspected during the visit included the surface footprint of the proposed open pits, the access roads, the proposed portal locations for the plant site and underground tunnels network, and the core storage and geological sample facilities at the Cosalá office. Observations from the site visit have been incorporated into the mining design, schedule and risk assessment presented in this section.

The effective date of this section is April 25, 2026, matching the Technical Report's effective date.

16.1.3 Design basis

The mining design basis for the Project is summarized in Table 16-1. The design basis reflects the PEA-level simplifications applied to the mining inventory — principally the adoption of a single deposit-wide average for mining dilution and mining recovery, a uniform 15-metre level spacing across the underground mine, retention of the originally-planned western plant location, and use of long-hole open stoping as the single underground method across all zones.

Table 16-1: Mining design basis (PEA)

Parameter	Value
Mill throughput (nominal)	5,000 t/d
Processing-plant location	Adjacent to ZTM Trend
Primary mining method (underground)	Long-hole open stoping with paste backfill
Level spacing (underground)	15 m, deposit-wide
Total mining dilution (underground)	~14% (1.1 m ELOS skin + 3% zero-grade), deposit-wide average
Total mining dilution (open pit)	5%
Mining recovery (stopes and open pit)	95%, deposit-wide average
Mining recovery (development)	100%
Access	Underground tunnel network with five portal locations; decline grade 1:7 (14%)
Open-pit method	Conventional drill-and-blast with excavator-truck loading
Backfill	Paste, 100% extraction/100% fill, 5% flat cement content
Mine life	13 years (LOM, including ramp-up and stockpile reclaim tail)
Operating mode	Concurrent underground and open-pit feeding a single mill

The headline scenario numbers that emerge from the mine plan built to this design basis are summarised in Table 16-2. Capital is quoted as Project Capital only; sustaining capital and total life-of-mine cost figures are presented in Section 21 and are not repeated in Section 16.

Table 16-2: PEA base-case headline numbers

Quantity	Value
Total mineralised-material inventory to mill (LOM)	26.07 Mt
Underground mineralised-material inventory	14.82 Mt
Open-pit mineral inventory	11.25 Mt
Open-pit waste	76.03 Mt
Open-pit strip ratio	6.76:1
Total underground lateral development	127.9 km
Total underground vertical development (raisebored)	~6 km
Paste backfill volume (LOM)	6.5 Mm ³
Peak truck fleet	15 trucks (Year 2; 7 UG + 8 OP)
Peak operating workforce	780 (Year 5; 570 UG + 210 OP)
Project Capital	US\$98.2M
Project Capital contingency (24%)	US\$23.3M
Project Capital (inclusive of contingency)	US\$121.5M

Year convention. Throughout this section, Year 0 is the first year of mining (haulage-network establishment, first open-pit pre-strip, initial low-grade stockpile build) and Year 1 is the first year of processing. Plant construction runs Year –1 through Year 0; the processing plant is ready at the start of Year 1. The Section 22 financial model uses a different convention (Year –1 or Year 0 = construction, Year 1 = first production year); Section 16.5.1 sets out the translation.

16.1.4 PEA mining inventory

The PEA mining inventory feeding the mine plan is presented in Table 16-3 (split by source, underground versus open-pit) and in Table 16-4 (split by Mineral Resource classification). The inventory is derived from the Mineral Resource Estimate presented in Section 14, with modifying factors applied as described in Section 16.8 and after stope-shape optimisation as described in Section 16.4.4. Inferred Mineral Resources are considered too geologically speculative to support Mineral Reserve categorisation under NI 43-101, and no Mineral Reserves are reported for the Project; the scheduled Inferred component is the principal mechanism by which the PEA is preliminary in nature, as set out in Section 15. The mining inventory is a preliminary, PEA-level estimate derived from the Mineral Resource by applying the selected mining methods (longitudinal long-hole open stoping underground, conventional open-pit mining) and PEA modifying factors; it is conservative in that it carries dilution and mining-recovery factors and excludes material below the applied cut-off.

Within the mining inventory, a reclassification of resource material occurred as a result of the increase in underground mining and reduction in open-pit mining relative to the shapes developed for the Mineral Resource Estimate (“MRE”) in Section 14. The MRE was generated from Revenue Factor 1 pits and residual underground shapes, while the PEA mine design uses smaller pits in favour of a larger underground component.

The drill spacing classification requirements for indicated material in the open pit models uses 40-metre spacing, while the underground model uses a stricter 30-metre spacing. As the underground tonnage increased in the resultant mine design, so did the amount of inferred material in the PEA mining inventory that did not meet the stricter underground model indicated classification threshold. As such, the total indicated and inferred numbers in Table 16-4 cannot be directly compared with the Mineral Resource Estimate in Section 14 and should be treated as a conservative treatment of the contained resource.

Table 16-3: PEA mining Inventory by Source

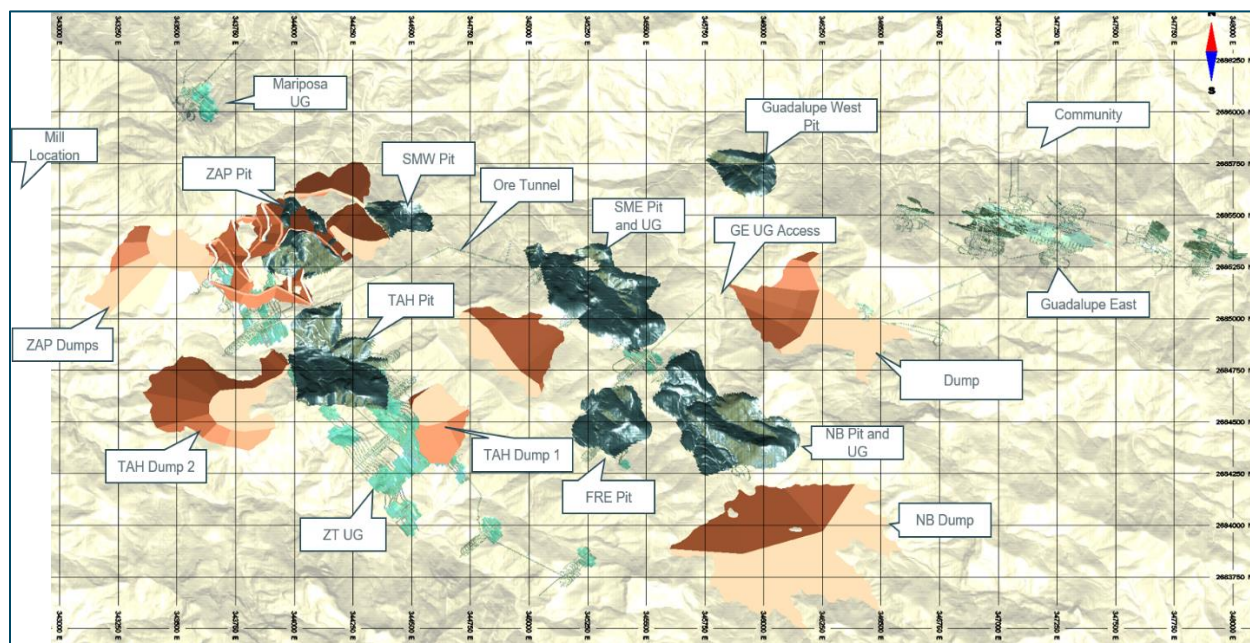
Source	Tonnes (Mt)	AuEq (g/t)	Au (g/t)	Ag (g/t)	AuEq (koz)	Au (koz)	Ag (koz)
Underground (LHOS)	14.8	3.20	2.12	92.00	1,525	1,011	43,842
Open-pit	11.3	1.43	1.11	25.7	516	400	9,280
Total PEA mining inventory	26.1	2.43	1.68	63.4	2,040	1,411	53,122

Notes: AuEq grades and ounces are derived from the PEA schedule on a diluted basis at the mill feed, after application of the modifying factors described in Section 16.8. The AuEq formula is given in Section 16.1.11.

Table 16-4: PEA mining inventory by Mineral Resource classification

Classification	Tonnes (Mt)	AuEq (g/t)	Au (g/t)	Ag (g/t)	AuEq (koz)	Au (koz)	Ag (koz)
Measured							
Indicated	16.5	2.37	1.62	59.5	1,254	861	31,545
Measured + Indicated	16.5	2.37	1.62	59.5	1,254	861	31,545
Inferred	9.6	2.55	1.79	70.0	786	551	21,577
Total PEA mining inventory	26.1	2.43	1.68	63.4	2,040	1,411	53,122

Figure 1-1: Overall mine layout — underground deposits, open pits, portals and haulage to plant



Note: Approximately 63% of the scheduled tonnage is drawn from Measured and Indicated material, with the balance (approximately 37% by tonnage; ~39% by contained AuEq ounces) from Inferred; the Inferred component is concentrated in later LOM years and is a candidate for conversion through in-fill drilling as the Project progresses through subsequent study stages. The Table 16-4 total tonnage will reconcile with Table 16-3.

16.1.5 Method-selection summary

Long-hole open stoping with paste backfill has been selected as the primary mining method for all four underground zones (Zapote Tahonitas Main, Guadalupe East, San Manuel-area satellites, and minor satellite orebodies). The selection is supported by the mining-method trade-off presented in Section 16.1.8. Geotechnical inputs (Section 16.2) and the stope-optimised mining inventory (Section 16.4.4) support the selected method. The selection will be revisited and referenced at PFS.

16.1.6 Method-selection trade-off

A mining-method trade-off was carried out for the Guadalupe East (GE) zone, with the Zapote Tahonitas (Z-T) zone reviewed qualitatively. Three underground mining methods were evaluated for GE: long-hole open stoping (LHOS), sub-level caving (SLC), and cut-and-fill (CAF).

LHOS has been selected as the single mine-wide underground mining method for the PEA. The selection is supported by three considerations beyond the GE-zone value proxy: (1) the satellite deposits outside GE are best-suited to LHOS, so a mine-wide LHOS approach avoids a two-method operation; (2) method standardisation simplifies the fleet, crew skill matrix, and ventilation and services design; and (3) paste-filling the historical voids in GE and mining through them under LHOS maximises metal extraction and minimises the safety risks of unfilled voids. A more detailed method trade-off for the GE zone specifically will be carried out in future study phases; that work may reinstate SLC as the GE method if the detailed geotechnical domain modelling and void-condition data support it. Stopping is predominantly longitudinal. Transverse long-hole open stoping is applied only at Guadalupe East, where the mineralised width exceeds approximately 20 m; the narrower zones (the other three deposits) are mined longitudinally.

16.1.7 Cut-off grade

A cut-off grade analysis was carried out to set the mining cut-off applied to both the underground stope optimisation and the open-pit pit optimisation. The cut-off-grade basis is presented here as part of the Section 16 design basis; because this is a PEA and Section 15 is reduced to a stub, the cut-off grade discussion that would normally appear in Section 15 of a full 43-101 Technical Report has been placed in this section.

The cut-off-grade basis uses a gold-equivalent (AuEq) formulation calculated at the mill feed (i.e. after mining dilution). The AuEq formula combines gold and silver grade using the ratio of their metal prices and plant recoveries:

$$\text{AuEq (g/t)} = \text{Au (g/t)} + \text{Ag (g/t)} \times (\text{Ag price} \times \text{Ag recovery}) / (\text{Au price} \times \text{Au recovery})$$

With a gold price of US\$1,950/oz, a silver price of US\$25.24/oz, a mill gold recovery of 94.5% and a mill silver recovery of 78.5%, the silver-to-gold equivalence ratio is approximately 93 — that is, 93 g/t Ag carries the same recovered value at the processing plant as 1 g/t Au.

Break-even cut-off grades have been calculated for each method as the grade at which marginal revenue equals marginal cost — the point below which mining a unit of material would not recover its incremental delivered cost.

‘Processing-plant-undiluted’ (Mill Undiluted) refers to the in-situ grade of the mine-design solids considered for plant feed, before mining dilution is applied.

Table 16-5: PEA cut-off grade basis

Method	Break-even AuEq (g/t, Mill Undiluted)	Applied AuEq (g/t, Mill Undiluted)	Basis for applied value
Open pit	0.37	0.37	Break-even adopted
Underground LHOS	1.22	1.50	Higher cut-off selected on NPV grounds

The applied cut-off for the underground mine plan is 1.50 g/t AuEq Mill Undiluted, which is above the LHOS break-even values. A cut-off sensitivity was carried out on the stope-optimisation output, testing cut-off values above and below the break-even. The sensitivity showed that the 1.50 g/t cut-off improved project net-present-value relative to the break-even cut-off: the higher cut-off slightly reduces the mining inventory but raises average feed grade and lowers operating cost per ounce, with a net positive NPV impact, and the 1.50 g/t value has been adopted for the PEA mine plan.

For the open pit, the break-even cut-off of 0.37 g/t AuEq Mill Undiluted has been adopted directly; no cut-off sensitivity has been run on the open-pit optimisation at PEA stage because the break-even cut-off on a low-cost per tonne operation is not materially NPV-sensitive. The 0.37 g/t AuEq cut-off is the direct-mill-feed cut-off; open-pit material above the marginal cut-off but below direct-feed grade is stockpiled and reclaimed late in the life of mine (the schedule shows open-pit material stockpiled in early years and reclaimed in Years 12 to 15).

The mining operating costs used to set the cut-off grades were checked against the final Section 21 estimates (underground approximately US\$57/t and open pit approximately US\$4/t total material) and are consistent at PEA level; no re-optimisation of the cut-off grades was required.

The mining inventory presented at Table 16-3 and Table 16-4 reflects these applied cut-off grades.

16.1.8 Commodity-price sensitivity

One observation from the stope-optimisation work bears on the design basis and is noted here for completeness. A sensitivity on the stope-optimisation cut-off tested the mine plan's response to a 50% increase in the metal price used to generate stope shapes. The response was modest — the stope-optimised AuEq tonnage increased by approximately 15%. The stope shapes generated for the PEA already capture most of the mineable envelope within the current geometry of the underground zones; further price upside converts mostly into marginal additions to existing stopes rather than into newly economic material. This result is consistent with a mining inventory that is grade-driven rather than price-driven, and it underpins the decision to hold the stope-optimisation cut-off flat through the sensitivity cases presented at Section 22.

16.2 Geotechnical design basis

This section sets out the geotechnical design basis adopted for the PEA, including the rock-mass quality and rock mass structure characterization and recommendations on the underground stope dimensions, overbreak, level-spacing, and open-pit slope geometry parameters. The design outputs that follow from these parameters — open pit geometry, ramp and berm geometry, stope layouts, development profiles — are presented in Section 16.4.

The geotechnical inputs were provided by Knight Piésold (KP). Additional site characterization and stability analyses are required at PFS to refine the PEA-level parameters.

16.2.1 Rock-mass quality

The rock mass across the four underground deposits is characterised as poor to fair, with Rock Mass Rating (RMR, per Bieniawski 1989; hereafter RMR89) values ranging from approximately 25 to 70 depending on lithology, location within each deposit, and proximity to the Estaca fault. The rock-mass quality was evaluated using the available RQD data, estimates of intact strength and a review of core photos. Norwegian Geotechnical Institute (NGI) Tunnelling Quality Index (Q, per Barton) values were derived from the RMR89 classification using the standard relationship $RMR89 = 9 \times \ln(Q) + 49$. The characterisation draws on the

available geotechnical data across the four deposits, comprising RQD logging, intact-strength estimates and core-photo review; the data density reflects the PEA stage and will be supplemented by the geotechnical drilling recommended in Section 26.

The zone-by-zone characterisation is summarised below.

Zapote-Tahonitas (Z-T) deposit. The rock-mass quality of the immediate hanging wall varies spatially. The rock mass is of lower quality (RMR89 30 to 45) in the northwest and of better quality (RMR89 approximately 60) in the southeast. The mineralised zone itself has variable rock mass quality without clear spatial control; therefore, the most common rock-mass quality, RMR89 35, has been selected as the design value for the mineralised zone.

Central deposit. The rock-mass quality of the immediate hanging wall varies by lithology. In the Andesite the typical RMR89 is 60 or better while in the Rhyolite the typical RMR89 is 35 to 45. The mineralised zone is variable but typically of lower quality, with RMR89 between 30 and 40.

Guadalupe deposit. The rock-mass characteristics are strongly influenced by the presence of the Estaca fault within the main vein. The fault is associated with an interval of poor-quality rock mass tens of metres thick. The immediate hanging wall and the portion of the mineralised zone within the fault zone have RMR89 values between 25 and 35. The remainder of the main vein and the splay veins are associated with significantly better rock-mass quality, with typical RMR89 values between 55 and 70 in both the mineralised zone and the hanging wall.

Table 16-6: Rock-mass rating ranges by deposit (RMR89; derived Q')

Deposit	Zone	HW RMR89	Mineralised-zone RMR89	Design HW Q'	Design mineralised-zone Q'
Z-T	Northwest	45	35	0.6	0.2
Z-T	Southeast	60	35	3.4	0.2
Central	Andesite HW	60	35	3.4	0.2
Central	Rhyolite HW	45	35	0.6	0.2
Guadalupe	Fault-zone HW and mineralised zone	25	25	0.1	0.1
Guadalupe	Main vein outside fault and splays	60	60	3.4	3.4

The KP assessment of rock-mass quality is consistent with the requirements of a PEA study and is expected to be refined at PFS once further drilling and structural mapping allow the lithological and structural controls on rock mass quality to be better understood and expanded to all deposits.

16.2.2 Rock mass structure

Discontinuity orientation data from oriented core drilling were used to evaluate the dominant structural orientations for each deposit.

At the Z-T deposit, the dominant discontinuity orientation strikes northwest-to-southeast and dips between 50° and 70° to the southwest, with considerable variation in both dip and strike. This orientation agrees well with the main veins, which dip between 45° and 60° to the southwest.

At the Central deposit, the dominant discontinuity orientation also strikes northwest-to-southeast and dips to the southwest between 60° and 70°, consistent with the main veins dipping between 50° and 70° to the southwest. A sub-horizontal set, a set dipping at 70° to the west-northwest, and a sub-vertical set striking east-west may also be present.

At the Guadalupe deposit, the dominant discontinuity orientation strikes southeast-to-northwest and dips to the northeast between 50° and 70°, consistent with the main veins dipping to the northeast. Evidence of the southwest-dipping Estaca vein/fault was not readily observed in the oriented-core data but the Estaca is assumed to be the dominant structural orientation in the zone. A north-south striking set dipping at 70° to the east may also be present.

The Estaca fault is the most important structural feature at Los Reyes from a mining geotechnical perspective, and because the fault runs along the mineralised main vein at Guadalupe it is both a geotechnical factor and a mineralising-structure factor. The framing of the Estaca at the mine-design level is given in Section 16.2.5.

16.2.3 Underground stope dimension basis

Achievable underground stope dimensions were assessed by KP using the empirical Stability Graph method (Potvin 1988; Nickson 1992), with expected overbreak estimated using the Equivalent Linear Overbreak/Slough (ELOS) method (Clark 1998; Capes 2009). Four geotechnical cases were developed to cover two longitudinal mining configurations at Z-T and Central and two transverse mining configurations at Guadalupe. The HW stope dimensions in Table 16-7 are designed to the mid-point of the unsupported Transition Zone on the Stability Graph chart. Therefore, neither pattern HW cables nor other HW support are assumed in the mine plan, which is consistent with the paste-filled stoping method proposed for Los Reyes. However, ground support (e.g., cable bolts or Super Swellex bolts) could be installed from the sill drive to improve HW performance in areas with poorer rock mass quality.

Table 16-7: Geotechnical stope cases

Case	Type	Applicable to	HW RMR89	Mineralised-zone RMR89	Height (m)	HW-FW span (m)	Strike (m)	ELOS HW (m)	ELOS end (m)
1	Longitudinal	Z-T NW and Central Rhyolite HW	45	35	20	8	10	1.0	0.5
2	Longitudinal	Z-T SE and Central Andesite HW	60	35	20	8	15	0.75	0.5
3	Transverse	Guadalupe stopes along the Estaca fault	25	25	15	10	15	2.0 to 3.0	0.5
4	Transverse	Guadalupe HW splay veins	—	60	15	20	15	—	0.5

Case 3 carries the highest ELOS hanging wall overbreak (2.0 to 3.0 m) on account of the poor-quality rock mass associated with the Estaca fault; however, because the Estaca fault runs within the mineralised main vein, the majority of the overbreak in Case 3 is expected to be mineable inventory rather than waste. This

observation is the basis for the Section 16.2.6 structural-control discussion and for the dilution framework in Section 16.8.

In order to manage the poor rock mass quality associated with the Estaca Fault at Guadalupe, Case 3 applies an underhanded sequence — new stopes are mined below the previous stopes, so that the stope back is established against cemented paste backfill placed in the previous stope rather than against the poor-quality rock mass. The hanging wall of the initial A-panel stopes will be established in rock, but the hanging wall of the subsequent B-panel stopes will be established in cemented backfill.

For the PEA the four cases were consolidated into a single design basis applied uniformly across all underground deposits, while the case-specific ELOS values remain available to inform the zone-by-zone dilution framework at PFS stage (see Section 16.8). The consolidated basis — 15 m stope height, maximum 15 m HW-FW span, 15 m strike length, 0.7 m hanging wall ELOS and 0.4 m footwall ELOS for a total of 1.1 m ELOS per stope — is summarised in Table 16-8. Stope width for fractional-dilution purposes is taken at 3 m minimum, 10 m average and 15 m maximum. The consolidated basis standardises the mine layout and the schedule without materially compromising the stope-dimensions envelope recommended by the geotechnical review; it is a deliberate PEA-level simplification and is suitable for re-optimisation at PFS once the KP assessment is refined.

Table 16-8: PEA stope dimension basis (applied uniformly)

Parameter	Value
Height	15 m
HW-FW span (longitudinal)	8m maximum
HW-FW span (transverse)	15 m maximum
Strike length	15 m
ELOS hanging wall	0.7 m
ELOS footwall	0.4 m
Total ELOS	1.1 m
Minimum stope width (for fractional dilution)	3 m
Average stope width	10 m
Maximum stope width	15 m

16.2.4 Level spacing

The geotechnical review considered both 15 m and 20 m level spacings depending on the characteristics of each zone. A single deposit-wide level spacing of 15 m has been conservatively adopted for the PEA, standardising across the four underground deposits.

The standardised spacing is a deliberate PEA-level simplification. A uniform spacing simplifies the mine layout, the development schedule, the ventilation design and the operating-crew skill requirement; zone-by-zone optimisation of the level spacing is deferred to PFS once the geotechnical data package is refined. Adopting the conservative end of the recommended range leaves headroom for downside outcomes from the additional geotechnical work without forcing a re-optimisation of the schedule, and the economic penalty of a slightly tighter spacing at the deposits that could have supported 20 m is absorbed by the PEA cost build-up without altering the method selection.

16.2.5 Wide Longitudinal Stopes

Where the mineralised vein in a longitudinal mining configuration is wider than the 8 m maximum HW-FW span supported geotechnically, two design options are available. The PEA mine plan adopts option (a):

(a) Mine the wider portion of the vein in 8 m panels along strike, with each panel mined and paste-filled in sequence before the next panel is taken. Panel widths up to 10 m have been demonstrated in industry practice and may be considered in detailed design where the local rock-mass quality supports it.

(b) Locally switch from longitudinal to transverse mining for the wider portions of the vein, accepting the underhanded sequence and the cemented-backfill-in-back configuration described at Section 16.2.3.

The selection between (a) and (b) at each location is a PFS-stage design choice driven by the local vein width, rock-mass quality and the schedule cost of the additional development required by the transverse alternative.

16.2.6 Estaca fault at Guadalupe

The Estaca fault runs along the mineralised main vein at the Guadalupe deposit and exerts two distinct effects on mine design. From a rock-mass perspective, the fault is associated with an interval of poor-quality ground tens of metres thick, with RMR89 values in the range of 25 to 35. Outside of the fault and its associated damage zone, the rock mass quality improves, with RMR89 values ranging from 55 to 70 (see Section 16.2.1). From a structural-geological perspective, the fault is a conduit for the mineralising fluids that formed the Guadalupe deposit — so the geologically poor-quality ground associated with the fault is, to a large extent, itself mineralised. Knight Piésold's assessment, however, indicates that the structural damage zone extends beyond the modelled grade shells and into the hanging wall and footwall waste.

The Guadalupe mine design intercepts the Estaca fault many times, both with capital and operating development and with stopes. The Case 3 configuration (Table 16-7) is specifically designed for stopes that intercept the Estaca fault zone at Guadalupe. The fault therefore does not govern the orientation of the stopes — stopes are oriented by the mineralised vein geometry — and mining at Guadalupe goes through the fault rather than around it.

The Estaca-fault zone is poor-quality ground and is expected to be challenging to mine. For those stopes positioned directly on the mineralised-material/waste boundary — rather than stopes sitting between other stopes within the wide mineralised zones — the ELOS overbreak adjacent to the fault could comprise a substantial proportion of waste in addition to mineralised material, and dilution for those stopes may be higher than the deposit-wide average. The deposit-wide 14% dilution remains the appropriate basis for the PEA, and the fault zone is treated as a dilution risk to be quantified through zone-specific dilution and recovery modelling at PFS (Section 16.12.1). The framework for this refinement is set out in Section 16.8.5.

16.2.7 Crown pillar

The dimensions of a permanent crown pillar between the underground workings and surface were evaluated using the empirical Critical Scaled Span method (Carter, 2014). A Probability of Failure (PoF) of between 5 and 10% was used as the acceptance criterion and is applicable for a short term, semi-temporary crown, under non-sensitive mine infrastructure (Carter, 2008). For the purposes of the PEA, a crown pillar thickness of 45 m is recommended.

The 45 m crown-pillar recommendation is a Knight Piésold geotechnical input. The PEA mine plan does not currently incorporate a permanent crown pillar between the underground workings and the open-pit final

pit floor. Incorporation of the recommended crown pillar will be revisited at PFS, with the schedule and inventory adjusted accordingly. The implication is flagged at Section 16.12.1 as a PFS-stage design item.

16.2.8 Open-pit slope geotechnical parameters

Open-pit slope geotechnical parameters were developed for the PEA by KP. Overall slope angles were provided for the hanging wall and the footwall of the deposit for Zapote-Tahonitas, Central, and Guadalupe East. The bench face angle, inter-ramp slope angle, bench height, berm width and ramp width for each of the four pit sectors were calculated by AGP from the overall slope angle. Ramp widths, berm widths, final-wall geometry and stockpile placements are carried into the pit design in Section 16.4.5; they are summarized in Table 16-9 here as part of the geotechnical design basis. Note that the design parameters for Guadalupe are based on geotechnical data acquired from the Central trend.

Table 16-9: Open-pit slope geotechnical parameters by pit sector

Parameter	Unit	Zapote-Tahonitas	Central	Guadalupe
Footwall slope angle	°	42	47	47
Hanging wall slope angle	°	45	47	47
Face angle	°	70FW/75HW	80	80
Inter-ramp slope angle (with ramp)	°	48	49	51
Bench height	m	10	10	10
Berm width (with ramp)	m	7.5	7.5	7.5
Ramp width	m	20	20	20
Average benches per ramp	#	10	10	10

The open-pit slope parameters are at a scoping/PEA level of definition. Additional slope-stability investigation — including further geotechnical drilling, structural mapping within the pit footprints, and pit-sector-specific kinematic and limit-equilibrium analysis — is recommended for the PFS. The PEA slope parameters are appropriately conservative pending that additional work.

16.3 Hydrogeology and dewatering

This section sets out the dewatering strategy adopted for the PEA mine plan and describes how the expected hydrogeological conditions are incorporated into the underground and open-pit mining designs. Inflow modelling, aquifer characterisation, recharge and groundwater quality are presented in the Project's hydrogeology section and are cross-referenced from this section where relevant. The design-basis framing adopted here is that of the mining engineer receiving the hydrogeological model as an input and translating it into a dewatering strategy, pumping duty and discharge arrangement compatible with the mine schedule.

16.3.1 Approach

Los Reyes is a shallow, low-altitude operation. The underground mine is developed from surface through a decline network with five portal locations (two doubles, three singles, described in Section 16.4.2), with total mining depth below the highest portal less than approximately 500 m across all four underground deposits. Regional climate and the limited hydrogeological observations made during the Project's exploration programme indicate that the underground workings are expected to be fairly dry by underground-mine standards, and that the dewatering design is dominated by the containment and handling of intercepted

water rather than by aggressive proactive depressurisation of a groundwater system. Open-pit mining proceeds in parallel with the underground operation and is subject to the usual wet-season surface-water inflows into an active pit in addition to any intercepted groundwater.

The dewatering design presented in this section is a PEA-level allowance sized to the expected inflow conditions. Proactive depressurisation via dedicated dewatering wells, ground-freezing around the portal or decline, and any form of grout-curtain or diversion infrastructure are not included in the PEA basis; none of these is expected to be required on the information currently available. The PFS hydrogeology programme should confirm this expectation through dedicated pumping tests, piezometer installations in the footprint of the open pits and along the planned decline alignments, and a refined inflow-forecast model.

16.3.2 Underground dewatering strategy

The underground dewatering strategy is based on a staged sump-and-pump chain discharging to surface via the main decline. Intercepted water is collected at sumps located at each sub-level, routed to a staged-elevation collection sump near the active mining horizon, and lifted to surface in a series of pump stages rather than in a single lift. This configuration is standard practice for a shallow decline-access mine and avoids the capital and schedule impact of a dedicated pumping raise.

The PEA dewatering system is sized for an approximate steady-state total inflow of 60 L/s pumped to surface across the underground mine as a whole. The 60 L/s figure is an engineering allowance consistent with the dry-mine framing described in Section 16.3.1; it is not an inflow model output and should be treated as a PEA placeholder. Pump-station capacity and sump volumes are sized to accommodate short-term excursions above the steady-state figure during wet-season events and during active interception of minor water-bearing structures, with a conservative margin applied to the steady-state design flow.

Pumping infrastructure is carried into the Section 16.10.4 underground services discussion and into the underground services cost build-up. Dewatering piping runs alongside the decline in a dedicated services utility channel, and dedicated electrical feeds to each major pump station are taken from the underground reticulation described in Section 16.10.1.

16.3.3 Open-Pit dewatering strategy

Open-pit dewatering follows a conventional in-pit strategy. Each active pit sector carries a pit-floor sump; pumps at the pit-floor sump lift intercepted groundwater and accumulated surface water up the pit via a series of pumping stages along the haul-road network to a collection point at the pit rim. From the pit rim, water is routed to the surface water-management system described in Section 16.3.4 and in the Project Infrastructure section.

Open-pit dewatering duty is dominated by the wet-season surface-water inflow to the pit rather than by intercepted groundwater, consistent with the shallow-pit configuration and the limited groundwater observations available at PEA stage. The PEA cost build-up for open-pit dewatering includes allowances for wet-season pumping above the dry-season baseline; these allowances are cross-referenced to the open-pit operating cost build-up in Section 21.

As with the underground dewatering strategy, proactive depressurisation (for example via dedicated pit-slope dewatering wells) is not included in the PEA basis. The PFS should confirm that the PEA-level slope parameters in Section 16.2.6 remain valid under the expected pit-dewatering conditions and should dimension any additional depressurisation capacity required for sectors where the PFS slope-stability analysis identifies a groundwater-driven factor of safety that is not otherwise acceptable.

16.3.4 Surface discharge and integration

Water pumped to surface from the underground workings and water removed from the active open pits is routed to the Project's surface water-management system, which comprises settlement ponds, water-quality monitoring and discharge infrastructure sized and described in the Project Infrastructure section. The mining-side dewatering design is sized to deliver water to the surface collection point at the rates described in Section 16.3.2 and Section 16.3.3; the downstream handling, storage and discharge of that water is outside the scope of this section.

Water balance over the life of mine is tracked by the Project's integrated water-balance model (cross-referenced to the Project Infrastructure section). The water balance integrates mining dewatering inflow, process-plant water demand (including paste-fill make-up water described in Section 16.11), pit surface-water inflow, camp and ancillary demands, and controlled discharge. Interactions between the mining dewatering strategy and the surface water-management system — for example, the seasonal profile of the water balance — should be confirmed at PFS once the hydrogeology programme has dimensioned the inflow-forecast model for each of the underground workings and the open pits.

16.4 Mine design

This section presents the mine design developed for the PEA, covering both the underground and open-pit operations. The section is organised around the design tools used, the access and portal arrangement, the lateral and vertical development geometry, the stope optimisation and design, the open-pit design, the materials-handling system, and the overall mine layout that integrates the four underground deposits, the open pits and the processing plant. The design outputs presented here are consistent with the geotechnical design basis in Section 16.2 and with the cut-off-grade basis in Section 16.1.11.

The open-pit/underground boundary was determined from an operating-profit analysis at multiple candidate transition depths (AGP). Because the deposits are relatively flat-lying, there is no sharp economic cross-over between open-pit and underground mining; the boundary was set at the transition depth that approximately maximises operating profit.

16.4.1 Mine design and scheduling software

The underground mine design and schedule were carried out in the Deswik suite. Stope optimisation was carried out in Deswik SO; stope, development and infrastructure design was carried out in Deswik CAD; schedule preparation and sequencing were carried out in Deswik IS; and the underground production schedule was generated in Deswik Sched.

Open-pit work was carried out in separate software. Pit optimisation was carried out in Datamine NPVS; pit design was carried out in Datamine Studio OP; and the open-pit production schedule was generated in Hexagon Mine Plan.

The underground and open-pit schedules were integrated in an Excel workbook that reconciles the two schedules on a year-by-year basis and carries the combined mill-feed profile that drives Section 22. A single set of models carries the Project's mine-design, geological and scheduling information, with controlled exchange between the mining, geology and scheduling models at each revision cycle.

16.4.2 Access and portal arrangement

The underground mine is accessed through a tunnel and drive network that includes multiple declines and ramps serving the four underground deposits. Surface access is provided at five portal locations:

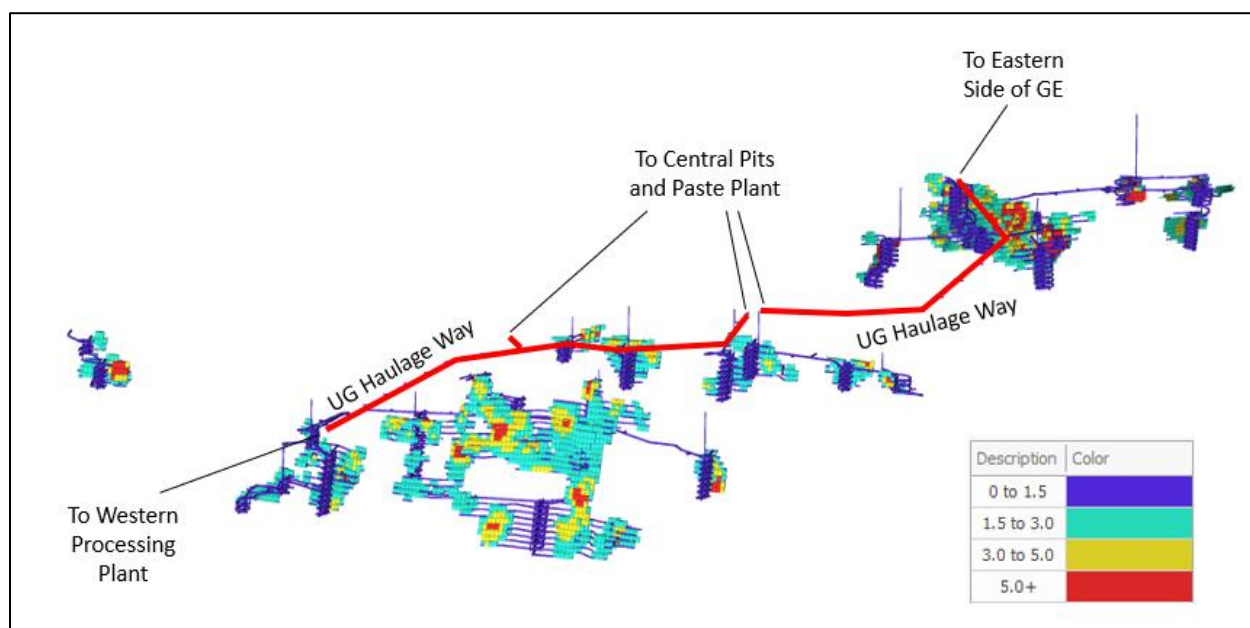
- One double portal at the western end of the Project area, at the processing-plant end, providing the principal intake and the main truck-haulage exit for mineralised material moving from the underground workings to the processing plant.
- Three single portals along the central trend, providing intake access to the central-trend deposits (Central and parts of Zapote-Tahonitas) and serving as the principal entry points for open-pit mineralised material from the Central-Valley pits into the underground haulage network.
- One double portal east of the Guadalupe East deposit, providing access to the Guadalupe East workings and the entry point for Guadalupe East open-pit mineralised material into the underground haulage network.

The double-portal configuration at the west and east ends separates fresh-air intake from return-air exhaust between the two portal openings at each location. The five portal locations together provide sufficient intake and return capacity to accommodate the peak airflow demand described in Section 16.9 and to permit concurrent operation of all four underground deposits. The underground 3D view, showing the four deposits, the main decline alignment and the portal network, is shown on Figure 16-1.

Declines and ramps within the network are developed to a maximum grade of 1 in 7 (14%). Local flattening below the maximum grade is permitted where the detailed design requires it. Development-profile dimensions for declines, ramps and the other development types are carried in Section 16.4.3.

Portal placement rationale is driven by four considerations: proximity to the underground deposit each portal serves, to minimise the length of the associated decline and capital development; proximity to the open pits, particularly the central-valley pits, to streamline material flow from the pits through the underground haulage network to the processing plant; local topography, which at Los Reyes favours portals cut into the valley walls rather than portals daylighting into low-lying ground; and alignment with the overall east-to-west material-flow pattern described in Section 16.4.6 and Section 16.4.7.

Figure 16-1: Underground 3D view and portal network.



Source: AGP, 2026

16.4.3 Lateral and vertical development

Lateral and vertical development for the PEA mine plan totals approximately 128 km of lateral development plus approximately 6 km of vertical raisebore development over the life of mine. The development types used, with their design-profile dimensions, are summarized in Table 16-10.

Table 16-10: Lateral and vertical development profiles

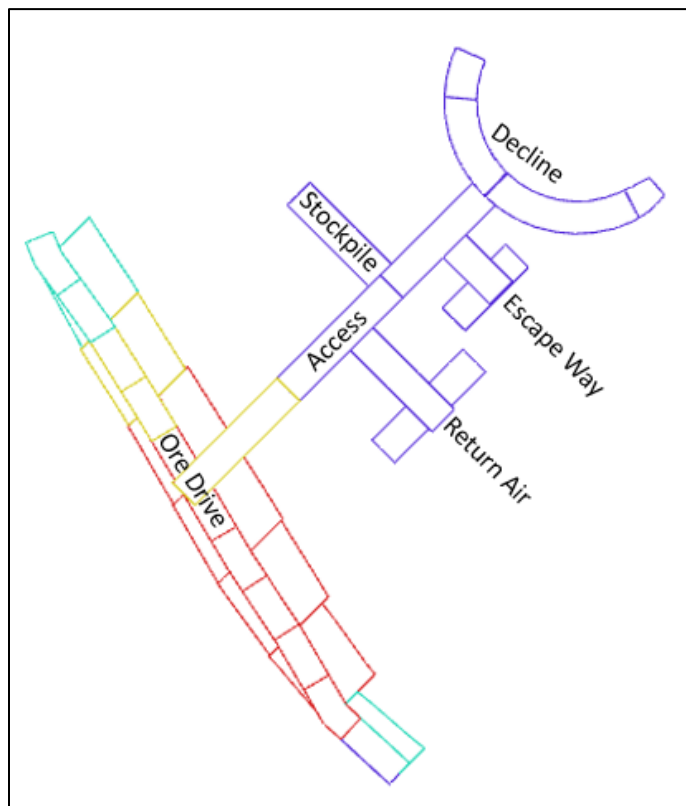
Development type	Width (m)	Height (m)	Arc radius (m)	Gradient
Decline	5.5	5.8	1.0	Maximum 1 in 7
Access/footwall drive/service point	5.5	5.5	1.0	Flat
Other infrastructure drives	5.0	5.0	1.0	Flat
Production drives	5.0	5.0	1.0	Flat
Return-air/fresh-air raise	4.5 m diameter (raisebore)	—	—	70°–90°
Escapeway/slot raise	1.5 m diameter (raisebore)	—	—	70°

Lateral-development total of approximately 128 km is distributed across capital development (decline, ramps, main access drives, return-air raise (RAR) and fresh-air raise (FAR) access) and operating development (production drives, level access, stope-access drives) across the life of mine. Approximately 6 km of the lateral development is pre-developed in Year 0 as early works, establishing the portals and the main underground haulage ways so that mineralised material can flow from the various open-pit and underground areas to the processing plant from Year 1 onwards. This early-works development is a prerequisite for Year 1 mining and is scheduled accordingly.

Vertical development is carried in raisebored raises. The raisebore programme includes the return-air raises (4.5 m diameter, fan-connected to surface), the fresh-air raises where a second intake route supplements the portal intakes, the slot raises used to initiate each stope, and the escapeways required for egress under regulation. The raisebore total is approximately 6 km across the four deposits. Raisebore productivity is assumed at 5 m per day per machine.

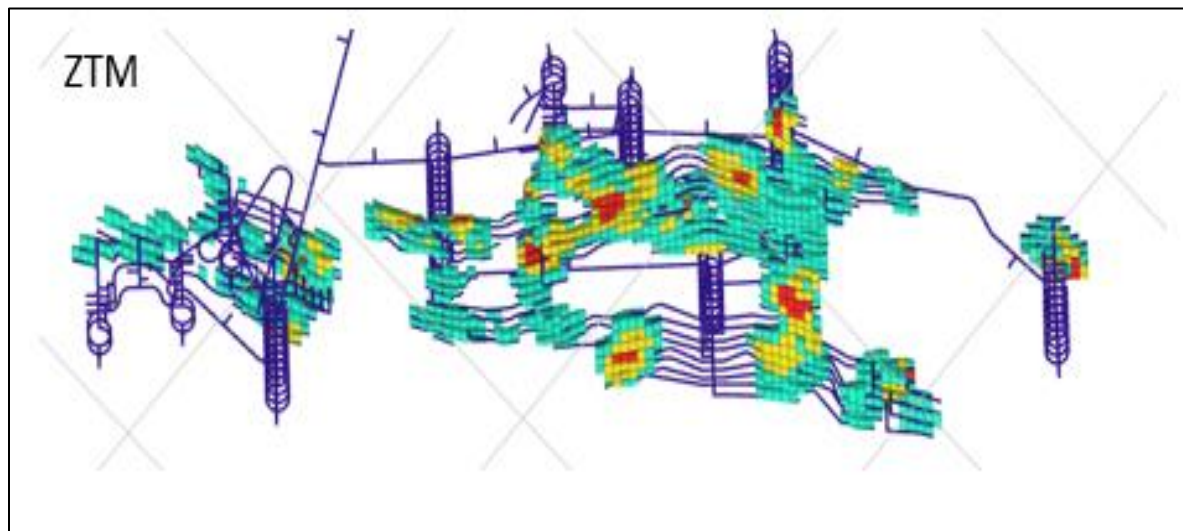
Levels are designed flat in the PEA; dewatering gradients on levels will be detailed at later study stages. Production drives are developed at the 5.0 m × 5.0 m profile; level access from the footwall-drive side is developed at the 5.5 m × 5.5 m profile to accommodate the loader-truck haulage described in Section 16.4.6. Detailed level-layout geometry — including the positions of the access drive, vent connections, stockpile, sump, escapeway and production drives relative to the underground haulageway — is shown on Figure 16-2, and per-zone underground plan views for Zapote-Tahonitas, Guadalupe East and San Manuel are presented across the three per-zone figures beginning at Figure 16-3.

Figure 16-2: Typical underground level plan view



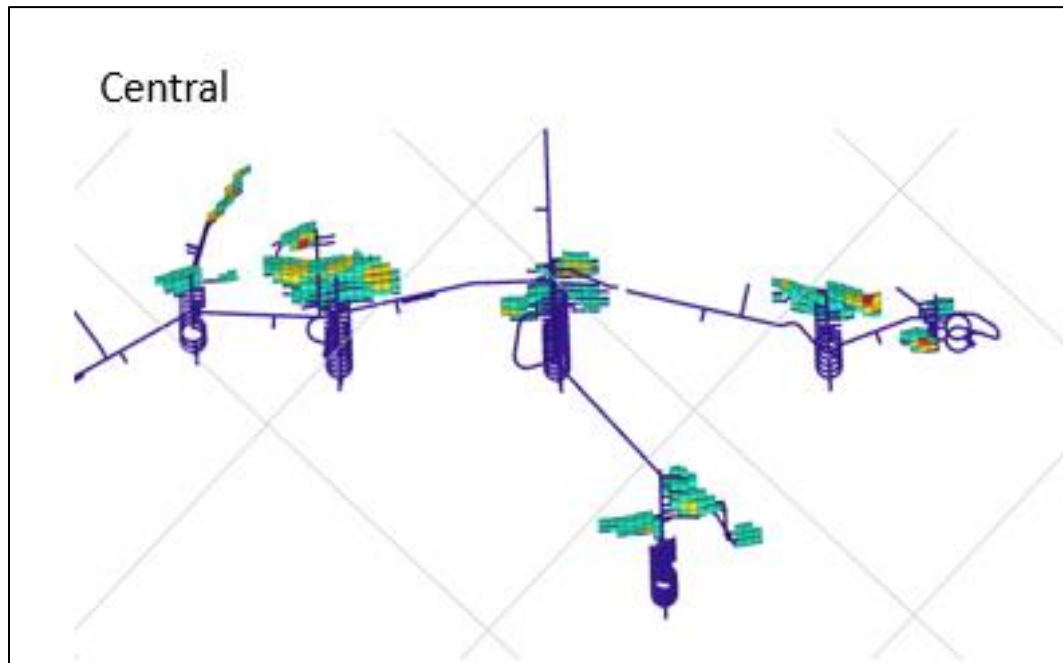
Source: AGP, 2026

Figure 16-3: ZTM underground plan views



Source: AGP, 2026

Figure 16-4: Central trend underground plan view



Source: AGP, 2026

Figure 16-5: GE underground plan view



Source: AGP, 2026

16.4.4 Stope optimisation and design

Stope optimisation for the underground mine plan was carried out in Deswik SO. The stope-optimisation parameters are summarized in Table 16-11.

Table 16-11: Stope optimisation parameter summary.

Category	Parameter	Value
Financial	Au price	US\$1,950/oz
Financial	Ag price	US\$25.24/oz
Financial	Plant Au recovery (mill)	94.5%
Financial	Plant Ag recovery (mill)	78.5%
Financial	AuEq ratio (Ag to Au by mass)	93
Financial	Royalty	0% (PEA assumption)
Operational	UG mining cost, mineralised material/stoping	US\$50/t
Operational	UG mining cost, lateral development	US\$3,000/m
Operational	Processing cost (mill)	US\$17.24/t
Operational	G&A	US\$4.50/t
Design/geotech	Stope height	15 m
Design/geotech	Maximum HW-FW span	15 m
Design/geotech	Strike length	15 m
Design/geotech	Total ELOS (HW 0.7 m + FW 0.4 m)	1.1 m
Design/geotech	Minimum mining width	3 m
Design/geotech	Break-even cut-off, AuEq Mill Undiluted	1.2 g/t
Design/geotech	Applied cut-off for scheduling	1.5 g/t AuEq Mill Undiluted

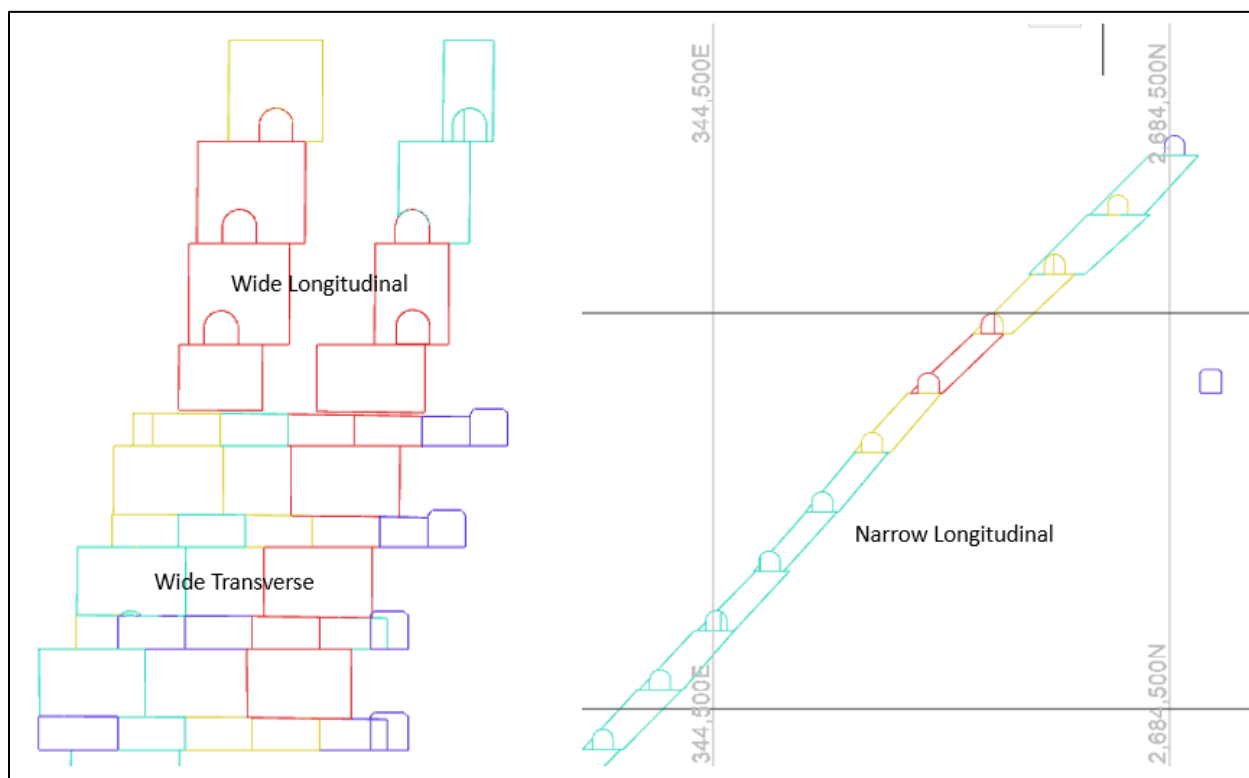
General and administrative (G&A) cost is applied as a single US\$/t-processed rate (total annual G&A divided by total mill feed) and is therefore identical for underground and open-pit feed; see Section 21.

No royalty was applied in the stope-optimisation cut-off; royalties are accounted for in the economic analysis (Section 22).

The stope optimisation was run at the break-even cut-off of 1.2 g/t AuEq Mill Undiluted. The resulting stope shapes were designed in Deswik CAD, with longitudinal stopes developed at Zapote-Tahonitas and Central and transverse stopes developed at Guadalupe reflecting the Section 16.2.3 geotechnical cases. A cut-off-grade sensitivity was then run on the stope-optimised inventory, testing cut-off values above and below the break-even. The sensitivity identified 1.5 g/t AuEq Mill Undiluted as the cut-off at which Project net-present-value was maximised, on the grounds discussed in Section 16.1.11. Stope shapes with AuEq Mill Undiluted grade below 1.5 g/t were filtered out of the inventory, and the filtered inventory was progressed to scheduling. The typical stope geometry, in plan and section, is shown on Figure 16-6.

Stopes are mined at 100% extraction with 100% paste backfill, consistent with Section 16.11. No pillars are carried between stopes in either longitudinal or transverse configurations. For longitudinal mining at Zapote-Tahonitas and Central, each panel is mined with the stoping face retreating towards the access point and paste backfill placed into each mined stope as the face advances. For transverse mining at Guadalupe, primary and secondary stopes across the orebody width are mined in an underhanded sequence: the initial A-panel stope is taken with the stope back in rock; subsequent B-panel stopes below are taken with their back in the cemented paste backfill placed in the previous stope rather than against the poor-quality rock mass associated with the Estaca fault.

Figure 16-6: Typical stope layout — plan and section



Source: AGP, 2026

16.4.5 Open-pit design

Pit optimisation was carried out in Datamine NPVS. The pit-optimisation parameters are summarized in Table 16-12; the open-pit geotechnical inputs (slope angles by sector) are carried in Section 16.2.6 (Table 16-9) and are cross-referenced here rather than repeated.

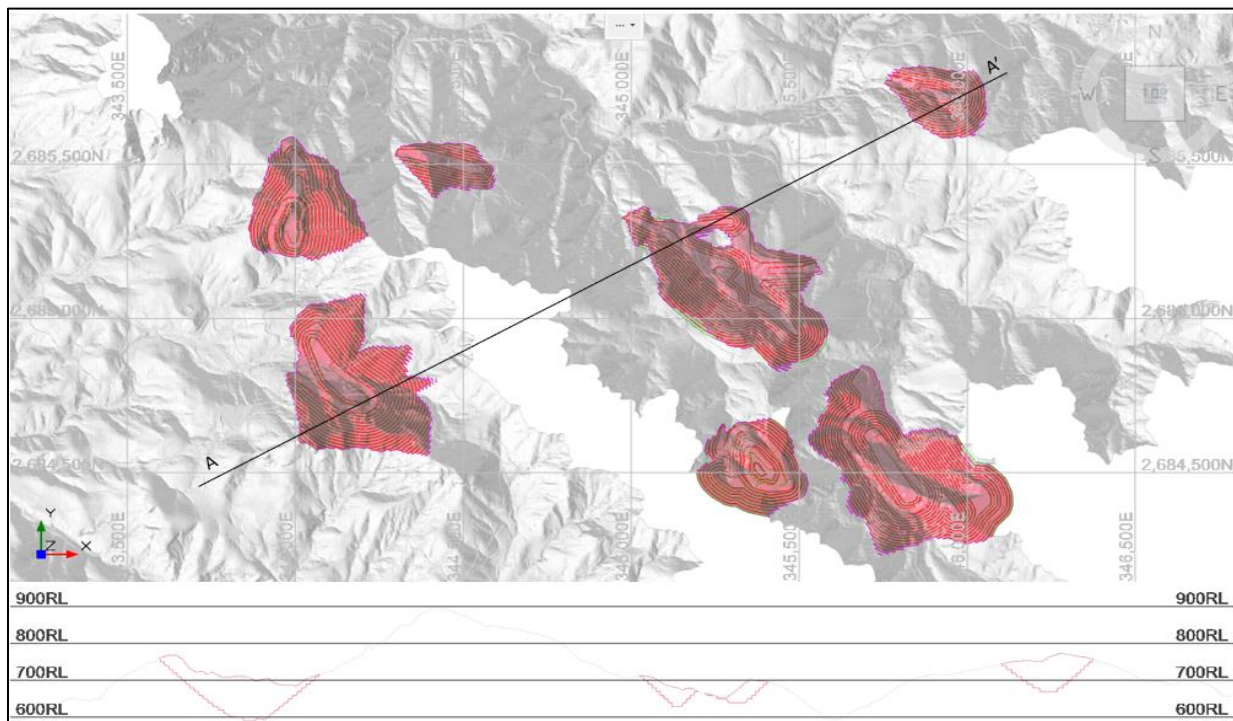
Table 16-12: Open-pit optimisation parameter summary

Category	Parameter	Value
Financial	Au price	US\$1,950/oz
Financial	Ag price	US\$25.24/oz
Financial	Plant Au recovery (mill)	94.5%

Category	Parameter	Value
Financial	Plant Ag recovery (mill)	78.5%
Financial	Selling cost	1%
Financial	Royalty	0% (PEA assumption)
Operational	OP mining cost, mineralised material/stoping	US\$4.50/t
Operational	OP mining cost, waste/lateral	US\$2.25/t
Operational	Processing cost (mill)	US\$17.24/t
Operational	G&A	US\$4.50/t
Design/geotech	Slope angles by sector	See Table 16-9
Design/geotech	Bench height	10 m
Design/geotech	Ramp width	20 m

Pit design was carried out in Datamine Studio OP using the Datamine NPVS pit-shell output as the basis. The pit-design outputs — final-wall geometry by pit, ramp configurations, bench and berm geometry, run-of-mine and in-pit stockpile placements — are shown in Section 16.4.7 (integrated layout) and on the pit-complex plan-view (Figure 16-7). Cut-off grade is applied within the final pit design to determine the mineralised-material/waste classification at the mining face; the applied cut-off is 0.37 g/t AuEq Mill Undiluted as discussed in Section 16.1.11.

Figure 16-7: Open-pit complex plan view



Source: AGP, 2026

16.4.6 Materials handling system

The materials-handling system routes the majority of open-pit mineralised material through the underground haulage network, which is a project-specific feature driven by the Project's topography. Figure 16-9 shows the overall materials-handling flow.

Underground mineralised material. Stopped mineable inventory is drawn from each stope by a load-haul-dump loader to the access-level stockpile immediately adjacent to the stope. No material passes are required over the life of mine; surge capacity is provided by truck haulage and the access-level stockpiles, consistent with the shallow decline-access configuration. From the access-level stockpile the mineable inventory is loaded into articulated trucks and tramped up the decline and along the underground haulage network to the **western portal**, from which it is trucked across the surface haul-road network the short distance to the processing plant. All underground mineable inventory from all four underground deposits routes through the western portal, using the east-to-west underground haulage that runs beneath the ridge separating the central and eastern deposits from the processing plant.

Open-pit mineable inventory from the central-valley pits. Mineable inventory from the Central and Guadalupe open pits, and from the Guadalupe East open pit, is loaded at the pit floor and trucked up the in-pit ramps to the pit rim. From the pit rim the mineralised material travels a short surface haul to the nearest underground portal — one of the three central-trend portals for the Central and Guadalupe pits, and the eastern double portal for the Guadalupe East pit — and then enters the underground haulage network alongside the underground mineralised material. It travels east-to-west through the underground haulage, exits the western portal, and is trucked to the processing plant.

Open-pit mineable inventory from the western pits. Zapote-Tahonitas sits on the western side of the ridge that separates the central valleys from the processing-plant valley. Because Zapote-Tahonitas is already on the plant side of the ridge, open-pit mineable inventory from Zapote-Tahonitas is trucked directly to the processing plant by surface haul rather than routed through the underground haulage. The surface-haul distance from Zapote-Tahonitas to the plant is short compared with the alternative of crossing the ridge to reach the underground haulage.

Pit waste. Waste from each pit is trucked to the closest surface waste dump by surface haul. Dump placements are sited to minimise surface-haul distance from each pit.

Underground waste. Waste generated from underground development is trucked to the closest portal and dumped at a small surface waste dump located near that portal. Detailed waste-dump placement for the underground-sourced waste is deferred to the PFS.

Year 1 transition. During Year 1, while the underground haulage network through the ridge is being completed, a portion of the central-valley open-pit mineable inventory may temporarily be trucked by surface haul directly to the processing plant. Once the underground haulage network is in service, the open-pit mineable inventory routes through the underground haulage as described above.

Routing central-valley open-pit mineable inventory through the underground haulage avoids the long ramp-up/ramp-down surface profile and substantial cut-and-fill road construction that would otherwise be needed to cross the ridges (see also Section 16.1.10). The 60-tonne-payload Volvo A60H articulated truck has been selected as a single haulage model for both underground and open-pit fleets. The 60-tonne class suits both the decline profile and the open-pit ramp geometry, and a common fleet reduces the truck count on the main haulage ways and simplifies traffic management at peak production.

16.4.7 Overall mine layout

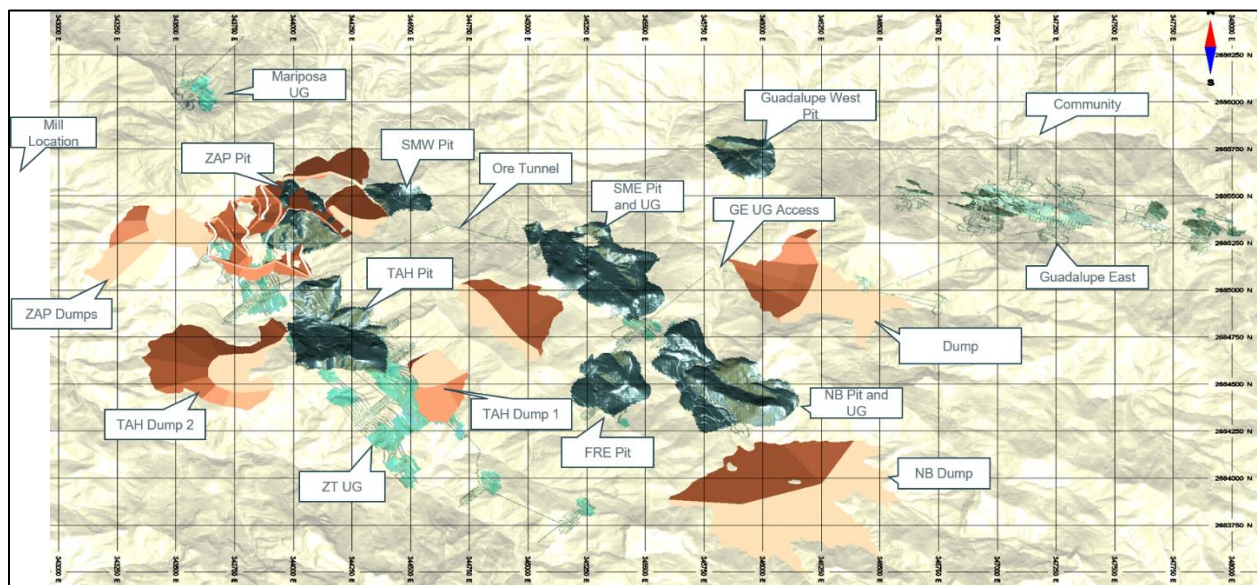
The overall mine layout integrates the four underground deposits, the four open pits, the surface haul-road network, the five portal locations, the waste-dump and run-of-mine stockpile footprints, and the processing plant into a single operating configuration. The layout is driven by three considerations: the east-to-west material-flow pattern established by the original western location of the processing plant (see Section 16.1.10); the topography of the Project area, which favours access routes along the valleys rather than across the ridges; and the need to allow concurrent operation of the underground and open-pit components without interference between underground and open-pit activities at any given deposit.

At the deposit level, the underground and open-pit operations are planned to operate concurrently at Zapote-Tahonitas, Central and Guadalupe, with underground stope access established from the portal network below and open-pit benching proceeding from surface downward. The open-pit and underground operations at each deposit are scheduled such that the underground workings are positioned below the final pit limit with an allowance for a temporary crown-pillar below the pit floor which will be mined later in the LOM; the crown-pillar allowance is consistent with the Section 16.2 geotechnical basis. At Guadalupe East the underground operation is accessed both through the dedicated eastern portal as well as underground haulage way and operates in parallel with the Guadalupe open pit.

Surface haul roads connecting the pit limits, the portal locations, the waste-dump footprints and the processing plant are designed to the 20 m ramp-width standard at grades compatible with the Volvo A60H truck fleet. The haul-road network aligns along the east-west valley trends that run through the Project area, and where cross-topography connections are required, cross-ridge haul-road sections are kept short and are designed at the flatter end of the grade envelope to protect truck productivity.

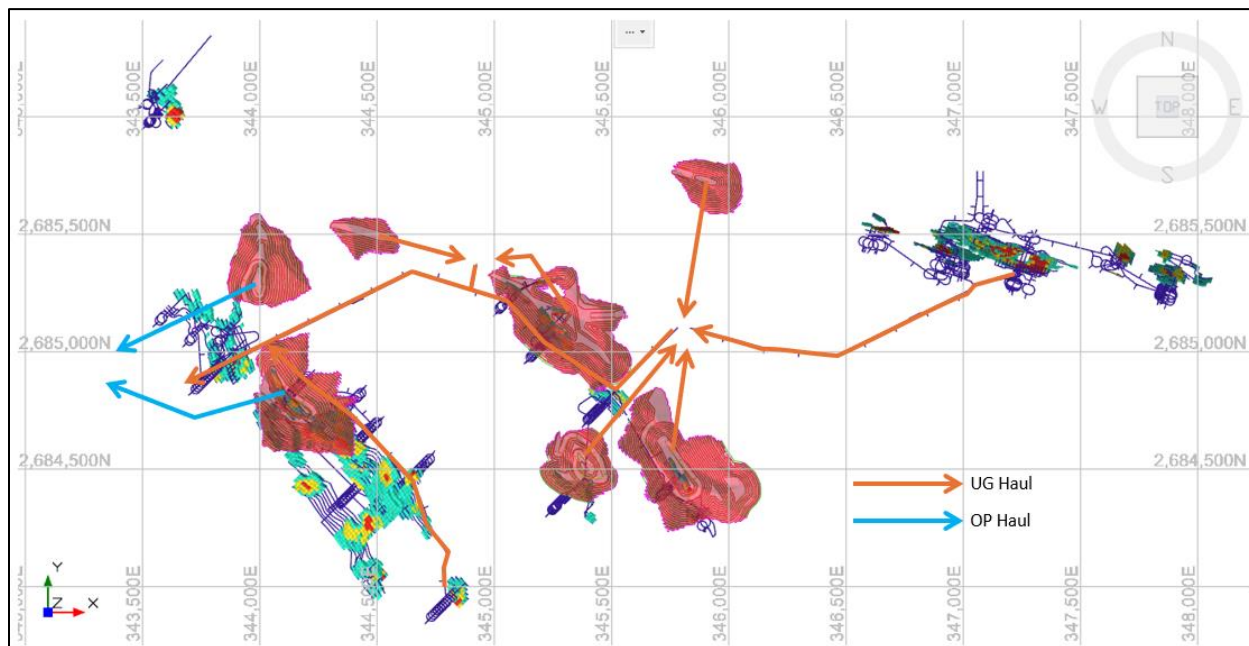
The overall site layout is shown on Figure 16-8, and the materials-handling flow that integrates the underground and open-pit operations through the haulage network is shown on Figure 16-9.

Figure 16-8: Overall site layout



Source: AGP, 2026

Figure 16-9: Materials-handling flow plan.



Source: AGP, 2026

16.5 Production schedule

This section presents the life-of-mine production schedule developed for the PEA. The schedule covers the concurrent underground and open-pit operations, the stockpile build-up and reclaim, and the annual mill-feed profile that the processing plant receives. The schedule has been generated using the design basis described in Section 16.1.3, the mine design described in Section 16.4, the modifying factors described in Section 16.8, and the productivity assumptions described in Section 16.5.3 below.

16.5.1 Year convention

The schedule year convention used in this section is that **Year 0 is the first year of mining**. Year 0 carries the establishment of the underground haulage network, the first open-pit pre-strip and mineralised-material-exposure work, and the first underground development metres. A portion of open-pit mineable inventory is mined during Year 0 and stockpiled on surface for processing in Year 1. The processing plant is under construction during Year 0 and is not yet receiving mineralised material.

Processing-plant construction begins in Year -1 and runs for two years across Year -1 and Year 0. The plant is ready to receive mineable inventory at the start of Year 1. **Year 1 is therefore the first year of processing**; the processing plant commissions and ramps up through Year 1, processing the mineable inventory stockpiled during Year 0 together with the early mineable inventory from the ramping underground operation and the active open-pit faces.

The life of mine runs from Year 0 (first mining) to the end of the open-pit-and-stockpile-reclaim tail in Year 15. Of this span, Year 0 is the pre-processing mining year, Year 1 is the processing-plant commissioning and ramp-up year, Years 2 through 11 are the underground-dominant steady-state window at nameplate throughput, Year 12 is the underground tail-out, and Years 13 through 15 are the final open-pit and stockpile-reclaim window.

16.5.2 Sequencing strategy

The production schedule has been sequenced to three objectives. The first is to keep the processing plant full at nameplate throughput over the operating life of mine. The second is to maximise project net-present-value, which in practice drives the prioritisation of the higher-grade Guadalupe East zone in the underground sequence and the early extraction of low-strip-ratio starter pits in the open-pit sequence. The third is to manage the operational interactions between the concurrent underground and open-pit operations. This means limiting the number of active mining fronts, matching the paste-fill system to the stoping rate, and sizing the fleet and workforce to the combined workload — so the scheduled tonnage is deliverable under the capacities described in Sections 16.6, 16.9, 16.10 and 16.11.

The annual mill-feed mix evolves over the life of mine as follows.

- **Early years (Year 1 to Year 2)** — Open-pit dominates the mill feed via low-strip-ratio starter pits at Zapote-Tahonitas and the western pit complex, while the underground ramps up. Year 1 delivers ~1.15 Mt OP mineable inventory against 0.18 Mt UG; OP material mined above mill feed is stockpiled.
- **Ramp-up (Year 2 to Year 3)** — The underground reaches peak annual rate by end-Year 2. From Year 3 onwards UG dominates the feed at ~1.48 Mt/a against ~0.35 Mt/a OP across most of the steady-state window.
- **Steady state (Year 3 to Year 11)** — UG provides ~80% of annual mill feed, OP the balance. Higher-grade Guadalupe East is brought forward to lift the early grade profile; lower-grade satellites are scheduled in the later part of the window.
- **Tail years (Year 12 to Year 15)** — UG ends at Year 12. Mill feed comes from final-year OP and reclaim of the low-grade stockpile, sustaining nameplate throughput through Year 14 and reduced throughput in Year 15.

Stope and production sequencing. Stope-extraction sequencing within each active zone follows the Section 16.2.3 geotechnical stope cases. For longitudinal mining at Zapote-Tahonitas and Central (Cases 1 and 2), each panel is mined with the stoping face retreating toward the access point; paste-fill is placed into each mined stope as the face advances, maintaining a fresh paste-fill boundary immediately behind the active stoping face for the duration of each panel. For transverse mining at Guadalupe (Cases 3 and 4), stopes are mined in primary–secondary lines across the orebody width, with each line retreating toward the footwall drive; primary stopes along each line are taken first, paste-filled and cured, and secondary stopes between them are taken subsequently with their stope walls in paste-fill rather than in rock. At Guadalupe the underhanded sequence described in Section 16.2.3 also applies: the A-panel stopes are taken with the stope back in rock, and subsequent B-panel stopes below are taken with their back in the cemented paste from the A-panel. Stope-cycle times, inter-stope activity gaps and paste-fill/cure durations are set by the Section 16.5.3 productivity assumptions; the resulting stoping rates are consistent with the annual underground tonnages presented at Table 16-7. Stoping is predominantly overhand (ascending), transitioning to underhand (descending) in the Estaca-fault-influenced portions of Guadalupe.

16.5.3 Productivity assumptions

The productivity-rate assumptions used to generate the production schedule are summarized in Table 16-13. The assumptions are PEA-level values, appropriate to the level of engineering definition at this stage; they will be refined at PFS against testwork, operator data and contractor benchmarking. Charged metres are the charged length of the blastholes.

Table 16-13: Productivity rate and physicals assumptions

Activity	Rate/physical	Value
Lateral development — decline	Advance rate	100 m/month
Lateral development — level drives	Advance rate	50 m/month
Raisebore	Advance rate	5 m/d
Production drilling	Drilling rate	250 m/d per rig
Mucking and loading	Rate	1,000 t/d per unit
Paste fill (UG-delivered maximum)	Flow rate	3,000 m ³ /d
Paste fill curing	Cure time	14 days
Production drilling density	Charged metres per stoped tonne	5 cm/t
Slot raise length	Fixed slot length for all stopes	11 m

Stope-cycle activities are scheduled in the following sequence: slot-drive development (including paste dig-out where a stope is adjacent to a previously backfilled stope), slot-raise establishment, production drilling, explosives charging, mucking, paste filling, and paste curing. Operational delays of 0 to 9 days are allowed between activities in the stope cycle to account for shift transitions, equipment availability and inter-activity logistics.

The paste-fill volume required per stope is calculated using:

$$\text{paste-fill volume (m}^3\text{)} = \text{in-situ stope volume} \times (1 + \text{paste dilution}) \times \text{mining recovery} + 250 \text{ m}^3$$

The additional 250 m³ allowance covers the drive volume immediately in front of the paste wall, which is filled as part of the pour. The resulting life-of-mine paste volume is approximately 6.53 Mm³, which is carried into the Section 16.11 paste backfill discussion.

16.5.4 Ramp-up and steady state

The processing plant is commissioned and ramped up during Year 1, reaching nameplate throughput of 5,000 t/d over the course of the year. Year 1 delivers approximately 1.33 Mt to the mill against the nameplate-year target of 1.825 Mt, corresponding to an operating-time-weighted ramp-up during the first year of processing that brings the plant to full nameplate from Year 2 onwards.

The underground operation is at its lowest annual output in Year 1 (approximately 0.18 Mt) reflecting the need to complete the decline establishment, portal infrastructure, initial access drives and the first raisebored vertical development before stoping can begin at rate. Stopping commences in Year 1 and ramps steeply through Year 2; by the end of Year 2 the underground operation reaches its peak annual production rate of approximately 1.48 Mt per year and sustains that rate through the steady-state window.

The open-pit operation carries the mill feed during the underground ramp-up. Year 1 is the open-pit peak-delivery year at approximately 1.15 Mt of mill feed from the pits, augmented by approximately 0.58 Mt of open-pit mineable inventory mined in excess of mill demand and stockpiled for later reclaim. From Year 2 the open-pit contribution to mill feed falls to approximately 0.52 Mt as the underground operation takes over the dominant share, and the open-pit contribution then sits in the 0.33 Mt to 0.35 Mt per year range through the underground-dominant steady-state window.

Steady state runs from Year 3 to Year 11, during which the processing plant processes approximately 1.83 Mt per year at a consistent grade profile driven principally by the underground ore-body grade. The tail-year profile (Year 12 to Year 15) is characterised by the under-ground tapering to zero at Year 12, followed by a final open-pit mining window (Year 12 to Year 14) and a stockpile-reclaim window (Year 14 to Year 15).

16.5.5 Life-of-mine physicals

The life-of-mine physicals for the PEA mine plan are summarized in Table 16-14. The totals reconcile with the PEA mining inventory in Table 16-3.

Table 16-14: Life-of-mine physicals summary

Source	Mineralised material delivered to mill (Mt)	Waste (Mt)	Strip/stope ratio
Underground (LHOS)	14.8	7.3	—
Open pit	11.3	76.0	6.76:1
Total mineralised material to mill	26.1	83.3	—

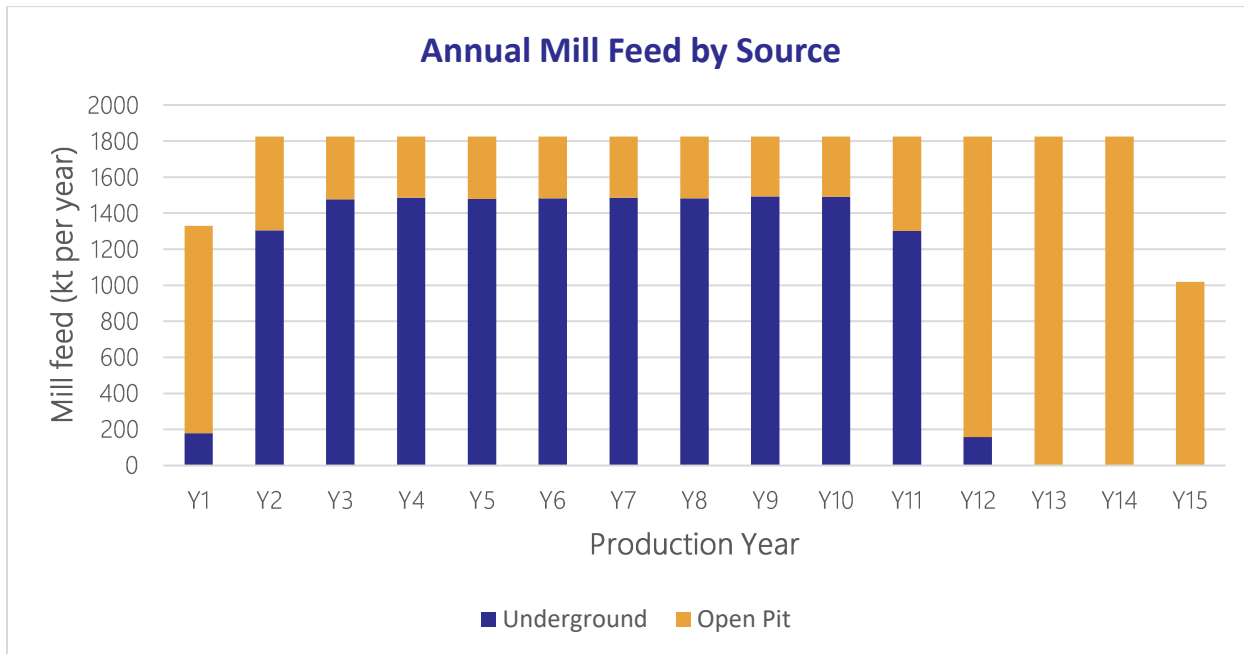
Underground lateral development over the life of mine is approximately 127.9 km (capital plus operating development), concentrated in the first five years where development intensity is high and tapering in the later half of the life of mine. Approximately 6 km of lateral development is pre-developed in Year 0. Vertical raisebored development is approximately 6 km over the life of mine. Underground stope production over the life of mine is approximately 13.02 Mt, with the balance of the 14.82 Mt underground mineable inventory figure coming from mineable inventory delivered by mineable-inventory drive development (approximately 1.80 Mt over life of mine).

Open-pit material movement over the life of mine is approximately 87.28 Mt (mineralised material plus waste), peaking in Year 1 at approximately 7.9 Mt total movement and sustaining at approximately 6 to 8 Mt total movement per year through the active mining window. The open-pit strip ratio of 6.76: 1 is an LOM average; individual pit complexes and individual pit phases sit at different strip ratios within that LOM average.

16.5.6 Annual mill feed

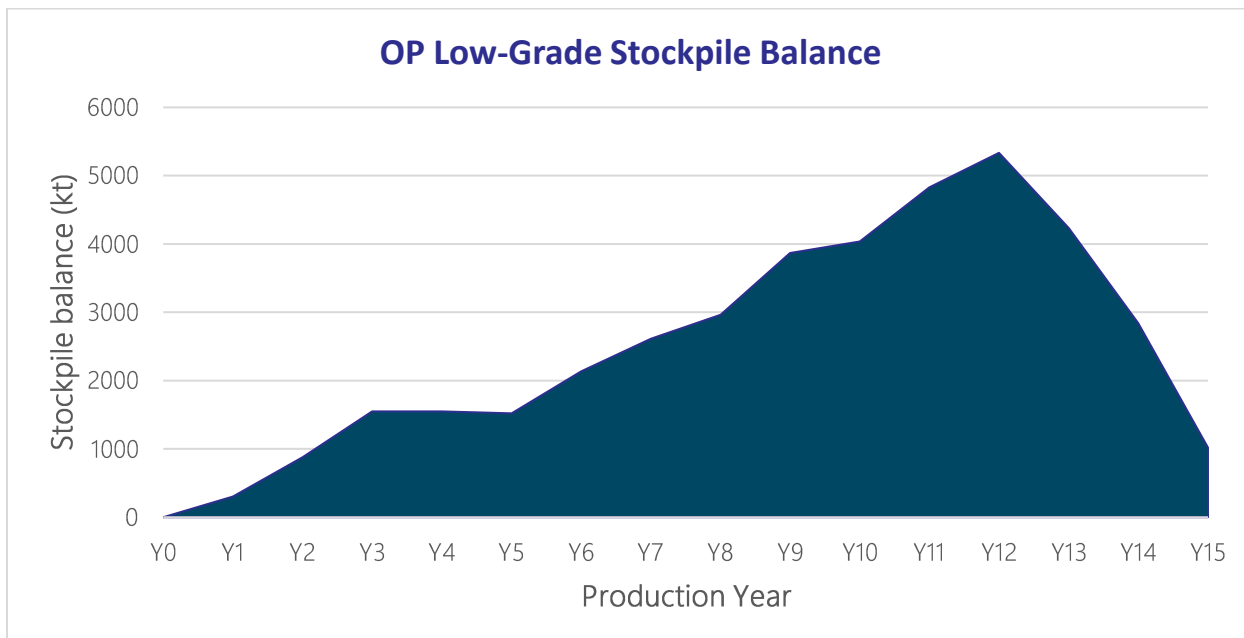
The annual mill-feed profile is presented in Table 16-15 and on Figure 16-10. The stockpile balance over the life of mine is presented on Figure 16-11.

Figure 16-10: Annual mill feed by source with AuEq head grade.



Source: AGP, 2026

Figure 16-11: Stockpile balance over life of mine



Source: AGP, 2026

Table 16-15: Annual mill feed (PEA base case)

Year	UG mill feed (Mt)	OP mill feed (Mt)	Total mill feed (Mt)
0	0.00	0.00	0.00
1	0.18	1.15	1.33
2	1.30	0.52	1.83
3	1.48	0.35	1.83
4	1.49	0.34	1.83
5	1.48	0.35	1.83
6	1.48	0.34	1.83
7	1.49	0.34	1.83
8	1.48	0.34	1.83
9	1.49	0.33	1.83
10	1.49	0.33	1.83
11	1.30	0.52	1.83
12	0.16	1.67	1.83
13	—	1.83	1.83
14	—	1.83	1.83
15	—	1.02	1.02
LOM total	14.82	11.25	26.07

16.6 Mobile equipment and fleet

This section presents the mobile-equipment and fleet basis developed for the PEA. The fleet is sized to the production schedule in Section 16.5 and to the materials-handling configuration in Section 16.4.6 and is drawn from standardised equipment classes that are compatible across the underground and open-pit operations. Fleet numbers evolve over the life of mine with the mining rate. The mobile fleet and selected fixed equipment are leased, so the fleet cost in Section 21 follows the year-by-year active-unit profile rather than a one-off capital purchase; lease payments are carried in capital and sustaining costs. The peak-year fleet, summarized in Section 16.6.4, sizes the workshop, ventilation and services rather than a capital purchase.

16.6.1 Fleet philosophy

The fleet philosophy for the PEA is built on three principles. The first is standardisation across the underground and open-pit operations. A single articulated-truck haulage model — 60-tonne payload class — is used across the underground decline haulage, the in-pit haulage at each open-pit complex, and the surface haul-road connections between portals and the processing plant. Standardisation reduces the number of operator certifications, the spare-parts inventory, the maintenance-crew skill matrix and the number of fleet interfaces on the project. The 60-tonne class is compatible with both the 1 in 7 decline grade and the 20 m ramp-width standard adopted for the open pits.

The second principle is the selection of fleet sizes to match the materials-handling routing in Section 16.4.6. The 60-tonne class reduces the peak truck count on the main underground haulageways compared with

the smaller truck classes originally considered during earlier study phases, which directly eases traffic management and passing-bay frequency on the through-the-ridge haulage. Open-pit fleet sizing is similarly matched to the pit geometry and the haul profile for each pit complex.

The third principle concerns the delivery of ground support. Ground-support installation at Los Reyes is carried out using the development jumbo and the longhole rig, with support manually installed, rather than with a dedicated mechanised bolter or a dedicated cable-bolt rig; no cable bolts are used. This is a PEA-level scheduling and fleet-optimisation choice that keeps the underground fleet lean and is consistent with common practice in Mexican underground operations of comparable scale. The allocation will be revisited at PFS against the refined development schedule, the bolting-pattern requirements from the KP geotechnical programme and the contractor-productivity benchmarking.

16.6.2 Underground fleet

OEMs have not been selected at this stage; brand and model names are given only as a reference for equipment sizing and lease/cost estimation.

The underground fleet covers four equipment categories: haulage, development, production and support.

Underground haulage is provided by 60-tonne articulated trucks, operating in the decline, the main underground haulage network and the production-drive-to-access-level movements within each active zone. A single articulated-truck model is specified across the underground mine. Main-haulage trucks are dedicated to the east-to-west underground haulage flow described in Section 16.4.6. Refuelling is carried out at surface, with an allowance for a small underground service bay.

Development fleet comprises two-boom electric-hydraulic development jumbos for drive development and ground-support drilling, and 14-tonne-class diesel LHDs for primary muck-out on production drives and stope-access drives. The LHD fleet is shared between development and production duty cycles; the same LHD class services both functions and the fleet allocation across the two duty cycles is planned into the Section 16.6.4 fleet profile.

Production fleet comprises longhole production drills (ITH or top-hammer top-drill, 76–89 mm hole diameter class) for stope drilling, and 14-tonne-class diesel LHDs for stope muck-out. The production LHDs are drawn from the same fleet as the development LHDs.

Support fleet includes Normet Spraymec 6050 WP shotcrete sprayers for ground-support installation, Normet MC 605 D(V) and MF 050 D(V) emulsion-charging units for development and stope blasting, Normet LF600 agitator trucks for shotcrete supply, Cat 962K integrated tools for utility work, a Cat 12F grader for underground road maintenance, Isuzu N75 stores trucks for materials movement, and a Toyota Hilux light-vehicle pool for personnel and supervisor transport. No underground rockbreaker is required; utility tasks are handled by the Cat 962K integrated-tool carrier, and the Isuzu N75 stores truck is boom-equipped.

16.6.3 Open-pit fleet

The open-pit fleet comprises haulage, loading, drill and blast, and ancillary support. Haulage is provided by 60-tonne articulated trucks, shared across all four pit complexes with fleet allocation following the annual mining rate at each pit. Loading is provided by hydraulic excavators at each active face, with smaller front-end loaders used for clean-up and ROM-stockpile duty. Drill-and-blast is provided by top-hammer drill rigs sized to the 10 m bench height, with standard emulsion blasting practice. Ancillary support includes graders for haul-road maintenance, water trucks for dust suppression, service vehicles for fuel and lube delivery, and a light-vehicle pool.

Open-pit trucks are dedicated to each pit complex for the duration of the complex's active mining life rather than shared across pits on a daily basis. This simplifies the fleet management and allows pit-specific haul distances to drive the fleet sizing and the fleet-hour profile at each pit. The pit-to-plant haul profile for each pit is a function of the materials-handling routing: Zapote-Tahonitas mineralised material and all pit waste are handled on surface haul roads, whereas mineralised material from the central-valley pits (Central, Guadalupe, Guadalupe East) enters the underground haulage network via the nearest portal, at which point the open-pit truck transfers its load to the shared underground articulated-truck fleet.

16.6.4 Peak fleet and utilisation

The peak fleet configuration is reached in Year 4 to Year 5, at the point where the underground operation is ramping to its peak annual production rate while the open-pit operation is still delivering a substantial annual tonnage from the starter pits. Peak truck count is 15 Volvo A60H trucks across the combined fleet: 7 trucks allocated to the underground operation and 8 trucks allocated to the open-pit operation. Total fleet count per fleet class at the peak year, including development, production and support equipment, is summarized in Table 16-16.

Table 16-16: Peak-year fleet by fleet class (Year 4-5).

Fleet class	Class	Rated power (kW)	Peak concurrent units	Duty
Articulated truck	Volvo A60H	577	15 total (7 UG + 8 OP)	Haulage — UG main + OP in-pit and surface
Jumbo	Epiroc Boomer E20 S	180	8	UG development, ground-support drilling
Longhole drill	Simba SM60 S	119	6	UG stope drilling
LHD	Scooptram ST14 S	354	16	UG development + production muck-out
Shotcrete sprayer	Normet Spraymec	—	3	UG ground support (spraying)
Charge unit	Normet Charmec	—	4	UG stope + development charging
Integrated tool	Cat 962K	—	3	UG utility, clean-up
Grader	Cat 12F	—	2	UG haul-road maintenance
Stores truck	Isuzu N75	—	4	UG materials movement
Light vehicle	Toyota Hilux (pool)	—	30+	Personnel and supervision
OP excavator	Hydraulic class	—	4	OP loading at face
OP top-hammer drill	Standard bench-drill class	—	4	OP drill-and-blast
OP ancillary	Grader, water truck, service	—	~10	OP road maintenance, dust, service

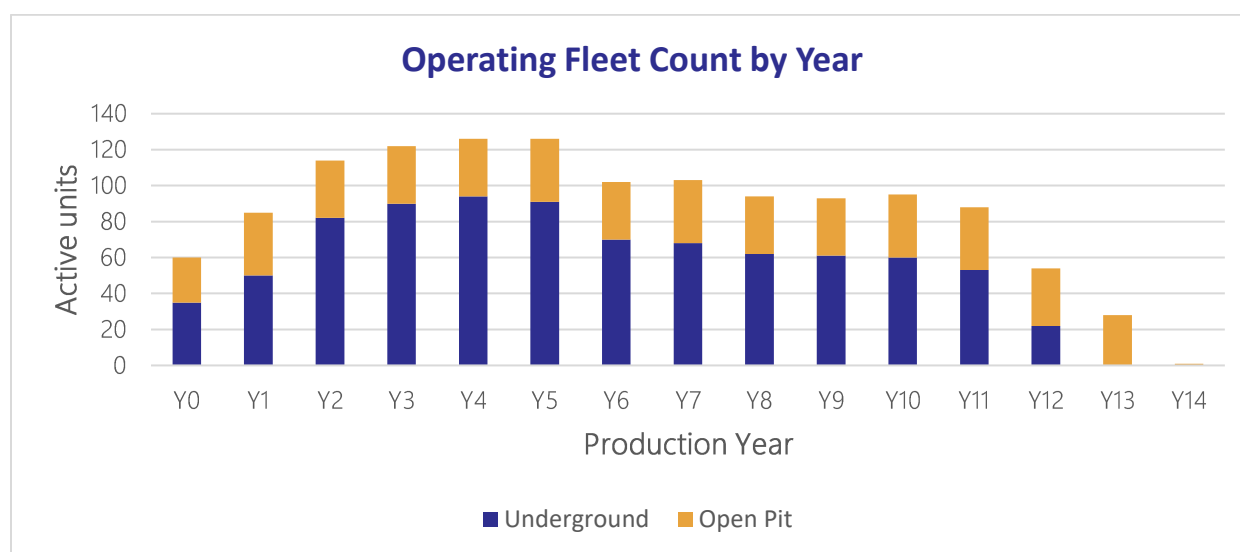
Utilisation is expressed as an Overall Equipment Efficiency (OEE) figure for each fleet class, where $OEE = Availability \times Utilisation \times Operational\ Efficiency$. OEE values for the principal fleet classes used in the PEA cost model are summarised in Table 16-17. The annual-operating-hour profile for each fleet class is derived from the OEE value and the assumed calendar-hour base for each fleet class.

Table 16-17: OEE values for the principal fleet classes (PEA cost model).

Class	OEE
Epiroc Boomer E20 S jumbo	56%
Simba SM60 S longhole drill	60%
Scooptram ST14 S LHD (development and production)	53%
Volvo A60H articulated truck	56%
Normet Spraymec 6050 WP	52%
Normet LF600 agitator truck	52%
Normet charging unit (development and production)	52%
Cat 962K integrated tool	35%
Cat 12F grader	53%
Isuzu N75 stores truck	32%
Toyota Hilux light vehicle	18%

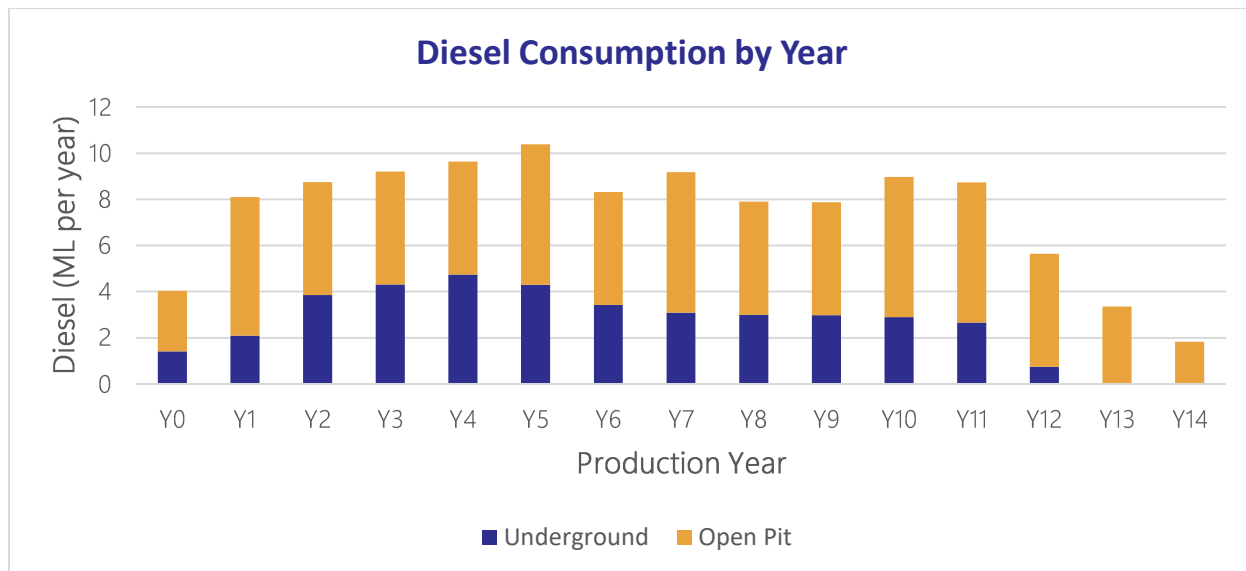
The operating fleet count by year, split between underground and open-pit, is shown on Figure 16-12, and the corresponding annual diesel consumption — used as a proxy for fleet utilisation — is shown on Figure 16-13.

Figure 16-12: Operating fleet count by year — underground and open-pit.



Source: AGP, 2026

Figure 16-13: Diesel consumption by year.



Source: AGP, 2026

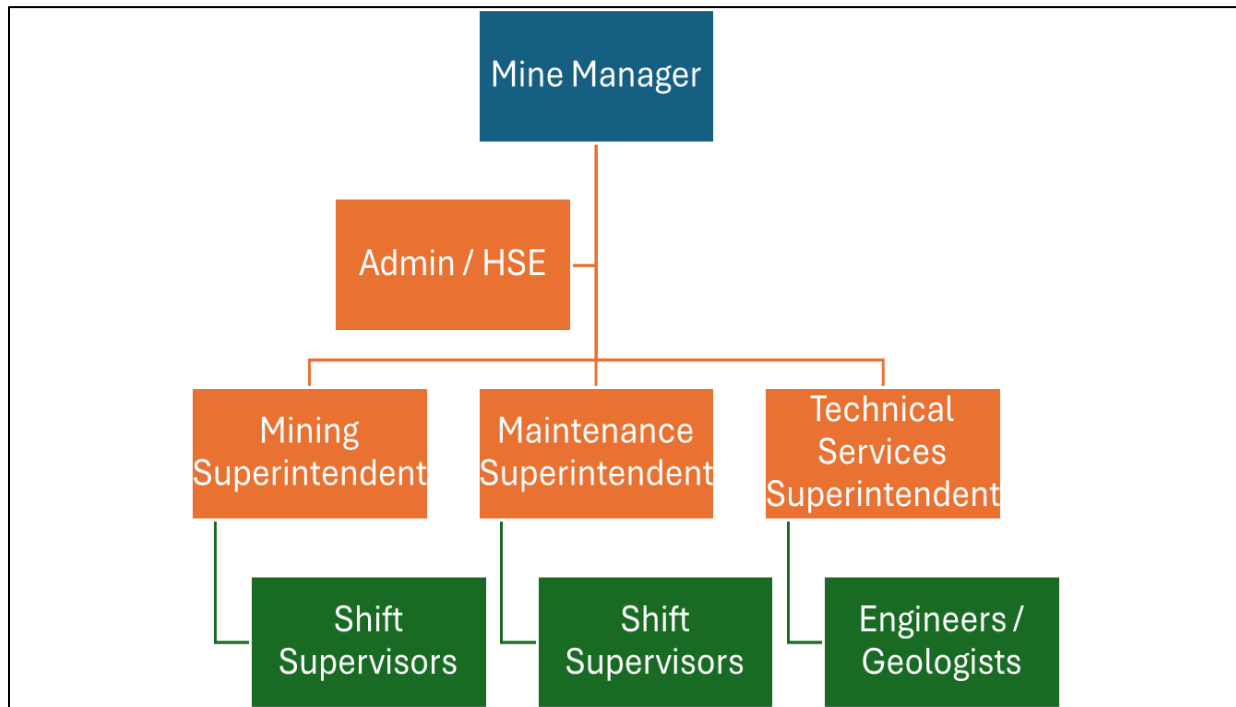
16.7 Workforce and organisation

This section presents the operating-phase workforce build-up and the high-level organisation structure developed for the PEA. The workforce schedule is sized to the production schedule in Section 16.5 and the fleet configuration in Section 16.6, and carries the operator, trades, supervision, technical and management roles required to deliver the mining scope across the life of mine.

16.7.1 Organisation and high-level roles

At a PEA level of detail, the operating-phase organisation at Los Reyes is structured around a single Mine Manager with responsibility for the combined underground and open-pit operation and for the downstream interfaces with processing, infrastructure and external stakeholders. Reporting to the Mine Manager are five operational and technical functions: Mining, Maintenance, Technical Services, Health-Safety-Environment, and Administration. The high-level organisation chart is shown as Figure 16-14.

Figure 16-14: Operating-phase organization chart



Source: AGP, 2026

The Mining function carries the day-to-day responsibility for the underground and open-pit mining operations. An Underground Superintendent manages the underground operation, reporting through shift supervisors to the stope, development and services crews at each active zone. An Open-Pit Superintendent manages the open-pit operation, reporting through shift supervisors to the drill-and-blast, load-and-haul and ancillary crews at each active pit complex. The two superintendents coordinate the shared fleet allocation and the portal-entry coordination that the materials-handling routing in Section 16.4.6 requires.

The **Maintenance** function carries responsibility for the mobile-fleet and fixed-plant maintenance across both operations. Maintenance is organised into three sub-functions — Fixed Plant (underground and surface fixed plant, pumps, ventilation, compressed air), Mobile (the Volvo haulage fleet, LHDs, jumbos, drills and support equipment) and Auto-Electrical (electrical and instrumentation across both fixed and mobile) — each led by a superintendent-level role.

The **Technical Services** function carries responsibility for mine planning, survey, geology, geotechnical engineering and engineering coordination. At PEA level Technical Services is a flat structure reporting to a Technical Services Superintendent.

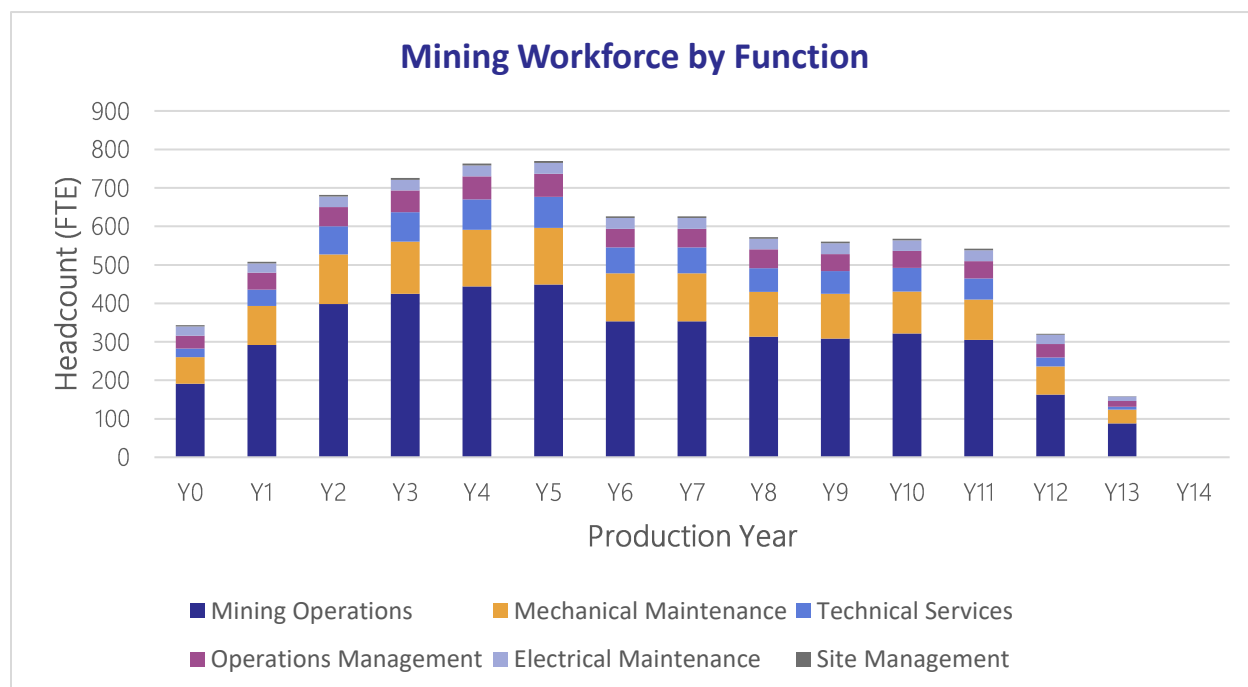
The **Health, Safety Environment & Community (HSEC)** function is organised as a stand-alone line function reporting directly to the Mine Manager.

The Human Resources & Administration function covers human resources, procurement, stores, general-administrative and training roles.

16.7.2 Workforce build-up

Total operating-phase workforce builds from the commissioning year to a peak of approximately 780 FTE in Year 5, reflecting the underground-dominant steady-state window where the combined underground and open-pit activity is at its highest. The year-by-year workforce profile, split by function, is shown on Figure 16-15. The peak is sustained through the steady-state window before tapering from Year 11 as the underground operation winds down and the open-pit-and-stockpile-reclaim tail demands a smaller workforce.

Figure 16-15: Workforce by year, split by function.



Source: AGP, 2026

The underground component of the workforce peaks at approximately 570 FTE in Year 5, driven by the combined stoping, development, services, maintenance, supervision and technical-services demand at peak underground production. The open-pit component peaks at approximately 210 FTE in Year 5, driven by the combined drill-and-blast, load-and-haul, maintenance and supervisory demand across the active pit complexes.

Year 0 carries a small workforce sized to the early-works underground development, the first open-pit pre-strip, the initial stores and surface-work establishment, and the on-site supervision and technical-services presence required to commission the operation. Year 1 workforce scales up with the fleet and the mining rate and reaches approximately 600 FTE by the end of Year 1 as the operation transitions to regular-rate production.

16.7.3 Role split

Table 16-18 summarizes the peak-year (Year 5) workforce split by function across the combined underground and open-pit operation.

Table 16-18: Peak-year workforce by function (Year 5).

Function	Underground (FTE)	Open-pit (FTE)	Combined (FTE)	% of peak
Mining operators	280	110	390	50%
Maintenance (fleet + fixed plant)	130	50	180	23%
Technical services (planning, survey, geology, geotech, engineering)	55	15	70	9%
Supervision (shift and section)	55	20	75	10%
HSE	20	5	25	3%
Administration (HR, stores, procurement, training)	30	10	40	5%
Total	570	210	780	100%

The Mining operators line includes jumbo operators, longhole drill operators, LHD operators, truck drivers, shotcreters, charging-unit operators, pit drill operators, pit loader operators, service-vehicle operators and related direct-operating roles. The Maintenance line covers fitters, boilermakers, auto-electricians, fixed-plant operators for the dewatering and compressed-air systems, and the Maintenance supervisors.

The workforce ratio of approximately 75% operators-plus-maintenance to 25% supervision/technical/HSE/administration is consistent with industry benchmarks for a Mexican operation of comparable scale and complexity.

16.7.4 Rostering, work patterns and labour oncost

A seven-days-a-week operating schedule is adopted for the PEA, with two twelve-hour shifts per day providing continuous underground and open-pit cover. Operating crews work a week-on/week-off roster pattern, which requires four crews — two day-shift crews (Day-A and Day-B) and two night-shift crews (Night-A and Night-B) — to cover the twenty-four-hour operation with each crew working one week and resting the next. The week-on/week-off cycle is consistent with standard Mexican underground mining practice and is reflected in the fleet-utilisation assumptions in Section 16.6.4.

Leave coverage (vacation and statutory leave) is provided by additional Service Crew Operators at approximately 16% of the direct-operator headcount. The Service Crew Operator allocation covers planned leave, training absence, medical absence and the rotating-rest-day allocation across the four-crew roster; these operators are counted in the total FTE figures reported in Section 16.7.2 and Table 16-18.

Local-commute arrangements apply for the majority of the workforce, who live in the Cosalá area and commute to site daily via bus-in/bus-out transport arranged from the Project. The location of the Project within commuting distance of established population centres makes a fly-in/fly-out roster unnecessary for the main operating workforce; a small contingent of specialist and supervisory personnel may use a drive-in/drive-out or fly-in/fly-out roster where the role requirements make local-commute impractical.

A small number of senior specialist roles (the Mine Manager, the Administration Manager, the Technical Services Superintendent, the Mining Superintendent and the Maintenance Superintendent) may be difficult to source from the local labour market in the early years and are costed accordingly. The Company's intent is to maximise local employment, and the balance of the workforce — including all operators and trades — is assumed to be hired locally and trained over time.

Labour cost in the PEA cost model carries a total oncost of 48.3% on base salary for Mexican-market roles, comprising Social Security (Instituto Mexicano del Seguro Social, IMSS), Retirement Savings (Sistema de Ahorro para el Retiro, SAR), the National Housing Fund (Instituto del Fondo Nacional de la Vivienda para los Trabajadores, INFONAVIT), the State Payroll Tax (Impuesto sobre Nóminas, ISN), and the 13th-salary annual bonus. The Senior Specialist roles carry an equivalent oncost aligned with each individual contract basis; the PEA cost model applies a comparable 48.3% oncost for simplicity at this study stage.

16.8 Modifying factors

This section presents the mine-plan-applied modifying factors used in the PEA mine plan — specifically the mining dilution and the mining recovery figures applied during stope optimisation, schedule generation and inventory reconciliation. The modifying factors are applied mine-wide as single deposit-wide averaged values, which is a deliberate PEA-level simplification. Refinement to zone-specific values is appropriate work for the PFS.

The term “modifying factors” is used here in the CIM sense of mining-specific modifying factors — that is, the factors applied to the Mineral Resource model to produce the mining inventory. Processing recovery is a separate concept that sits in Section 17 and is not discussed here.

16.8.1 Underground mining dilution

A total underground mining dilution of approximately 14% is applied uniformly across all four underground deposits. The total is the sum of two components of different character:

- **ELOS skin dilution** of 1.1 m per stope (hanging wall 0.7 m + footwall 0.4 m), derived from the geotechnical stope cases discussed in Section 16.2.3. The skin dilution is applied as a physical solid on the hanging wall and footwall sides of each stope and is evaluated against the geological block model at that location. The ELOS volume therefore carries whatever the block model indicates at the stope-wall position — normally this is waste rock, but in specific stope configurations (and in particular within the Estaca-fault zone at Guadalupe, see Section 16.8.3) the ELOS volume may overlap mineralised material. Applied against the PEA stope-width range (3 m minimum, 10 m average, 15 m maximum), the ELOS component yields an average fractional dilution of approximately 11%.
- **Zero-grade dilution** of 3% added as a flat schedule-level factor, representing paste-fill boundary material, operational-contamination material intercepted during muck-out, and small waste inclusions within the stope volume that the geological model does not resolve. The zero-grade factor is at zero grade by definition and does not reference the block model.

The 14% total — comprising ELOS skin dilution (approximately 11%) plus zero-grade schedule dilution (3%) — is applied deposit-wide as a single averaged value. The four geotechnical stope cases in Section 16.2.3 give different individual ELOS values — from 0.75 m hanging wall in Case 2 (longitudinal Southeast andesite and Central Andesite) to 2.0 to 3.0 m hanging wall in Case 3 (transverse along the Estaca fault at Guadalupe). A zone-specific dilution model is deferred to PFS once the geotechnical domain modelling and the detailed stope-shape optimisation have been refined.

16.8.2 Open-pit mining dilution

Open-pit mining dilution is applied as a flat 5% zero-grade dilution across all four open-pit complexes for the PEA. The 5% zero-grade factor reflects scoping-to-PEA-level allowance for face-cleanup inefficiency, contamination at the mineralised-material/waste boundary during load-and-haul, and small waste inclusions within the mineable inventory geometry that the pit design does not resolve. No separate ELOS-equivalent physical skin dilution is applied to the open-pit volumes at PEA stage; the dilution that would otherwise be captured by an ELOS analysis is embedded in the pit-design geometry and in the Section 14 resource-modelling process.

16.8.3 Mining recovery

Mining recovery is applied as a single deposit-wide averaged pair of values, distinguishing between stoped material and developed material:

- **Stope mining recovery:** 95%. Applied uniformly across the four underground deposits and across both longitudinal and transverse stope configurations. The 5% loss represents stope-wall material left against the paste-fill boundary, drawpoint losses during muck-out, and underbreak that is not recovered during the stope cycle.
- **Development mining recovery:** 100%. Applied to mineable-inventory drives and to mineralised-horizon development where mineralisation is intercepted by capital or operating development. Development material is tightly bounded by the development profile and is recovered in full.

The underground mineable inventory figure of 14.82 Mt delivered to the mill over the life of mine (see Section 16.5.5) reflects these recovery values applied to the stope-optimised inventory plus the production-drive development tonnage at 100% recovery. Approximately 13.02 Mt of the underground mineable inventory is from stoping; the balance is from production-drive development.

Open-pit mining recovery is assumed at 95% uniformly across the four open-pit complexes for the PEA, consistent with standard scoping-to-PEA allowance for pit-floor losses, overbreak management and face-cleanup efficiency. Open-pit dilution is embedded in the pit design and resource-modelling process at Section 14 and is not separately applied in Section 16.8 as a modifying factor.

16.8.4 Modifying-factor summary

The modifying factors applied across the PEA mine plan are summarized in Table 16-19. The factors apply uniformly across the four underground deposits and the four open-pit complexes respectively, consistent with the Section 16.8.1, Section 16.8.2 and Section 16.8.3 framework.

Table 16-19: Modifying-factor summary.

Component	Applied value	Basis
Underground ELOS skin dilution (HW + FW)	1.1 m (0.7 m + 0.4 m)	Geotechnical stope cases consolidated to PEA basis (Section 16.2.3)
Underground zero-grade dilution	3%	Schedule-level allowance for paste-fill boundary, operational-contamination, small waste inclusions
Underground total mining dilution	~14%	Sum of ELOS skin and zero-grade components

Component	Applied value	Basis
Open-pit zero-grade dilution	5%	Face-cleanup inefficiency, mineralised-material/waste boundary contamination, small waste inclusions
Underground stope mining recovery	95%	Stope-wall losses against paste-fill boundary, drawpoint losses, underbreak
Underground development mining recovery	100%	Mineable inventory sill material recovered in full
Open-pit mining recovery	95%	Pit-floor losses, overbreak management, face-cleanup efficiency

16.8.5 Structural control at Guadalupe and the dilution framework

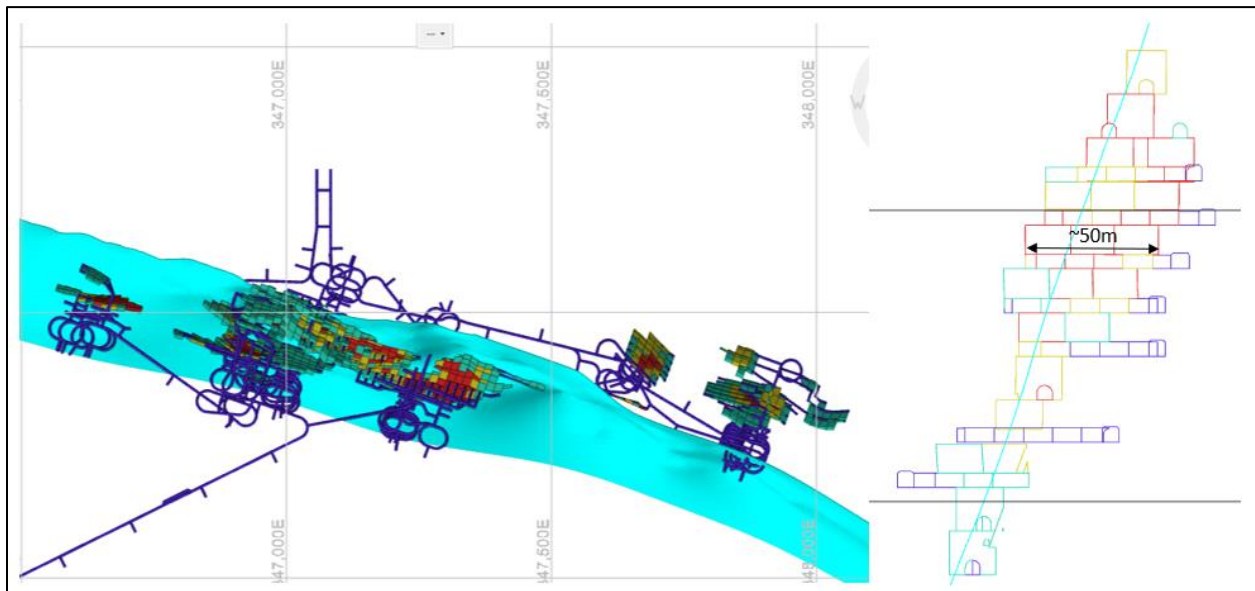
The Estaca fault runs parallel to the mineralised main vein at Guadalupe and sits in the middle of the orebody. The main zone of Guadalupe where the Estaca is most prominent is approximately 20 to 60 m wide across hanging wall to footwall. Because the PEA stope basis limits stope width to a maximum of 15 m (Section 16.2.3), the main-zone hanging wall-to-footwall span is broken into multiple stopes across the width — typically two to four stopes per level depending on the local width of the main zone. The consequence for mining dilution is material.

When the poor-quality rock mass associated with the Estaca fault fails during mining and contributes to stope-wall overbreak, the material that falls into the active stope is predominantly sourced from the adjacent stope or from the stope above (in the underhanded-sequenced portions), rather than from true waste rock. In economic terms, this overbreak is not adding metal to the mine plan; it is relocating metal from a neighbouring stope that will subsequently either not be mined in the same location (because the adjacent stope has effectively been pre-drawn) or, where the adjacent stope is mined, will be taken against a paste-fill boundary rather than a rock boundary. The dilution model captures the mass of the overbreak but not the fact that the overbreak is not mineralogically distinct from the mine's own inventory.

Stopes positioned directly on the mineralised-material/waste boundary — those whose hanging wall or footwall touches genuine waste rock at the margins of the main zone — could see the fault-related overbreak comprise a substantial proportion of waste in addition to the mineralised material, and dilution for those stopes may be higher than the deposit-wide average. Stopes that sit between other stopes within the wide mineralised main zone are less exposed, because their walls abut adjacent mineralised stopes or paste-fill rather than waste. Distinguishing the two cases is part of the zone-specific work recommended below.

At PEA stage this structural-control observation is treated qualitatively and is not propagated into a zone-specific modifying-factor model. The deposit-wide 14% dilution and 95% stope recovery/100% development recovery modifying factors are used consistently across the stope optimisation in Section 16.4.4, the production schedule in Section 16.5, and the capital and operating cost build-up referenced at Section 21, without differentiation between fault-influenced and unaffected portions of Guadalupe. Quantitative refinement — by domain modelling the fault-influenced volume, and by assigning selective dilution and recovery factors inside and outside the main-zone envelope — is recommended work for the PFS geotechnical and mine-planning programmes and will inform the zone-specific modifying-factor model at that stage.

Figure 16-16: Estaca fault surface and section plot



Source: AGP, 2026

16.9 Primary and secondary ventilation

This section presents the primary and secondary ventilation design basis for the PEA. The ventilation design is sized at a peak airflow demand derived from the underground fleet and is distributed across the five portal locations and the active return-air raise network. Year-by-year airflow profiling and detailed primary/secondary fan network calculations are appropriate work for the PFS; the PEA basis is sized to the peak demand and is consistent with the cost-model fan counts used to cost the installed ventilation infrastructure.

16.9.1 Design basis

Primary-airflow demand is sized from the peak concurrent underground diesel fleet using a design factor of $0.06 \text{ m}^3/\text{s}$ per installed diesel kW (consistent with the diesel-airflow design bases commonly applied under Canadian and Western Australian hard-rock guidance, adopted here as the PEA basis in the absence of a Mexican statutory per-kW design factor), applied to the rated power of each underground diesel unit, multiplied by the concurrent-unit count and weighted by the OEE-based utilisation figure for each equipment class (per Section 16.6.4). The $0.06 \text{ m}^3/\text{s}/\text{kW}$ factor is standard Australian and Canadian practice for a Mexican-type ventilation jurisdiction and is the same factor that underpins the fan sizing in the cost model.

A leakage allowance of +25% is applied to the fleet-driven base airflow to account for duct, raisebored-raise and stopping leakage under a shallow decline-access mine configuration. A further contingency of +15% is applied for simultaneity across the active zones, personnel demand, fixed-plant demand and fixed heat load. The combined allowance brings the design total airflow to approximately 140% of the fleet-driven base figure.

16.9.2 Peak airflow demand

The peak total airflow demand, established from the peak-year underground fleet (see Section 16.6.4), is approximately 726 m³/s (approximately 1.54 Mcfm). This is a peak-year value; year-by-year airflow profiling is a PFS deliverable.

Airflow enters the mine across the five portal locations described in Section 16.4.2 — the western double portal, the three central-trend single portals, and the eastern double portal at Guadalupe East. During the early development phase, when the return-air raise network is not yet in service, the double portals operate with one opening as fresh-air intake and the other as return-air exhaust; this provides the temporary ventilation circuit used during the Year 0 decline establishment and early Year 1 development. Once the return-air raises are in service with the primary exhaust fans installed on surface at the RAR collars, all seven portal openings across the five locations (both openings at each of the two double portals plus the three central-trend single portals) operate as fresh-air intakes, and the return-air raises become the sole exhaust airways for the mine. The distribution of airflow across the seven intake portal openings and the active return-air raises is managed through regulators located at each RAR-level connection; the regulators control the airflow permitted at each working level and balance the network against the active-zone configuration in any given year.

16.9.3 Primary ventilation system

The primary ventilation system is built around three to six active RARs operating concurrently at any one time, depending on the active-zone configuration. Each RAR is a 4.5 m diameter raisebored development (per Section 16.4.3) connected to a primary exhaust fan on surface. The number of concurrent RARs reflects the horizontal footprint of the mine — four underground deposits spread across the east-west axis of the Project area — rather than a single deep-shaft configuration.

The financial model for the PEA carries three primary exhaust fans as the installed-fan PEA assumption, with the three fans installed on surface at three of the active return-air raises at any given time in the life of mine. The operating configuration is selected year-on-year to match the active zones and the airflow-distribution requirement: fans move between RARs as the active mining footprint evolves over the life of mine. Fan shaft power per RAR ranges from approximately 580 kW in a minimum-spread configuration with three RARs active (242 m³/s per RAR), down to approximately 290 kW in a maximum-spread configuration with six RARs active (121 m³/s per RAR). Installed motor power is sized at approximately 1.15 times the shaft-power duty point, giving installed motor ratings in the range 335–670 kW per primary fan.

Primary-fan total pressure is sized at approximately 1,800 Pa, consistent with a shallow decline-access mine and short intake columns. Primary-fan total efficiency (hydraulic × motor) is taken at 75% for PEA sizing. Table 16-20 summarises the primary-fan sizing range.

Table 16-20: Primary fan sizing by RAR configuration (PEA).

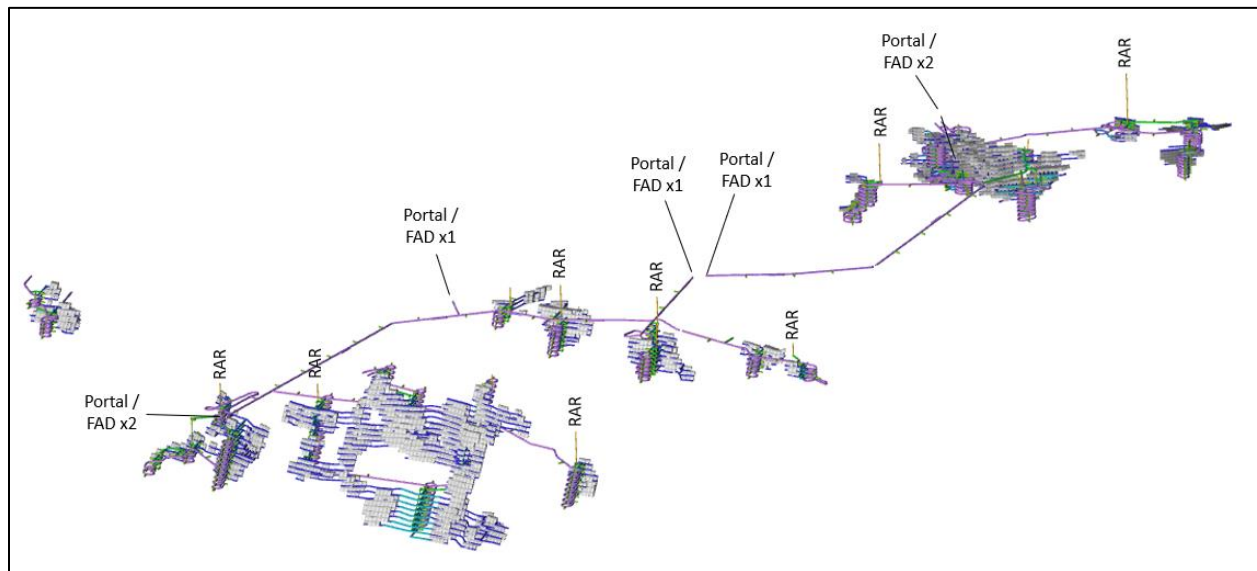
Configuration	Active RARs	Airflow per RAR (m ³ /s)	Fan shaft power per RAR (kW)	Installed motor per fan (kW)
Minimum spread	3	242	580	670
Maximum spread	6	121	290	335

16.9.4 Secondary ventilation

Secondary ventilation is provided by 110 kW auxiliary fans at each active sub-level access and by 55 kW auxiliary fans at individual working headings. The secondary fans are mounted on the decline at the level of each active sub-level and are ducted into the face. Over the life of mine, the financial model carries an LOM-average of approximately 55 secondary fans (combined 110 kW sub-level and 55 kW heading fans) as the installed-fan assumption, consistent with the peak-year underground fleet and the number of active working headings and sub-levels across the four deposits.

Secondary-fan duty points are a direct consequence of the heading length, the heading profile and the airflow demand at each face. At PEA stage the secondary-fan fleet is costed on an averaged basis without detailing the duty-point per face; PFS will schedule the secondary-fan fleet with finer time resolution and match individual fan sizing to each active face. Consistent with PEA-level scope, only peak underground airflow demand has been calculated; a year-by-year LOM airflow profile will be developed at PFS stage from the time-resolved fleet-hour and utilisation schedule.

Figure 16-17: Ventilation schematic



Source: AGP, 2026

16.9.5 Limitations

The PEA ventilation design is a peak-airflow sizing, not a year-by-year network simulation. A full-network simulation in Ventsim or equivalent software is recommended for the PFS and will provide year-by-year airflow distribution, resistance and regulator settings, fan duty-point tracking, and heat-load and auto-compression treatment. The following items are flagged for PFS attention:

- **Year-by-year airflow demand profiling** — the 726 m³/s figure is a peak value; year-by-year demand tracks the fleet profile and will be lower than peak in the early and late years of the mine life.
- **Heat load and auto-compression** — Los Reyes is a low-altitude operation with total mining depth below surface of less than approximately 500 m, so heat and auto-compression effects are expected to be small; the PEA does not explicitly model these loads and the PFS should confirm them.

- **Primary cooling/heating** — the shallow low-altitude configuration is not expected to require primary cooling, but the PFS should confirm with in-situ temperature measurements at depth.
- **Gas monitoring** — default regulatory minima under EPA MSHA/Mexican NOM apply; the PFS should confirm the specific gas-monitoring requirements and the radon monitoring basis.
- **Network simulation and regulator design** — the detailed airflow-distribution network, the regulator placements, the fan-duty selection by operating year and the emergency/escape ventilation scenarios are all PFS deliverables.

16.10 Underground services

This section presents the underground services design basis for the PEA, covering electrical reticulation, compressed-air distribution, communications and positioning, dewatering reticulation and paste-fill reticulation. Service capacities are sized to the peak-year underground fleet and the peak-year mining rate. Open-pit services are limited in scope: open-pit dewatering is described in Section 16.3.3, and open-pit power for dewatering pumps, lighting and ancillary loads is supplied from the site electrical reticulation described in the Project Infrastructure section.

16.10.1 Electrical reticulation

Electrical power is supplied from the main surface substation through a dedicated underground reticulation that distributes power to each active zone of the mine. Main substation supply, surface transformers and the main surface-to-underground feeder arrangement are described in the Project Infrastructure section and are cross-referenced rather than repeated here.

Underground distribution is built around a main-line cable in the decline that carries power from the surface substation to a series of underground-transformer substations at strategic points along the decline and at the entry to each active mining zone. Each underground substation steps the feeder voltage down to the working-level distribution voltage used for the fixed-plant loads (dewatering pumps, primary and secondary fans, paste-fill reticulation pumps, compressed-air compressors, communications backbone) and for the mobile-fleet plug-in points at each active face.

Peak underground electrical demand is driven by the combined load of the dewatering pumps (Section 16.3.2), the secondary ventilation fans (Section 16.9.4), the paste-fill reticulation pumps (Section 16.10.5), the compressed-air compressors (Section 16.10.2), and the personnel and lighting loads across the active development faces and stoping headings. The peak-year installed electrical capacity is summarised in Table 16-21.

16.10.2 Compressed air

A central compressed-air supply is provided from a compressor house located on surface near the western portal and distributed underground through a main air-reticulation line running along the decline. Compressed air is used for jumbo and longhole drilling, for stope-service air at each active stope, for paste-fill line cleaning and flushing, and for ancillary-air duty across the active headings.

Main compressor house capacity is sized to the peak concurrent-drilling demand across the underground operation, with compressors sized at a 15-to-20% margin above the peak demand to allow for maintenance outages and for transient peaks in drilling activity. The peak-year installed capacity is summarised in Table 16-21. Air-reticulation pressure is maintained at approximately 700 kPa at the working face, with pressure drop across the reticulation line managed through line-diameter sizing and through receiver tanks at strategic points in the network.

16.10.3 Communications and positioning

Underground communications are provided by a leaky-feeder backbone running the length of the decline and extending into each active zone, augmented by a Wi-Fi mesh network at strategic locations to support the fleet-management system, the personnel-tracking system and the supervisory data links. Voice communication is carried on the leaky-feeder as the primary channel, with Wi-Fi providing data and video channels where required.

Fleet positioning and tracking are carried on the same Wi-Fi mesh, with active transponders on each mobile unit reporting position, status and productivity data back to the Technical Services group at surface. The fleet-management platform is standard off-the-shelf software; vendor selection is deferred to the PFS.

Emergency communications and personnel tracking are integrated with the leaky-feeder backbone and the Wi-Fi mesh, with redundant pathways between the underground communications and the surface control room. Refuge stations are positioned consistent with the regulatory egress-distance requirements and are equipped with dedicated communications to surface.

16.10.4 Dewatering reticulation

Dewatering reticulation carries intercepted groundwater from the underground workings to surface through a gravity-drainage and pump-chain configuration. The strategy is described in Section 16.3.2. The underground dewatering chain pumps to surface via the main decline to the surface water-management ponds. The 60 L/s is the groundwater-inflow pumping allowance; mine service water is recirculated and is not additive. The installed-capacity figures are summarized at Table.

Water drains by gravity from each working level through dedicated drainholes that connect the sump access at each level downward to a settling sump. Settling sumps are located at every four to five levels along the decline and collect water from the drainhole network above. At each settling sump, water is cleaned by decanting — solids settle out in the sump and the clarified water is pumped onward — before being lifted to the next pump station up the decline.

The pump chain within the decline is built around a travelling centrifuge pump at each settling sump that pumps clarified water to a mono-pump (screw positive-displacement pump) station, which in turn pumps the water up the decline to the next mono-pump station. The mono-pump chain continues up the decline in a series of stages to the surface collection point. The mono pumps are selected for their positive-displacement handling of the intercepted water and for their compatibility with the sump-to-sump elevation difference between stages.

At each working heading — whether a development face, a stope drawpoint or an active services-installation area — a travelling pump is either stationed on the heading or travels with the jumbo or longhole rig servicing the heading. The travelling heading pumps lift accumulated heading water to the nearest gravity-drainhole access and discharge it into the drainhole network, from which the water is collected at the next settling sump down the decline.

Steady-state dewatering design flow is approximately 60 L/s, with short-term peak capability sized to approximately 90 L/s to accommodate wet-season peaks and incident-response scenarios.

16.10.5 Paste-fill reticulation

Paste-fill reticulation covers the underground piping that delivers paste-fill from the central-valley paste plant to each active stope. The paste plant is located on surface in the central valley at approximately the same elevation as the central portals, and the paste-fill reticulation runs directly from the paste plant into

the underground network through one of the central portals — no vertical boreholes from surface are required to enter the underground network.

Underground, paste is conveyed horizontally along the main haulage ways that connect the various orebodies to the central portals. Where paste must move from one level to another within the underground network — for example from the main haulage level to a higher or lower working level — short internal boreholes between levels are used rather than routing the piping up or down the declines. The horizontal-piping-with-internal-boreholes configuration keeps the paste reticulation off the busy decline haulage corridor and avoids the pressure-drop management that a decline-routed paste line would require.

Pour rings at each active stope access the stope from the overcut development level above the stope via a dedicated paste-delivery pipe, with isolation valves and flushing capability at each pour ring to allow line clearance between pours and to accommodate the 14-day curing period before a pour ring is rotated to the next stope in sequence.

Peak underground paste-delivery rate is summarised in Table 16-21 and is consistent with the 3,000 m³/day maximum delivered rate in Section 16.5.3.

16.10.6 Installed service capacities

Table 16-21: Installed underground service capacities (peak-year basis).

Service	Peak-year installed capacity	Unit
Underground electrical distribution	25	MVA
Compressed air supply	200	m ³ /min
Leaky-feeder voice backbone	Full-mine coverage	—
Wi-Fi mesh data backbone	Full-mine coverage	—
Dewatering, steady state	60	L/s
Dewatering, short-term peak	90	L/s
Paste-fill delivery, peak	3,000	m ³ /d

Service capacities are sized to the peak-year underground fleet and the peak-year mining rate described in Section 16.5. Year-by-year service-capacity profiling is deferred to PFS; the peak-year figures in Table 16-21 are the installed-capacity basis used for the capital cost estimate and drive the mine-services cost line in Section 21.

16.11 Paste backfill

This section presents the paste-backfill strategy adopted for the PEA, including the fill strategy, the paste-plant location and distribution approach, the paste specification, the underground reticulation, the filling of the historical voids in the Guadalupe East zone, and the cross-references to the Project Infrastructure section for paste-plant process design.

16.11.1 Fill strategy

Stopes at Los Reyes are mined at 100% extraction with 100% paste backfill. No pillars are carried between stopes in either longitudinal or transverse stope configurations. The fill strategy is consistent across all four underground deposits.

For longitudinal stope mining at Zapote-Tahonitas and Central (Cases 1 and 2 of Section 16.2.3), each stope panel is mined with the stoping face retreating toward the access, and paste fill is placed into each mined stope as the face advances. Paste is poured from the overcut access through dedicated paste-delivery holes and the stope is filled behind the advancing face on a cycle compatible with the 14-day paste curing period described in Section 16.5.3.

For transverse stope mining at Guadalupe (Cases 3 and 4 of Section 16.2.3), stopes are mined in an underhanded sequence. The initial A-panel stope is taken with the stope back in rock, mucked out and paste-filled; subsequent B-panel stopes below the A-panel are taken with their back in the cemented paste backfill placed in the A-panel rather than against the poor-quality rock mass associated with the Estaca fault. This sequence is the basis for the stope-shape inputs described in Section 16.4.4.

Life-of-mine paste volume is approximately 6.53 Mm³ delivered underground. The volume is calculated using the paste-volume formula described in Section 16.5.3 (paste-fill volume equals in-situ stope volume times (1 + paste dilution) times mining recovery, plus a 250 m³ allowance for the drive volume immediately in front of the paste wall). The 6.53 Mm³ is the operating stope-backfill volume only; the historical-void fill at Guadalupe East is additional and is capitalised separately as a dedicated capital line item (see Section 16.11.5), and is not included in this total.

16.11.1.1 Paste plant location and distribution

The paste plant is located on surface in the central valley of the Project area, at approximately the same elevation as the central portals. The central-valley location minimises the average horizontal-pumping distance from the paste plant to the orebodies being filled, since the central valley sits between the western-end processing plant and the eastern (Guadalupe/Guadalupe East) end of the underground mine.

Tailings are pumped from the processing plant at the western end of the Project area to the paste plant in the central valley via the underground haulage network. The tailings-transfer piping enters the underground through the western portal, runs horizontally along the underground haulage way that connects the western end of the mine to the central portals, and exits the underground through one of the central portals to the paste plant. Routing the tailings-transfer piping through the underground haulage way — rather than across surface — avoids the long surface haul across the ridges that separate the western processing-plant valley from the central valley, in the same way that the material-flow routing in Section 16.4.6 routes mineable inventory through the underground haulage.

Buffer storage at the paste plant absorbs short-term mismatches between tailings production at the processing plant and paste delivery to the underground stopes.

From the paste plant, paste is pumped into the underground network through one of the central portals (the paste plant sits at approximately the same elevation as the portals, so no vertical entry boreholes are required). Underground, paste is distributed horizontally along the main haulage ways, with short internal boreholes used between levels as described in Section 16.10.5. Given the long horizontal runs from the paste plant to the furthest underground stopes, gravity-feed paste delivery is not feasible; paste delivery is pumped from the paste plant to the active stope through the paste-fill reticulation. The peak paste-delivery rate underground is 3,000 m³/d, consistent with the Section 16.5.3 productivity basis.

16.11.2 Cement and services to the paste plant

Cement is delivered by road to a surface transfer facility near the western portal and carried underground on mine-spec haulage along the main haulage way to the central-valley paste plant; highway trucks do not travel underground. The cement-logistics arrangement will be confirmed at PFS. Routing cement through the underground haulage rather than constructing a dedicated surface access road to the central-valley plant avoids a significant surface-works scope — the same ridge-crossing haul profile discussed in Section 16.4.6 for mineable inventory flow applies to cement delivery to the central valley. Cement storage at the paste plant is sized to buffer approximately two weeks of consumption, accommodating supply-chain variability without constraining underground paste-fill delivery.

Power to the paste plant is supplied either from the underground electrical circuit through a branch reticulation routed via the underground haulage way, or from a dedicated surface power line routed alongside the cement-delivery route. The final power-supply configuration is a PFS deliverable and is carried as a design option for the paste plant in the Project Infrastructure section.

16.11.3 Paste specification

Paste cement content is set at a flat 5% binder across all stopes for the PEA. The flat 5% is a PEA-level simplification that envelopes the full range of pour-by-pour binder requirements across the four deposits under a single conservative value. Paste strength development at 5% binder is consistent with the 14-day curing period adopted in the stope cycle (Section 16.5.3) and is compatible with the underhanded sequence at Guadalupe transverse mining where subsequent stope backs bear against previously placed paste. Paste backfill is designed to develop approximately 400 kPa for adjacent side-wall exposure and approximately 3.0 MPa for undercut exposure, where a subsequent stope is mined beneath cured paste. The flat 5% binder is adopted for the PEA cost basis; the binder content and curing schedule required to achieve these exposure strengths will be confirmed by paste testwork at PFS. Adjacent-stope extraction against a cured paste-fill wall is sequenced after the 14-day curing period, by which point the fill has developed the approximately 400 kPa side-exposure strength (and approximately 3.0 MPa where the adjacent stope is mined beneath cured paste).

A design upside that has been identified but not incorporated into the PEA basis is the reduction of cement content in the bulk pour above the initial 7 m paste plug at the base of each stope. Under a reduced-cement bulk-pour basis, the lower 7 m of each stope would be poured at a higher cement percentage to provide the initial structural strength of the stope fill, and the balance of the stope would be poured at a reduced cement percentage. This refinement is contingent on paste testwork and is appropriate work for PFS or FS stage; it carries a material cost-reduction opportunity at the paste-cement line that is not captured in the PEA cost build-up.

16.11.4 Historical voids in the Guadalupe east zone

The Guadalupe East zone contains a set of historical underground workings that must be filled before the modern mine plan can mine through them safely. The historical voids at Guadalupe East are filled using the paste-fill system in a dedicated programme sequenced across the early years of the mine plan. The historical voids are reasonably well surveyed (a best-efforts survey to date, to be refined at PFS). The void-fill volume is tracked separately as a dedicated capital line item and is not included in the 6.53 Mm³ life-of-mine operating paste-fill volume. Paste is delivered to the voids from the top (overcut) drive where access is available; where it is not, by upholes from the bottom (undercut) drive with a breather hole into the void or stope crown to vent displaced air as the void fills.

Paste-filling of the historical voids is carried out in three phases across Year 1, Year 2 and Year 3. Mining of the Guadalupe East zone begins in Year 1 and proceeds in mineable inventory away from the historical voids; there is ample mineralised material available in the zone outside the immediate void volume to support Year 1 production while the first phase of void-filling is carried out. From approximately Year 2 the progress of Guadalupe East mining brings the working faces closer to the historical voids, and void-filling begins to be on the critical path. By approximately Year 3 the filling of the historical voids is completed and mining proceeds through the previously-void volumes in the standard stoping sequence.

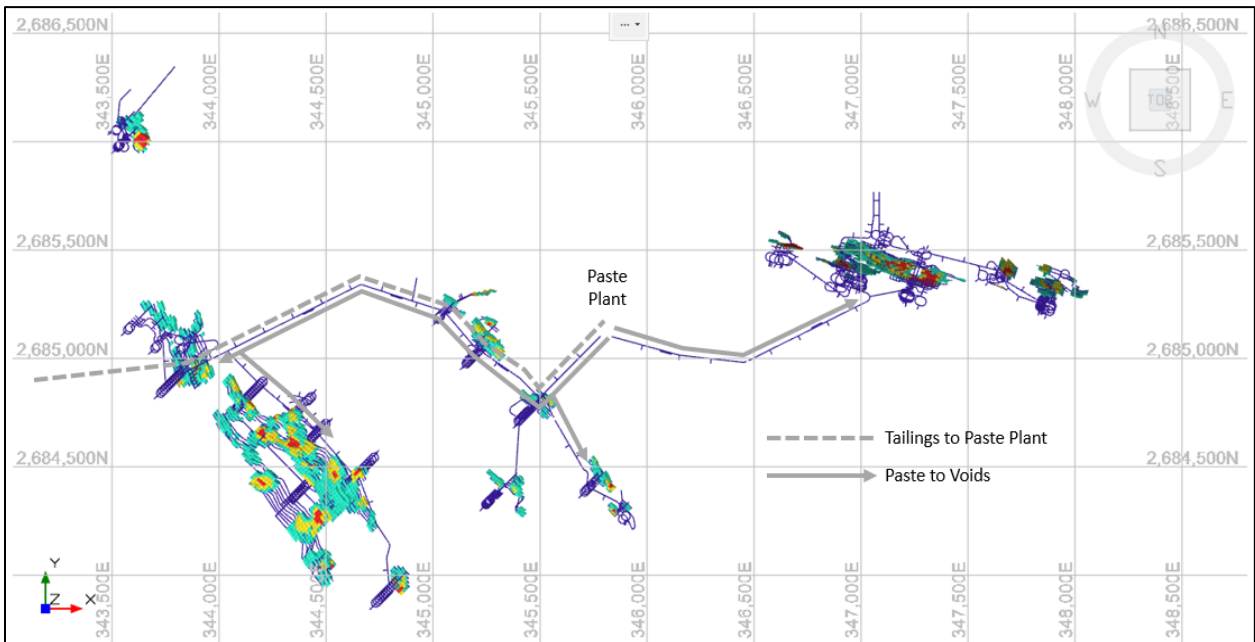
The three-year filling window provides a contingency of approximately one year for commissioning or capacity delays on the paste plant: if paste-plant commissioning slips by up to a year relative to the current schedule, Guadalupe East mining can continue on the ample mineable inventory available away from the voids, and the void-filling programme can be completed within the early-year window without materially affecting the production schedule. Beyond approximately one year of slip, the Guadalupe East mining face would begin to be materially constrained by the unfilled voids; this timing buffer is explicitly reflected in the Section 16.12 factors-that-may-affect discussion.

16.11.5 Underground reticulation

Paste-fill reticulation underground comprises the horizontal piping that runs along the main haulage ways from the central-portal entry point to each active orebody, short internal boreholes between levels where the paste line has to change elevation, and the pour-ring distribution from each overcut access to the active stopes. The horizontal reticulation follows the same east-to-west alignment as the material-haulage network described in Section 16.4.6 and uses the same main haulage corridor. No vertical boreholes from surface are required because the paste plant sits at approximately the same elevation as the central portals and the reticulation enters the underground through the portal rather than from above.

Pour rings at each active stope are accessed from the overcut development level above the stope. Each pour ring has isolation valves and flushing capability to allow line clearance between pours and to accommodate the 14-day curing period before a pour ring serving a completed stope is de-commissioned and the line is routed to the next stope in sequence. The annual paste-fill delivered tonnes across the LOM, including the Guadalupe East void-filling programme in Years 1 through 3, are shown on Figure 16-19.

Figure 16-18: Paste plant and underground distribution schematic



Source: AGP, 2026

Figure 16-19: Paste-fill delivered tonnes per year.



Source: AGP, 2026

16.11.6 Paste plant process design — cross-reference

Process design and detailed metallurgical engineering for the paste plant — including the thickening, mixing and pumping equipment, the binder handling and dosing, the water balance around the plant, the tailings reception and buffer storage, the instrumentation and control philosophy, and the plant layout — are presented in the Project Infrastructure section and are not repeated here.

16.12 Factors that may affect the mining method

This section summarises the PEA-level factors that may affect the mining method, the production schedule or the mining inventory, together with the actions recommended for each factor at PFS or FS stage. The factors are drawn from the preceding sections of this section and from the broader design-basis and modifying-factor framework. Table 16-22 summarizes the factors and the direction of sensitivity.

16.12.1 Geotechnical uncertainty

The geotechnical design basis (Section 16.2) rests on the current rock mass characterization which is based on limited data. Additional geotechnical work is recommended as the project advances to PFS, including zone-specific geotechnical drilling, laboratory strength testing, and structural mapping to refine the rock mass characterization. The design input can then be updated. If the rock-mass conditions are found to be worse than the PEA basis in a particular zone, the stope dimensions basis for that zone could decrease, the expected overbreak could increase, and the ground support requirements could increase. This would have a corresponding impact on mining costs and a decrease in extracted tonnage per stope. Conversely, confirmation of better rock-mass conditions in specific zones could support larger stope dimensions or decreased overbreak, and a modest tonnage uplift.

16.12.2 Overhand sequencing at Guadalupe east

The stopes at Guadalupe East are predominantly sequenced underhand (top-down), consistent with the geotechnical design basis in Section 16.2. The exception is a minor portion at the top of the Guadalupe East zone, which has been sequenced in the PEA as overhand (bottom-up) for local geometry reasons. The portion involved has been assessed as immaterial to the overall production schedule, and a sequence revision to underhand is not expected to materially change the life-of-mine tonnage, the schedule window or the economic outcome. The sequencing will be revised at PFS against the refined geotechnical model for Guadalupe East and the refined stope inventory; the revision is a localised optimisation rather than a structural change to the mine plan.

16.12.3 Stopes close to the western pit walls

A small number of underground stopes are located immediately behind the final open pit wall of the Zapote-Tahonitas pit complex. At PEA stage the pit wall and the underground stopes have been designed independently and the interaction between them has not been managed in detail. At PFS either the stopes in question will be mined with additional stand-off and geotechnical monitoring, or the pit-wall design at that sector will be locally adjusted to increase the separation. Both options are immaterial to the overall production schedule; the number of stopes involved is small and the tonnage implication is within the PEA-level precision of the mining inventory.

16.12.4 Ventilation-raise and pit interactions

A small number of ventilation raises in the PEA design intersect the final pit footprints of the open-pit complexes. At PEA stage the raise locations and the pit limits have been designed independently. At PFS the intersecting raises will either be scheduled with finer time resolution so that raise use and pit mining do not coincide, or the raises will be relocated outside the pit footprint. Both options are immaterial to the overall production schedule or to the mine-wide airflow distribution.

16.12.5 Throughput sensitivity

The 5,000 t/d mill throughput adopted for the PEA was tested against 4,000 t/d and 6,000 t/d alternatives on a mining-cost basis in Section 16.1.9. The three cases are close on the mining-cost comparison and the 5,000 t/d case is the marginal preference. At PFS the throughput selection should be revisited once the processing and infrastructure cost build-ups are refined and the Mineral Resource estimate is updated, both of which materially affect the ranking.

16.12.6 Sub-level caving

Sub-level caving (SLC) has been retained in the Project design as an alternative mining method for the Guadalupe East zone only. If SLC is considered in future mining plans, it would require re-optimisation of the Guadalupe East mining inventory under the caving method, revision of the Guadalupe East production schedule, and updating of the capital and operating cost build-ups for the Guadalupe East portion of the mine. The long-hole open stoping method and paste-fill strategy adopted for the remaining three underground deposits would be unaffected. Under an SLC mine plan for Guadalupe East, paste backfill of the Guadalupe East zone would no longer be required, which would remove the Guadalupe East paste-fill demand and could reduce paste-plant capacity and the associated capital; the cavability and surface-subsidence implications would require assessment. This is an option only and is not in the PEA base case. This is the second and final mention of sub-level caving in Section 16 of the Technical Report.

16.12.7 Open-pit design dependencies

The open-pit design basis in Section 16.4.5 rests on PEA-level slope parameters (Section 16.2.6) and on pit-optimization inputs including price, royalty and plant recovery. The PFS open-pit program should confirm the slope parameters through pit-sector-specific geotechnical characterization and analysis and should refine the pit-optimisation inputs — particularly the price basis and the detailed operating cost per tonne by pit — to support a re-optimised pit shell. Material changes to the pit shell at PFS would feed back into the Section 16.4.5 pit design and into the Section 16.5 production schedule.

16.12.8 Resource model dependencies

The mining inventory presented in Section 16.1.4 is derived from the Mineral Resource Estimate at Section 14. A number of Section 14 placeholder values and modelling assumptions that the PEA mine plan is built on should be confirmed at PFS: the classification split between Measured, Indicated and Inferred within each of the four underground deposits, the spatial distribution of the Inferred component (particularly in the late-LOM scheduled windows), the grade-thickness distribution within the Estaca-fault-influenced volume at Guadalupe (which underpins the Section 16.8.3 structural-control mitigant), and the metallurgical characterisation of any zones that have not been testwork-sampled at PEA stage. Material changes to the Mineral Resource Estimate at PFS would feed back into the stope optimization in Section 16.4.4 and into the production schedule in Section 16.5.

16.12.9 Commodity-price sensitivity

The mine plan is relatively insensitive to metal price within the range tested at PEA stage. A 50% increase in the AuEq metal price applied to the stope-optimisation cut-off (as a test of the mine plan's response to upside price conditions) resulted in an approximately 15% increase in the stope-optimised AuEq tonnage. The result indicates that the mine plan is grade-driven rather than price-driven, and that further price upside converts into marginal additions to existing stope shapes rather than into the opening-up of previously-uneconomic zones. This observation was developed in Section 16.1.12.

16.12.10 Factors summary

Table 16-22: Factors that may affect the mining method — summary

Factor	Section	Direction of sensitivity	PFS-stage action
Rock-mass quality (by zone)	16.2/16.8	Worse conditions → tighter stope dimensions, lower tonnage	Zone-specific geotechnical drilling and domain modelling
Estaca-fault structural control at Guadalupe	16.8.3	Upside on dilution (mineralised overbreak) not in PEA basis	Domain model the fault zone; apply selective modifying factors
Overhand sequencing at Guadalupe East	16.12.2	Immaterial to LOM; localised sequence refinement	Revise sequence with refined geotechnical model
Stopes close to western pit walls	16.12.3	Immaterial to LOM; local stand-off or pit adjustment	Manage interaction in PFS
Ventilation-raise/pit interactions	16.12.4	Immaterial to LOM; schedule or relocation	Refine raise schedule or relocate outside pit footprint
Throughput (4/5/6 kt/d)	16.1.9/16.12.5	Close on mining-cost basis; integrated economic basis at PFS	Re-test with refined processing and infrastructure cost
SLC Option	16.1.8/16.12.6	Activation would require Guadalupe East re-optimisation	PFS geotechnical program outcome-dependent
Open-pit slope parameters	16.2.6/16.4.5	Scoping-level slopes; PFS refinement	Pit-sector-specific geotechnical and slope-stability analysis
Mineral Resource classification split	16.1.4/16.12.8	Inferred-to-Indicated conversion → schedule and NPV upside	Infill drilling and updated Mineral Resource Estimate
Commodity-price sensitivity	16.1.12/16.12.9	50% price → ~15% AuEq tonnage change; grade-driven plan	Hold sensitivity as LOM upside
Cut-off grade (underground)	16.1.11	1.5 g/t AuEq applied (above 1.2 g/t break-even on NPV grounds)	Re-test cut-off at PFS with refined schedule and cost inputs
Paste-plant commissioning timing (Guadalupe voids)	16.11.5/16.12.11	~1-year buffer against paste-plant delay	Track paste-plant critical path at PFS
Cement content reduction (bulk pour above 7 m plug)	16.11.4	Cost-reduction upside not in PEA basis	Testwork programme; apply at FS if supported

16.12.11 Paste-plant commissioning and the Guadalupe east buffer

The Guadalupe East mining sequence is built on the assumption that the paste plant commissions on schedule and that the three-phase filling of the historical voids at Guadalupe East (Section 16.11.5) is completed by approximately Year 3. A delay of up to approximately one year in paste-plant commissioning is absorbed by the available mineable inventory in Guadalupe East away from the historical voids, as discussed in Section 16.11.5. Delays beyond approximately one year would begin to materially constrain the Guadalupe East mining face and would require a re-sequencing of the zone's production against the then-updated paste-plant commissioning profile. The paste-plant commissioning is on the critical path at PFS and should be tracked with appropriate scrutiny as the Project advances. A paste-plant delay would also affect the other underground zones, which are mined at full extraction with paste backfill — principally Zapote-Tahonitas Main (ZTM) and the satellites. In a delay scenario, stoping across these zones would be constrained by available paste-fill capacity and would need to be re-sequenced against the paste-plant ramp-up; the one-year buffer and early-years stoping flexibility apply mine-wide, not only to Guadalupe East.

17 Recovery methods

17.1 Overview

The process flowsheet for Los Reyes Project is based on the preliminary metallurgical testwork discussed in Section 13. The plant is designed to process 5 kt/d of mineralised material to produce gold and silver doré. The processing design consists of crushing, grinding, cyanide leaching, counter-current decantation (CCD) and precious metal recovery via the Merrill-Crowe process. Tailings materials are detoxified using the SO_2/O_2 cyanide detoxification process, thickened, and filtered before being dry-stacked on a filtered tailings storage facility. When cemented paste is required for backfilling underground, detoxified tailings are diverted to the paste plant near the underground mine portal and cemented paste is prepared before pumping to the underground reticulation system.

The design is based on completed test work programs performed on the deposit, Ausenco's database of reference projects, and in-house process modelling. The plant has been designed with assumed availability of 65% for crushing stage, 85% for tailings filtering, 92% for grinding, and 65% for paste plant. The plant will operate on a two 12-hour shifts per day schedule, 365 days per year.

17.2 Process Design Criteria

Table 17-1 presents the design criteria developed for the process plant.

Table 17-1: Process Design Criteria

Parameter	Units	Value
Plant capacity	t/d	5,000
Life of mine	years	15
Gold head grade, LOM average	g/t	1.68
Silver head grade, LOM average	g/t	63.4
Gold head grade, design	g/t	2.27
Silver head grade, design	g/t	90.1
Sulphur, design	%	1.4
Crushing availability	%	65
Grinding Availability	%	92
Filtering Availability	%	85
Paste Plant Availability	%	65
JK Axb, design	-	37.2
Bond ball work index, design	kWh/t	18.5
Bond abrasion index, average	-	0.396
Specific gravity	-	2.57
ROM feed size, F_{100}	mm	600
ROM feed size, F_{80}	mm	317
Ball mill recirculating load	%	400

Parameter	Units	Value
Grinding product size, P_{80}	μm	75
Primary cyclone overflow density	% solids (w/w)	35
Leach residence time	h	72
Pre-leach thickener settling rate	t/h/m^2	0.7
Pre-leach thickener underflow density	% solids w/w	50
CCD suspended solids in last thickener overflow	ppm	<500
CCD settling rate	t/h/m^2	0.7
CCD underflow density	% solids w/w	55
CCD wash ratio	$\text{m}^3 \text{ water: m}^3 \text{ water in feed slurry}$	3.5
Clarification filter suspended solids in product	mg/L	<10
Clarifier rise rate	m/h	1.0
Clarification filter specific capacity	$\text{m}^3/\text{m}^2/\text{h}$	2.2
Tailings thickening, solids flux loading rate	$\text{t/m}^2/\text{h}$	0.7
Tailings thickener underflow density	% solids w/w	60
Tailings filter stock tank residence time	h	4
Cyanide detoxification residence time	min	90
Detox Circuit Pulp Density (solids), nominal	% w/w	43
Specific tailings filtration rate	kg/h/m^2	126-150
Paste plant capacity, design	Mm^3/a	0.71
Paste plant capacity, design	Mt/a	1.21

17.3 Process Plant Description

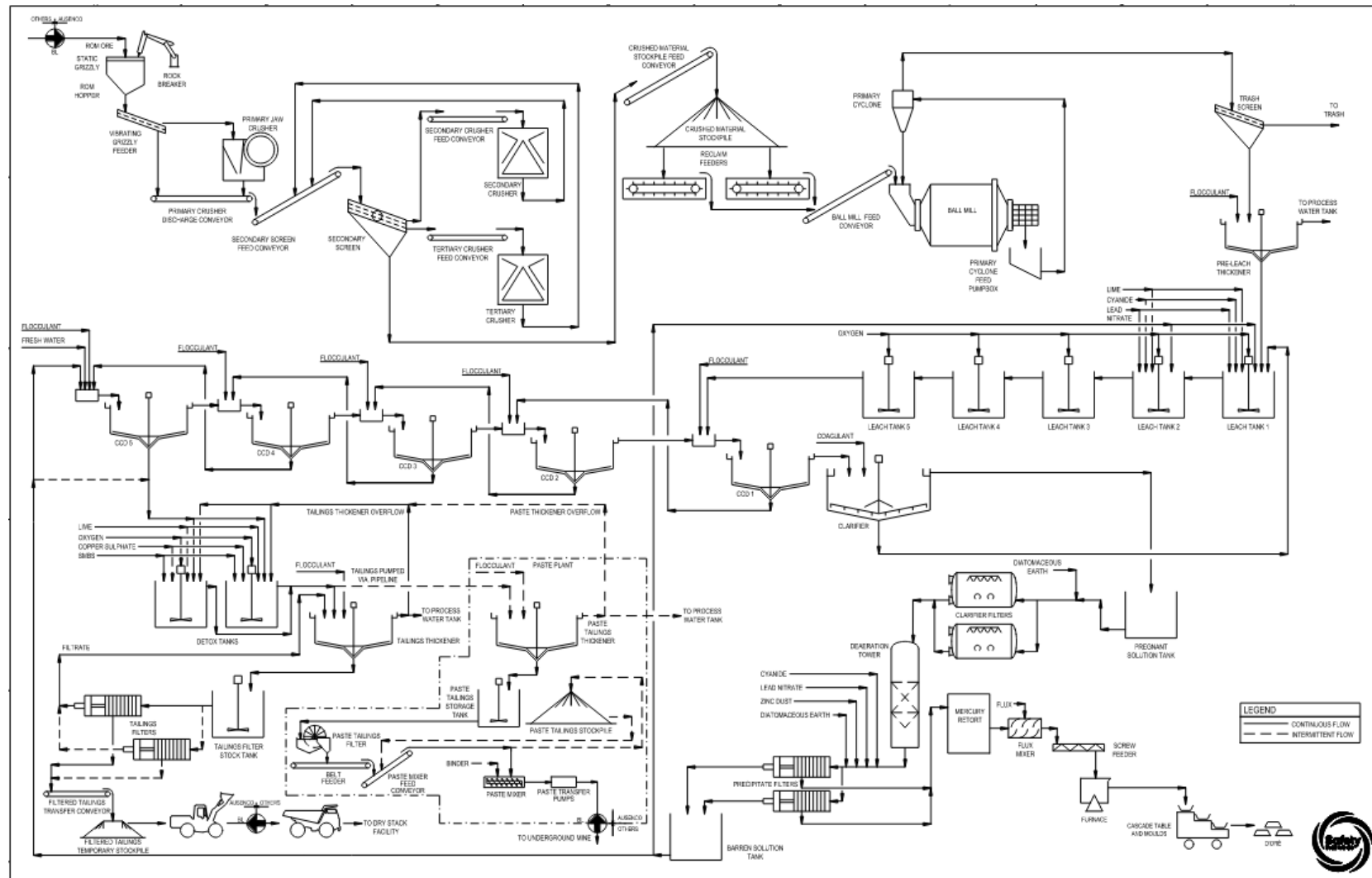
The process design consists of the following major components:

- Three-stage crushing of run of mine (ROM) material
- Ball milling in closed circuit with a classifying cyclone pack
- Whole-ore leaching of the cyclone overflow
- Counter-current decantation (CCD)
- Merrill-Crowe circuit
- Retorting and smelting to produce doré
- Tailings cyanide detoxification
- Tailings thickening and filtering
- Paste backfill plant.

17.4 Process Flowsheet

An overall process flow diagram showing the unit operations in the selected process flowsheet is presented in Figure 17-1.

Figure 17-1: Process Flowsheet



Source: Ausenco, 2026.

17.5 Plant Design

17.5.1 Crushing Circuit

The crushing plant uses a three-stage closed circuit to reduce the size of run of mine (ROM) material from an 80% passing feed size (F_{80}) of 317 mm to an 80% passing product size (P_{80}) of 10 mm.

ROM material will be hauled from the mine and dumped into a hopper that feeds a vibrating grizzly screen. Grizzly oversize reports to a jaw crusher, after which the crushed product is recombined with the grizzly undersize and conveyed to a double-deck vibrating screen. Oversize from the upper deck is directed to the secondary crusher, and lower-deck oversize to the tertiary crusher. The discharge of secondary and tertiary crusher returns to the secondary screen feed conveyor. Screen undersize is conveyed to the stockpile.

Major equipment in this area include:

- Static grizzly (above the ROM hopper)
- Vibrating grizzly feeder
- Primary jaw crusher and discharge conveyor
- Secondary screen and secondary screen feed conveyor
- Secondary crusher and secondary crusher feed conveyor
- Tertiary crusher and tertiary crusher feed conveyor
- Crushed material stockpile feed conveyor
- Reclaim feeders and ball mill feed conveyor.

17.5.2 Grinding Circuit

The grinding circuit receives fresh feed from the crushed material stockpile via reclaim feeder and mill feed conveyor. The mill discharge is screened by a trommel to remove scats. The trommel undersize reports to the cyclone feed pump box, then diluted and pumped to the primary cyclones for classification. The cyclone underflow returns to the ball mill for further grinding. The cyclone overflow gravities to the trash screen, where debris are removed to ensure clean slurry feed to the pre-leach thickener.

Major equipment in this circuit includes:

- Ball mill
- Primary cyclones
- Trash screen.

17.5.3 Whole Ore Leaching (WOL)

Undersize from the trash screen is advanced to a pre-leach thickener where flocculant will be added to facilitate solids settling. Thickener overflow will be transferred to the process water tank, while underflow will be pumped to the leaching tanks. Leaching will occur in a series of five tanks that provide a total of 72 hours of residence time. Discharge from the final bulk leaching tank reports to the counter current decantation (CCD) circuit.

Hydrated lime, sodium cyanide, and lead nitrate will be added to the first two tanks in the series to provide pH control, to maintain the required levels of cyanide, and to act as a catalyst, respectively. Barren solution

from the Merrill Crowe circuit is also added to the first two leaching tanks as a recycle stream. Oxygen is sparged into all the tanks to facilitate leaching of gold and silver from the material.

Major equipment in this area will include:

- Pre-leach thickener
- Five mechanically agitated leaching tanks.

17.5.4 Counter Current Decantation

Leached solids are washed of pregnant solution through a series of five counter current decantation thickeners. In each thickener, solids are washed through the addition of overflow from the subsequent stage, while settling is facilitated through flocculant addition. Thickener overflow is advanced to the preceding stage, while the first thickener in series advances to the pregnant solution clarifier. Pregnant solution clarifier overflow advances to a pregnant solution tank for feeding to the Merrill Crowe circuit. Underflow from the clarifier is pumped as required back to the start of the bulk leaching circuit.

Thickener underflow is pumped to the subsequent CCD stage for further washing, until it passes out of the circuit from the last thickener in series. The target for gold concentration in the tailings discharge from the last thickener is 0.02 ppm, while the target silver concentration is less than 0.76 ppm. Barren solution will be added to the last thickener to achieve the target wash ratio and solution losses. Flocculant streams for the CCD thickeners and pregnant solution will be made from flocculant reagent and barren solution from the Merrill Crowe circuit.

Major equipment in this area will include:

- Five counter current decantation thickeners
- One pregnant solution clarifier.

17.5.5 Merrill Crowe Circuit

Overflow from the clarifier advances to the pregnant solution tank for storage prior to precious metal recovery via zinc precipitation. Solution is pumped from the pregnant solution tank through clarifier filters and subsequently a deaeration tower to reduce dissolved oxygen content. Deaeration tower discharge is then pumped to precipitation filters to recover the precious metal rich precipitate. Solids from the precipitate filters are discharged and manually moved to the smelting furnace for doré production, while filtrate advances to the barren solution tank. Barren solution is then distributed to the CCD circuit as wash water, flocculant dosing streams, and the detoxification circuit for dilution as necessary.

Antiscalant is added to the pregnant and barren solution tanks to inhibit scale formation within the Merrill Crowe circuit. Diatomaceous earth is added to both the clarifying and precipitation filters to improve filter performance.

Major equipment in this area will include:

- Pregnant solution tank,
- Two clarifying filters,
- Deaeration tower, and
- Barren solution tank.

17.5.6 Refinery

Zinc precipitate discharged from the precipitation filters is manually transferred to the refinery for retorting and smelting into silver-gold doré bars. Retorting and smelting will occur in batch mode in a secure, enclosed area with restricted access.

Zinc powder is added to the precipitation filter feed to provide the necessary levels of zinc for precipitation. Cyanide is also added to the precipitation filter feed as necessary, to ensure proper cyanide levels before filtration.

The refinery will be a vendor package and will include the following major equipment:

- Two precipitation filters
- Smelting furnace
- Flux dosing and mixing system
- Mercury retort
- Doré scale and storage vault
- Slag handling equipment
- Dust collection system.

17.5.7 Cyanide Detoxification and Tailings Handling

Underflow from the final CCD thickener is pumped to the cyanide detoxification tanks where weak-acid-dissociable cyanide (CNWAD) is detoxified using the SO_2 /air process. Barren solution from the Merrill Crowe circuit and overflow solution from the tailings thickener or paste thickener is added as needed to the tank to achieve the desired solids density. Cyanide detoxification is conducted using parallel tanks that provide a total residence time of 90 minutes and reduce CNWAD concentration to below 5 mg/L.

The discharge slurry from the detoxification tanks flows by gravity to the tailings thickener where flocculant is added to support solids settling. Overflow from the thickener is pumped to the process water tank and back to the detoxification circuit, while underflow reports to the tailings filter storage tanks. The material in the storage tanks reports to the tailings filters in batches and filter cake discharges to a stockpile. The material in the stockpile is reclaimed by an excavator and sent to the dry stack facility via trucks. The filtrate from the filters goes back to the tailings thickener feed.

Sodium metabisulphite (SMBS) will be added to the cyanide detoxification tanks as the main reactant, copper sulphate will be added to act as catalyst for the destruction reaction, and hydrated lime will be added to the tanks to maintain the required reaction pH. Oxygen is sparged into the tanks to support the reaction.

Major equipment in this area will include:

- Two mechanically agitated cyanide detoxification tanks
- Tailings thickener
- Tailings storage tanks
- Two tailings filters.

17.5.8 Paste Backfill Plant

The paste plant will operate as required for underground backfill. The plant is sized to accommodate the highest year of paste demand, equivalent to approximately 50% of tailings material from the process plant. Material exiting the cyanide detoxification circuit will bypass the tailings thickener and will be pumped to the paste plant on an as required basis. At the paste plant, tailings are thickened and filtered and then added to the paste mixer unit with cement binder to create paste backfill. Filtrate from the paste tailings filters is recycled to the paste tailings thickener feed. Overflow from the paste tailings thickener reports to the process water tank and is recycled to the cyanide detoxification circuit as needed. Following mixing, paste is pumped to the underground reticulation distribution system.

Major equipment for the plant will include:

- Paste tailings storage tank
- Paste tailings thickener
- Paste tailings filters
- Cement binder silo
- Cement binder dosing system
- Paste mixer and hopper
- Two positive displacement paste pumps.

17.6 Product/Materials Handling

The reagent handling systems (Table 17-2) include unloading and storage facilities, mixing and storage tanks, and feeding equipment as needed for each of the required plant reagents. Each set of compatible reagents is located in a dedicated containment area to prevent environmental contamination and mixing of incompatible reagents. Appropriate ventilation, fire, safety protection, eyewash stations, and safety data sheet (SDS) stations are located throughout the area. Sumps and sump pumps are provided for each containment area for spillage control.

Table 17-2: Reagents Handling

Reagent	Preparation Method	Use
Quicklime	Received as powder in bulk shipments and stored in a silo; slaked with raw water in a vertical mill and pumped to a storage tank; distributed via ring main to the leaching circuits and detox circuit	pH control
Sodium cyanide	Received as powder in bulk bags; dissolved and transferred to a storage tank; distributed to cyanide leaching circuits	Leaching reagent
Copper sulphate	Received as powder in bulk bags; dissolved and transferred to a storage tank; distributed to cyanide detoxification circuit	Cyanide destruction catalyst
Lead nitrate	Received as powder in bulk bags; dissolved and transferred to a storage tank; distributed to Merrill Crowe circuit	Leaching and precipitation reagent
Diatomaceous earth	Received as powder in bulk bags; dissolved and transferred to a storage tank; distributed to Merrill Crowe circuit	Filter aid

Reagent	Preparation Method	Use
Zinc powder	Received as powder in drums; dissolved and transferred to a storage tank; distributed to Merrill Crowe circuit	Precipitation reagent
SMBS	Received as powder in bulk bags; dissolved and transferred to a storage tank; distributed to cyanide detoxification circuit	Cyanide destruction reagent
Flux Additives	Received as powder in bulk bags; mixed with calcined charges for smelting	Smelting reagents
Antiscalant	Received as intermediate bulk containers; distributed neat to pregnant and barren solution tanks	Scale inhibitor
Flocculant	Received as powder in bulk bags; dissolved and transferred to a storage tank; distributed to thickeners across the plant	Flocculation promotor
Cement binder	Received as powder in bulk shipments and stored in a silo; added to the paste mixer at the paste plant	Paste mixing

17.7 Energy, Water, and Process Materials Requirements

17.7.1 Process Reagents and Consumables

A summary of reagents and consumables required for plant operation and their estimated annual consumption is presented in Table 17-3.

Table 17-3: Reagents and consumables nominal annual consumption

Item	Unit	Value
Quicklime (leaching)	t/a	777
Quicklime (detoxification)	t/a	4,130
Sodium cyanide	t/a	3,947
Copper sulphate	t/a	143
Lead nitrate (leaching)	t/a	365
Lead nitrate (precious metal recovery)	t/a	38
Diatomaceous earth	t/a	839
Zinc powder	t/a	218
SMBS	t/a	5,900
Flux Additives	t/a	143
Antiscalant	t/a	67
Flocculant (tailings)	t/a	18
Flocculant (CCD thickeners)	t/a	365
Flocculant (pre-leach)	t/a	18
Binder (paste)	t/a	38,877

Item	Unit	Value
Primary crusher liners	sets/a	8
Secondary/Tertiary crusher liners	sets/a	12
Grinding media	t/a	1,789
Ball mill liners	sets/a	1
Screen panels	sets/a	24
Clarifying filter cloths	units/a	2
Precipitation filter cloths	units/a	2
Refinery baghouse cartridges	units/a	1
Refinery crucibles	units/a	6
Tailings and paste filter cloths	sets/a	361

17.7.2 Water requirements

17.7.2.1 Raw water

Raw water will be obtained either from collection ponds on site and mine dewatering, or alternatively from groundwater wells in the region of Potrerillo de los Torres, pumped to the plant and stored in a raw water storage tank where it will be distributed to the various users across the plant site such as reagent preparation, gland seal water, potable water, and plant make-up. Raw water tank designed for 12-hour live storage capacity. During normal operation (i.e., when the paste plant is not in use, and throughput of the process plant is maintained at 5,000 tpd) the nominal fresh / raw water requirement is approximately 27 m³/h. When the paste plant is in operation, the estimated water requirement increases to approximately 62 m³/h.

17.7.2.2 Process water

Process water for the plant will consist of pre-leach thickener overflow, and tailings thickener overflow. Raw water will be provided as required for makeup. Process water will be stored in a storage tank before being pumped across the plant site to the various end users. Tank designed for live capacity of one hour.

17.7.2.3 Gland seal water

Gland seal water for the plant will be sourced from the raw water tank and pumped to the various pumps across the plant site.

17.7.2.4 Fire water

Fire water for the process plant will be stored in a dedicated volume within the raw water tank. A dedicated pump skid consisting of an electrical pump, jockey pump, and diesel pump will supply water from the fire water reserve volume to the distribution system.

17.7.2.5 Fresh water

Fresh water will be collected, wherever possible, from on-site water management facilities and supplemented by groundwater wells located to the south-west of the project.

17.7.2.6 Potable water

Potable water will be produced by an on-site potable water plant which processes water from the raw water tank. Potable water will be stored in a dedicated potable water storage tank before being distributed to the various end users in the process plant. Dedicated tank with 48-hour live capacity.

17.7.3 Air

High-pressure and low-pressure air will be produced by compressors to meet plant requirements of 750 and 250 kPa, respectively. The air supply will be collected in a plant air receiver before being distributed across the plant. A dryer will be fed by the plant air receiver and discharge into an instrument air receiver prior to plant distribution.

17.7.4 Oxygen

Oxygen gas for the cyanide leaching and detoxification circuits will be generated from an on-site oxygen plant. The oxygen plant will employ vacuum swing adsorption (VSA) technology and will produce an oxygen stream of 93% purity at 100 psig.

17.7.5 Power

The total installed power requirement for the process plant is estimated at 13.0 MW. Further discussion regarding the power supply and distribution system is available in Section 18. Further discussion around the operating costs associated with the plant power consumption is available in Section 21.

18 Project infrastructure

18.1 Introduction

Infrastructure at the Los Reyes Project includes on-site infrastructure such as earthworks development, site facilities and buildings, on-site roads, water management systems, a TSF and site electrical power facilities. Off-site infrastructure includes the site access road, fresh water supply including off-plot piping, and power supply. The site infrastructure includes:

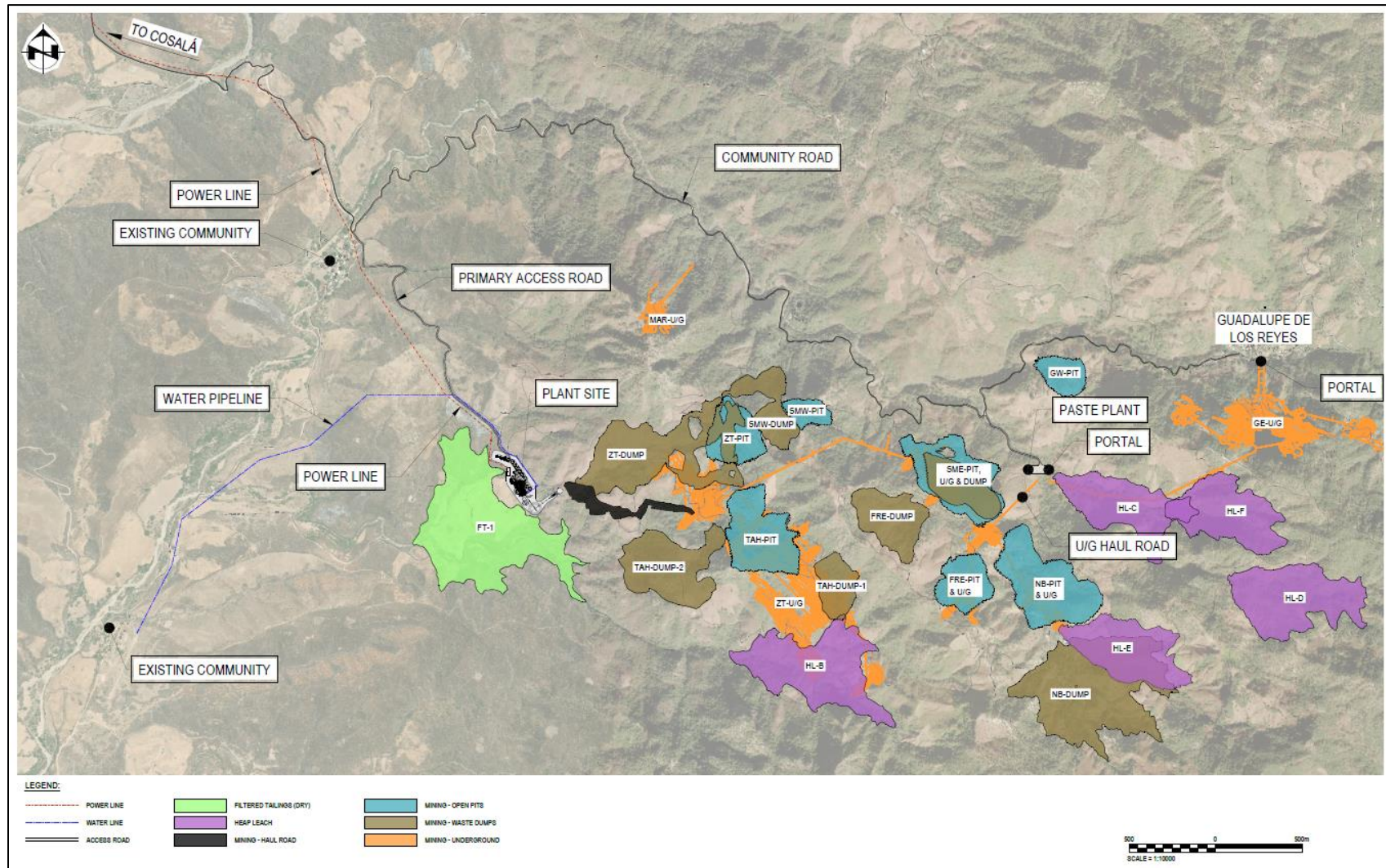
- mine facilities, including truck shop & wash bay and mine office, explosive storage facilities, warehouse, paste backfill facilities, waste rock storage and low-grade stockpiles, ROM (run-of-mine) pad, pits and Underground portals;
- bulk earthwork required to develop the process plant pad, paste plant pad, interconnecting roads and additional laydown areas to facilitate temporary facilities and storage;
- infrastructure for power distribution;
- fuel storage for mine, process, and support fleets;
- sewage and effluent treatment for site;
- offices, workshops, warehouses, security buildings, accommodations as well as additional support buildings;
- surface mobile fleet for non-mining site crew and staff;
- process plant including coarse crushed materials storage, grinding and classification, leaching and gold processing, detox, filtering, storing and handling, tailings thickening, a reagent mixing and distributing facility, assay laboratory and process plant workshop and warehouse;
- tailings management including filtration, haulage and a TSF and
- water management and balance for the entire site.

The overall site layout was developed using the following criteria and factors:

- The processing facilities described above must be located to minimize mine haul costs, initial capital costs (earthworks, roads, power, water), environmental and community impacts.
- The TSF should be located as close to the process plant as possible, taking advantage of the sloped natural terrain, down-gradient from tailings filtering facilities, and surface runoff managed accordingly.
- The arrangement of the administration buildings, mine workshops, processing plant, and additional offices should be optimized for foot and vehicle traffic.
- The project is located at a high altitude with very steep terrain. Consideration has been given to these conditions when locating buildings and access roads.

The Los Reyes Project layout is shown in Figure 18-1.

Figure 18-1: Infrastructure layout plan



Source: Ausenco, 2026

18.2 Site Access

There are two key accesses to the site. The main site entry is in the north-west of the site and is connected to the township of Cosalá by a 27 km gravel road, to the proposed site location. It is proposed to upgrade this road in the initial stages of the project. This access is close to the proposed process plant and site camp facilities. The other site access is on the eastern side of the project and is connected by an existing public road that supports the nearby community of Guadalupe. This access is close to the proposed paste plant and Guadalupe East underground portal. Both access roads, once they are within the project boundaries, are private and not shared with the public.

18.3 Mining infrastructure

18.3.1 Mining infrastructure

The underground operation is accessed from surface by a primary decline driven at a nominal 1-in-7 gradient, together with additional portal entries serving the four underground zones; five portal locations in total provide access and ventilation connection across the deposits. Surface mining infrastructure at each portal comprises a run-of-mine pad and mineralised-material/waste transfer point, a mine dry and office, a maintenance workshop and washdown, a warehouse and laydown, a fuel and lube facility, and a magazine for explosives storage. Underground fixed infrastructure includes the primary ventilation installation (intake and return raises with surface primary fans), the dewatering system (sumps, settlers and a pumping reticulation discharging to surface), fixed electrical reticulation and substations, compressed-air, communications and dewatering reticulation, and the paste-fill reticulation network distributing backfill from the paste plant to the active stopes. The paste backfill plant itself is covered under the process facilities (Section 21, Area 3000). Stockpiles comprise short-term run-of-mine (ROM) piles ahead of the primary crusher and low-grade stockpiles drawn down through the life of mine, located in the vicinity of the ROM pad.

18.3.2 Waste rock storage facilities

Open-pit waste rock is placed in engineered waste rock storage facilities (WRSFs) located adjacent to the open-pit complexes to minimise haulage distance. Over the life of mine approximately 76 Mt of open-pit waste is mined at an average strip ratio of 6.8:1 and placed in the WRSFs; underground development waste is, where practicable, used underground as backfill aggregate or placed in the WRSFs. The WRSF designs, including foundation preparation, lift geometry, surface-water diversion and progressive rehabilitation, are developed by AGP Mining Consultants in conjunction with the geotechnical and environmental work; final landform and closure integration is addressed in the environmental and closure sections. At closure the WRSFs will be regraded to stable, free-draining slopes, capped and revegetated to a final landform compatible with the surrounding topography and post-closure land use, consistent with the closure plan (Section 20); the final landform and closure design will be advanced at PFS.

18.4 Tailings Storage Facility

18.4.1 Filtered Tailings Storage Facility

The processing of mineralised material will generate approximately 25.7 million tonnes (Mt) of tailings over a 15-year mine life, at a throughput of approximately 5,000 tonnes per day (t/d). Approximately 15.9 Mt of filtered tailings will be managed in a permanent above-ground TSF located immediately south of the Plant

Site, and the remaining 9.8 Mt of tailings will be used for underground paste backfill. Runoff from upslope undisturbed and natural areas will be diverted around the TSF where possible and discharged downstream of the TSF. The TSF will be designed and constructed in accordance with the International Council on Mining and Metals (ICMM) Global Industry Standard on Tailings Management (ICMM, 2020).

Filtered tailings will be transported from the Plant Site to the TSF by truck and will be placed and compacted in lifts. A stabilization buttress will be constructed at the downstream toe to enhance slope stability under static and seismic conditions while limiting long-term deformation. It also protects against erosion from surface water runoff and weathering. A designated structural zone of filtered tailings will be established at the downstream toe of the facility. This zone will be constructed to a higher compaction specification as part of the overall TSF stability design. The final filtered tailings surface will be graded at a 4H:1V slope on the downstream slope.

A smaller starter facility will be constructed prior to mill start-up, with the Starter Stability Berm located approximately 0.5 km upstream of the ultimate TSF downstream toe. A key objective of the starter facility is to reduce contact water management and the stability berm size at start-up. The starter facility is designed to store up to six months of tailings, fully confined behind the starter berm. This provides operational flexibility during filter plant commissioning and reduces risk by preventing off-specification material from being placed in the critical structural zone at the downstream toe of the final facility footprint. Construction of the Final Stability Berm at the ultimate downstream toe will commence in Year 1, by which time the filter plant will be producing in-specification tailings suitable for placement at the ultimate downstream toe. The starter facility can also be expanded to manage up to two-years of filtered tailings stacked behind the Starter Stability Berm should in-specification tailings material be produced immediately at mill-start up.

18.4.2 TSF Stability Berm Design

The TSF Starter and Final Stability Berms will be constructed primarily of earthfill and/or rockfill sourced from local borrow areas. Foundation preparation will involve the complete removal of unsuitable foundation materials down to an approved bedrock foundation. Excavated material will be hauled to stockpiles outside the TSF footprint.

A Drainage Zone will be constructed on the upstream face of the stability berm to promote drainage and limit the migration of tailings fines downstream. The zone will consist of processed sand and gravel sourced from local borrow sources.

Geotechnical instrumentation will be installed within the stability berms, tailings, and foundation materials to monitor performance during construction and operations. An Operation, Maintenance and Surveillance (OMS) Manual will be prepared prior to commissioning of the TSF.

18.4.3 TSF Seepage Control Measures

A foundation drainage network will be constructed along existing creeks and natural drainage pathways to control groundwater pressure and collect seepage from the filtered tailings facility. Both TSF stages will include a seepage collection pond at the downstream toe of the TSF stability berm. The seepage collection ponds will collect seepage from the TSF foundation drainage system, as well as runoff from the berm and tailings slope. The seepage collection ponds will be sized to accommodate runoff from the design storm event, with flows from larger events safely conveyed through an emergency spillway.

18.4.4 Closure Considerations

TSF closure and reclamation will commence at the end of milling operations. The primary objective is to restore the site to a self-sustaining facility with pre-mining land capability. The TSF surface will be regraded to direct surface run-off towards a closure spillway located at the north end of the facility, conveying any surface water discharge into the adjacent drainage. The TSF will be designed to maintain long-term geochemical and physical stability, protect the downstream environment, and manage surface runoff. Key activities that will be carried out during operations and at closure to achieve these objectives are listed below:

- Selective placement of filtered tailings in the final year of operations to establish a final tailings surface that will facilitate surface water management and reclamation
- Removal of the mechanical systems, structures, and equipment not required beyond mine closure
- Construction of a final dry closure capping layer on the entire area of the TSF
- Construction of a closure channel on the closer surface cover
- Construction of a permanent closure spillway from the TSF
- Regrading of facility surfaces to promote surface drainage towards the closure spillway
- Removal and re-grading of all access roads, ponds, ditches and borrow areas not required beyond mine operations
- Long-term stabilization and rehabilitation of all exposed erodible surfaces

18.5 Water management

18.5.1 Overview

The principal water management objectives for the project are to:

- Provide sufficient water supply to support milling at start-up and throughout operations through a combination of underground and open pit dewatering.
- Divert non-contact water upstream of project infrastructure to reduce the contributing catchment areas, wherever possible.
- Store contact water on site for use as process water with surplus contact water treated prior to discharge to the downstream environment.
- Maximize the recycling of both process and contact water to reuse in the process.

Mean annual precipitation (MAP) at the Project site is approximately 1,800 mm/a, with most of the rainfall occurring from July to September and relatively dry conditions prevailing during the remainder of the year. Project facilities, including the TSF and WRSFs, have large upslope catchments requiring site water management infrastructure to manage storm water inflows.

Erosion and sediment control Best Management Practices (BMPs) will be implemented throughout the Life of Mine to protect downstream watercourses.

18.5.2 Upstream Water Control Dam (UWCD)

The UWCD, located immediately upstream of the TSF, will serve as the primary control point for contact water containment and process water storage. This includes surplus from underground and open pit dewatering activities not used for processing. A diversion channel will be constructed around the UWCD to reduce surface water runoff from the contributing catchment into the facility.

The UWCD will include a cross-valley embankment constructed from earthfill and rockfill sourced from local borrow areas and basin shaping activities within the reservoir footprint. A continuous geomembrane liner will be installed in the basin and upstream face of the embankment to minimize seepage and support long-term water storage. A foundation drainage system beneath the liner will collect groundwater and connect to the TSF foundation drainage network downstream of the UWCD. An emergency spillway will be constructed on the left abutment to manage large storm events.

18.5.3 TSF Water Management

Contact water from the TSF and its upslope disturbed areas will be collected in a seepage collection pond downstream of the TSF. The seepage collection pond will be sized for runoff inflows from the selected design storm event, and inflows greater than this will be discharged via emergency spillway. Non-contact water from TSF upslope undisturbed areas will be diverted, where possible. Contact water from the TSF seepage collection ponds will be pumped directly to the Plant Site for use as process water.

18.5.4 Waste Rock Water Management

Additional water management infrastructure will be constructed to manage runoff from WRSFs where there is no natural drainage to the open pits or the UWCD. This will include sediment ponds at the downstream toes of WRSFs, collection channels to route contact water into the open pits, and diversion channels to route non-contact water to the downstream environment. Contact water from waste rock facility sediment ponds and open pits will be pumped directly to the Plant Site or Paste Plant for use as process water.

18.5.5 Water Balance

A site wide annual water balance was developed for the Project to estimate the water deficit or surplus throughout the 15-year operating mine life. Key results from the water balance are summarized below:

- The Project is expected to be in a water surplus throughout all 15 years of Operations with process water requirements largely met by underground and open pit dewatering activities.
- The largest water loss at the Project is attributed to losses to tailings voids, which equates to approximately 600,000 m³/a from both filtered tailings to the TSF and paste underground backfill. Other water losses considered in the water balance include evaporation, camp water, and dust control.
- The UWCD will provide storage for surplus water from underground and open pit dewatering. The UWCD water level may need to be drawn down (i.e., water may need to be discharged) to provide capacity for storm water attenuation. Water treatment of contact water from mine dewatering is expected to be required prior to discharge from the UWCD.

18.5.6 Water Supply

External potable water sources are required to meet the needs of the on-site camp. It is also recommended that external make-up water sources are identified and available for intermittent use during Operations if climate, process, or dewatering conditions are unfavorable, particularly in the drier months from October through June. The current design allows for drawing water from a well system to be located approximately 3 km south-west of the process plant with overland pipelines and associated pumping facilities. The external make-up water sources for the Project have not yet been fully identified, secured, or permitted. It is anticipated that the make-up water requirements will be supplied by local groundwater aquifers or surface water sources.

18.6 Built infrastructure

18.6.1 Gate House and Security Facility

The gate house is a trailer building with a lockable gate and communications to the main site. An additional modular security building is located near the gate house. The security facilities include rooms for personnel screening during rotations in and out of site.

18.6.2 Main Administration Building

The main administration building is modular comprised of 12 units including a change/lunch facility, offices, meeting rooms, and washrooms. The offices will have space for process, and general and administrative employees. This building also includes medical facilities consisting of first aid and emergency response rooms for on-site treatment and headquarters for mine rescue team. These facilities are equipped with fire protection and an alarm system.

18.6.3 Warehouse and Maintenance

The warehouse/shop is a fabric building with concrete floor, overhead doors, fire protection, and alarm systems. This building will be used for general storage, storing equipment spares for the process plant

18.6.4 Truck Shop/Wash

The truck shop/wash is a pre-engineered building with a concrete floor, overhead crane, and overhead doors with fire protection and alarm systems. There will be a total of four maintenance bays. Two maintenance bays will be assigned to preventative maintenance, one will be for corrective maintenance, and the last bay will be multipurpose. Additionally, a single welding bay and truck wash will be located at the front of the truck workshop building.

18.6.5 Mine Dry & Operations Building

The mine dry and office facility is a modular building shared by both open pit and underground operations. It includes lockers, washrooms, and showers for men and women.

18.6.6 Assay Laboratory

The assay laboratory is a modular building to primarily support the quality testwork requirements of the process plant.

18.6.7 Site Preparation

All infrastructure areas will be cleared of vegetation, and the topsoil will be removed and stockpiled for closure use. Drains, safety bunds, and backfilling with granular material and aggregates for road construction are all elements of the initial site development. Site civil work includes design for the following infrastructure:

- on-site roads for light vehicles and heavy equipment
- site access roads
- stockpile areas
- mine and process facilities pads
- ROM stockpile and crusher facility
- Water management facilities, including ditches, underdrains and diversion channels
- TSF.

Initial pre-strip material from the mine operations will be used, wherever possible, to establish some of the main fills required for the project.

18.6.8 On-site Roads

The project site has unpaved roads connecting the access road to the gatehouse. In addition to any existing roads on site, new roads will be constructed linking the guard house, the administration building, the process plant, the underground portals, the primary crusher, and the TSF.

18.7 Accommodation

Accommodation for site personnel will be a combination of camp facilities and travelling, by project supply buses, from the township of Cosalá. The camp facility will be located on site, close to the main entry and will consist of modular accommodation units.

18.8 Power and electrical

Permanent electrical power is provided by a hydro dam located to the north of Cosalá. A 138 kV transmission line will connect into the grid near Cosalá and run to the site main substation located near the process plant before being stepped down for distribution to meet the power requirements across the project site.

The project power requirements is 13MW, and the design allows for power lines from the proposed reliable hydro dam power source where there is sufficient power for the site.

18.9 Fuel storage and distribution

Fuel storage has been provided for diesel, propane and gasoline. It features above ground tanks and is serviced by tankers and dispensers.

19 Market studies and contracts

19.1 Production specification

The project will produce gold and silver doré bars at the processing facility on site. These bars will be further processed in a refinery located in North America or Europe.

19.2 Market studies

The Company has not conducted a formal market study for production from the Project. The market for gold and silver doré bars is very liquid and the product can easily be sold on a spot basis at market prices. The Company currently has refining relationships for gold doré that can be leveraged for the sale of the gold and silver doré.

Expected maximum annual doré production is 437 kg/d and nominal production is 309 kg/d from the gold room.

The expected purity of the gold-silver doré (before refining) is ~95%. It is expected that this is shipped to a refinery to be upgraded to 99.99% Au and 99.9% Ag.

The QP for this section has reviewed the market analysis and, in the QP's opinion, the data supports the assumptions in this technical report.

19.3 Commodity price projections

Project economics were estimated based on long-term metal prices of US\$3,600/oz Au, and US\$50/oz Ag, which was based on consensus forecasts from various financial institutions.

The QP notes that the pricing used in the cash flow analysis is reasonably aligned with various long-term forward-looking estimates from major Canadian and international banks.

19.4 Contracts

The Company does not currently have contracts for the gold and silver doré production at Los Reyes. The Company will leverage its experience and existing relationships in the gold doré market to enter into future contracts for the sale of gold and silver doré. We expect the terms in the contract to be market-based terms.

20 Environmental studies, permitting, and social or community impact

20.1 Environmental considerations

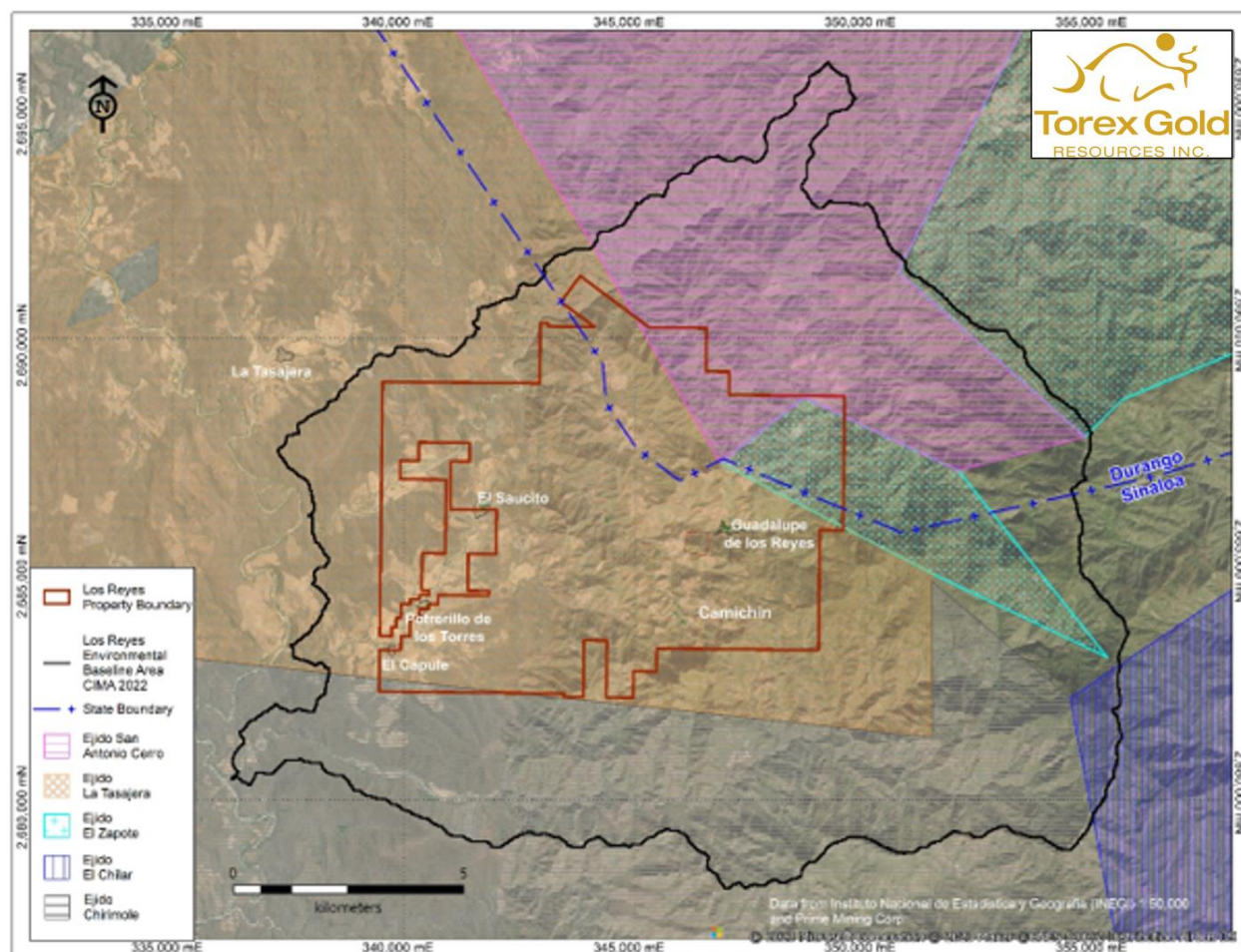
The Los Reyes Project site is situated in a rural setting characterized by a low population density. It is located in the southeasternmost part of the municipality of Cosalá, in central-eastern Sinaloa state, near its boundary with Durango state. Low-intensity livestock farming and rainfed agriculture, as well as mining activities, have traditionally been carried out in the area. These economic activities have perturbed, in varying degrees, the vegetation cover and the soils in the project site, leading to impacts in the local hydrology and ecology.

The project site is positioned away from any federal, state, municipal or private area legally designated for protection as well as from areas with some level of environmental priority by the federal regulatory agency (Secretaría de Medio Ambiente y Recursos Naturales, SEMARNAT) or international agreements, such as the RAMSAR Convention on Wetlands of International Importance (Sitios RAMSAR), areas of importance for bird conservation (Áreas de Importancia para la Conservación de las Aves, AICAS), priority hydrological regions (regiones hidrológicas prioritarias) and priority land regions (regiones terrestres prioritarias).

Regarding the federal General Ecological Territorial Planning Program (Programa de Ordenamiento Ecológico General del Territorio, POEGT), the project is mostly located within the biophysical unit number 94, denominated Cañones Duranguenses del Sur (Southern Durango Canyons) and to a lesser extent in unit 12, denominated Pie de la Sierra Sinaloense Centro (Foot of the Central Sinaloan Mountain). Both units have forestry and mining as their guidelines for development.

The observations made in this the Technical Report covered an area of 21,079 ha, which extends beyond the delimitation of the Los Reyes claim area (see Figure 20-1).

Figure 20-1: Environmental Area of Interest



Source: Instituto Nacional de Estadística y Geografía (INEGI) and Prime, 2024

20.1.1 Geology

The location of the Los Reyes Project is dominated mainly by andesitic rocks and rhyolitic tuffs. These rocks belong, respectively, to the so-called Lower Volcanic and Upper volcanic sequences of the Sierra Madre Occidental geologic province. An unconformity separates both volcanic packages. The Lower Volcanic sequence is considered to be of late Cretaceous to early Tertiary in age and the Upper Volcanic sequence is of a younger Tertiary age.

The andesites rest unconformably on a late Cretaceous granodioritic body. Close to the project site a rhyolitic stock crops out, perhaps of Oligocene age. Felsic dykes and quartz veins cut the andesitic rocks, to which the gold and silver mineralization appears to be related. North-northwest to northwest striking structures are found within and neighboring the project site and have played an important role in controlling the mineralization.

20.1.2 Physiography

The Los Reyes project area is located in the low reaches of the “Gran Meseta y Cañadas Duranguenses” sub-province of the “Sierra Madre Occidental” physiographic province. The latter is a long mountain chain, running NNW-SSE parallel to the western seaboard of Mexico, from the U.S.A. border to the central part of the country. The sub-province is characterized by a complex setting of broad high plateaus with deep canyons and topographic prominences. In this scenery, the area of the project is in a context dominated by ridges and canyons where landforms are sculptured mainly on volcanic rocks.

The project site is characterized by a moderate to steep topography with slopes varying mostly between 11.1% and 100% rise and elevations ranging mostly from around 380 metres ASL to over 970 metres ASL. The lowest elevations are found in the western part of the project site as well as the less inclined hillslopes.

The Los Reyes project is in a region for which the Mexican government considers to be a high level of danger for mass movement on the slopes. However, no evidence of this phenomenon has been observed in satellite images.

The local landforms have only been slightly modified by anthropic actions such as access roads and mining works.

20.1.3 Soils

According to the latest soil cartography (National continuum, series III, ed. 2024, esc. 1:250,000) of INEGI (National Institute of Statistics and Geography), based on the World Reference Base for Soil Resources International Soil Classification of the International Union of Soil Sciences (ed. 2014), three soil types are found in the Los Reyes Project site. The two dominant types are regosol and leptosol, both of which show a poor solum development.

The regosol type is a weakly evolved mineral soil, lacking horizon B and no diagnostic horizons. In the area, it is limited by continuous rock within the first 50 cm from the surface and contains abundant gravel and other large fragments in its profile. It is delicate soil.

Leptosol is a thin soil limited by hard rock within 25 cm from the surface and contains a lot of coarse fragments. It is typical of rugged topography. In the area, leptosol plays two roles. As the dominant type in one soil unit, it has an effective base saturation and on average 1% of organic carbon. It is also found as a secondary soil associated with regosol. In this case, it also shows a good base saturation but is rich in coarse fragments. Leptosol is very susceptible to erosion.

Phaeozem is the third type of soil present in the area, but it is only found associated with the dominant leptosol described above and, as such, it only plays a lesser role. This is a more developed and fertile soil with a dark surface horizon rich in bases, containing more than 5% of organic carbon. It is limited by hard rock between 25 cm and 50 cm from the soil surface.

The natural condition of soil has been modified in much of the area. In addition to exploration activities and old mining works, the soil has also been affected by the economic activities carried out by the locals. In several places, it lacks the protective cover of natural vegetation, due to rainfed agriculture or perhaps overgrazing by cattle, and therefore subjected to accelerated erosion. In addition, there are widespread indications of a slow downslope movement of the soil, which typically is associated with cattle trampling on moderate to steep slopes. This movement creates a microtopography on the slopes.

A soil sampling program is required to better characterize the properties of soil in the project area.

20.1.4 Climate

The climate of the project area is warm subhumid with a rainy season in the summer. The mean annual temperature at the site is close to 24 °C and the mean annual precipitation is close to 1,800 mm. At the meteorological station 25036 Guadalupe de los Reyes, located in the eastern part of the project site, the reported extreme temperatures (T) are 44°C (07/29/2009) for the maximum T and 2°C for the minimum T, as well as 192 mm of rain recorded on 08/07/1966 as the extreme precipitation in a 24-hour period. The area is within a zone that may experience the effects of tropical cyclones. The only data for wind is for Cosalá, located about 20 km in straight line to the NW, where prevailing winds come from the NE and from WSW depending on the epoch of the year.

No data on air quality is available for the project site. However, it is in general considered that good air quality prevails since this is a rural area. The operation of a weather station is recommended at the project location.

20.1.5 Hydrology

20.1.5.1 Surface Water

The Los Reyes project is within the territory of the hydrological interconnected basin 1001 Río Habitas, established by the Mexican federal government for managing purposes. According to the last published official report (12/28/2023), this basin has a yearly mean availability of 92.503 million cubic metres of water at the basin exit.

Hydrographically, the project site belongs to the basin of the Elota river, which is the main water body of the region and is located about 4.7 km in straight line to the west of the site. Surface runoff in the project location flows freely, since there is no exploitation of surface water and no man-made structures to retain water other than a few small ponds for cattle. The area runoff is conducted by creeks, of different levels, to the Elota River. The most important is the intermittent Guadalupe-El Saucito-Capule creek that flows directly to the Elota River and runs in part in the vicinity of some of the planned facilities. The whole project site is entirely located within the watershed of this creek.

The main disturbances to surface water are the alteration of runoff caused by the microtopography created by the slow downslope movement of soil, previously mentioned, as well as the increased runoff in areas lacking vegetation cover where soils are being eroded faster.

As part of the national program aimed at the planning, construction, and operation of irrigation works, seeking the efficient use of the country's water resources, the then Ministry of Water Resources (now SEMARNAT) issued an agreement declaring an indefinite ban on granting concessions for the use of water from the Elota River in the states of Durango and Sinaloa. This agreement was published in the Official Gazette of the Federation on August 31, 1954. Even though the conditions of the basin have changed since the promulgation of the agreement and the planned works have been completed, the ban remains in effect and covers the entire tributary basin from the headwaters of the main stream, throughout its course in the states of Durango and Sinaloa, to its mouth in the Pacific Ocean.

To establish a surface water quality baseline, two sampling rounds were carried out in 2022, along different creeks and ditches in and around the project site. However, the laboratory results were compared to maximum permissible levels indicated in Mexican standard NOM-001-SEMARNAT-1996 (now NOM-001-SEMARNAT-2021), which is not adequate since such standard is for discharges of residual waters into receiving surface water bodies. Therefore, no conclusions can be made on surface water quality.

20.1.5.2 Underground Water

Los Reyes Project area is within the Rio Elota aquifer (ID 2506) which, according to the most recent official report (2024), has no underground water availability due to a deficit of -10.033776 million cubic metres. Although the aquifer is not under any banning ordinance, drilling of new water wells is not allowed.

No records are known, if they exist at all, as to the volume of exploitation of underground water in the project site area. However, it is assumed to be at a low scale because this is a rural area and no industrial or intensive agricultural uses exist.

Another impact to underground water, is the alteration to the rate and/or volume of infiltration derived from both the exposed soils in zones with no protective vegetation cover and the microtopography developed by the previously mentioned slow downslope movement of soils on moderate to steep slopes. Flash floods are common during the rainy season as water runoff rapidly down hill towards the coastal plains.

Article 19-V of the Mexican Mining Act establishes that holders of a mining concession have the right to use the water from mining operations for the extraction and processing of the minerals or substances obtained and for domestic use by the personnel employed therein, provided that the National Water Commission is notified and the corresponding fees are paid. Since Los Reyes Project has many historical adits and raises leading to flooded underground works, water can be pumped out and stored in dams or ponds for future use in construction, development and operations.

Similarly to the case of surface water, two underground water sampling rounds were performed in year 2022 and the laboratory results were compared to the Mexican standard NOM-001 SEMARNAT 1996 for water discharge, so it is not applicable. The samples were taken from mine works in and around the project site and from one water well located downstream over 3 km in a straight line to the NW. No conclusions as to the quality of underground water can be reached from the corresponding report. However, it is commonly known that water coming from old underground mines usually does not have a good enough quality for human consumption

20.1.6 Biodiversity

20.1.6.1 Flora

The environmental system of the Los Reyes project extends over region of low deciduous forest and its transitioning to oak and pine forests. At the project site, the low deciduous forest extends over a little more than 50% of the area of the planned facilities and only very small zones of oak forest and oak-pine forest are found in the eastern part of the site. In addition, a small zone with secondary shrub vegetation of low deciduous forest is located in the west. At the site, the natural vegetation continuity has been fragmented by subsistence farming and cattle raising as well as by mining activities, both historical and recent. Consequently, the natural vegetation is now found in irregular patches, of different size, shape and interconnectivity, dispersed throughout the area of the planned facilities, where also a network of rural dirt roads has contributed to the fragmentation. In general, the interconnectedness of the patches has not been completely lost. In a preliminary manner, it is estimated than more than half of the facilities area show signs of vegetation perturbation.

At the level of the environmental system, a high flora diversity was found based on 155 species recognized, of which 51% are trees, 19% shrubs and 30% herbs. No endemic or protected species, according to the Mexican standard NOM-059-SEMARNAT-2010, were observed in the environmental system.

There is no flora sampling data specific to the project site. Due to the fragmentation of the vegetation, it may be surmised that a less flora species richness and diversity, among other aspects, should be expected. A systematic and quantitative study of the local flora is recommended.

20.1.6.2 Fauna

The only available data on fauna is for the environmental system, where 68 species of vertebrate wildlife were detected of which 13 are mammals, 45 birds, 4 amphibians and 6 reptiles. The most diverse group are the birds whereas mammals show a medium to high diversity. In turn, amphibians and reptiles have a low diversity. On the basis of the Mexican standard NOM-059-SEMARNAT-2010, one mammal species is endangered. Two birds and two reptile species are under a special protection status. No endemic species were found. This data is summarized in Table 20-1.

The fauna species and populations have not been studied at the project site. However, it is clear that human activities in the area and the reduction of their habitat by the previously mentioned fragmentation of the ecosystem, may have locally affected the behavior and number of individuals as well as the interactions between species, such as predation and competence, among other aspects. It is recommended to carry out a study of the local fauna.

Table 20-1: Study Area Species of Interest

No.	Technical Name	Common Name	Status	
			NOM-059-SEMARNAT-2010	CITES
1	Lynx rufus Baileyi	Bobcat		AP II
2	Puma concolor azteca	Puma		AP II
3	Leopardus pardalis	Leopard	P	
4	Accipiter cooperi	Dove Hawk	Pr	AP II
5	Buteo jamaicensis	Red Tailed Hawk		AP II
6	Falco sparverius	Falcon		AP II
7	Glaucidium brasilianum	Little Owl		AP II
8	Progne sinaloae	Sinaloan Swallow	Pr	
9	Amazilia violiceps	Violet-headed Hummingbird		AP II
10	Cynanthus latirostris	Thick-billed Hummingbird		AP II
11	Stellula calliope	Stripe-throated Hummingbird		AP II
12	Iguana iguana	Green Iguana	Pr	AP II
13	Crotalus basiliscus	Pacific rattlesnake	Pr	
14	Crotalus lepidus maculosus	Gray Rattlesnake	Pr	

Notes:

P: in danger of extinction.

Pr: subject to special protection.

AP II: threatened species with populations that have been greatly reduced, although they are not in danger of extinction.

20.1.6.3 Landscape

The Los Reyes project is set in a mountainous area where the traditional economic activities of the rural sparse population of the zone have modified the primary vegetation cover and therefore have diminished the naturalness and aesthetic appeal of the landscape. Originally, the primary vegetation consisted, from west to east, of low deciduous forest, oak forest and oak-pine forest. Livestock raising, subsistence rainfed agriculture and historical and recent mining activities have fragmented the ecosystem and now consists of a mosaic of irregular patches, both in shape and size, of natural flora communities unequally interspersed with agricultural land and/or parts with scarce vegetation, in addition to the effects of the network of dirt roads developed for mining activities and for communication purposes of small communities and farms. In the area where mine facilities for the Los Reyes Project are planned, it is considered qualitatively that the landscape have so been modified in more than 50% of the surface area. Consequently, the oak forest and oak-pine forest are only found in small sectors in the east and secondary shrub vegetation of low deciduous forest is developing in the west. A more detailed and quantitative study of the project site is required to assess the landscape situation.

20.1.7 Cultural Heritage

In addition to the vegetative and faunal conditions of the area, sites of cultural interest have been evaluated. The town of Guadalupe de Los Reyes is considered a cultural heritage site and modifications to its buildings, which include the Chapel of Nuestra Señora de Guadalupe, the cobbled streets, and the facades of the buildings are prohibited.

20.2 Permitting Considerations

Environmental regulations for mining in Mexico have been subjected to updates in recent years through amendments of the Mining Law (Ley Minera), the General Law of Ecological Equilibrium and Protection to the Environment (LGGEPA), as well as other related laws and norms that apply to mining. There was also a reform initiative to prohibit open pit mining, which was not adopted and is no longer under debate in the legislature.

All the stages of development of a mining project must comply with applicable government regulations, mainly at the federal level, with some formalities at the state and municipal level.

Torex Gold Resources owns several mining concessions that comprise an area of 6,273 hectares within several ejido surface land, in the states of Sinaloa and Durango, Mexico. Although the environmental system outlined for this project encloses areas in both states, the Los Reyes project will be developed within the ejido La Tasajera, located in the municipality of Cosalá, Sinaloa. Therefore, in terms of permits requirements to develop the mining project it will only be necessary to deal with the state of Sinaloa and the municipality of Cosalá, in addition to the requirements at the federal level. This will be different for any associated project such as access roads or electric lines that cross both states.

The environmental baseline study reported no environmental liabilities in the area of the project; however, it is known that mining activities took place back in the 1950's. Some disturbance has been observed within the environmental system due to old mining works such as adits, small waste rock piles and cyanidation vat, as well as exploration roads, agricultural and livestock activities.

The Los Reyes project will consist of open pit and underground exploitation of a gold and silver deposit characterized by a low sulphidation epithermal system containing silica veins, stockwork veins and breccias. According to the metallurgical test work, the various deposits at Los Reyes property show amenability to cyanide leaching (agitated leach), with improved recoveries with fine crushing/grinding of the ore. Main

chemical reagents and substances to be used during the operation stage of the project are diesel fuel, ammonium nitrate (anfo), sodium cyanide, lead nitrate, zinc powder, copper sulfate, sodium metabisulfite, among other.

Fresh water source for the project has yet to be determined due to legal limitations and availability in the hydrological basin (surface waters) and the aquifer (groundwaters); however, Mexican mining legislation allows holders of mining concessions to use water from mining works. Water from mine dewatering can be used for the operations, provided that CONAGUA is notified and the fees for volume of water to use are paid for. Occasional water discharges of treated effluents could be required for managing water balance only.

Other operations in Mexico have secured water needed for mining and milling operations through agreements to purchase treated municipal discharges.

The main permanent or residual environmental impacts anticipated for this project would be the open pits, the waste rock dumps and the tailings disposal facility. No data was available regarding geochemical characterization tests of the waste rock, mined material and tailings, to know their potential to generate acid drainage and/or toxic metal leaching. Acid base accounting (ABA) tests, humidity cell tests and metal mobility tests should be run to representative samples of the tailings and waste rock materials in accordance with the standards NOM-141-SEMARNAT-2003 and NOM-157-SEMARNAT-2009.

The area of the project is not located within any natural protected area, conservation area or fragile environment. There are no restrictions either regarding state or municipal ecological ordinance of the land at Los Reyes project area.

The Los Reyes project will mainly be regulated at the federal level throughout all stages of site preparation, development, construction, operations, and closure. Main environmental permits for construction and operation are those related to the environmental impact, environmental risk analysis (because of the use of hazardous materials), land use change, water use and effluent discharge, hazardous residues management and air emissions. Also, for the closure and abandonment stage there are some federal regulations to comply with.

At the state and municipal level, there are some important formalities to undertake before the construction of the project, such as obtaining the license for construction and land use, an endorsement letter for the use and storage of explosives and a permit to dispose non-hazardous residues at the municipal or private landfills.

An important issue to clarify is regarding the presence of indigenous population at the project area, since some additional formalities, such as a community consultation process, need to be conducted when a project is to be developed in areas with indigenous communities.

The socioeconomic and cultural baseline study prepared by Prime in 2022 reported two indigenous communities (Los Llanos and Cosalá) within the project area; however, this information is not consistent with the database information of INPI (Instituto Nacional de los Pueblos Indígenas). This information will be confirmed when an update of the socioeconomic study is carried out.

The baseline studies indicate that there are apparently no areas of archaeological interest in the project area. It is recommended that an archeological survey be conducted in the area by INAH (National Institute of History and Anthropology) personnel, to obtain an archeological release letter before starting any construction at the project site.

The environmental permits to be obtained for the different stages of the Los Reyes project, at the federal, state and municipal level, are listed below:

Table 20-2: Permits at the Municipal Level

Mining stage	Formality	Agency/ Administrative unit	Approximate response time (labor days)	Description
Construction/Operation	Land use licence	Municipio de Cosalá, Sin. (Desarrollo Urbano, Obras y Servicios Públicos)	3-5	Procedure to obtain the license for use of the land. Document that states the permitted uses and construction characteristics for a specific property, construction restrictions, and any impacts, as established in urban plans.
	Endorsement letter to use and storage explosives	Municipio de Cosalá, Sin. (Protección Civil)	3-5	An endorsement letter should be obtained from the municipality as part of the requirements from SEDENA to permit the use of explosives.
	Nonhazardous domestic residues disposal permit	Municipio de Cosalá, Sin. (Desarrollo Urbano, Obras y Servicios Públicos)	3-5	A permit letter should be obtained from the Municipality for the use of the municipal landfill to dispose of the non-hazardous domestic waste generated at the project site.

Table 20-3: Permits at the State Level

Mining stage	Formality	Agency/ Administrative unit	Approximate response time (labor days)	Description
Construction/operation	Authorization for the management of special handling waste	Secretaría del Bienestar y Desarrollo Sustentable (Dirección de Protección al Ambiente)	30	An authorization should be obtained that covers the generation and comprehensive management plan of special handling waste such as: Construction debris, tires and wood scraps, among others.
	Endorsement letter to use and storage of explosives	State government (Protección Civil)		An endorsement letter should be obtained from the state government as part of the requirements from SEDENA to permit the use of explosives.

Table 20-4: Permits at the Federal Level

Mining stage	Formality		Agency/ Administrative unit	Approximate response time (labor days)	Description
Construction	Option 1	Submittal of the Environmental Impact Manifest (MIA-Regional modality) for its evaluation and Resolution.	SEMARNAT (Dirección General de Impacto y Riesgo Ambiental)	120	A comprehensive environmental manifest shall be submitted for evaluation and Resolution. The MIA Resolution will establish conditions and will require a financial guarantee (bond) to ensure compliance with each of the proposed measures and requested conditions.
		Submittal of the Risk Study (ER) for its evaluation and Resolution.	SEMARNAT (Dirección General de Impacto y Riesgo Ambiental)	120	A risk analysis shall be submitted and will be evaluated together with the MIA when high risk substances such as cyanide is used in the process.
		Submittal of the Technical study (ETJ) to obtain the authorization for the change of land use in forest land.	SEMARNAT (Oficina de Representación en Sinaloa)	60	A detailed forestry inventory and a technical study shall be submitted for evaluation and Authorization. * The authorization of the land use change will be subject to a compensation payment for the hectares that will be deforested.
	Option 2	Submittal of the Unified Technical Document (DTU) for its evaluation and Resolution.	SEMARNAT (Dirección General de Impacto y Riesgo Ambiental)	120	A unified technical document (DTU) that integrates information from the MIA, ER and ETJ shall be submitted for evaluation and obtention of a global Resolution. Both the environmental bond and the compensation payment for deforestation will be required when authorizing the DTU.
	Application for water use permit.		CONAGUA	90	A permit shall be obtained from CONAGUA (water concession title) to use water for mining industrial purposes, covering the volume of water required for the project. Water from mine works dewatering is allowed to use for the mining operation, as long as CONAGUA is notified and the water fees are paid.

Mining stage	Formality	Agency/ Administrative unit	Approximate response time (labor days)	Description
	Application for explosives handling permit	SEDENA	90	An application shall be submitted to SEDENA to get authorization for the storage and use of explosives (endorsement letters from the state and municipality are necessary). An inspection of the magazine will be done by SEDENA personnel.
Construction	Archeological clearance letter	INAH	Not defined	A request letter should be submitted to INAH to conduct a survey and issue an archeological release letter. If there is some archeological interest a rescue and documenting program will be performed.
Operation	Register of hazardous waste generator	SEMARNAT (Dirección General de Gestión Integral de Materiales y Actividades Riesgosas)	Not defined	Registration as a hazardous waste generator, such hydrocarbons residues, chemical substances containers, etc.
	Water discharge permit	CONAGUA	90	An application needs to be filed before CONAGUA with estimated annual volume and the quality of the discharge. This may include the sanitary service water discharge on any other water discharge to septic tanks or the natural environment.
	Accident prevention Plan (PPA)	SEMARNAT (Dirección General de Gestión Integral de Materiales y Actividades Riesgosas)	90	Allows to obtain the approval of the PPA for activities that uses high risk substances such as cyanide.
	Annual operation certificate (COA)	SEMARNAT (Dirección General de Industria, Energías Limpias y Gestión de la Calidad del Aire)	Not defined	This formality is intended to report emissions and transfers from large generators of hazardous waste, those discharging waste water and those emitting 25,000 ton or more of CO ₂ equivalent of greenhouse gases and compounds.
	Mining wastes management plan (PMRM)	SEMARNAT (Oficina de Representación en Sinaloa)	Not defined	A comprehensive plan must be prepared and submitted for the management of the mining wastes (waste rock, tailings, etc.)

Mining stage	Formality	Agency/ Administrative unit	Approximate response time (labor days)	Description
				considering processes, methods and technologies that are environmental, economical and technically feasible.
	Hazardous residues management plan (PMRP)	SEMARNAT (Dirección General de Gestión Integral de Materiales y Actividades Riesgosas)	45	This formality allows the registration and approval of the hazardous residues management plan.
Abandonment	Mine Restoration, closure and post-closure Plan.	SEMARNAT (Oficina de Representación en Sinaloa)	Not defined	A comprehensive closure and reclamation plan should be prepared and submitted before closing the mine.
	Mine closure plan	SE (Secretaría de Economía)	Not defined	It is the obligation of concession holders to carry out the closure of their operation through a Mine Closure Plan. The closure plan should be submitted to the Ministry of Economy within two years an up to one year prior to cessation of operations. The Ministry of Economy will approve the Mine Closure Plan with the opinion of SEMARNAT, in relation to the corresponding Mine Restoration, Closure, and Post-Closure Program.

*In case of an indigenous community, a free consultation process is required to obtain the consent of the community.

An Environmental Impact Statement (Manifestación de Impacto Ambiental or MIA) to cover exploration activities was submitted to SEMARNAT for evaluation on September 30, 2025, and approved December 9, 2025.

Some of the Mexican environmental standards that may apply or be used as a reference for this project are:

- NOM-141-SEMARNAT-2003, that establishes the procedures for tailings characterization, as well as the specifications and criteria for the tailings dam site characterization, construction, operation and closure.
- NOM-157-SEMARNAT-2009, that establishes elements and procedures to instrument mining residues management plans.
- NOM-059-SEMARNAT-2010, Mexican wild native flora and fauna species. Establishes the list of endangered species and specifications for their inclusion, exclusion, or change.
- NOM-161-SEMARNAT-2011, which establishes the criteria for classifying Special Handling Waste and determining which are subject to a Management Plan.

- NOM-138-SEMARNAT/SSA1-2012, that establishes the maximum permissible limits of hydrocarbons in soils and guidelines for sampling in characterization and specifications for remediation.
- NOM-147-SEMARNAT/SSA1-2004, which establishes criteria to determine remediation concentrations of soils contaminated by arsenic, barium, beryllium, cadmium, hexavalent chromium, mercury, nickel, silver, lead, selenium, thallium, and/or vanadium.
- NOM-127-SSA1-2021, Water for human use and consumption. Permissible limits of water quality.
- NOM-001-SEMARNAT-2021, which establishes the maximum permissible limits of pollutants in the discharge of wastewater into national waters and property.
- CE-CCA-001/89, Ecological Quality Criteria for Water. Establishes the water quality criteria, on which the competent authority may classify bodies of water.
- NOM-052-SEMARNAT-2005, which establishes the characteristics, the identification procedure, classification and the lists of hazardous waste.
- NOM-053-SEMARNAT-1993, which establishes the procedure to carry out the leaching test to determine the constituents that make a waste hazardous due to its toxicity to the environment.
- NOM-080-SEMARNAT-1994, which establishes the maximum permissible limits for noise emissions from fixed sources and their measurement method.

20.3 Social Considerations

According to the Cultural and Socioeconomic Base Line Study (CSBLS), the recurring issues at the surveyed communities were the lack of water, poor road conditions, lack of public transportation service to the municipal seat, unemployment, inadequate health care facilities, and absence of support for students. They are also concerned about dust emissions caused by the extraction and transportation of sand and gravel from creeks at the Palo Verde and Las Habitas communities.

Torex Gold, through its subsidiary Minera Amari, works closely with the ejidos regarding development, access improvements, water supply, potential employment and other considerations.

There was no knowledge of any commitment from Minera Amari to the communities, other than the agreements for the use of the surface land with the ejidos. One of the most important issues to address is related to water availability, since very often the communities do not have water for drinking and for agriculture or raising cattle.

The Los Reyes project seems to be very well accepted by the communities. Generation of jobs, improvements of service infrastructure, such as roads and water supply, as well as technical training of local people, are some of the most important benefits that the development of the project will provide to the region.

Considering that the community of El Camichín (57 inhabitants) would be directly affected since it is located inside the project area, Minera Amari would need to reach an agreement with the community for the relocation of the people.

20.4 Closure Considerations

20.4.1 Closure and Reclamation Plans

Closure planning for the Project is at a conceptual level, consistent with a PEA study. A detailed Closure and Reclamation Plan (CRP) will be developed at future stages of study and will be included in the Environmental Impact Manifest (MIA) to obtain the environmental permit from SEMARNAT. The closure strategy is based on standard industry practices and applicable Mexican regulatory requirements and is intended to return the site to a stable and non-polluting condition, with land uses compatible with surrounding environments and communities.

The conceptual closure approach includes the following key activities:

- Decommissioning and removal of plant, infrastructure, and ancillary facilities
- Protect water and soil resources by controlling any pollutant exposure. Any effluent to the environment will comply with the discharge standard NOM-001-SEMARNAT-2021
- Stabilization and recontouring of disturbed areas to support surface water drainage
- Rehabilitation of open pits, WRSFs, and TSF
- Placement of cover materials, where required, to limit infiltration and reduce potential impacts to water quality
- Revegetation of disturbed areas using native species where practical
- Closure of mine openings (e.g., adits, portals and shafts) to prevent access and ensure safety
- Implementation of post-closure monitoring and maintenance programs
- Preparation of the land for long term productive use and/or establishment of wildlife habitat.

20.4.1.1 Open Pit

Surface water diversion structure will be constructed around the pit. The end of mine design will leave stable pit walls. A protection fence and/or berms will be installed around the open to restrict the access of persons and fauna. Land scarification and seeding will be done on the pit surroundings to promote the development of vegetation cover.

Pit backfilling with waste rock will be evaluated for the mining plans.

No groundwater modeling report was available to know if a pit lake will be formed after the mine ceased. It is recommended to conduct a hydrogeological study that evaluates and predicts the effects of the project on the groundwater quality and quantity (geochemical and hydrogeological modeling).

20.4.1.2 Underground

Pastefill will be used as fill material for underground excavations. The underground mining works will be prepared to provide long-term stability conditions. Mine shafts and adits will be properly sealed or protected. Warning signs and fences will be installed to restrict free access to non-stable mine areas.

Access roads and mine works surrounding areas, no longer utilized, will be scarified and seeded with native species.

20.4.1.3 Waste Rock Dumps

Surface water diversion structures will be constructed around the waste rock dumps. The end of mine design will leave stable dumps slopes. The slope angles at the waste dumps for the abandonment stage would be

adequate for vegetation establishment (28° to 34°), the disposal methods of the waste rock would seek for those slope angles.

Geochemical characterization tests results on representative samples of the waste rock will define if it would be necessary to isolate any material that may potentially generate acid drainage or lixiviate toxic metals.

20.4.1.4 Tailings Storage Facility

The criteria for closing and restoration of the TSF should be based on the standard NOM-141-SEMARNAT-2003 which establishes the procedure to characterize tailings, and the specifications and criteria to characterize and prepare the site, project, construction, operation, and post-operation of the tailings dam.

The tailings storage wall slope should be adjusted to provide a static and dynamic stability condition. Erosion control structures will be constructed as necessary, such as rock armoring on the exterior slopes of the embankment and slope grading for stability. The tailings dry stack will be contoured and graded for adequate surface water drainage. Water diversion features to control runoff around and within the tailing facility will be enhanced to avoid long term impact on the surface and underground water.

The tailings will be characterized according to the NOM-141-SEMARNAT-2003 to determine the potential for acid drainage generation and metal leaching. Based on the results of the geochemical characterization of the tailings, a cover with inert and impermeable material may be necessary to avoid water infiltration and formation of toxic leachate to the natural environment. On top of this layer, another layer of topsoil or organic material will be placed and native seeds planted to promote the development of a vegetation cover.

Downstream sedimentation ponds will function as sediment traps for runoff water, before reaching the natural water courses

20.4.1.5 Processing Facility

All waste materials and remnants will be withdrawn and the equipment, machinery and installations in general will be cleaned. In the case of tanks containing chemicals, process sludges or hydrocarbons, all of the product will be used up and the sludge or sediments that have accumulated in the bottom of the tanks will be sent to their final disposal site.

All process plant equipment will be removed, and associated buildings will be demolished.

20.4.1.6 Post Closure Activities

A periodic monitoring and maintenance program will continue for the necessary years until stable and safe conditions are achieved in all disturbed areas.

Mine design, construction and operation should incorporate measures to minimize the requirements for care and maintenance at the abandonment stage.

The post-closure monitoring actions will include: slope stability, safety conditions, pit water condition, surface and underground water quality, air quality, vegetation development and wildlife re-establishment.

20.4.2 Closure cost estimate

A conceptual closure cost estimate has been developed for the Project based on benchmarking against similar operations and preliminary facility layouts and allocation of costs across key closure components.

The total closure cost is estimated at approximately US\$35 million

The estimate includes the following major cost components:

- Plant and infrastructure decommissioning and removal
- Rehabilitation of plant site and general surface areas
- Pit and mine infrastructure closure (including sealing of underground openings)
- WRSF and TSF rehabilitation (representing the largest cost component)
- Water management and pond rehabilitation
- Post-closure monitoring and care and maintenance
- Preliminary and general costs.

Waste rock and tailings storage facility rehabilitation represents the largest proportion of closure costs, followed by plant demolition and general site rehabilitation.

The estimate includes allowances for indirect costs and contingencies and is considered appropriate for a PEA-level study. No detailed quantity take-offs or site-specific unit rate build-ups have been completed at this stage. A detailed table of costs can be found in Section 21 of the report.

21 Capital and operating costs

21.1 Introduction

The capital and operating cost estimates presented in this PEA provide substantiated costs that can be used to assess the preliminary economics of the Los Reyes Project. The estimates are primarily based on an underground mine operation, process plant construction and operation, TSF construction and operation and owner's costs and provisions. The processing plant nameplate capacity is 5,000 t/d (1.83 Mt/a) with a life of mine (LOM) of 15 years.

All operational cost estimates are presented in United States dollars (US\$), with exchange rate from early Q2 2026 being used.

21.2 Capital costs

21.2.1 Overview

The capital cost estimate was developed in Q2 2026 to Class5 estimate in accordance with the Association for the Advancement of Cost Engineering International (AACE international) for an accuracy of $\pm 50\%$. The estimate was developed with Ausenco's in-house database of projects and studies, including processing, onsite infrastructure, offsite infrastructure, project indirect costs, project delivery, owner's costs and provisions. Mining costs was built up by AGP Mining, tailings and water management was developed by Knight Piésold.

Table 21-1: Capital Cost Summary

WBS	Description	Capital Cost (US\$M)	Sustaining Cost (US\$M)	Total Cost (US\$M)
1000	Mining	\$98.2	\$414.9	\$513.1
2000	Process Plant	\$113.3	\$89.4	\$202.7
3000	Additional Process Facilities	\$30.9	\$25.2	\$56.0
4000	On-Site Infrastructure	\$54.1	\$2.0	\$56.1
5000	Off-Site Infrastructure	\$16.5	\$0	\$16.5
Total Direct Costs		\$313.0	\$531.4	\$844.4
6000	Project Preliminaries	\$36.5	\$0	\$36.5
7000	Project Delivery	\$44.2	\$0	\$44.2
Total Indirect Costs		\$80.7	\$0	\$80.7
Total Direct + Indirect Costs		\$393.6	\$531.4	\$925.0
8000	Owner's Costs	\$18.5	\$0	\$18.5
9000	Contingency	\$102.6	\$47.2	\$149.8
Total Capital Cost		\$514.8	\$578.6	\$1,093.4

Note: Total may not add up due to rounding.

21.2.2 Exchange Rates

The estimate is prepared in the base currency of US dollars (USD). Foreign currency pricing has been converted to USD using the exchange rate in Table 21-2. Exchange rates were used to convert the equipment and contract pricing to US dollars for the development of the estimate. The foreign currency exchange rates were obtained at the beginning of Q2 2026.

Table 21-2: Estimate Exchange Rates

Code	Exchange Rate (per USD)
USD	1.00
CAD	1.37
AUD	1.40
EUR	0.86
MXN	17.54

21.2.3 Area 1000 – Mining Capital Costs

The mining capital cost estimate was developed by AGP Mining Consultants from first principles, built up from the mine design and production schedule described in Section 16. The estimate is presented on an owner-operated basis in which the mobile fleet and selected fixed equipment are leased rather than purchased; equipment lease payments are carried in project and sustaining capital cost. As a result, mining capital is dominated by equipment costs, capitalised underground development and fixed infrastructure.

Total mining capital is estimated at US\$513.1 million of direct cost, comprising US\$98.2 million of initial (pre-production) capital and US\$414.9 million of sustaining capital. Sustaining capital is dominated by leasing costs and ongoing underground lateral and vertical development, which continues throughout the mine life. The mining capital estimate by component is summarised in Table 21-3.

Table 21-3: Mining Capital Cost Summary

Component	Initial Capital Cost (US\$M)	Sustaining Capital Cost (US\$M)	Total Capital Cost (US\$M)
Underground	76.9	339.6	416.5
Open Pit	21.3	75.3	96.6
Total	98.2	414.9	513.1

Note: equipment is leased; lease payments are reported under capital and sustaining costs. Numbers may not sum exactly due to rounding.

21.2.3.1 Underground development capital

Capitalised development comprises US\$124.0 million of lateral development (capital-classified ramps, drives and accesses) and US\$18.8M of vertical development (capital raises). Development classified as operating (ore drives and in-stope slots) is carried in operating cost.

21.2.3.2 Fixed plant and infrastructure

Underground fixed plant (US\$5.9 million) and underground infrastructure and support (US\$33.4 million) cover primary ventilation fans and reticulation, dewatering and water-management installations, fixed electrical reticulation, the explosives magazine, workshops, warehouse and mine offices that are not leased. Open-pit infrastructure and support (US\$14.7 million) covers the main haul roads, pit dewatering, the explosives facility and mine support buildings that are not leased. The paste backfill plant is in Ausenco's scope (Area 3000) and is not included in mining capital.

Operating costs incurred prior to mill start-up — in Year -1 and Year 0 — are capitalised as pre-production operating cost (principally open-pit pre-production mining and underground operating development). Operating costs from Year 1 onward, including the ramp-up period, are not capitalised. Capitalised pre-production operating cost including PTU is approximately US\$14.4 million, and total pre-production cost (initial capital plus capitalised pre-production operating plus contingency) is approximately US\$135.9 million.

21.2.4 Area 2000 - Process Plant Capital Costs

The selection and sizing of process equipment requirements was based on process flowsheets and process design criteria as defined in Section 17. All major equipment was sized based on the process mass balance as dictated by the process design criteria, to develop a mechanical equipment list. The mechanical equipment list was then developed through historical budgetary quotations from the Ausenco database, further benchmarked against recent Mexican gold and silver project and studies.

Bulk quantities for the process plant were derived for earthworks and platework and priced from other benchmark projects. All other quantities for electrical, instrumentation, concrete, steel, piping, cable and minor platework were factored and priced accordingly to the benchmarked circuits and processes. Sustaining capital cost pertains to the capitalized operating earthworks required for the solid tailings handling from the plant as well as an allowance to replace major equipment during the LOM.

The process capital cost breakdown is summarized in Table 21-4, including all contractor's direct and indirect labour, permanent equipment, materials, and freight associated with the physical construction of the areas.

Table 21-4: Process Capital Costs

WBS	Process Plant Areas	Initial Capital Cost (US\$M)	Sustaining Capital Cost (US\$M)	Total Capital Cost (US\$M)
2000	Process Plant (General)		25.8	25.8
2100	Crushing	7.3		7.3
2200	Stockpile and Reclaim	8.7		8.7
2300	Grinding	14.0		14.0
2600	Leaching and Gold Processing	36.2		36.2
2700	Detox and Tailings Handling	27.5	63.6	91.2
2800	Reagents	10.1		10.1
2900	Plant Services	9.4		9.4
Total	Process Plant	113.3	89.4	202.7

Note: Total may not add up due to rounding.

21.2.5 Area 3000 – Additional Process Facilities

21.2.5.1 Introduction

Additional facilities that are required for the mining and process operations include the TSF, paste plant, and paste plant piping. The initial capital and sustaining capital costs associated with this are provided in Table 21-5.

21.2.5.2 Tailings Storage Facility Development

The TSF was developed through the material take-offs derived from drawings and conceptual layouts of the TSF. MTOs generated for the TSF include site preparation heavy civil construction, piping and instrumentation work activities. Unit rates were developed and benchmarked based on comparable project experience. Unit rates are inclusive of labour, equipment rental, operating, insurance, and contractor markup costs. Material costs are included in the unit rates, where applicable. Unit rates were developed assuming all construction works are to be completed by a contractor owned equipment fleet.

Unit rates for geosynthetics, including non-woven geotextile and high-density polyethylene (HDPE) geomembrane, include supply and installation costs. Installation will consist of the placement of the material, the welding of seams, and Quality Assurance testing.

Lump sum budgetary allowances have been included in the cost estimate for the following tasks and single unit items:

- Instrumentation installation including Vibrating Wire Piezometers (VWPs), survey monuments, inclinometers, and data loggers
- Construction water management, including dewatering and temporary construction water management structures
- Emergency Dump Pond construction
- Upstream Water Control Dam spillway construction
- TSF Sediment Pond construction

- Waste Rock Water Management Structures

21.2.5.3 Paste Plant Development and Paste Plant Off-Plot Piping

The sizing of the paste plant was based upon Ausenco's historical projects. An equipment list was developed to suit the paste production as stipulated from the mine plan.

Bulk quantities of earthwork were derived from the layout of the paste pad in relation to the main plant site. Other bulk quantities were factored and priced accordingly to the benchmarked paste plant as seen in other historical projects.

Off-plot piping for the transport of tailings to the paste plant and paste to the underground portal were developed MTOs. MTOs were generated for piping with unit rates developed and benchmarked based on projects in the region.

Table 21-5: Additional Process Facilities

WBS	Description	Initial Capital Cost (US\$M)	Sustaining Capital Cost (US\$M)	Total Capital Cost (US\$M)
3100	Tailings Storage Facility	10.4	25.2	35.6
3200	Paste Plant	18.5		18.5
3300	Paste Plant Off Plot Piping	2.0		2.0
Total Capital Cost		30.9	25.2	56.0

Note: Total may not add up due to rounding.

21.3 Area 4000 – Onsite Infrastructure Capital Costs

Onsite infrastructure cost includes the major support infrastructure on the project site necessary for the operation of the mine. This estimate includes:

- bulk earthwork required to develop the process plant pad, paste plant pad, interconnecting roads and additional laydown areas to facilitate temporary facilities and storage.
- water management and balance for the entire site.
- infrastructure for power distribution.
- fuel storage for mine, process, and support fleets.
- sewage and effluent treatment for site.
- offices, workshops, warehouses, security buildings, accommodations as well as additional, support buildings.
- surface mobile fleet for non mining site crew and staff.

The costs are presented in Table 21-6.

The bulk earthworks and water management for the site were developed through the MTO quantities derived from the layout drawings and layouts for each respective section. Ausenco developed bulk earthwork MTOs from their plant site layout drawings, and sourced unit rates from historical projects in the region.

KP developed water management costs using the same methodology in Section 21.2.4.1, deriving MTO's from conceptual layouts and drawings, while utilizing KP's database of the comparable projects.

Infrastructure for this section was derived from historical quotes infrastructure and facilities to suit the project, with benchmarking for project specific quantities.

Table 21-6: On-site Infrastructure Capital Costs

WBS	On-Site Infrastructure	Initial Capital Cost (US\$M)	Sustaining Capital Cost (US\$M)	Total Capital Cost (US\$M)
4100	Bulk Earthworks	8.7	0	8.7
4200	Water Management	16.9	2.0	18.9
4300	Hv Power Switchyard and Power Distribution	11.8	0	11.8
4400	Fuel Storage	0.9	0	0.9
4500	Sewage	1.1	0	1.1
4600	Infrastructure Buildings	14.7	0	14.7
Total	On-Site Infrastructure	54.1	2.0	56.1

Note: Total may not add up due to rounding.

21.3.1 Area 5000 Off-Site Infrastructure Capital Costs

Offsite infrastructure costs includes the major costs associated with upgrading the main access road, water supply pipeline, and power supply. Material takeoffs were taken off the initial process pant and civil layouts and then priced based on historical data. There are no sustaining costs in this area, and the off-site infrastructure costs are summarized in Table 21-7.

Table 21-7: Off-site Infrastructure Capital Costs

5000	Off Site Infrastructure	US\$M
5100	Main Access Road	4.6
5200	Water Supply	2.4
5300	Power Supply	9.5
Total	Off Site Infrastructure	16.5

Note: Total may not add up due to rounding.

21.3.2 Area 6000-8000 – Project Indirects

Indirect costs are calculated as a percentage of direct costs, with additional costs built up from first principles to ensure in the execution of the project.

Project indirect costs include:

- temporary construction facilities and services
- messing and catering during construction

- vendor reps and assistance
- equipment Spares
- first fills and initial charges
- project delivery
- owner's costs.

Additionally, provisions were also included in the form of contingency.

The summarized costs are shown in Table 21-8.

Table 21-8: Project Indirect Costs

WBS	Description	Initial Capital Cost (US\$M)	Sustaining Capital Cost (US\$M)	Total Capital Cost (US\$M)
6100	Temporary Construction Facilities and Services	34.3		34.3
6200	Commissioning Assistance and Ops Readiness	0.2		0.2
6300	Spares (Included Capitalized Project OPEX)			
6400	First Fills (Included Capitalized Project OPEX)			
6500	Miscellaneous Distributable Cost & Vendor Representatives	2.0		2.0
	Project Preliminaries	36.5		36.5
7100	Engineering and Procurement	23.8		23.8
7200	Construction Management	20.4		20.4
	Project Delivery	44.2		44.2
8000	Owner's Cost	18.5		18.5
	Total Project Preliminaries, Project Delivery, and Owner's Cost	99.2		99.2
9000	Contingency	102.6	47.2	149.8

Note: Total may not add up due to rounding.

21.3.2.1 6100 - Temporary Construction Facilities and Services

This estimate covers the preproduction costs during the construction period, including:

- temporary construction facilities
- construction camp
- camp accommodations for the construction staff
- travel for construction staffing from out of state locations
- camp catering, maintenance and housekeeping costs
- construction equipment including mobile equipment, scaffolding, safety supplies.

The total cost is estimated to be \$34.3 million.

21.3.2.2 Commissioning Assistance & Operations Readiness

Commissioning assistance and operation readiness covers the vendors representatives and contractor support before commissioning. These costs incur during the installation of the purchased equipment, and the cost are benchmarked based on previous Mexican projects. The total cost is estimated to be \$0.2 million.

21.3.2.3 Spares (Capital Commissioning, Operating)

Commissioning spares and operating spares quantities were recommended and priced by equipment suppliers. Where quotations were not solicited and obtained from vendors, factor were applied based on Ausenco's historical database and project factors. The capital cost is captured in the capitalized operating cost.

21.3.2.4 First Fills

Process first fills quantities, such lubricants, media, reagents were calculated based on engineering design and priced using requested vendor quotes. The capital cost is captured in the capitalized operating cost.

21.3.2.5 Project Delivery (EPCM)

Project delivery costs, including engineering, procurement, and construction management costs for the process plant have been estimated based on first principles. The total EPCM costs is \$44.2 million, consisting of the following:

- engineering
- procurement
- construction management
- field inspection and expediting
- site and office expenses
- corporate overhead and fees
- commissioning support.

21.3.2.6 Owner capital costs

Owner costs for preproduction have been estimated at \$18.5 million to include:

- owner's project team and expenses
- administration, finance, insurance and legal fees
- environmental consultation and management
- human resources, recruiting, and training
- stakeholder relations
- and site security.

21.3.3 Area 9000 – Contingencies

Provisional costs accounts for the difference in costs between the estimated and actual costs of material and equipment. The level of contingency varies depending on client requirements as well as the nature of the contract for the project construction. Due to the uncertainties at the time of this estimate (As dictated

by the level of engineering definition, basis of estimate, schedule development, etc.), it is imperative that a contingency is included to cover the risk from these uncertainties.

The following contingencies were applied:

- TSF and site wide water management, 40%
- Process plant infrastructure, onsite infrastructure, off-site infrastructure, and paste Plant, 37%
- Mining capital costs, 25% for initial, 10% for sustaining.

The total contingencies estimated for this project consists of \$102.6 million in the initial phase and a total LOM cost of \$149.8 million.

21.3.4 Closure Costs

The estimated the total closure net costs for the Los Reyes project is to be \$35 million. These are the owner's costs to be incurred after production and it is inclusive of all necessary demolition, rehabilitation, revegetation, earthworks, disposal and tipping fees, as well as post-closure monitoring.

21.4 Operating costs

21.4.1 Overview

The costs considered on-site operating costs are those related to mining, processing, tailings handling, maintenance, power and general and administrative (G&A) activities.

A summary of the operating costs is presented below in Table 21-9.

The LOM average annual operating cost is US\$79.93/t processed, including a G&A cost of US\$8.43/t

Table 21-9: LOM Average Operating Cost

Cost Area	US\$/t Processed
Mining	37.36
Process	23.39
G&A	8.43
Total	79.93

Note: Total may not add up due to rounding. Unit rates exclude PTU.

21.4.2 Basis of Estimate

The following key assumptions were made to estimate the operating costs for the project:

- Cost estimates are based on Q2 2026.
- Costs are expressed in United States dollars (US\$).
- Power cost of US\$0.13 per kilowatt-hour (kWh) was assumed.
- A diesel cost of US\$1.14 per litre was assumed based on on-site cost benchmarks from projects in the same region.

- A gasoline cost of US\$1.18 per litre was assumed based on on-site cost benchmarks from projects in the same region.
- A throughput of 5000 t/d or 1.83 Mt/a was used for the processing plant.
- Crushing circuit availability is assumed to be 65%, grinding availability is assumed to be 92%, filtration availability is assumed to be 85%, and paste plant availability is assumed to be 65%.
- ROM and concentrate grades and recoveries are based on metallurgical test work results described in Section 13.
- Material and equipment are purchased as new.
- Reagent consumption rates are based on metallurgical test work results and in-house benchmarks.
- Grinding media consumption rates are based on mineral material characteristics as described in Section 13.
- Mining costs were estimated by MOS/AGP Mining on an owner-operated basis with a leased mobile fleet included in the capital estimate.

21.4.3 Mining Operating Costs

21.4.3.1 Mining Operating Costs

Mining operating costs were developed by MOS Mining/AGP Mining from the mine design, production schedule and first-principles build-up described in Section 16, on an owner-operated basis with a leased mobile fleet. Lease payments for the mobile fleet and selected fixed equipment are included in mining capital cost (see Section 21.2.3). The life-of-mine mining operating cost is estimated at US\$1,122.5 million, split US\$840.5 million underground and US\$282.0 million open pit inclusive of capital period operating expenses. Mining operating costs incurred after commercial production is achieved total US\$1,108.5 million, equivalent to US\$42.45/t of material processed. Mining operating costs by category are summarised in Table 21-10.

Table 21-10: Mining Operating Costs Summary (Life of Mine)

Category	Underground \$M	Open Pit US\$M	Total US\$M
Labour	160	91	250
Maintenance	203	80	283
Auxiliary equipment	–	8	8
Fixed plant	13	–	13
Consumables	368	19	386
Diesel/power/gas	97	85	182
Total mining operating	841	282	1,123

Note: of the Labour line, US\$55 million is PTU payments (UG US\$37 million plus OP US\$18 million); the balance is salaries. Numbers may not sum exactly due to rounding.

The underground mining costs are comprised of development at US\$4,011/m (fully allocated), stoping production at US\$57.09/t and pastefill costs at \$11.55/t mined. Underground operating cost is driven by 127,880 m of lateral development and 13.0 Mt of stope production over the mine life.

The open-pit mining cost is US\$4.34/t of total material moved, equivalent to US\$9.31/t of mineralised material and US\$3.60/t of waste on a fully-allocated per-material basis. Open-pit cost reflects 87.3 Mt of total material at a 6.76:1 strip ratio (76.0 Mt waste, 11.3 Mt ore).

The estimate is to a PEA level of accuracy (Class 5, $\pm 50\%$, consistent with the rest of the Section 21 estimate).

21.4.3.2 Paste Plant Operating Costs

Paste plant operating costs are based on the estimated annual volumes of paste backfill required for the project. Benchmarks and in-house values were assumed for physical parameters such as bulk density, moisture content of mixed paste, and cement binder content. Costs for the paste plant were estimated for each cost category as described below for the process plant, including power, reagents, maintenance, and labour. The LOM average paste plant operating cost is estimated to be US\$11.9 million.

21.4.4 Process Plant Operating Costs

The process operating cost estimate is based on a throughput of 5,000 t/d (1.83 Mt/a). The plant will produce doré gold through a process consisting of crushing, grinding, leach, CCD, Merrill Crowe, detox and tailings handling. The LOM average process operating costs are estimated to be US\$42.1 million.

Table 21.9 summarizes the operating costs for the process plant.

Table 21-11: Process Plant Operating Cost Summary

Cost Area	LOM Average Annual Costs (US\$M)
Power	8.7
Reagents	21.1
Consumables	3.7
Maintenance	2.1
Labour	3.0
Mobile Equipment	1.2
Lab Services	0.2
Fresh Water Supply	1.9
Total	42.1

Note: Total may not add up due to rounding

21.4.4.1 Reagents and Consumables

Reagents, grinding media and various consumables are required to process the mineralised material from the active zones in the Los Reyes deposit. Consumption rates of each of the consumable items are based on the metallurgical test work outlined in Section 13 and based on the planned process plant throughput of 5000 t/d. The total costs of the reagents and consumables by area as well as the costs of mobile equipment used are shown below in Table 21-12. The LOM average reagents and consumables annual operating cost is US\$24.8 million.

Table 21-12: Reagents and Consumables Cost Summary

Cost Area	Annual Costs (US\$M)
Crushing	0.6
Grinding	4.3
Leaching	15.1
Precious Metal Recovery	7.5
Detox	7.7
Tailings	0.2
Total	24.8

Note: Total may not add up due to rounding

21.4.4.2 Maintenance Consumables

Annual maintenance consumable costs were calculated based on a total installed mechanical capital cost by area using a weighted average factor from 3 to 4%. The process area includes the three-stage crushing circuit, one stage grinding circuit, whole ore leach and Merrill Crowe circuit, reagent handling and plant services areas. The LOM average maintenance consumables annual operating cost is US\$2.1 million.

21.4.4.3 Power

Power operating costs are calculated from an estimate of annual power consumption using a unit cost of US\$0.13/kWh. The annual power consumption for the processing plant is based on the average utilization of each motor on the electrical load list for the process plant.

The process plant energy consumption is estimated to be 98,338 MWh/a. The LOM average annual total power operating cost is US\$8.7 million.

21.4.4.4 Labour

Labour includes all processing and maintenance labour costs.

Processing production labour was developed using benchmarks from similar projects and includes operation departments such as metallurgy, mill operations, maintenance and the assay lab.

Each position was defined and classified as salary and wages. Costs included taxes and benefits. For process operations labour, the LOM average annual labour operating cost is US\$3.0 million.

Including G&A personnel, the total estimated labour force was estimated at 154 people for the process plant, paste plant, and associated areas, not including any mining personnel. The estimate was based on providing a labour force to support continuous operations at 24 hours/day and 365 days/year.

21.4.4.5 Mobile Equipment

Vehicle costs are based on a scheduled number of light vehicles and mobile equipment for the process plant. The costs include fuel, annual maintenance, spares and tires, annual insurance, and equipment leasing costs. The LOM average annual mobile equipment costs for the process plant are estimated at US\$1.2 million.

21.4.4.6 On-site Laboratory Services

Operating costs for the assay lab were estimated based on a review of benchmark projects with similar flowsheets and sampling requirements. The lab is expected to handle grade control samples, mill solid and

aqueous samples, water testing, doré quality testing, and other miscellaneous tests as required. The LOM annual operating costs for the laboratory are estimated at US\$0.2 million.

21.4.4.7 Water Treatment Costs

Water supply costs were calculated using the expected annual makeup requirements of the process plant from the mass balance. The total volume of water from mine dewatering and site collection ponds was estimated to be sufficient to supply the process plant with the required fresh water each year.

Effluent treatment costs were assumed based on benchmarking from the area. Costs to treat fresh water for potable water use were estimated based on the number of personnel within the process plant and associated areas. Sewage treatment costs were treated as a yearly allowance based on benchmarks from the region.

The LOM annual average water costs are estimated at US\$1.9 million.

21.4.5 General & Administrative Costs

The G&A operating costs cover the expenses of the operating departments, and a summary is presented in Table 21-13.

G&A costs were developed using Ausenco's in-house data on existing Mexican operations. The costs were estimated based:

- Site maintenance (including maintenance-related mobile equipment, on-site road maintenance)
- Mobile equipment costs associated with G&A areas and personnel
- Human resources and health & safety (including personnel, recruiting, training, community relations, personal protective equipment and first-aid)
- Environmental (including sampling and effluent testing)
- Onsite operations camp (including flights in/out of the nearby town)
- Mercury retort and disposal costs for handling mercury contamination from the process plant
- Administration costs (including TOA payments, office supplies, IT hardware & software, and support services)
- Contract services (including insurance, sanitation, licence fees and legal fees).

The LOM average total annual G&A cost was estimated at US\$15.6 million.

Table 21-13: G&A Cost Summary

G&A Expenses	Average Annual Costs (US\$M)
Site Maintenance	0.2
Personnel	2.7
Human Resources	1.1
Mobile Equipment	1.1
Health and Safety	0.2
Environmental Costs	0.6
IT & Telecommunications	0.5
Contract Services, Insurance, Legal	3.9

G&A Expenses	Average Annual Costs (US\$M)
Camp Costs (including Travel)	2.8
Administrative Costs	2.4
Mercury Storage and Disposal	0.0
Total	15.6

Note: Total may not add up due to rounding

22 Economic analysis

22.1 Forward-looking information

The results of the economic analyses discussed in this section represent forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here. Information that is forward-looking includes the following:

- Mineral resource estimates
- Assumed commodity prices and exchange rates
- Proposed mine production plan
- Projected mining and process recovery rates
- Assumptions as to mining dilution and estimated future production
- Sustaining costs and proposed operating costs
- Assumptions as to closure costs and closure requirements
- Assumptions as to environmental, permitting, and social risks.
- Additional risks to the forward-looking information include:
 - Changes to costs of production from what is assumed
 - Unrecognized environmental risks
 - Unanticipated reclamation expenses
 - Unexpected variations in quantity of mineralised material, grade, or recovery rates
 - Accidents, labour disputes and other risks of the mining industry
 - Geotechnical or hydrogeological considerations during mining being different from what was assumed
 - Failure of mining methods to operate as anticipated
 - Failure of plant, equipment, or processes to operate as anticipated
 - Changes to assumptions as to the availability of electrical power, and the power rates used in the operating cost estimates and financial analysis
 - Changes to site access, use of water for mining purposes and to time to obtain environment and other regulatory permits
 - Ability to maintain the social licence to operate
 - Changes to interest rates
 - Changes to tax rates.

Readers are cautioned that the 2026 PEA is preliminary in nature. It includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the 2026 PEA will be realized.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

22.2 Methodologies used

The Project was evaluated using a discounted cash flow (DCF) analysis based on a 5% discount rate. Cash inflows consisted of quarterly revenue projections. Cash outflows consisted of capital expenditures, including pre-production costs; operating costs; taxes; and royalties. These were subtracted from the inflows to arrive at the annual cash flow projections. Cash flows were taken to occur at the mid-point of each period. It must be noted that tax calculations involve complex variables that can only be accurately determined during operations and, as such, the actual post-tax results may differ from those estimated. A sensitivity analysis was performed to assess the impact of variations in metal prices, discount rate, operating costs, initial capex, metal recovery, and head grade.

The capital and operating cost estimates are presented in Section 21 of this Report in Q2 2026 United States dollars. The economic analysis was run on a constant dollar basis with no inflation.

22.3 Financial model parameters

The economic analysis was performed assuming a gold price of US\$3,600/oz and a silver price of US\$50/oz. These metal prices were based on consensus analyst estimates and recently published economic studies. The forecasts used are meant to reflect the average metals price expectation over the life of the Project. No price inflation or escalation factors were considered. Commodity prices can be volatile, and there is the potential for deviation from the forecast.

22.3.1 Assumptions

The economic analysis also used the following assumptions:

- Construction period of two years with half year ramp up in year 1
- Mine life of 15 years (last year is a partial year)
- Results based on 100% ownership
- Capital cost funded with 100% equity (no financing cost assumed)
- All cash flows discounted to start of construction period using mid-period discounting convention
- All metal products are sold in the same year they are produced
- Project revenue is derived from the sale of gold and silver doré
- No contractual arrangements for refining currently exist
- All costs reported in United States dollars with no Mexican peso denominated costs.

22.3.2 Taxes

The project has been evaluated on an after-tax basis to provide an approximate value of the potential economics. The tax model was compiled with assistance from Torex.

- The calculations are based on the tax regime in place as of the date of the preliminary economic analysis.
- Income tax uses a 30% tax rate inclusive of SMD mining royalty at 8.5%.
- Total Opening tax pool of \$44.7 million and total exploration tax pool of \$53M was used over the life of mine.
- Total tax depreciation of \$777 million was used for income tax calculation and excludes mine equipment leasing.

- Mine leasing costs of \$304M were excluded from gross revenue.

The overall effective tax rate over the life of mine is 27.3%.

At the base case price assumption, total tax payments are estimated to be \$898.7 million over the life of mine.

22.3.3 Working capital

An estimate of working capital has been incorporated into the economic analysis based on the following assumptions:

Table 22-1: Estimate of working capital

Item	Time
Accounts Receivable	0 days
Inventories	30 days
Accounts Payable	30 days

22.3.4 Royalties

The mine is subject to a 4% NSR royalty and EMD royalty of 1% gross revenue.

The total royalty costs were estimated to be \$346.1 million over the life of mine.

22.3.5 Closure costs and salvage value

Closure costs are applied at the end of the life of mine. Closure costs were estimated to be \$35 million.

22.3.6 Off-site costs

Off-site costs and payabilities were determined using industry benchmarks and standard industry practices. with input from Torex. A gold refining charge of US\$5.00/oz and a silver refining charge of US\$0.50/oz was used. Life of mine total offsite costs were estimated to be US\$28 million.

22.4 Economic analysis

The pre-tax NPV discounted at 5% is US\$2,098 million; the IRR is 46.72%; and the payback period is 1.4 years. On a post-tax basis, the NPV discounted at 5% is US\$1,491 million; the IRR is 37.32%; and payback period is 1.9 years. A summary of project economics is summarized in Table 22-2 and illustrated in Figure 22-1. The analysis was done on a quarterly cashflow basis; the cashflow output is shown in Table 22-3. Readers are cautioned that the PEA is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Table 22-2: Economic analysis summary

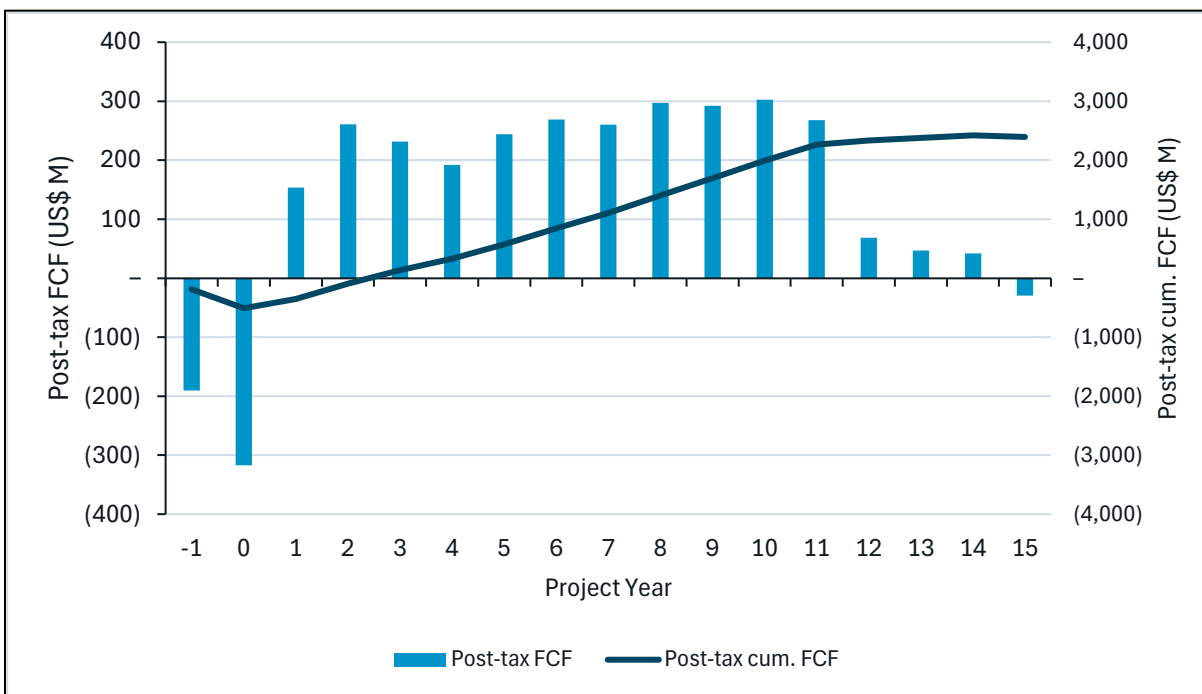
Item	Units	Total/Avg
Macro		
Gold price	US\$/oz	3,600.0
Silver price	US\$/oz	50.0
Discount Rate	%	5.0%
Production		
Project Life	yrs	14.4
Mine life at nameplate capacity	yrs	10.0
Total Resource Mined	kt	26,074
LOM Au Grade	g/t	1.68
LOM Ag Grade	g/t	63.37
Total Waste Mined	kt	76,034
Average Annual Mined Resource	ktpa	1,808
Total ROM feed	kt	26,074
Gold recovery	%	94.8%
Silver recovery	%	81.1%
Total Recovered Au	koz	1,338
Total Recovered Ag	koz	43,082
Total Recovered Au Eq	koz	1,936
Average Annual Recovered Au Eq at nameplate capacity	koz/a	166
Total Payable Au	koz	1,337
Total Payable Ag	koz	42,651
Total Payable Au Eq	koz	1,929
Average Annual Payable Au Eq at nameplate capacity	koz/a	166
Average Annual Payable Au Eq	koz/a	134
Revenue/Costs		
Total Revenue	US\$ M	6,945
Average Annual Revenue	US\$ M/a	482
EBITDA ⁽⁵⁾	US\$ M	4,439
Average Annual EBITDA ⁽⁵⁾	US\$ M/a	308
"Total On Site Operating Costs (Mining, Process, G&A)"	US\$ M	2,131
Open Pit Unit Cost*	\$/t mined OP	2.89
Open Pit Unit Cost*	\$/t processed	9.67
Underground Unit Cost*	\$/t mined UG	54.15
Underground Unit Cost*	\$/t processed	30.81
Paste Plant Unit Cost*	\$/t mined UG	11.52
Paste Plant Unit Cost*	\$/t processed	6.55
Mining + Paste Plant Unit Cost*	\$/t mined UG	65.68
Mining + Paste Plant Unit Cost*	\$/t processed	37.36
Process Unit Cost*	\$/t processed	23.39
G&A Unit Cost*	\$/t processed	8.43

Item	Units	Total/Avg
Total operating unit cost ^{*(4)}	\$/t processed	79.93
"Total Off Site Operating Costs (Transport, Treatment & Refining)"	US\$ M	28
Cash costs		
Total cash costs - gold equivalent ⁽¹⁾	US\$/oz Au Eq	1,260
Mine-site all-in sustaining costs - gold equivalent ⁽²⁾	US\$/oz Au Eq	1,578
Capital costs		
Initial CAPEX	US\$ M	515
Sustaining CAPEX	US\$ M	579
Closure CAPEX	US\$ M	35
Salvage Value	US\$ M	–
Pre-production OPEX	US\$ M	19
Pre-Tax Economics		
NPV (5%)	US\$ M	2,098
IRR	%	46.7%
Payback	yr	1.4
NPV/Initial Capital	-	4.1
Post-Tax Economics		
NPV (5%) ⁽³⁾	US\$ M	1,491
IRR	%	37.3%
Payback	yr	1.9
NPV/Initial Capital	-	2.9

Notes

- 1) Total cash costs include operating cost, offsite costs and royalties excluding PTU profit share.
- 2) Mine-site AISC include total cash costs, closure costs and sustaining capital expenditures.
- 3) The effective tax rate is 27.3% (This combines corporate tax and mineral tax).
- 4) Total operating unit cost includes mine, mill, paste plant, G&A and offsite costs (excludes PTU profit share).
- 5) EBITDA, Average Annual EBITDA, AISC, total cash costs, and sustaining capital expenditures are non-GAAP financial measures or ratios. These measures have no standardised meaning under IFRS Accounting Standards and may not be comparable to similar measures used by other issuers. As the Project is not in production, Torex does not have historical non-GAAP financial measures nor historical comparable measures under IFRS, and therefore the foregoing prospective non-GAAP financial measures or ratios may not be reconciled to the nearest comparable measures under IFRS. Refer to the "Non-GAAP Financial Measures" in Section 2.8 of this Technical Report for more information, including a detailed description of each of these measures.

Figure 22-1: LOM post-tax free cashflow ¹



Notes:

1. Post-tax free cashflow is a non-GAAP financial measure or ratio. This measure has no standardised meaning under IFRS Accounting Standards and may not be comparable to similar measures used by other issuers. As the Project is not in production, Torex does not have historical non-GAAP financial measures nor historical comparable measures under IFRS, and therefore the foregoing prospective non-GAAP financial measure or ratio may not be reconciled to the nearest comparable measures under IFRS. Refer to the "Non-GAAP Financial Measures" in Section 2.8 of this Technical Report for more information, including a detailed description of post-tax free cashflow.

Source: Ausenco, 2026

Table 22-3: Cashflow statement on an annualized basis

	Units	Total/Avg	-1	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Macro assumptions																			
Gold price - flat	US\$ /oz	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600
Silver price - flat	US\$ /oz	50	50	50	50	50	50	50	50	50	50	50	50	50	50	50	50	50	50
Free cash flow valuation																			
Revenue	US\$ M	6,944.6	–	–	398.4	629.9	573.1	532.4	610.6	600.3	577.6	619.9	609.6	625.8	583.1	236.5	165.2	134.8	47.3
Operating cost	US\$ M	(2,131.3)	–	–	(106.3)	(179.2)	(185.7)	(182.6)	(188.8)	(175.8)	(178.7)	(171.5)	(169.5)	(169.8)	(153.4)	(100.1)	(75.1)	(60.0)	(34.7)
Refining charges	US\$ M	(28.0)	–	–	(1.0)	(2.4)	(2.5)	(2.1)	(2.4)	(2.4)	(2.3)	(2.5)	(2.7)	(2.7)	(2.7)	(1.1)	(0.6)	(0.5)	(0.2)
Royalties	US\$ M	(346.1)	–	–	(19.9)	(31.4)	(28.6)	(26.5)	(30.4)	(29.9)	(28.8)	(30.9)	(30.4)	(31.2)	(29.0)	(11.8)	(8.2)	(6.7)	(2.4)
EBITDA	US\$ M	4,439.1	–	–	271.2	416.9	356.4	321.1	389.0	392.2	367.8	415.1	407.0	422.2	398.0	123.6	81.3	67.6	10.0
Pre-production OPEX	US\$ M	(19.4)	–	(19.4)	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Initial capex	US\$ M	(514.8)	(190.4)	(297.9)	(26.5)	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Sustaining capex	US\$ M	(578.6)	–	–	(51.7)	(66.9)	(53.2)	(68.8)	(66.3)	(43.1)	(35.0)	(31.7)	(31.2)	(32.4)	(33.2)	(30.8)	(18.1)	(11.7)	(4.3)
Closure capex	US\$ M	(35.0)	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	(35.0)
Change in working capital	US\$ M	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Pre-tax unlevered free cash flow	US\$ M	3,291.4	-190.4	-317.2	192.9	349.9	303.1	252.3	322.6	349.0	332.8	383.3	375.8	389.8	364.8	92.8	63.2	55.8	(29.3)
Pre-tax cumulative unlevered free cash flow	US\$ M		(190.4)	(507.6)	(314.6)	35.3	338.4	590.8	913.4	1,262.4	1,595.2	1,978.5	2,354.3	2,744.1	3,108.9	3,201.6	3,264.8	3,320.7	3,291.4
Unlevered cash taxes	US\$ M	(898.7)	–	–	(39.2)	(89.3)	(71.4)	(60.5)	(78.3)	(79.7)	(72.6)	(85.9)	(83.5)	(87.1)	(96.8)	(24.3)	(16.2)	(13.9)	–
Post-tax unlevered free cash flow	US\$ M	2,392.7	-190.4	-317.2	153.8	260.7	231.8	191.9	244.3	269.3	260.2	297.4	292.3	302.7	267.9	68.5	47.0	41.9	(29.3)
Post-tax cumulative unlevered free cash flow	US\$ M		(190.4)	(507.6)	(353.8)	(93.2)	138.6	330.5	574.7	844.0	1,104.2	1,401.7	1,693.9	1,996.6	2,264.5	2,333.0	2,380.0	2,421.9	2,392.7
Production																			
Total OP mineralised material mined	kt	11,251	–	303	1,728	1,187	350	309	958	822	690	1,245	501	1,125	1,031	564	437	–	–
Total OP waste	kt	76,034	–	1,697	6,141	4,813	5,650	5,691	7,042	5,178	7,310	4,768	5,499	6,875	6,969	5,436	2,967	–	–
OP gold head grade	g/t	1.11	–	1.99	1.44	1.30	0.81	0.73	0.94	1.04	1.12	1.45	0.52	0.76	1.14	0.41	1.06	–	–
OP silver head grade	g/t	25.65	–	18.66	15.98	17.04	12.40	18.34	26.72	29.07	21.16	26.97	22.47	24.93	67.20	19.25	18.28	–	–
Strip ratio	waste:mineralised material	6.76	–	5.59	3.55	4.05	16.13	18.40	7.35	6.30	10.59	3.83	10.97	6.11	6.76	9.64	6.79	–	–
Total material mined	kt	87,286	–	2,000	7,869	6,000	6,000	6,000	8,000	6,000	8,000	6,013	6,000	8,000	8,000	6,000	3,404	–	–
Total UG mineralised material mined	kt	14,822	–	–	180	1,305	1,478	1,487	1,478	1,483	1,486	1,483	1,492	1,492	1,301	158	–	–	–
UG gold head grade	g/t	2.12	–	–	2.73	2.22	2.13	2.10	2.23	2.14	2.05	2.05	2.07	2.14	2.03	2.13	–	–	–
UG silver head grade	g/t	92.00	–	–	111.96	94.73	97.76	80.39	84.58	83.43	85.89	88.71	102.58	101.44	97.31	107.46	–	–	–
Total mineralised material mined	kt	26,074	–	303	1,908	2,492	1,828	1,796	2,436	2,305	2,176	2,728	1,994	2,617	2,332	721	437	–	–
Mill feed	kt	26,074	–	–	1,330	1,825	1,825	1,825	1,825	1,825	1,825	1,825	1,825	1,825	1,825	1,825	1,825	1,825	1,019
Gold head grade	g/t	1.68	–	–	2.35	2.27	1.90	1.85	2.14	2.09	2.00	2.17	1.98	2.06	1.84	0.76	0.63	0.48	0.30
Silver head grade	g/t	63.37	–	–	32.71	75.00	81.84	68.93	77.38	77.09	75.27	79.04	90.68	90.40	91.35	35.80	16.95	16.13	10.14
Gold equivalent head grade	g/t	2.43	–	–	2.74	3.15	2.87	2.67	3.06	3.01	2.89	3.10	3.05	3.14	2.92	1.19	0.83	0.68	0.42
Contained Gold	koz	1,411	–	–	100	133	112	109	126	123	117	127	116	121	108	45	37	28	10
Contained Silver	koz	53,122	–	–	1,399	4,401	4,802	4,045	4,540	4,524	4,416	4,638	5,320	5,304	5,360	2,101	994	946	332
Contained Au Eq	koz	2,041	–	–	117	185	168	156	179	176	170	182	179	184	171	70	49	40	14
Gold Recovery	%	94.8%	–	–	94.8%	94.8%	94.8%	94.8%	94.8%	94.8%	94.8%	94.8%	94.8%	94.8%	94.8%	94.8%	94.8%	94.8%	94.8%
Silver Recovery	%	81.1%	–	–	81.1%	81.1%	81.1%	81.1%	81.1%	81.1%	81.1%	81.1%	81.1%	81.1%	81.1%	81.1%	81.1%	81.1%	81.1%
Gold Recovered	koz	1,338	–	–	95	126	106	103	119	116	111	121	110	115	102	42	35	27	9
Silver Recovered	koz	43,082	–	–	1,134	3,569	3,894	3,280	3,682	3,669	3,582	3,761	4,315	4,302	4,347	1,704	807	768	269
Au Eq Recovered	koz	1,936	–	–	111	176	160	148	170	167	161	173	170	175	163	66	46	38	13
Gold Payable	koz	1,337	–	–	95	126	106	103	119	116	111	120	110	115	102	42	35	27	9
Silver Payable	koz	42,651	–	–	1,123	3,533	3,855	3,247	3,645	3,632	3,546	3,724	4,272	4,259	4,303	1,687	798	760	267
Au Eq Payable	koz	1,929	–	–	111	175	159	148	170	167	160	172	169	174	162	66	46	37	13
Gold Revenue	US\$ M	4,812	–	–	342	453	380	370	428	419	400	434	396	413	368	152	125	97	34
Silver Revenue	US\$ M	2,133	–	–	56	177	193	162	182	182	177	186	214	213	215	84	40	38	13

	Units	Total/Avg	-1	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Gross Revenue	US\$ M	6,945	–	–	398	630	573	532	611	600	578	620	610	626	583	237	165	135	47
Operating costs																			
Open pit operating costs	US\$ M	269.3	–	–	23.4	20.2	19.4	19.4	23.1	19.9	23.0	20.0	19.7	23.4	23.4	19.8	14.5	–	–
Underground operating costs	US\$ M	838.7	–	–	33.1	87.0	90.6	88.7	92.1	80.1	80.1	74.9	72.2	68.7	54.1	16.7	0.5	–	–
Process plant operating costs	US\$ M	621.2	–	–	31.9	42.4	42.4	42.7	42.9	43.1	43.4	43.4	43.6	43.6	43.6	43.9	44.1	44.1	25.9
General and Administration	US\$ M	231.0	–	–	15.9	15.9	15.9	15.9	15.9	15.9	15.9	15.9	15.9	15.9	15.9	15.9	15.9	15.9	8.9
Paste Plant	US\$ M	171.1	–	–	1.9	13.7	17.4	16.0	14.8	16.8	16.4	17.3	18.1	18.2	16.4	4.0	–	–	–
Total OPEX	US\$ M	2,131.3	–	–	106.3	179.2	185.7	182.6	188.8	175.8	178.7	171.5	169.5	169.8	153.4	100.1	75.1	60.0	34.7
including PTU profit share	US\$ M	75.2	–	–	5.2	6.3	6.7	7.0	7.0	5.9	5.9	5.6	5.5	5.6	5.4	4.0	2.6	1.5	0.9
Refining & royalties																			
Treatment and refining charges	US\$ M	28.0	–	–	1.0	2.4	2.5	2.1	2.4	2.4	2.3	2.5	2.7	2.7	2.7	1.1	0.6	0.5	0.2
Royalties	US\$ M	346.1	–	–	19.9	31.4	28.6	26.5	30.4	29.9	28.8	30.9	30.4	31.2	29.0	11.8	8.2	6.7	2.4
Cash costs																			
Total OPEX excl. PTU profit share	US\$ M	2,056.0	–	–	101.1	172.9	178.9	175.6	181.8	169.9	172.8	165.9	164.0	164.2	148.0	96.2	72.5	58.5	33.9
Treatment and refining charges	US\$ M	28.0	–	–	1.0	2.4	2.5	2.1	2.4	2.4	2.3	2.5	2.7	2.7	2.7	1.1	0.6	0.5	0.2
Royalties	US\$ M	346.1	–	–	19.9	31.4	28.6	26.5	30.4	29.9	28.8	30.9	30.4	31.2	29.0	11.8	8.2	6.7	2.4
Total cash costs	US\$ M	2,430.2	–	–	122.0	206.7	209.9	204.3	214.6	202.2	203.9	199.2	197.1	198.1	179.7	109.0	81.3	65.7	36.4
Sustaining capital	US\$ M	578.6	–	–	51.7	66.9	53.2	68.8	66.3	43.1	35.0	31.7	31.2	32.4	33.2	30.8	18.1	11.7	4.3
Closure cost	US\$ M	35.0	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	35.0
Mine-site all-in sustaining costs	US\$ M	3,043.8	–	–	173.7	273.6	263.2	273.1	281.0	245.3	238.9	231.0	228.3	230.5	212.9	139.8	99.4	77.4	75.7
Total cash costs - gold equivalent ⁽¹⁾	US\$ /oz AuEq	1,259.77	–	–	1,102.12	1,181.45	1,318.84	1,381.59	1,265.40	1,212.58	1,270.80	1,157.08	1,163.80	1,139.47	1,109.56	1,658.84	1,771.13	1,754.61	2,770.47
Mine-site all-in sustaining costs - gold equivalent ⁽²⁾	US\$ /oz AuEq	1,577.87	–	–	1,584.78	1,573.98	1,664.75	1,859.31	1,668.18	1,483.63	1,502.07	1,354.58	1,362.06	1,340.02	1,330.70	2,169.85	2,229.53	2,149.49	3,343.37
Capital expenditures																			
Initial capital	US\$ M	514.8	190.4	297.9	26.5	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Open pit	US\$ M	21.3	11.9	5.5	3.9	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Underground	US\$ M	76.9	11.8	42.4	22.6	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Process plant	US\$ M	113.3	45.3	68.0	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Additional process facilities	US\$ M	30.9	12.3	18.5	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
On-site infrastructure	US\$ M	54.1	21.6	32.5	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Off site infrastructure	US\$ M	16.5	6.6	9.9	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Total project preliminaries	US\$ M	36.5	14.6	21.9	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Total project delivery	US\$ M	44.2	17.7	26.5	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Owner's cost	US\$ M	18.5	7.4	11.1	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Total provisions	US\$ M	102.6	41.1	61.6	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Sustaining capital	US\$ M	578.6	–	–	51.7	66.9	53.2	68.8	66.3	43.1	35.0	31.7	31.2	32.4	33.2	30.8	18.1	11.7	4.3
Open pit	US\$ M	75.3	–	–	3.9	5.7	5.5	5.5	8.6	5.6	6.5	5.6	5.5	6.6	6.3	5.6	4.4	–	–
Underground	US\$ M	339.6	–	–	22.6	42.4	38.6	52.1	46.7	29.0	20.4	18.6	18.5	18.6	18.9	12.1	0.9	–	–
Process plant	US\$ M	89.4	–	–	6.3	6.3	5.4	5.7	6.0	5.5	5.6	5.4	5.2	5.2	5.6	8.9	8.4	7.3	2.6
Additional process facilities	US\$ M	25.2	–	–	10.3	6.1	0.6	0.6	0.6	0.6	0.6	0.6	0.5	0.5	0.6	1.0	1.0	1.0	0.5
On-site infrastructure	US\$ M	2.0	–	–	1.8	–	–	0.3	–	–	–	–	–	–	–	–	–	–	–
Off site infrastructure	US\$ M	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Total project preliminaries	US\$ M	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Total project delivery	US\$ M	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Owner's cost	US\$ M	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Total provisions	US\$ M	47.2	–	–	6.9	6.3	3.2	4.6	4.4	2.5	1.8	1.6	1.5	1.5	1.8	3.3	3.4	3.4	1.3
Closure cost	US\$ M	35.0	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	35.0
Total capital expenditures including salvage value	US\$ M	1,128.4	190.4	297.9	78.3	66.9	53.2	68.8	66.3	43.1	35.0	31.7	31.2	32.4	33.2	30.8	18.1	11.7	39.3

Notes: Cash flows discounted to December 31, 2026. Dollar figures in Real 2026 US dollars unless otherwise noted.

1. Total cash costs include operating cost excluding PTU profit share, offsite costs and royalties.
2. Mine-site AISC include total cash costs, closure costs and sustaining capital expenditures.

22.5 Sensitivity analysis

A sensitivity analysis was conducted on the pre-tax and post-tax NPV and IRR of the project using the following variables: gold price, silver price, discount rate, operating costs, initial capex, metal recovery, and head grade. Table 22-4 and Table 22-5 show the pre-tax sensitivity analysis results for gold and silver, respectively. Likewise, Table 22-6 and Table 22-7 show the post-tax sensitivity results for gold and silver, respectively.

Table 22-4: Pre-tax sensitivity analysis (gold)

	Pre-Tax NPV Sensitivity to Discount Rate (US\$M)									Pre-Tax IRR Sensitivity to Discount Rate							
	Gold Price									Gold Price							
Discount Rate		(30%)	(20%)	(10%)	–	10%	20%	30%	Discount Rate		(30%)	(20%)	(10%)	–	10%	20%	30%
	3.00%	1,419	1,781	2,142	2,504	2,865	3,227	3,588		3.00%	30.5%	36.2%	41.5%	46.7%	51.7%	56.6%	61.4%
	5.00%	1,161	1,473	1,786	2,098	2,410	2,722	3,035		5.00%	30.5%	36.2%	41.5%	46.7%	51.7%	56.6%	61.4%
	8.00%	858	1,112	1,366	1,620	1,874	2,128	2,382		8.00%	30.5%	36.2%	41.5%	46.7%	51.7%	56.6%	61.4%
	10.00%	699	922	1,146	1,369	1,592	1,815	2,038		10.00%	30.5%	36.2%	41.5%	46.7%	51.7%	56.6%	61.4%
	12.00%	567	764	962	1,159	1,356	1,553	1,750		12.00%	30.5%	36.2%	41.5%	46.7%	51.7%	56.6%	61.4%
	Pre-Tax NPV Sensitivity to Opex (US\$M)									Pre-Tax IRR Sensitivity to Opex							
	Gold Price									Gold Price							
Opex		(30%)	(20%)	(10%)	–	10%	20%	30%	Opex		(30%)	(20%)	(10%)	–	10%	20%	30%
	(30.0%)	1,595	1,907	2,220	2,532	2,844	3,156	3,469		(30.0%)	38.0%	43.3%	48.4%	53.3%	58.2%	62.9%	67.5%
	(20.0%)	1,451	1,763	2,075	2,387	2,699	3,012	3,324		(20.0%)	35.6%	40.9%	46.1%	51.2%	56.1%	60.8%	65.5%
	(10.0%)	1,306	1,618	1,930	2,243	2,555	2,867	3,179		(10.0%)	33.1%	38.6%	43.9%	49.0%	53.9%	58.7%	63.4%
	–	1,161	1,473	1,786	2,098	2,410	2,722	3,035		–	30.5%	36.2%	41.5%	46.7%	51.7%	56.6%	61.4%
	10.0%	1,017	1,329	1,641	1,953	2,265	2,578	2,890		10.0%	27.9%	33.7%	39.2%	44.4%	49.5%	54.5%	59.3%
	20.0%	872	1,184	1,496	1,809	2,121	2,433	2,745		20.0%	25.2%	31.1%	36.8%	42.1%	47.3%	52.3%	57.2%
	30.0%	727	1,039	1,352	1,664	1,976	2,288	2,600		30.0%	22.4%	28.5%	34.3%	39.8%	45.0%	50.1%	55.1%
	Pre-Tax NPV Sensitivity to Initial Capex (US\$M)									Pre-Tax IRR Sensitivity to Initial Capex							
Initial Capex	Gold Price								Initial Capex	Gold Price							
		(30%)	(20%)	(10%)	–	10%	20%	30%			(30%)	(20%)	(10%)	–	10%	20%	30%
	(30.0%)	1,307	1,619	1,932	2,244	2,556	2,868	3,180		(30.0%)	41.6%	48.7%	55.6%	62.3%	68.7%	75.0%	81.0%
	(20.0%)	1,258	1,571	1,883	2,195	2,507	2,820	3,132		(20.0%)	37.2%	43.7%	50.0%	56.0%	61.9%	67.6%	73.1%
	(10.0%)	1,210	1,522	1,834	2,147	2,459	2,771	3,083		(10.0%)	33.6%	39.6%	45.4%	51.0%	56.4%	61.6%	66.7%
	–	1,161	1,473	1,786	2,098	2,410	2,722	3,035		–	30.5%	36.2%	41.5%	46.7%	51.7%	56.6%	61.4%
	10.0%	1,113	1,425	1,737	2,049	2,361	2,674	2,986		10.0%	27.9%	33.2%	38.3%	43.1%	47.8%	52.4%	56.9%
	20.0%	1,064	1,376	1,688	2,001	2,313	2,625	2,937		20.0%	25.7%	30.7%	35.5%	40.0%	44.5%	48.8%	53.0%
	Pre-Tax NPV Sensitivity to Recovery Mill (US\$M)									Pre-Tax IRR Sensitivity to Recovery Mill							
Recovery Mill	Gold Price								Recovery Mill	Gold Price							
		(30%)	(20%)	(10%)	–	10%	20%	30%			(30%)	(20%)	(10%)	–	10%	20%	30%
	(30.0%)	105	324	542	761	979	1,198	1,417		(30.0%)	8.1%	13.8%	18.9%	23.5%	27.9%	32.1%	36.1%
	(20.0%)	457	707	957	1,207	1,456	1,706	1,956		(20.0%)	16.8%	22.2%	27.2%	31.9%	36.5%	40.9%	45.1%
	(10.0%)	809	1,090	1,371	1,652	1,933	2,214	2,495		(10.0%)	24.0%	29.5%	34.6%	39.6%	44.4%	49.0%	53.5%
	–	1,161	1,473	1,786	2,098	2,410	2,722	3,035		–	30.5%	36.2%	41.5%	46.7%	51.7%	56.6%	61.4%
	10.0%	1,415	1,744	2,073	2,403	2,732	3,061	3,391		10.0%	34.8%	40.5%	46.0%	51.2%	56.4%	61.4%	66.2%
	20.0%	1,549	1,878	2,207	2,537	2,866	3,195	3,525		20.0%	36.8%	42.4%	47.8%	53.0%	58.0%	62.9%	67.7%
	Pre-Tax NPV Sensitivity to Head Grade (US\$M)									Pre-Tax IRR Sensitivity to Head Grade							
Head Grade	Gold Price								Head Grade	Gold Price							
		(30%)	(20%)	(10%)	–	10%	20%	30%			(30%)	(20%)	(10%)	–	10%	20%	30%
	(30.0%)	105	324	542	761	979	1,198	1,417		(30.0%)	8.1%	13.8%	18.9%	23.5%	27.9%	32.1%	36.1%
	(20.0%)	457	707	957	1,207	1,456	1,706	1,956		(20.0%)	16.8%	22.2%	27.2%	31.9%	36.5%	40.9%	45.1%
	(10.0%)	809	1,090	1,371	1,652	1,933	2,214	2,495		(10.0%)	24.0%	29.5%	34.6%	39.6%	44.4%	49.0%	53.5%
	–	1,161	1,473	1,786	2,098	2,410	2,722	3,035		–	30.5%	36.2%	41.5%	46.7%	51.7%	56.6%	61.4%
	10.0%	1,513	1,857	2,200	2,544	2,887	3,230	3,574		10.0%	36.5%	42.4%	48.0%	53.5%	58.7%	63.9%	68.9%
	20.0%	1,865	2,240	2,615	2,989	3,364	3,739	4,113		20.0%	42.2%	48.3%	54.2%	59.9%	65.4%	70.8%	76.1%
Head Grade		(30%)	(20%)	(10%)	–	10%	20%	30%	Head Grade		(30%)	(20%)	(10%)	–	10%	20%	30%
	30.0%	2,217	2,623	3,029	3,435	3,841	4,247	4,653		30.0%	47.6%	53.9%	60.1%	66.0%	71.8%	77.4%	82.9%

Table 22-5: Pre-tax sensitivity analysis (silver)

	Pre-Tax NPV Sensitivity to Discount Rate (US\$M)									Pre-Tax IRR Sensitivity to Discount Rate							
	Silver Price									Silver Price							
Discount Rate		(30%)	(20%)	(10%)	–	10%	20%	30%	Discount Rate		(30%)	(20%)	(10%)	–	10%	20%	30%
	3.00%	2,029	2,188	2,346	2,504	2,662	2,820	2,978		3.00%	41.0%	42.9%	44.9%	46.7%	48.5%	50.3%	52.1%
	5.00%	1,692	1,827	1,963	2,098	2,233	2,368	2,504		5.00%	41.0%	42.9%	44.9%	46.7%	48.5%	50.3%	52.1%
	8.00%	1,295	1,403	1,512	1,620	1,728	1,837	1,945		8.00%	41.0%	42.9%	44.9%	46.7%	48.5%	50.3%	52.1%
	10.00%	1,085	1,180	1,274	1,369	1,463	1,557	1,652		10.00%	41.0%	42.9%	44.9%	46.7%	48.5%	50.3%	52.1%
	12.00%	911	994	1,076	1,159	1,241	1,324	1,407		12.00%	41.0%	42.9%	44.9%	46.7%	48.5%	50.3%	52.1%
	Pre-Tax NPV Sensitivity to Opex (US\$M)									Pre-Tax IRR Sensitivity to Opex							
	Silver Price									Silver Price							
Opex		(30%)	(20%)	(10%)	–	10%	20%	30%	Opex		(30%)	(20%)	(10%)	–	10%	20%	30%
	(30.0%)	2,126	2,261	2,397	2,532	2,667	2,802	2,938		(30.0%)	48.0%	49.8%	51.6%	53.3%	55.0%	56.7%	58.3%
	(20.0%)	1,981	2,117	2,252	2,387	2,523	2,658	2,793		(20.0%)	45.7%	47.6%	49.4%	51.2%	52.9%	54.6%	56.3%
	(10.0%)	1,837	1,972	2,107	2,243	2,378	2,513	2,648		(10.0%)	43.4%	45.3%	47.1%	49.0%	50.7%	52.5%	54.2%
	–	1,692	1,827	1,963	2,098	2,233	2,368	2,504		–	41.0%	42.9%	44.9%	46.7%	48.5%	50.3%	52.1%
	10.0%	1,547	1,683	1,818	1,953	2,088	2,224	2,359		10.0%	38.5%	40.6%	42.5%	44.4%	46.3%	48.1%	49.9%
	20.0%	1,403	1,538	1,673	1,809	1,944	2,079	2,214		20.0%	36.1%	38.1%	40.2%	42.1%	44.0%	45.9%	47.7%
	30.0%	1,258	1,393	1,529	1,664	1,799	1,934	2,070		30.0%	33.5%	35.7%	37.8%	39.8%	41.7%	43.6%	45.5%
	Pre-Tax NPV Sensitivity to Initial Capex (US\$M)									Pre-Tax IRR Sensitivity to Initial Capex							
Initial Capex	Silver Price								Initial Capex	Silver Price							
		(30%)	(20%)	(10%)	–	10%	20%	30%			(30%)	(20%)	(10%)	–	10%	20%	30%
	(30.0%)	1,838	1,973	2,109	2,244	2,379	2,514	2,650		(30.0%)	55.3%	57.7%	60.0%	62.3%	64.5%	66.6%	68.7%
	(20.0%)	1,789	1,925	2,060	2,195	2,330	2,466	2,601		(20.0%)	49.5%	51.8%	53.9%	56.0%	58.1%	60.1%	62.1%
	(10.0%)	1,741	1,876	2,011	2,147	2,282	2,417	2,552		(10.0%)	44.9%	47.0%	49.0%	51.0%	52.9%	54.8%	56.6%
	–	1,692	1,827	1,963	2,098	2,233	2,368	2,504		–	41.0%	42.9%	44.9%	46.7%	48.5%	50.3%	52.1%
	10.0%	1,643	1,779	1,914	2,049	2,184	2,320	2,455		10.0%	37.7%	39.5%	41.4%	43.1%	44.9%	46.5%	48.2%
	20.0%	1,595	1,730	1,865	2,001	2,136	2,271	2,406		20.0%	34.8%	36.6%	38.3%	40.0%	41.7%	43.3%	44.9%
	30.0%	1,546	1,681	1,817	1,952	2,087	2,222	2,358		30.0%	32.3%	34.0%	35.7%	37.3%	38.9%	40.4%	41.9%
	Pre-Tax NPV Sensitivity to Recovery Mill (US\$M)									Pre-Tax IRR Sensitivity to Recovery Mill							
Recovery Mill	Silver Price								Recovery Mill	Silver Price							
		(30%)	(20%)	(10%)	–	10%	20%	30%			(30%)	(20%)	(10%)	–	10%	20%	30%
	(30.0%)	477	571	666	761	856	950	1,045		(30.0%)	17.8%	19.8%	21.7%	23.5%	25.3%	27.0%	28.6%
	(20.0%)	882	990	1,098	1,207	1,315	1,423	1,531		(20.0%)	26.3%	28.3%	30.1%	31.9%	33.7%	35.4%	37.0%
	(10.0%)	1,287	1,409	1,530	1,652	1,774	1,896	2,017		(10.0%)	34.0%	35.9%	37.8%	39.6%	41.4%	43.1%	44.8%
	–	1,692	1,827	1,963	2,098	2,233	2,368	2,504		–	41.0%	42.9%	44.9%	46.7%	48.5%	50.3%	52.1%
	10.0%	1,956	2,105	2,254	2,403	2,552	2,700	2,849		10.0%	45.2%	47.3%	49.3%	51.2%	53.2%	55.0%	56.8%
	20.0%	2,050	2,212	2,374	2,537	2,699	2,861	3,024		20.0%	46.5%	48.7%	50.9%	53.0%	55.0%	57.0%	58.9%
	30.0%	2,081	2,247	2,414	2,581	2,748	2,915	3,081		30.0%	47.0%	49.2%	51.4%	53.5%	55.6%	57.6%	59.6%
	Pre-Tax NPV Sensitivity to Head Grade (US\$M)									Pre-Tax IRR Sensitivity to Head Grade							
	Silver Price									Silver Price							
Head Grade		(30%)	(20%)	(10%)	–	10%	20%	30%	Head Grade		(30%)	(20%)	(10%)	–	10%	20%	30%
	(30.0%)	477	571	666	761	856	950	1045		(30.0%)	17.8%	19.8%	21.7%	23.5%	25.3%	27.0%	28.6%
	(20.0%)	882	990	1098	1207	1315	1423	1531		(20.0%)	26.3%	28.3%	30.1%	31.9%	33.7%	35.4%	37.0%
	(10.0%)	1287	1409	1530	1652	1774	1896	2017		(10.0%)	34.0%	35.9%	37.8%	39.6%	41.4%	43.1%	44.8%
	–	1692	1827	1963	2098	2233	2368	2504		–	41.0%	42.9%	44.9%	46.7%	48.5%	50.3%	52.1%
	10.0%	2097	2246	2395	2544	2692	2841	2990		10.0%	47.6%	49.6%	51.5%	53.5%	55.3%	57.2%	58.9%
	20.0%	2502	2665	2827	2989	3152	3314	3476		20.0%	53.9%	55.9%	57.9%	59.9%	61.8%	63.7%	65.5%
	30.0%	2907	3083	3259	3435	3611	3787	3962		30.0%	59.9%	62.0%	64.0%	66.0%	68.0%	69.9%	71.8%

Table 22-6: Post-tax sensitivity analysis (gold)

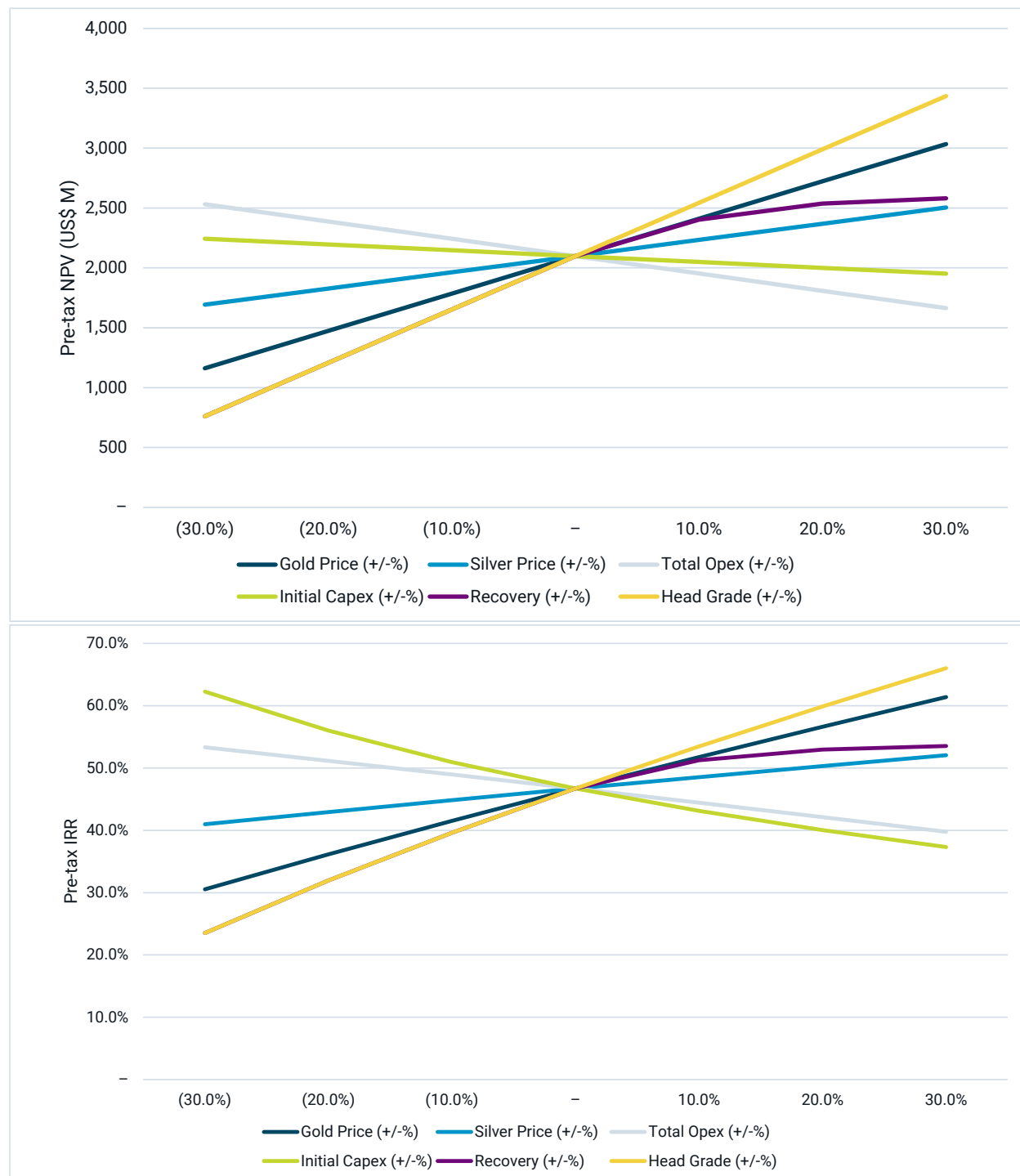
	Post-Tax NPV Sensitivity to Discount Rate (US\$M)									Post-Tax IRR Sensitivity to Discount Rate							
	Gold Price									Gold Price							
Discount Rate		(30%)	(20%)	(10%)	–	10%	20%	30%	Discount Rate		(30%)	(20%)	(10%)	–	10%	20%	30%
	3.00%	1,009	1,272	1,535	1,798	2,060	2,323	2,586		3.00%	24.4%	28.9%	33.2%	37.3%	41.3%	45.2%	48.9%
	5.00%	810	1,037	1,264	1,491	1,718	1,945	2,171		5.00%	24.4%	28.9%	33.2%	37.3%	41.3%	45.2%	48.9%
	8.00%	576	761	946	1,130	1,315	1,499	1,683		8.00%	24.4%	28.9%	33.2%	37.3%	41.3%	45.2%	48.9%
	10.00%	454	616	778	940	1,102	1,265	1,426		10.00%	24.4%	28.9%	33.2%	37.3%	41.3%	45.2%	48.9%
	12.00%	353	496	639	782	926	1,069	1,212		12.00%	24.4%	28.9%	33.2%	37.3%	41.3%	45.2%	48.9%
	Post-Tax NPV Sensitivity to Opex (US\$M)									Post-Tax IRR Sensitivity to Opex							
	Gold Price									Gold Price							
Opex		(30%)	(20%)	(10%)	–	10%	20%	30%	Opex		(30%)	(20%)	(10%)	–	10%	20%	30%
	(30.0%)	1,127	1,354	1,581	1,808	2,034	2,261	2,487		(30.0%)	30.5%	34.7%	38.7%	42.7%	46.5%	50.2%	53.9%
	(20.0%)	1,022	1,249	1,475	1,702	1,929	2,155	2,382		(20.0%)	28.5%	32.8%	36.9%	40.9%	44.8%	48.6%	52.2%
	(10.0%)	916	1,143	1,370	1,597	1,824	2,050	2,277		(10.0%)	26.5%	30.9%	35.1%	39.1%	43.1%	46.9%	50.6%
	–	810	1,037	1,264	1,491	1,718	1,945	2,171		–	24.4%	28.9%	33.2%	37.3%	41.3%	45.2%	48.9%
	10.0%	704	931	1,158	1,385	1,612	1,839	2,066		10.0%	22.3%	26.9%	31.3%	35.5%	39.5%	43.4%	47.3%
	20.0%	598	825	1,052	1,279	1,506	1,733	1,960		20.0%	20.1%	24.9%	29.3%	33.6%	37.7%	41.7%	45.6%
	30.0%	493	720	947	1,174	1,400	1,627	1,854		30.0%	17.8%	22.8%	27.3%	31.7%	35.9%	39.9%	43.8%
	Post-Tax NPV Sensitivity to Initial Capex (US\$M)									Post-Tax IRR Sensitivity to Initial Capex							
Initial Capex	Gold Price								Initial Capex	Gold Price							
		(30%)	(20%)	(10%)	–	10%	20%	30%			(30%)	(20%)	(10%)	–	10%	20%	30%
	(30.0%)	923	1,150	1,377	1,604	1,830	2,057	2,284		(30.0%)	33.2%	38.9%	44.4%	49.7%	54.8%	59.8%	64.6%
	(20.0%)	885	1,112	1,339	1,566	1,793	2,020	2,246		(20.0%)	29.7%	34.9%	39.9%	44.7%	49.4%	53.9%	58.3%
	(10.0%)	848	1,075	1,301	1,528	1,755	1,982	2,209		(10.0%)	26.9%	31.7%	36.3%	40.7%	45.0%	49.1%	53.2%
	–	810	1,037	1,264	1,491	1,718	1,945	2,171		–	24.4%	28.9%	33.2%	37.3%	41.3%	45.2%	48.9%
	10.0%	772	999	1,226	1,453	1,680	1,907	2,134		10.0%	22.4%	26.6%	30.6%	34.5%	38.2%	41.8%	45.3%
	20.0%	735	962	1,189	1,416	1,643	1,870	2,096		20.0%	20.6%	24.6%	28.3%	32.0%	35.5%	38.9%	42.2%
	30.0%	697	924	1,151	1,378	1,605	1,832	2,059		30.0%	19.0%	22.8%	26.4%	29.8%	33.2%	36.4%	39.5%
	Post-Tax NPV Sensitivity to Recovery Mill (US\$M)									Post-Tax IRR Sensitivity to Recovery Mill							
Recovery Mill	Gold Price								Recovery Mill	Gold Price							
		(30%)	(20%)	(10%)	–	10%	20%	30%			(30%)	(20%)	(10%)	–	10%	20%	30%
	(30.0%)	38	200	360	519	678	837	996		(30.0%)	6.2%	10.9%	15.0%	18.7%	22.2%	25.5%	28.7%
	(20.0%)	298	480	661	843	1,025	1,206	1,388		(20.0%)	13.3%	17.7%	21.7%	25.5%	29.1%	32.6%	35.9%
	(10.0%)	554	758	963	1,167	1,371	1,575	1,780		(10.0%)	19.2%	23.6%	27.7%	31.6%	35.4%	39.1%	42.6%
	–	810	1,037	1,264	1,491	1,718	1,945	2,171		–	24.4%	28.9%	33.2%	37.3%	41.3%	45.2%	48.9%
	10.0%	994	1,234	1,473	1,712	1,952	2,191	2,430		10.0%	27.9%	32.4%	36.8%	41.0%	45.0%	49.0%	52.8%
	20.0%	1,092	1,331	1,570	1,810	2,049	2,288	2,527		20.0%	29.6%	34.0%	38.2%	42.4%	46.4%	50.3%	54.1%
	30.0%	1,124	1,363	1,603	1,842	2,081	2,320	2,559		30.0%	30.1%	34.5%	38.7%	42.8%	46.8%	50.7%	54.5%
	Post-Tax NPV Sensitivity to Head Grade (US\$M)									Post-Tax IRR Sensitivity to Head Grade							
	Gold Price									Gold Price							
Head Grade		(30%)	(20%)	(10%)	–	10%	20%	30%	Head Grade		(30%)	(20%)	(10%)	–	10%	20%	30%
	(30.0%)	38	200	360	519	678	837	996		(30.0%)	6.2%	10.9%	15.0%	18.7%	22.2%	25.5%	28.7%
	(20.0%)	298	480	661	843	1,025	1,206	1,388		(20.0%)	13.3%	17.7%	21.7%	25.5%	29.1%	32.6%	35.9%
	(10.0%)	554	758	963	1,167	1,371	1,575	1,780		(10.0%)	19.2%	23.6%	27.7%	31.6%	35.4%	39.1%	42.6%
	–	810	1,037	1,264	1,491	1,718	1,945	2,171		–	24.4%	28.9%	33.2%	37.3%	41.3%	45.2%	48.9%
	10.0%	1,066	1,316	1,565	1,815	2,064	2,313	2,563		10.0%	29.3%	33.9%	38.4%	42.7%	46.9%	51.0%	54.9%
	20.0%	1,322	1,594	1,866	2,138	2,410	2,682	2,954		20.0%	33.8%	38.7%	43.3%	47.8%	52.2%	56.5%	60.7%
	30.0%	1,578	1,873	2,167	2,462	2,756	3,051	3,345		30.0%	38.1%	43.2%	48.0%	52.8%	57.4%	61.9%	66.2%

Table 22-7: Post-tax sensitivity analysis (silver)

	Post-Tax NPV Sensitivity to Discount Rate (US\$M)									Post-Tax IRR Sensitivity to Discount Rate							
	Silver Price									Silver Price							
Discount Rate		(30%)	(20%)	(10%)	–	10%	20%	30%	Discount Rate		(30%)	(20%)	(10%)	–	10%	20%	30%
	3.00%	1,453	1,568	1,683	1,798	1,912	2,027	2,142		3.00%	32.6%	34.2%	35.8%	37.3%	38.8%	40.3%	41.7%
	5.00%	1,196	1,294	1,393	1,491	1,589	1,687	1,786		5.00%	32.6%	34.2%	35.8%	37.3%	38.8%	40.3%	41.7%
	8.00%	894	973	1,051	1,130	1,209	1,288	1,366		8.00%	32.6%	34.2%	35.8%	37.3%	38.8%	40.3%	41.7%
	10.00%	735	803	872	940	1,009	1,078	1,146		10.00%	32.6%	34.2%	35.8%	37.3%	38.8%	40.3%	41.7%
	12.00%	602	662	722	782	842	902	962		12.00%	32.6%	34.2%	35.8%	37.3%	38.8%	40.3%	41.7%
	Post-Tax NPV Sensitivity to Opex (US\$M)									Post-Tax IRR Sensitivity to Opex							
	Silver Price									Silver Price							
Opex		(30%)	(20%)	(10%)	–	10%	20%	30%	Opex		(30%)	(20%)	(10%)	–	10%	20%	30%
	(30.0%)	1,513	1,611	1,710	1,808	1,906	2,004	2,102		(30.0%)	38.3%	39.8%	41.2%	42.7%	44.1%	45.4%	46.8%
	(20.0%)	1,408	1,506	1,604	1,702	1,801	1,899	1,997		(20.0%)	36.4%	37.9%	39.4%	40.9%	42.3%	43.7%	45.1%
	(10.0%)	1,302	1,400	1,498	1,597	1,695	1,793	1,891		(10.0%)	34.5%	36.1%	37.6%	39.1%	40.6%	42.0%	43.4%
	–	1,196	1,294	1,393	1,491	1,589	1,687	1,786		–	32.6%	34.2%	35.8%	37.3%	38.8%	40.3%	41.7%
	10.0%	1,090	1,189	1,287	1,385	1,483	1,582	1,680		10.0%	30.6%	32.3%	33.9%	35.5%	37.0%	38.5%	40.0%
	20.0%	984	1,083	1,181	1,279	1,378	1,476	1,574		20.0%	28.6%	30.3%	32.0%	33.6%	35.2%	36.7%	38.2%
	30.0%	879	977	1,075	1,174	1,272	1,370	1,468		30.0%	26.6%	28.3%	30.0%	31.7%	33.3%	34.9%	36.4%
	Post-Tax NPV Sensitivity to Initial Capex (US\$M)									Post-Tax IRR Sensitivity to Initial Capex							
Initial Capex	Silver Price								Initial Capex	Silver Price							
		(30%)	(20%)	(10%)	–	10%	20%	30%			(30%)	(20%)	(10%)	–	10%	20%	30%
	(30.0%)	1,309	1,407	1,505	1,604	1,702	1,800	1,898		(30.0%)	43.9%	45.9%	47.8%	49.7%	51.5%	53.3%	55.1%
	(20.0%)	1,271	1,369	1,468	1,566	1,664	1,763	1,861		(20.0%)	39.4%	41.2%	43.0%	44.7%	46.4%	48.1%	49.7%
	(10.0%)	1,234	1,332	1,430	1,528	1,627	1,725	1,823		(10.0%)	35.7%	37.4%	39.1%	40.7%	42.3%	43.8%	45.3%
	–	1,196	1,294	1,393	1,491	1,589	1,687	1,786		–	32.6%	34.2%	35.8%	37.3%	38.8%	40.3%	41.7%
	10.0%	1,158	1,257	1,355	1,453	1,552	1,650	1,748		10.0%	30.0%	31.5%	33.0%	34.5%	35.9%	37.3%	38.6%
	20.0%	1,121	1,219	1,317	1,416	1,514	1,612	1,711		20.0%	27.7%	29.2%	30.6%	32.0%	33.3%	34.7%	35.9%
	30.0%	1,083	1,182	1,280	1,378	1,476	1,575	1,673		30.0%	25.8%	27.2%	28.5%	29.8%	31.1%	32.4%	33.6%
	Post-Tax NPV Sensitivity to Recovery Mill (US\$M)									Post-Tax IRR Sensitivity to Recovery Mill							
Recovery Mill	Silver Price								Recovery Mill	Silver Price							
		(30%)	(20%)	(10%)	–	10%	20%	30%			(30%)	(20%)	(10%)	–	10%	20%	30%
	(30.0%)	313	381	450	519	588	657	725		(30.0%)	14.0%	15.7%	17.2%	18.7%	20.1%	21.5%	22.9%
	(20.0%)	607	686	764	843	922	1,000	1,079		(20.0%)	20.9%	22.5%	24.0%	25.5%	26.9%	28.3%	29.6%
	(10.0%)	902	990	1,078	1,167	1,255	1,344	1,432		(10.0%)	27.0%	28.6%	30.1%	31.6%	33.1%	34.5%	35.9%
	–	1,196	1,294	1,393	1,491	1,589	1,687	1,786		–	32.6%	34.2%	35.8%	37.3%	38.8%	40.3%	41.7%
	10.0%	1,388	1,496	1,604	1,712	1,821	1,929	2,037		10.0%	36.0%	37.7%	39.4%	41.0%	42.5%	44.1%	45.6%
	20.0%	1,456	1,574	1,692	1,810	1,928	2,046	2,163		20.0%	37.1%	38.9%	40.7%	42.4%	44.0%	45.7%	47.3%
	30.0%	1,478	1,600	1,721	1,842	1,963	2,084	2,205		30.0%	37.4%	39.3%	41.1%	42.8%	44.5%	46.2%	47.8%
	Post-Tax NPV Sensitivity to Head Grade (US\$M)									Post-Tax IRR Sensitivity to Head Grade							
	Silver Price									Silver Price							
Head Grade		(30%)	(20%)	(10%)	–	10%	20%	30%	Head Grade		(30%)	(20%)	(10%)	–	10%	20%	30%
	(30.0%)	313	381	450	519	588	657	725		(30.0%)	14.0%	15.7%	17.2%	18.7%	20.1%	21.5%	22.9%
	(20.0%)	607	686	764	843	922	1,000	1,079		(20.0%)	20.9%	22.5%	24.0%	25.5%	26.9%	28.3%	29.6%
	(10.0%)	902	990	1,078	1,167	1,255	1,344	1,432		(10.0%)	27.0%	28.6%	30.1%	31.6%	33.1%	34.5%	35.9%
	–	1,196	1,294	1,393	1,491	1,589	1,687	1,786		–	32.6%	34.2%	35.8%	37.3%	38.8%	40.3%	41.7%
	10.0%	1,490	1,599	1,707	1,815	1,923	2,031	2,139		10.0%	37.9%	39.5%	41.1%	42.7%	44.2%	45.7%	47.2%
	20.0%	1,785	1,903	2,021	2,138	2,256	2,374	2,492		20.0%	42.9%	44.6%	46.2%	47.8%	49.4%	51.0%	52.5%
	30.0%	2,079	2,207	2,334	2,462	2,589	2,717	2,844		30.0%	47.7%	49.4%	51.1%	52.8%	54.4%	56.0%	57.6%

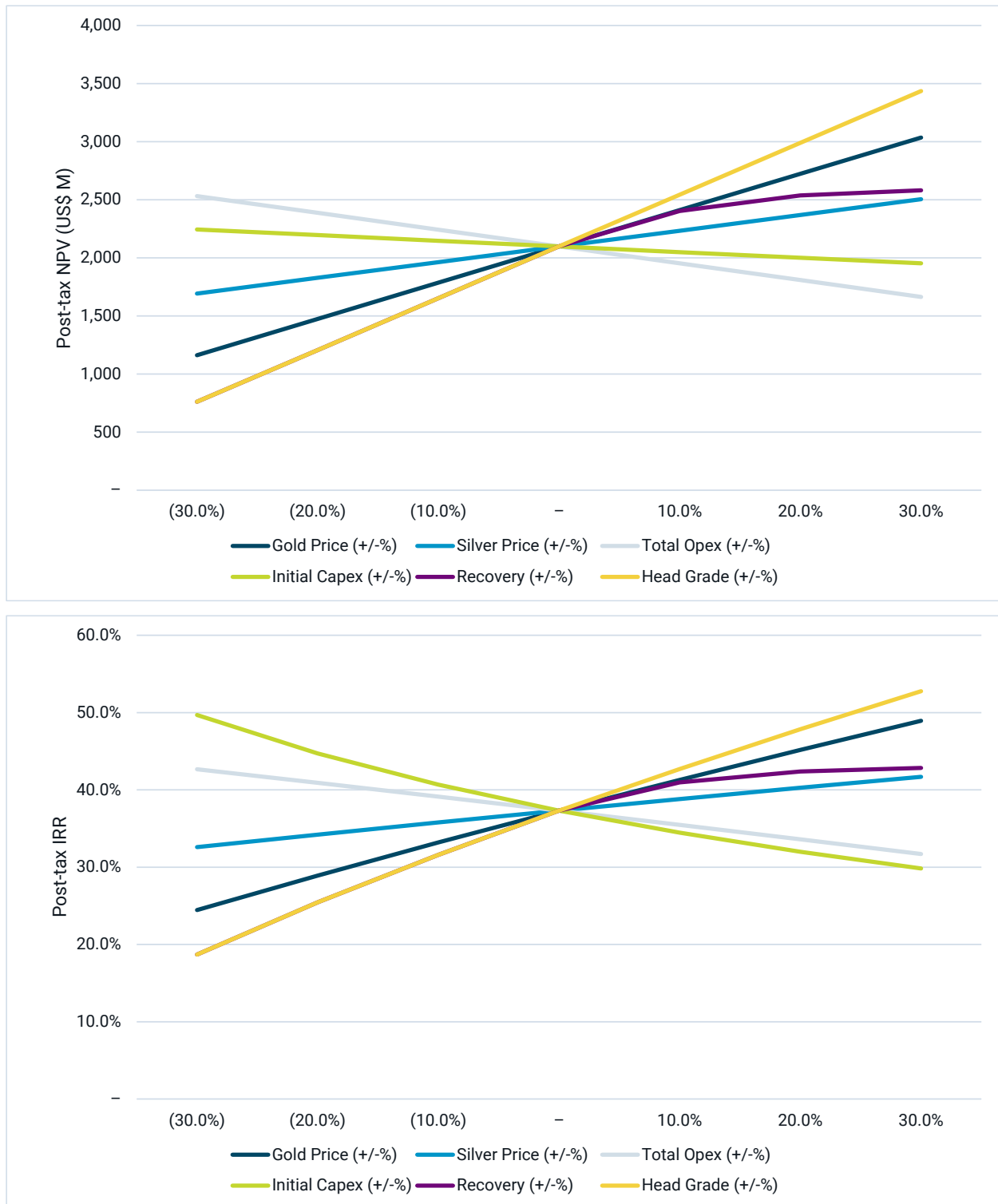
As presented Figure 22-2 and Figure 22-3, the sensitivity analysis showed that the project is most sensitive to head grade, gold price, and recovery.

Figure 22-2: Post-tax sensitivity summary



Note: Head grade and recovery overlap. Source: Ausenco, 2026

Figure 22-3: Post-tax sensitivity summary



Note: Head grade and recovery overlap. Source: Ausenco, 2026

23 Adjacent properties

Torex is not aware of any material adjacent properties relevant to this report.

24 Other relevant data and information

This section is not relevant to this report.

25 Interpretation and conclusions

25.1 Introduction

The QPs note the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for this report.

25.2 Mineral tenure, surface rights, royalties and agreements

Mineral tenure for the Los Reyes Project is well established and in good standing. The concessions, comprising a contiguous land package, are legally registered and supported by a title opinion completed in 2024 confirming their validity. Ownership has been consolidated through a series of transactions resulting in Minera Amari, an indirect wholly owned subsidiary of Torex, holding a 100% interest in the mining concessions. While the chain of title reflects multiple historical agreements and owners, no outstanding ownership disputes or third-party ownership claims have been identified in the available documentation.

Surface access to the Property is secured through agreements with Ejido La Tasajera, which provide for long-term use of land required for exploration, development, and potential mining operations. These agreements define staged payments linked to project advancement and are considered to provide sufficient access rights to support ongoing and future activities. In addition, the agreements are transferable, allowing continuity of access under current ownership.

The Property is subject to a number of net smelter return royalties that vary by concession and reflect the historical assembly of the land package. Certain royalties include repurchase provisions that may allow future reduction of royalty burdens. Historical agreements have also resulted in the removal of certain legacy encumbrances, including back-in rights previously associated with earlier ownership. Remaining encumbrances are limited to royalty obligations and surface use agreements, which are considered typical for a project of this nature and do not, based on available information, materially impair advancement of the Project.

25.3 Geology and mineralisation

The available geological and mineralisation information indicates a large, structurally controlled precious-metals system with multiple mineralised trends, demonstrated continuity, and favourable host relationships for additional exploration success. The combination of district-scale strike extent, variable but locally substantial widths, strong silicification and brecciation, and the association with felsic intrusions supports the view that the Property remains highly prospective for further delineation of gold-silver mineralisation within the principal corridors and adjacent targets.

The Los Reyes Property is underlain by a well-preserved volcanic and intrusive succession within the Sierra Madre Occidental, where lithological contrasts, felsic intrusions, and repeated structural reactivation have combined to localize a district-scale low-sulphidation epithermal gold-silver system. Mineralisation is concentrated within several laterally extensive, northwest- to west-northwest-trending corridors that exhibit strong continuity, variable but locally significant widths, and multiple styles of quartz-vein and breccia-hosted mineralisation. Together, the geological setting, alteration patterns, structural controls, and distribution of known mineralised zones indicate a robust mineralizing system with clear potential for continued delineation and expansion of mineralisation across both the principal trends and adjacent satellite targets.

25.4 Exploration and drilling

In the opinion of the Qualified Person, the exploration information reviewed for the Property indicate a coherent and district-scale low-sulphidation epithermal Au-Ag system with multiple mineralised trends and targets supported by geological mapping, surface geochemistry, geophysics, spectral mineralogy, magnetic susceptibility, and remote sensing. Exploration completed by Prime and previous operators has materially improved the understanding of structural controls, alteration patterns, and target continuity. The integration of LiDAR, mapping, geochemical, and geophysical datasets into the geological interpretation provides a reasonable basis for continued targeting and supports the conclusions presented in this report. Based on the information reviewed, the Qualified Person considers the exploration results to be relevant, internally consistent, and sufficiently reliable for the purposes of this technical report.

25.5 Metallurgical testwork

The metallurgical test work completed to date is reasonable to support this level of study. Composites and variability samples covering the majority of the deposits have been tested, and metallurgical performance was confirmed through testing at two different laboratories. Leach performance results are generally consistent within each resource zone. The range of primary grind sizes tested was sufficient to determine an appropriate leach feed sizing for the project. Sodium cyanide concentrations and leach residence times varied in the test programs, providing a basis to select conditions that achieved an economic balance between silver recoveries and operating costs. Gold leaching appeared to be complete within 48 hours, and these kinetics did not affect the leach residence time selection.

Comminution testing to date is somewhat limited in comparison to the quantity of leach tested samples, however the 8 samples tested provide a suitable dataset for comminution circuit design.

25.6 Mineral resource estimates

Based on the highly prospective geology, size and continuity of the mineralized structural corridors identified to date, including surface mapping and drilling results by previous operators, Property mineralization may be much more extensive than currently reported.

The Project contains Indicated and Inferred Mineral Resources that are associated with well-defined mineralized trends. All deposits are generally open along strike and at depth, and the Property has the potential for the delineation of additional Mineral Resources within the three main trends and that further additional exploration is warranted on high-priority targets identified from detailed mapping and surface sampling within the Property.

Under the supervision of John Sims of Sims Resources LLC., Independent QP, and based on NI 43 101 definitions and standards, the Project contains the following MRE, as of April 25, 2026.

This MRE is not a Mineral Reserve and following NI 43-101 does not have demonstrated economic viability.

25.7 Mining methods

Long-hole open stoping with paste backfill is a technically appropriate and well-established method for the four underground deposits, and conventional open-pit mining is appropriate for the near-surface mineralisation. Method standardisation — a single underground method and a single haulage-truck class across both operations — is a sound basis at PEA level and simplifies the fleet, the workforce skill matrix,

and the ventilation and services design. The mine plan, schedule and cost estimates demonstrate that the Project supports a positive preliminary economic outcome at the assumed throughput and cut-off grades.

25.8 Recovery methods

The selected flowsheet aligns with conventional practices in the industry. Comminution, whole-ore leaching, precious metal precipitation, retort and smelting of payable metals, cyanide detoxification, and handling of tailings are achieved through conventional processes that are commonly used in the industry for similar projects with no significant elements of technological innovation. Previous studies, coupled with available test work results and financial evaluations, were used to develop the resulting flowsheet.

25.9 Infrastructure

The Los Reyes Project includes on-site infrastructure such as civil, structural and earthworks development, site facilities and buildings, on-site roads, water management systems, and site electrical power facilities. Off-site infrastructure includes site access roads, raw water supply, and power supply. The site infrastructure will include:

- mine facilities, including truck shop & wash bay and mine office, explosive storage facilities, warehouse, paste backfill facilities, waste rock storage and low-grade stockpiles, ROM (run-of-mine) pad, pits and Underground portals
- Common facilities including an entrance/exit gatehouse, a security office, overall site administration building, potable water systems, compressed air, power distribution facilities, fuel tank storage, communications and sanitation systems,
- Process facilities housed in the processing plant, including crushing, grinding and classification, leaching, counter-current decantation, Merrill-Crowe circuit, retorting and smelting, cyanide detoxification, tailings thickening and filtering, assay laboratory and process plant workshop and warehouse,
- Other infrastructure including the TSF.
- There are two key accesses to the site. The main site entry is in the north-west of the site and is connected to the township of Cosalá by a 36km main road. It is proposed to upgrade this road in the initial stages of the project. This access is close to the proposed process plant and site camp facilities. The other site access is on the eastern side of the project and is connected by an existing public road that supports the nearby community of Guadalupe. This access is close to the proposed paste plant and Guadalupe East underground portal.
- Permanent electrical power is provided by a 138kV transmission line connecting into the grid near Cosalá.
- Raw water will be sourced from wells drilled to collect groundwater.
- The overall site layout was developed using the following criteria and factors:
 - The processing facilities described above must be located to minimize mine haul costs, initial capital costs (earthworks, roads, power, water), environmental and community impacts.
 - The TSF should be located as close to the process plant as possible, taking advantage of the sloped natural terrain, down-gradient from tailings filtering facilities, and surface runoff managed accordingly.
 - The arrangement of the administration buildings, mine workshops, processing plant, and additional offices should be optimized for foot and vehicle traffic.

- The project is located at a high altitude with very steep terrain. Consideration has been given to these conditions when locating buildings and access roads.

25.10 Market studies

The Company has not conducted a formal market study for production from the Project. The market for gold and silver doré bars is very liquid, and the product can easily be sold on a spot basis at market prices. The Company currently has refining relationships for gold doré that can be leveraged for the sale of the gold and silver doré.

Project economics were estimated based on long-term metal prices of US\$3,600/oz Au, and US\$50/oz Ag, which was based on consensus forecasts from various financial institutions.

The Company does not currently have contracts for the gold and silver doré production at Los Reyes. The Company will leverage its experience and existing relationships in the gold doré market to enter into future contracts for the sale of gold and silver doré. We expect the terms in the contract to be market-based terms.

25.11 Environmental, permitting and social considerations

The environmental setting of the Los Reyes Project is understood for a PEA. The project area supports a diverse ecosystem and includes nearby communities as part of its cultural landscape. Existing environmental liabilities are limited to minor historical mining features (such as old adits and small waste rock piles), and no acid rock drainage has been reported from past workings. Importantly, the site lies outside any protected or ecologically sensitive areas. Some prior disturbance from historic mining and agricultural activities (including local farming and grazing) is evident, but the author is unaware of any environmental constraints at this time.

The Project is in compliance with its current exploration-stage requirements, including the submission and approval of an Environmental Impact Statement (MIA) for exploration activities. To advance into development and construction, the Project will need to obtain additional permits and authorizations under Mexican regulations – notably including environmental impact approvals, land-use change permits, water use and discharge permits – as administered by the federal agencies (e.g. SEMARNAT for environment and land use, PROFEPA for compliance, and CONAGUA for water rights) and relevant state or local authorities. Completion of comprehensive baseline environmental studies is recognized as a key requirement for future permit applications and project de-risking.

Social aspects of the Project are underpinned by long-standing agreements with local ejidos, which provide secure land access and support for ongoing exploration as well as potential future development. The Project area covers lands of three ejidos (with Ejido La Tasajera encompassing the majority). These agreements form the foundation of the Project's social license to operate, outlining provisions for shared land use, and commitments to local benefits. The presence of nearby rural communities and the co-operation with ejido stakeholders highlight the importance of continued community engagement, particularly as the Project transitions from exploration to more advanced development phases.

Closure and reclamation planning is currently addressed at a conceptual level appropriate for a PEA. The preliminary closure strategy focuses on returning the site to a stable, non-polluting condition that is compatible with the surrounding environment. This will involve removal of project infrastructure, recontouring of disturbed areas to promote natural drainage, stabilization of mine landforms (such as WRSFs and TSF), and progressive revegetation. Overall, no material environmental, permitting, or social obstacles have been identified at this stage of evaluation.

25.12 Capital and operating cost estimates

25.12.1 Capital cost estimate

The capital cost estimates presented in this PEA presents substantiated costs to develop the preliminary economics for Los Reyes. The estimates consist of costs developed by multiple parties. AGP Mining developed the estimates for open pit and underground mining operation. The process plant, paste plant construction and on-site and offsite infrastructure by Ausenco. The tailings storage and surface water management facilities were developed by Knight Piesold.

The capital cost estimate conforms to Class 5 guidelines as dictated by the Association of the Advancement of Cost Engineering International (AACE International), with an estimated accuracy of +50%/-30%. The capital cost was developed in Q2 2026 dollars.

The total initial capital cost for the Los Reyes project is US\$514.8 million, with a LOM sustaining capital cost of US\$578.6 million, and a closure cost of US\$35.0 million. The capital costs are summarized in Section 21.

25.12.2 Operating cost estimate

The operating cost estimate was developed in Q2 2026 using data from projects, studies and previous operations from internal databases. The operating cost estimate is approximately +50%/-30% accurate, matching that of the capital cost estimate. The estimate covers the processing, tailings handling, maintenance, power and (G&A) activities. Section 21 includes a summary of the operating expenses.

The LOM average unit process operating cost is US\$23.39/t processed material and G&A cost is US\$8.43/t processed.

25.13 Economic analysis

The pre-tax NPV discounted at 5% is \$2,098 million; the IRR is 46.72%; and the payback period is 1.4 years. On a post-tax basis, the NPV discounted at 5% is \$1,491 million; the IRR is 37.32%; and payback period is 1.9 years.

The preliminary economic assessment is preliminary in nature, that it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

25.14 Risks and opportunities

25.14.1 Risks

25.14.1.1 Exploration and drilling

Exploration risks identified from the data reviewed primarily relate to the current level of advancement and the remaining uncertainty in target definition. Several mapped trends and structures, including the Orito trend and structures north of San Miguel East and Guadalupe East, remain at an early stage and require additional exploration to confirm continuity, geometry, and mineralised extent. Confidence in vertical geochemical and geological vectors in the Z-T and Central zones is also limited by the relative lack of deep drilling, such that the expected increase in Cu, Pb, and Zn with depth typical of low-sulphidation epithermal systems has not yet been confirmed. In addition, some datasets remain interpretive or incomplete, including

remote sensing work and geophysical results where VLF-EM effectiveness was partly constrained by interference from power transmission lines.

25.14.1.2 Metallurgical testing

The following risks have been identified for the metallurgical testwork:

- The quantity of samples subjected to comminution testing was limited and may not accurately determine the range of comminution properties in the deposits.
- No metallurgical testing was completed on samples from two of the deposits in the resource, Mariposa and Fresnillo. Metallurgical performance for these deposits is estimated from the nearby deposits of Zapote and Noche Bueno, respectively.
- Other deposits had very limited testing (Guadalupe West, San Miguel, Noche Buena, and Tahonitas) and the measured metallurgical performance may not be representative of each deposit.

25.14.1.3 Mineral resource estimate

This MRE could be influenced by changes in any of the contributing inputs and macroeconomic assumptions used to generate the estimate, including, but not limited to:

1. The geologic interpretation, modelling of geologic attributes such as lithology, faults, and mineralization controls, and the resulting MREs were prepared using the most accurate information available at the time this Technical Report was completed. However, additional drilling, data collection, and analysis may require revisions to wireframes, interpolation methodologies, density modelling, or other attributes which may impact future resource MREs.
2. Commodity price changes and capital and operating cost estimates could impact revenue and cost inputs used in the MRE, and overall economic interpretation of the viability of Project study and development.
3. The accuracy of historical mine workings cannot be verified in cases where collapse or other unsafe conditions result in limited access to the excavations. Construction of the current depletion solids therefore relied on a combination of LiDAR-surveyed workings where available, voids encountered in surveyed drillholes, and historical production maps. While the work was completed to the highest level of accuracy possible with the current dataset, the position of some underground workings may be inaccurate. Other historical mine workings may also be present which have not been documented.
4. Future technical studies, including geotechnical and metallurgical, could result in revisions to slope angle and process recovery assumptions.
5. Changes to the regulatory or permitting environment and laws in Mexico that affect operating or capital costs, taxes, royalties, anticipated environmental compliance regulations, closure costs and obligations among others.

25.14.1.4 Mining methods

- Inferred Mineral Resources: approximately 37% of the scheduled mining inventory is Inferred and is concentrated in later life-of-mine years; the economic outcome depends on conversion of this material through in-fill drilling
- Geotechnical / Estaca fault: the rock mass is poor to fair (RMR89 $\approx$ 25–60), and the Estaca fault running through the Guadalupe main zone introduces overbreak and dilution uncertainty. Slope dimensions, dilution and recovery are applied as deposit-wide PEA-level averages ($\approx$ 14% underground dilution, 95% stope/100% development recovery) and require zone-specific refinement.

- Throughput and cut-off: the 5,000 t/d throughput and the 1.50 g/t AuEq underground cut-off were selected on a mining-cost basis; the integrated ranking may shift once process and infrastructure costs, and the resource model are refined.
- Paste backfill and historical voids: paste-filling and mining through the historical voids in Guadalupe East carries execution and scheduling risk.

25.14.1.5 Recovery methods

The flowsheet was designed with limited metallurgical testing available, including but not limited to the following:

- Crushing and grinding equipment was selected based on the available comminution test data and may be undersized if actual values are higher than design
- The precious metals recovery flowsheet was selected based on the available test data and may not be optimized
- Solid liquid separation data were not available at the time of the study. Thickeners (including the counter-current decantation (CCD) thickeners and clarification thickener) and filters may be undersized if actual settling and filtration rates are lower than design
- Cyanide detoxification data were not available at the time of the study. Reagent addition systems and consumption rates may be undersized if actual rates are higher than design
- No oxygen uptake test results were available; oxygen consumption for both whole-ore leaching and cyanide detoxification is estimated based on benchmark data for similar material. The oxygen plant required for the process plant may be undersized or oversized if actual oxygen demand is higher or lower than the design, respectively
- No paste mixing and handling testwork was available at the time of study. Circuit design is based on benchmarks and the author's best estimate. Reagent addition systems and consumption rates may be undersized if actual rates are higher than design. Paste pumping and reticulation underground may require additional or larger pump equipment if paste material handling parameters vary from the design.

25.14.1.6 Infrastructure

The current design allows for drawing water from a well system to be located approximately 3km south-west of the process plant with overland pipelines and associated pumping facilities. The external make-up water sources for the Project have not yet been fully identified, secured, or permitted. It is anticipated that the make-up water requirements will be supplied by local groundwater aquifers or surface water sources.

Power supply is planned to be drawn from a hydro dam to the north-west of the site. The design allows for the construction of a power line from the grid near the town of Cosalá to the site. This will require permitting and could result in potential delays resulting from dealing with the CFE. The suitability of connecting to the power grid at this approximate location requires verification.

The current project costing has allowed for a construction camp on site. During operation it is expected that there will be a split in housing of personnel between a camp on site and accommodation in the nearby town of Cosalá. More detailed assessments of the accommodation model for the project should be carried out in future phases of the project.

25.14.1.7 Environmental, permitting, social and community

Closure planning is at a conceptual level and is subject to uncertainty due to limited technical definition. Key risks include incomplete geochemical characterization of waste rock and tailings, which may affect

predictions of water quality and the need for long-term mitigation. Uncertainty also remains regarding pit lake formation, closure cover design, and the long-term geotechnical stability of waste rock and tailings storage facilities.

An environmental baseline study completed in the future may impact the overall scope of the project, and impact costs related to

Additional risks include evolving regulatory requirements, the need for future permitting approvals, and the absence of finalized closure land use objectives with local stakeholders and ejidos. These factors may influence both closure design and environmental obligations, as well as project timelines and costs.

25.14.1.8 Capital and operating cost estimates

Benchmarking and in-house data was used to develop estimates for the capital and operating costs of the project based on available data. If additional data is available in future phases, the costs for the project may change as well.

The risks for the process plant and paste plant capital costs are that equipment may be oversized or undersized. If subsequent test work indicates that equipment sizing must increase, the capital costs of the processing plant will likely increase proportionally as well. On the other hand, if test work shows that equipment sizing can decrease, the costs of the process and paste plants will decrease as well.

Similarly, if future metallurgical test work indicates that reagent and consumable consumption rates increase or decrease, the operating costs for the processing plant will increase or decrease, respectively. If equipment sizing or model details change, power consumption for the processing plant or paste plant may also change.

For the mining capital estimate, lease costs and terms for mining equipment were estimated based on in-house data and previous agreements. These factors may change at subsequent stages of study and impact the overall capital cost of the project. Additionally, unstable ground conditions may impact development costs over the course of the life of mine. Geotechnical data will need to be gathered and analyzed during future study work and incorporated into the mine design to accurately estimate development costs.

Risks associated with the TSF and water management design relate to the uncertainties in the geotechnical and hydrogeological foundation conditions of the proposed infrastructure. Site investigation programs at future study phases may result in modifications to the current design of the TSF and water management structures, resulting in changes to the capital cost estimate. The water management strategy was developed based on the current understanding of the project climate, make-up water requirements, and estimated surplus water. Refinements to these inputs at future study phases may present changes to the water management strategy and project water balance, resulting in changes to the capital and operating cost estimates.

25.14.1.9 Financial model

The ability of mining companies to fund the advancement of their projects through exploration and development is influenced by commodity prices. Variations in the commodity prices may lead to reduced or elevated revenues compared to those projected in this study.

U.S. tariffs on Mexican exports and the potential for Mexico to impose tariffs on U.S. goods could significantly impact the project. Proactive measures including cost management, pricing adjustments, and market diversification are recommended.

Changes in government can impact policies relating to taxes which can also affect the project both adversely and beneficially.

25.14.2 Opportunities

25.14.2.1 Exploration and drilling

Exploration opportunities identified from the data reviewed are significant and relate to the demonstrated district-scale potential of the Property and the growing effectiveness of integrated targeting. Geological mapping, surface geochemistry, geophysics, spectral mineralogy, magnetic susceptibility, and remote sensing have collectively identified multiple high-priority target areas, including Tahonitas, Las Primas, Fresnillo, Zapote 300, Orito, and structures north of San Miguel East and Guadalupe East, several of which remain open or only partially tested. Surface sampling and geochemical studies have confirmed known mineralised trends and highlighted additional targets for follow-up, while geophysical and spectral datasets have improved the understanding of structural controls, alteration zonation, and vectors toward mineralisation. The availability of LiDAR coverage, detailed mapping, extensive rock and soil geochemistry, and expanding mineralogical datasets provides a strong technical basis to refine drill targeting, test extensions to known trends, and evaluate new zones with potential to add mineralised inventory.

25.14.2.2 Metallurgical testing

The following opportunities were identified for the metallurgical testwork:

- The metallurgical testing did not specifically evaluate the leach durations and NaCN dosages proposed in the design. Extractions on several samples in the recovery model database could be higher if tested at the design 72-hour residence time. As well, NaCN consumptions may be lower than the estimated values.
- There may be opportunities to reduce the NaCN dosages applied to the leach, thereby reducing the end of leach concentration that will report to cyanide destruction. Based on confirmation from further metallurgical testwork there is the potential this reduction could provide operational cost savings on both NaCN and SMBS consumptions.
- Froth flotation in conjunction with cyanide leaching may be investigated as means to reduce grinding energy. A flotation concentrate recovered at a coarser primary grind size could be reground prior to leaching, providing a targeted application of grinding energy.
- A limited amount of testing was completed without lead nitrate; it may be that this reagent could be eliminated from the circuit with sufficient pre-aeration of the leach slurry.
- Gravity concentration testing was somewhat limited, additional testing may demonstrate that it provides a net benefit to precious metal recoveries.
- Column leach testing on appropriate low-grade samples is limited. There may be an opportunity to demonstrate heap leaching as an economical means to process low grade materials.

25.14.2.3 Mineral resource estimate

Based on the highly prospective geology, size and continuity of the mineralized structural corridors identified to date, including surface mapping and drilling results by previous operators, Property mineralization may be much more extensive than currently reported.

25.14.2.4 Mining methods

- Resource conversion: in-fill drilling to convert Inferred material, particularly in the later-year inventory, would de-risk the schedule.

- Estaca-fault dilution: the structural-control analysis indicates that much of the modelled Guadalupe overbreak is mineralised material relocated from adjacent stopes rather than true waste; a zone-specific dilution/recovery model could recover apparent metal currently treated as dilution.
- Sub-level caving: SLC is retained as an alternative mining method for Guadalupe East and could improve productivity and reduce paste demand if the detailed geotechnical and void-condition data support it.
- Commodity-price leverage: the inventory is grade-driven; the stope envelope is relatively insensitive to price, which limits downside.

25.14.2.5 Recovery methods

There may be opportunities to optimize the process flowsheet once additional metallurgical test work is completed. Future studies should include the following engineering trade-off studies to confirm the following:

- Validation of primary grind size and comminution circuit selection
- Consideration of inclusion of gravity recovery prior to whole-ore leaching
- Consideration of using flotation in conjunction with cyanide leaching to reduce grinding requirements
- Validation of leaching retention time
- Validation of number of counter-current decantation thickeners
- Evaluation of detoxification conditions to determine if performance is significantly reduced at higher pulp density
- Evaluation of detoxification residence time
- Evaluation of tailings thickening performance and optimisation of flocculant addition
- Evaluation of tailings filtration performance
- Trade-off study of delivery method of tailings to the paste plant
- Determination of paste binder requirements.

25.14.2.6 Infrastructure

Due to additional land, previously not part of the project, becoming available there is an opportunity to potentially locate more site infrastructure in the north-west in an area that is closer to external facilities and could result in reduced earthworks. Also, consideration of the plant location to reduce tailings pumping equipment and operating costs.

25.14.2.7 Environmental, permitting, social and community

Opportunities exist to optimize closure outcomes as the Project advances in engineering. Integration of closure considerations into mine design and planning may reduce material rehandling, earthworks, and overall closure costs. Improved geochemical understanding may allow optimization of cover systems and reduce long-term environmental liabilities.

Opportunities also include early alignment of environmental, permitting, and social strategies to support project approvals and stakeholder engagement. Development of clear closure land use objectives in collaboration with local communities may improve social acceptance and reduce permitting risk, while enhanced environmental planning may improve overall project outcomes and ESG performance.

25.14.2.8 Capital and operating cost estimates

Opportunities to optimize the design of the processing plant may allow a reduction in capital and operating costs for the project, depending on the results of metallurgical testing in future. As outlined in Section 17, these tests may reduce reagent and consumable consumption, allow a coarser primary grind size of the processed material or change the process flowsheet significantly in future phases. Additions to the process flowsheet such as gravity recovery may increase the recovery of coarse gold and reduce throughput to whole-ore leaching and downstream tailings handling.

Opportunities to improve the mining costs are related to the reduction in overall development quantities by investigating alternative mining methods. Additionally, the amount of waste stripping required in the open pits is currently tied to conservative geotechnical parameters that may be optimized in future studies.

26 Recommendations

26.1 Introduction

The Los Reyes project shows positive economics, as demonstrated by the results presented in this technical report. It is recommended that the project be advanced through to PFS study. The following sections detail the recommended future work for the project. The estimated costs are summarized in Table 26-1.

Table 26-1: Recommendations cost summary

Program component	Estimated total cost (US\$M)
Geotechnical Drilling	8.80
Metallurgical Testwork	1.50
Geotechnical & Hydrological drilling & field work	3.05
Mining & Geotechnical Studies, incl. backfilling	1.70
Process & Infrastructure Engineering	1.45
Tailings Storage Facility	0.45
Environmental Studies	1.20
Others	1.75
Total	19.90

26.2 Exploration and mineral resource

26.2.1 Exploration

The drilling program includes a Phase I campaign of 20,000 m scheduled for 2026 with a budget of US\$8.09 million focused on de-risking and upgrading the existing Mineral Resource Estimate previously defined by Prime. A Phase II drilling campaign contingent upon positive results from the 2026 program is to further expand the resource base and advance the most prospective exploration opportunities identified during Phase.

The 2026 drilling campaign will prioritize areas identified as early mineable opportunities, as defined in the PEA mine design. The program proposes the deployment of six drill rigs across the Zapote, San Miguel, and Guadalupe Underground target areas. Drilling will be conducted on systematic sections using directional drilling techniques to optimize meterage, reduce the number of drill platforms required, and improve overall drilling efficiency.

In parallel, a district-scale delineation program with a cost of US\$0.96 million will be undertaken with the objective of defining and prioritizing target areas for follow-up target definition work in Phase II, ultimately establishing a district-wide exploration pipeline. Delineation activities will include development of a district-scale structural framework with emphasis on first-order controls, volcanic facies mapping, and the integration of systematic geochemistry, hyperspectral remote sensing, geophysics, and detailed geological mapping.

26.2.2 Mineral resource estimate

The Project contains Indicated and Inferred Mineral Resources that are associated with well-defined mineralized trends and models. All deposits are generally open along strike and at depth. The Company believes that the Project has the potential for the delineation of additional Mineral Resources within the three main trends and additional exploration is warranted on new high priority targets identified from detailed mapping, surface sampling and preliminary drilling within the Property.

The exploration program should include a phased approach of drilling along the extensions (along strike and at depth) of the known deposits (resource drilling) along with drilling other identified high-priority targets (discovery drilling) as well as other key objectives as listed below:

- Continue detailed field mapping and sampling, rock and soil geochemistry along currently defined and possible new structural corridors.
- Almost three-quarters of the MRE is at the Indicated level of confidence, which is already sufficient for inclusion in a PFS and potential conversion to Mineral Reserves. Prior to commencement of a PFS, exploration should focus on converting pit- and stope-constrained Inferred resources to the Indicated level of confidence. Approximately 20,000 metres of drilling are recommended in 2026 to de-risk and upgrade mineral resources along the three main trends (Gaudalupe, Z-T, and Central), and to collect data for metallurgical and geotechnical test-work to support additional studies.

26.3 Mining Methods

26.3.1 Mine geotechnical drilling

A dedicated geotechnical drilling and logging program is recommended across each of the deposits to characterize the rock masses expected to form the underground workings and open pit slopes. The program is expected to include HQ3 oriented core drilling with detailed geomechanical logging and laboratory strength testing to collect rock mass structure, quality and intact strength data. The program will target key areas, such as the Estaca Fault zone, the mineralized veins and surrounding rock masses, the host rock behind the open pit slopes, as well as the underground declines between the deposits. The number of drillholes will vary by deposit, with two to three drillholes in the smaller deposits (i.e. Guadeloupe West open pit, Mariposa underground, etc.) and six to twelve drillholes in the larger deposits (i.e. Z-T, Central and Guadeloupe East) and the underground declines between the deposits.

26.3.2 Waste rock storage facility

Engineering of the open-pit WRSFs is recommended, covering foundation conditions, geometry, surface-water management and progressive rehabilitation, developed together with the geotechnical and environmental programmes.

26.4 Metallurgical testing

Additional metallurgical testing should be conducted to advance the project to a higher level of understanding. Metallurgical test work prior to a potential Pre-Feasibility Study (PFS) should include the following:

- Characterization of metallurgical variability samples across all mineralised zones

- A greater number of variability samples should be tested across each deposit, with a focus on the first five years of production to confirm material hardness properties and metallurgical response. A minimum of 60 variability for leach tests, 46 variability samples for comminution tests, and 33 composite samples is recommended.
- Testing of master composites from the main vein system deposits, assembled from drill core intervals that represent greater spatial coverage of the potential mill feed material.
- Evaluation of potential alternate processing routes including heap leaching, flotation, gravity recovery, and alternate leaching methods
- Detailed characterization test work on feed samples
- Quantifying the presence of any carbonaceous or sulphidic material that could lead to preg-robbing
- Quantifying any deleterious elements in the material, including mercury, arsenic, antimony, etc.
- Comminution test work for material from all mineralised zones in the project area, including:
 - SMC tests
 - Bond Ball Mill Work Index (BWi) tests
 - Abrasion Index (Ai) tests
- Whole-ore leaching test work for material from all mineralised zones in the project area
- Heap leaching test work, including column leaching tests, permeability tests, agglomeration tests, and pregnant leach solution characterization tests
- Oxygen uptake tests for whole-ore leaching
- Cyanide detoxification testing on material representative of the final CCD underflow
- Dynamic thickener settling tests on leach feed, CCD circuit feed, and detoxified tailings for confirmation of thickener sizing
- Filtration tests on thickened detoxified tailings
- Paste tailings testing
- Environmental characterization of tailings.

26.5 Process and Infrastructure Engineering

The estimated cost for process and infrastructure engineering for a potential feasibility study (FS) is US\$2.0 million. Engineering deliverables would include:

- Process trade-off studies
- Flow diagrams
- Detailed equipment list
- Power listing and consumption estimate
- Architectural (building sizes) to estimate steel and concrete quantities
- Detailed material and water balance
- Detailed process design criteria
- General arrangements (GA) and elevation drawings
- Electrical single line drawing

- Equipment and supply quotations updated and sources determined
- Estimate of equipment and materials freight quantities
- Capital cost estimate
- Operating cost estimate
- Major equipment spares and warehouse inventory cost estimate
- Construction workhours estimate
- Construction schedule.

26.6 Infrastructure

26.6.1 On-site and off-site Infrastructure

To advance the Process Plant into the next phase of the project, the plant site will need to be frozen and initial geotechnical pits dug in critical locations to provide improved information for foundation designs around major structures.

Confirmation of the site water needs and potential water and power sources will be required to improve accuracy of infrastructure designs.

Assessment of the availability of accommodation in the nearby town of Cosalá will also improve accuracy of the site camp sizing.

26.6.2 Tailings management facility and water management

To advance the TSF and water management design beyond the Preliminary Economic Assessment level, additional engineering and field investigations are recommended to reduce technical uncertainty and support subsequent study phases. Key recommended activities to support the next phase of engineering are summarized below:

- Investigate foundation conditions in the TSF basin and underneath the stability berms by completing geotechnical site investigation programs.
- Complete geological mapping in the TSF basin to identify and characterize fault systems.
- Install vibrating wire piezometers in foundation materials so that foundation pore pressures and hydraulic gradients can be assessed.
- Complete laboratory testwork programs to geotechnically and geochemically characterize the tailings and construction borrow sources.
- Conduct a stability analysis on the filtered TSF
- Collect site-specific meteorological and hydrology data to refine seasonal runoff values, design storms, and water balance calculations.
- Complete hydrogeological site investigation programs to characterize the foundation and better understand the required seepage control measures.
- Identify construction material borrow sources and assess the suitability of waste rock from the open pit for potential use as a construction material for dam construction.

- Evaluate underground and open pit development and sequencing and refine estimates of the associated dewatering rates. Assess the implications of these dewatering rates to refine water balance projections and potential water treatment requirements.
- Develop an Operations, Maintenance and Surveillance (OMS) Manual and Emergency Preparedness and Response Plan (EPRP) for the tailings and water management systems based on final designs and operating criteria.
- Develop a detailed closure plan for the TSF and mine site based on the final design configuration.

26.7 Environmental studies

A permitting strategy will be advanced early to address the full set of federal, state, and local requirements identified for the project. This includes securing key approvals from SEMARNAT (e.g., MIA authorization for development and construction, land use change technical study, environmental license, and waste management plans), CONAGUA (water use concession and discharge permits), and local authorities (land use and construction licenses, civil protection programs, and residue management registration). Given the breadth of approvals required, coordination between agencies should be proactively managed to ensure alignment of application timing and supporting technical studies.

Regulatory risk management should continue to focus on maintaining compliance with environmental authorization frameworks with the submission of a MIA for exploration activities, followed by another subsequent mining activities.

Water availability and permitting represent a critical project constraint requiring focused evaluation. While the project area shows a significant deficit in groundwater availability within the Río Elota aquifer, a substantial surplus exists in the Rio Habitas basin for surface water. However, the longstanding restriction on granting water use concessions within the Elota River basin introduces permitting limitations, particularly given that Rio Habitas is a tributary of Rio Elota. As a result, water sourcing strategies, including reliance on mining-derived water or alternative supply options, should be clearly defined and supported by detailed future studies.

27 References

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