NI 43-101 Technical Report Prefeasibility Study for the Soto Norte Project, Santander, Colombia

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Forward looking information

This technical report may contain or incorporate by reference information that constitutes “forward-looking information” or “forward-looking statements” (collectively, forward-looking information) within the meaning of the applicable securities legislation. All statements, other than statements of historical fact, contained or incorporated by reference in this technical report including, but not limited to, statements related to those items listed below, constitute forward-looking information. Forward-looking information involves known and unknown risks, uncertainties, and other factors that may cause the actual results, performance or achievements of Aris Mining to be materially different from the forward-looking information contained herein. When used in this technical report, such information uses words such as “aims”, “anticipates”, “assumes”, “believes”, “budget”, “committed”, “continue”, “plans”, “project”, “endeavors”, “ensures”, “estimates”, “expects”, “focus”, “forecasts”, “forward”, “guidance”, “intends”, “likely”, “opportunity”, “outlook”, “pending”, “possible”, “potentially”, “predicts”, “proposed”, “scheduled”, “seeks”, “strives”, “targets” or variations of such words and phrases or statements that certain actions, events or results “can”, “could”, “generally”, “may”, “might”, “should”, “will” or “would” occur or be achieved and any other similar terminology.

The forward-looking information contained herein reflects current expectations regarding future events and operating performance and speaks only as of the date of this technical report. Generally, forward-looking information involves significant risks and uncertainties; therefore, it should not be read as a guarantee of future performance or results and will not necessarily be an accurate indication of whether or not such results will be achieved. Undue reliance should not be placed on such statements. A number of factors could cause the actual results to differ materially from the results discussed in the forward-looking information. Although the forward-looking information is based on what management of Aris Mining believes are reasonable assumptions, Aris Mining cannot assure readers that actual results will be consistent with the forward-looking information.

This technical report includes forward-looking information pertaining to, among other factors, the following: the projections, assumptions and estimates related to the Project, including, without limitation, those relating to development, capital and operating costs, production, grade, recoveries, metal prices, life of mine, mine sequencing, economic assumptions such as capital expenditures, cash flow and revenue, mine design, permitting and licensing, mining techniques and processes, timing of estimated production, equipment, staffing, emissions, use of land, estimates of mineral resources, use of energy storage technologies, the timing and expectations for other future studies, the operation of Aris Mining, the development and future operation of the Project, and the ability to secure state and local permits for the Soto Norte Project.

Forward-looking information is based upon a number of estimates and assumptions that, while considered reasonable by Aris Mining as of the date of such statements, are inherently subject to significant business, economic and competitive uncertainties and contingencies. With respect to forward-looking information contained herein, the assumptions made by Aris Mining include but are not limited to:

- the environmental liabilities to which Project is subject;
- political developments in any jurisdiction in which Aris Mining operates being consistent with Aris Mining’s current expectations;
- the validity of its existing title to property and mineral claims;
- Aris Mining’s ability to maintain surface rights and legal access to property and mineral claims;
- experts retained by Aris Mining, technical and otherwise, being appropriately reputable and qualified;
- the viability, economically and otherwise, of developing the Project;
- Aris Mining’s ability to obtain qualified staff and equipment in a timely and cost-efficient manner to meet Aris Mining’s demand; and
- the impact of acquisitions, dispositions, suspensions or delays on Aris Mining’s business.

Forward-looking information is based on current expectations, estimates and projections that involve a number of risks which could cause the actual results to vary and, in some instances, to differ materially from those described in the forward- looking information contained in this technical report. These material risks include, but are not limited to:

- local environmental and regulatory requirements and delays in obtaining required environmental and other licenses, including delays associated with local communities and indigenous peoples;
- changes in national and local government legislation, taxation, controls, regulations and political or economic developments in Canada or Colombia, including with respect to the TRA and Order 044;
- uncertainties and hazards associated with gold exploration, development and mining, including but not limited to, environmental hazards, industrial accidents, unusual or unexpected formations, pressures, cave-ins, flooding, polymetallic concentrate losses, and blockades and operational stoppages;
- risks associated with costs, supply chain disruptions, and financial risks due to changes in tariffs, trade policies, international trade disputes, or regulatory shifts;
- economic and political risks associated with operating in foreign jurisdictions, including emerging country risks, exchange controls, expropriation risks, political instability and corruption;
- risks associated with capital and operating cost estimates;
- dependence of operations on construction and maintenance of adequate infrastructure;
- fluctuations in foreign exchange or interest rates and stock market volatility;
- operational and technical problems;
- Aris Mining's ability to maintain good relations with employees and contract mining partners;
- reliance on key personnel;
- competition for, among other things, capital, and the acquisition of mining properties and undeveloped lands;
- uncertainties relating to title to property and mineral resource and mineral reserve estimates;
- risks associated with acquisitions and integration;
- risks associated with Aris Mining's ability to meet its financial obligations as they fall due;
- volatility in the price of salable metal or certain other commodities relevant to Aris Mining's operations, such as diesel fuel and electricity;
- risks that Aris Mining's actual production may be less than is currently estimated;
- risks associated with servicing Aris Mining's indebtedness and additional funding requirements for exploration, operational programs or expansion properties, as well as to complete any large scale development projects;
- risks associated with general economic factors, including ongoing economic conditions, investor sentiment, market accessibility and market perception;
- changes in the accessibility and availability of insurance for mining operations and property;
- environmental, sustainability and governance practices and performance;
- risks associated with climate change;
- risks associated with the reliance on experts outside of Canada;
- costs associated with the decommissioning of Aris Mining's mines and exploration properties;
- potential conflicts of interest among the directors of Aris Mining;
- uncertainties relating to the enforcement of civil labilities and service of process outside of Canada;
- risks associated with keeping adequate cyber-security measures;
- risks associated with operating a joint venture; and
- other factors further discussed in the section entitled "Risk Factors" in Aris Mining's Annual Information Form for the year ended December 31, 2024 which is available on Aris Mining's website at www.aris-mining.com, on SEDAR+ at www.sedarplus.ca and included as part of Aris Mining's Annual Report on Form 40-F, filed with the SEC at www.sec.gov.

Readers are cautioned that the foregoing lists of factors are not exhaustive. There can be no assurance that forward-looking information will prove to be accurate. Forward-looking information is provided for the purpose of providing information about management's expectations and plans relating to the future. The forward-looking information included in this technical report is qualified by these cautionary statements and those made in Aris Mining's other filings with the securities regulators of Canada including, but not limited to, the cautionary statements made in the "*Risks and Uncertainties*" section of Aris Mining's Management's Discussion and Analysis for the three and six months ended June 30, 2025 which is available on Aris Mining's website at www.aris-mining.com, on SEDAR+ at www.sedarplus.ca and on Aris Mining's profile with the SEC at www.sec.gov.

The forward-looking information contained herein is made as of the date of this technical report and Aris Mining assumes no obligations to update or revise it to reflect new events or circumstances, other than as required by applicable securities laws.

Non-GAAP financial measures

This technical report refers to a number of non-GAAP financial measures, which are not measures recognized under International Financial Reporting Standards (IFRS) and do not have a standardized meaning prescribed by IFRS. These non-GAA financial measures described below do not have standardized meanings under IFRS, may differ from those used by other issuers, and may not be comparable to similar financial measures reported by other issuers. Accordingly, these measures are intended to provide additional information and should not be considered in isolation or as a substitute for measures of performance prepared in accordance with IFRS.

Cash costs – Cash cost and cash cost per ounce ($ per oz) are a common financial performance measure and ratio in the mining industry; however, they have no standard meaning under IFRS. Cash cost $ per oz is calculated by dividing total cash costs by volume of gold ounces projected to be produced on a payable basis.

All-in sustaining costs (AISC) – AISC and AISC $ per oz sold are a common financial performance measure and ratio in the mining industry; however, they have no standard meaning under IFRS. AISC is calculated by dividing AISC by volume of gold ounces projected to be produced on a payable basis.

Earnings before interest, taxes, depreciation and amortization (EBITDA) – EBITDA is a common financial performance measure in the mining industry; however, it has no standard meaning under IFRS. EBITDA represents earnings before interest, income taxes and depreciation, depletion and amortization.

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1 Summary

1.1 Introduction

This technical report has been prepared for Aris Mining Corporation (Aris Mining) in compliance with the disclosure requirements of National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* (NI 43-101) to disclose material updates to the Soto Norte Project (Soto Norte, the Project, or the Property) resulting from updated mineral resource and mineral reserve estimates and the results of a prefeasibility study of a smaller, more efficient development plan for the Project than was previously considered in a feasibility study titled *NI 43-101 Feasibility Study of the Soto Norte Gold Project, Santander, Colombia* prepared by SRK Consulting (UK) Limited, Sociedad Minera de Santander S.A.S. and SNC-Lavalin Inc. with an effective date of January 1, 2021 (the 2021 feasibility study). Soto Norte is an undeveloped, prefeasibility study stage underground gold-copper project where environmental permitting activities are underway.

This prefeasibility study outlines a long life underground gold mine with robust economics, low operating costs, and a design that reflects industry best practices in environmental protection and community partnership. The Soto Norte development plan has been specifically designed to balance scale and sustainability, and engineered to avoid impacts on the Santurbán páramo and regional watercourses. The Project will also generate significant long term benefits for Colombia, the department of Santander, and the municipalities of California, Suratá, and Matanza.

Two parallel zones of gold, silver, and copper mineralization known as Gigante and Mascota have been defined by 904 diamond drillholes for a total of over 375,000 metres (m) over a strike length of 2.6 kilometres (km) each down to a depth of approximately 800 m below surface. The mine will be developed over a strike length of 1.8 km at Gigante and 1.6 km at Mascota over a vertical range of 700 m.

The underground mine has been designed to produce ore using longitudinal open stoping with backfill, a safe, modern mining method, at a rate of 2,750 tonnes per day (tpd). Nearly half of the process tailings will be filtered and reused in paste backfill for underground mining, which both supports ground stability and reduces the volume of process tailings stored on the surface. The process water will be reclaimed from the tailings and reused in the processing plant, lowering water needs. The mined material will be transferred from the mine portal to the processing plant via an aerial rope conveyor, minimizing truck traffic, road dust, and the risk of material spillage into waterways.

As part of the Project's commitment to local stakeholders, the process plant has been designed to produce at a rate of 3,500 tpd, of which 750 tpd is dedicated to processing mill feed purchased from contract mining partners from the local communities. This initiative provides a safe, regulated alternative to traditional artisanal processing and helps to reduce existing impacts in a region that suffers from untreated tailings releases into rivers. The designed throughput of 3,500 tpd, including the 750 tpd dedicated for the local community, supports strong investment returns, and is purposefully smaller than previous designs to reflect community input.

The processing plant will produce three saleable concentrate products including gold rich copper concentrates, pyrite concentrates, and gravity gold concentrates. No cyanide or mercury is required or will be used in the process, eliminating two of the most harmful pollutants in gold processing. By purchasing and processing material from contract mining partners, the Project will eliminate the need for the use of mercury in the neighbouring community, improve the quality of water, provide an efficient use of energy, and help ensure all tailings are managed in a safe, controlled manner in the local area.

The gold rich concentrates will be produced in a mechanically contained system, minimizing the risk of fine particles or process water entering local streams. No on site smelting or refining will take place, avoiding activities that generate harmful emissions or chemical residues. The concentrates are a stable, high grade semi-solid product, and will be securely stored, containerized, and shipped to international buyers.

The Project footprint is located 600 m horizontally outside of the current delimitation of the Santurbán páramo, where mining is prohibited, and approximately 350 m in elevation below it. While the Colombian government is in the process of redefining the Santurbán páramo boundaries, and these distances may change, the Project is outside of the Santurbán páramo boundaries. Comprehensive water studies have demonstrated that the underground mine will not affect the ecologically sensitive environment of the Santurbán páramo, as the shallow water at the páramo is not connected to the deeper groundwater at the mine.

The Project has been designed with robust water management and protection as guiding principles. There is sufficient and readily available water within the Project area for all aspects of the operation including mining, processing, the camp, and other activities. Construction water will be sourced under permits from the La Baja Creek and Suratá River, which both have abundant water supply. During operation, the main source of water for the underground mine and its associated infrastructure will be from groundwater and the main source of water for processing will be from the adjacent Suratá River.

The Project plans to reuse, recycle, and return approximately 96.5% of the Project's total water requirements back to the environment, resulting in a net water use estimated at 3.5%. The majority of the demand will be supplied by the Suratá River, which will supply approximately 2.8 litres per second of net make up water for the processing plant, representing 0.22% of the average flow of the Suratá River at the planned water access point near the village of Suratá, and 0.08% of the average flow of the Suratá River at Bucaramanga, the nearest city located 55 km downstream from the Project.

The Project includes a comprehensive water management plan that incorporates plans and programs tailored to each water application or activity with significant interaction with water to ensure that both the quality and quantity of local water resources are preserved throughout the life of the operation, and all potential impacts related to water use have been thoroughly identified, assessed, and addressed through prevention, mitigation, and compensation measures. The underground mine has been designed to minimize groundwater flows into the underground workings through advanced cover drilling and grouting ahead of mine development to identify and seal any water bearing structures before mining reaches them, greatly reducing potential inflow, and to manage, treat, and if required, safely return any captured water to the environment in compliance with the environmental standards and discharge permits. The process plant has been designed to minimize the use of water and to recycle both process water and the water obtained from the dewatering of the process tailings. The filtered tailings storage facility has been designed to divert rainwater runoff away from the facility and to drain and manage any subsequent dewatering of the tailings and any rainwater falling onto the facility. Continuous monitoring of water flow and quality will be undertaken with defined measures planned if any changes are detected. These strategies will help ensure that the current water flow rates within the ecosystem are maintained.

The peak workforce during Project construction is estimated at 2,292, plus 90 administrative staff and management. During operations, the workforce is estimated at 676 company personnel. The Project is targeting 60% of the workforce to be hired from the local community, 20% from the department of Santander, and 18% from other departments in Colombia.

The Project's expected mine life of 22 years is based on the current mineral reserve estimate, and there are opportunities for future mine extension if additional mineral reserves are defined through continued exploration. There are numerous areas of high grade inferred material located adjacent to stopes designed around indicated material that could be targeted for exploration. An initial exploration drilling program of 35 drillholes for approximately 12,500 m is recommended to target the highest grade areas of inferred mineral resources, and those located in the upper areas of the mine, comprising 1.2 Mt at 12.50 grams per tonne (g/t) Au for 482,000 ounces, to potentially convert those volumes to indicated mineral resources.

Over its life, the Project is estimated to produce 4.3 million ounces (Moz) of gold, 18.8 Moz of silver, and 84.0 million pounds (Mlb) of copper in concentrates. The average annual gold production is estimated at 263 thousand ounces (koz) between years 2 and 10, with an average annual gold production of 203 koz between years 1 and 21.

The total estimated upfront capital expenditure is \$625 million. The cumulative after tax net cash flow is estimated at \$5.0 billion, including all initial capital costs, pre-production costs, sustaining capital costs, closure costs, value added tax (VAT), and contingency. At the base case assumption of \$2,600 per ounce of gold, the life of mine average cash costs per ounce of gold are estimated at \$345 and all in sustaining costs (AISC) per ounce of gold are estimated at \$534. The Project has an after tax net present value at a 5% discount rate ($NPV_{5\%}$) of \$2.7 billion, an internal rate of return (IRR) of 35.4%, and a pay back period of 2.3 years.

The Project is estimated to contribute \$2.6 billion in income taxes and \$393 million in royalty payments to the Colombian government.

All dollar amounts presented in this technical report are expressed in U.S. dollars, unless otherwise indicated.

1.2 Property description, location, and access

1.2.1 Location and access

The Project is located in the historic California – Vetas mining district, approximately 350 km north of Bogotá, the capital city of Colombia, 55 km northeast of the city of Bucaramanga, and 9 km northeast of the town of California. The Project is readily accessed year round by vehicle from Bucaramanga via 54 km of paved and unpaved roads to California and then by 9 km of unpaved road that passes through the centre of the Project. Smaller roads and foot trails provide further access throughout the Property, including 13 km of road connecting the processing plant and the underground mine access area.

Bucaramanga is readily accessible from Bogotá by 397 km of paved road, and connected to the city of Barrancabermeja, a city with a river port that is the preferred option for transporting concentrates to overseas buyers, via 115 km of paved road. Barrancabermeja is located 650 km on the Magdalena River from the seaport at Cartagena, one of the largest in South America. Bucaramanga is also connected to the city of Santa Marta on the Caribbean coast by 539 km of paved road, which is another potential concentrate transport route.

There are several daily domestic flights from Bogotá and Medellín, where Aris Mining maintains its executive and shared services office, to Bucaramanga, which also offers direct international flights to Panama City and Fort Lauderdale.

1.2.2 Mineral tenure, Aris Mining's interest, surface rights, and obligations

There are 20 titles with a total area of 3,225.22 hectares (ha) associated with the Property, all of which are 100% owned by Proyecto Soto Norte S.A.S. (PSN), with the exception of title 14947, of which PSN owns an option for 80% of the title. PSN is a company existing under the laws of Colombia, of which Aris Mining is the Project operator and holds an indirect 51% joint venture interest with MDC Industry Holdings LLC (MDCIH), a wholly owned subsidiary of the investment company Mubadala Investment Company PJSC holding the remaining 49% joint venture interest.

The mining titles at the Project do not provide property or rights over the land, but the right exists to expropriate or impose an easement over the land through administrative and/or judicial proceedings if it is required to develop the Project. The preferred method for acquiring land at Soto Norte for project development will be to reach agreements with landowners following receipt of the environmental permits.

PSN owns land with a total area of 192 ha, of which 10 ha are shared with third parties, and leases a further 1 ha of land to support the activities at the Property.

Title 095-68 is the key title containing the Property's mineral resources and mineral reserves, the planned underground mining infrastructure, and the surface infrastructure at the mine area. It has an area of 379.4 ha and the title expiry date is June 8, 2028. PSN will request an extension of the title in accordance with the terms of the contract.

Additional land purchases will be required for Project development and operation.

The work required to maintain the titles is dependent on the stage of the contract, such as performing exploration activities during the exploration stage. The title owner is required to comply with the environmental regulations and obtain all the necessary permits during the exploration stage, and the environmental license must be obtained for the construction and mining stages. Economic requirements include paying a surface fee during the exploration and construction stages and paying the royalty and obtaining an environmental mining insurance policy during the mining stage.

1.2.3 Royalties, agreements, and encumbrances

Royalties due to the Colombian state include a 4% royalty on 80% of the gold and silver produced and a 5% royalty for copper on 100% of the copper produced.

1.2.4 Significant factors and risks

On January 30, 2024, the Colombian Ministry of Environment issued Decree 044 which allows the Ministry to declare temporary reserve areas in certain parts of Colombia. To establish a temporary reserve area, a resolution must be issued by the Ministry detailing the area that is to be temporarily reserved. The Ministry issued Resolution 221 of 2025, amended by Resolution 239 of 2025, by means of which it declared a Temporary Reserve Area (TRA) in the Soto Norte region. The TRA will be in effect for two years, with a possible two year extension. While the TRA is in force, no new concessions or environmental permits may be granted by the mining or environmental regulators. During this period, the Ministry of Environment and Sustainable Development of Colombia must conduct environmental studies to determine whether to make the reserve area permanent. Notwithstanding the TRA, the Soto Norte Project may continue environmental studies, provided no environmental permit is required. Decree 044 and the TRA may delay licencing of the Soto Norte Project.

Decree 044 and the TRA resolutions are presently being challenged in administrative courts, with actions led by the Colombian Disciplinary Office, artisanal and small mining units, the Colombian Mining Trade Association, and the National trade association.

Additionally, the Administrative Tribunal of Santander issued a ruling in July 2025 in a class action proceeding recognizing the Santurbán páramo as a subject of personal rights and designating the Ministry of Environment as its legal guardian. While there is no direct impact on the Soto Norte Project, environmental licensing proceedings for the Soto Norte Project may be delayed or hindered because the Tribunal ordered that: (i) the Ministry of Environment must actively participate and protect the páramo in any licensing process, including through the use of administrative injunctions, (ii) all relevant environmental authorities must identify critical transition areas to the páramo in the Soto Norte region for water protection, and (iii) zoning regulations must exclude mining activities in “buffer zones” in alignment with the 2014 delimitation process. The ruling was appealed.

The Soto Norte Project remains several years away from development. With the completion of this technical report, PSN intends to present a fully redesigned project to the Colombian regulators following the conclusion of the environmental and technical studies currently underway.

Environmental effects have arisen from historical workings and ongoing informal mining activities and processing plants within the Project titles. Monitoring data show impacts on the La Baja Creek, which runs through the Project area. These include discharges from informal mining affected by acid rock drainage and/or metal leaching, and there is erosion and mobilization of sediments near the informal mine workings and processing areas. None of the environmental liabilities resulting from these informal operations conducted after the Project acquired the titles are the legal responsibility of the Project. PSN is responsible for managing environmental effects related to its own activities and for the effects of the previous titleholder activities. PSN is and will continue to keep working with the regulatory authorities to remediate damage wherever possible.

Except for the risks mentioned herein and in Section 4.7, there are no other known significant factors or risks that may affect access, title, or the right or ability to perform ongoing work programs on the Property, including permitting and addressing environmental liabilities, aside from the requirement to obtain future environmental licenses and approvals.

1.3 History

Artisanal miners holding small tenements within the Property mined on a small scale in the past. No production records exist, but an estimated 50,000 to 75,000 tonnes are believed to have been mined from the Property. Between 2010 and 2012, small scale production was reported at a rate of between 10 and 30 tpd. There has been no formal production at the Property.

In December 2005, Ventana Gold Corporation (Ventana) acquired small scale tenements from the artisanal miners and formed the La Bodega project. In December 2005, Ventana began the first modern exploration program comprising geochemical sampling, geophysical surveys, and exploration diamond drilling. Ventana disclosed a historical scoping study in November 2010, and by March 2011, a total of 378 diamond drillholes for 134,078 m had been completed.

In March 2009, EBX Group (EBX) began purchasing shares of Ventana, and on May 25, 2011, AUX Canada Acquisition Inc. (AUX Canada), an affiliate of EBX, acquired Ventana and changed the Project name from La Bodega to El Gigante. In October 2012, AUX Canada’s local subsidiary merged with Sociedad Bodega Ventana Baja, consolidating the Project under AUX Colombia S.A.S. (AUX Colombia). AUX Colombia drilled 431 diamond drillholes for 198,660 m between 2011 and 2013. AUX Colombia excavated an exploration tunnel between July and October 2012, as well as four drives into the Gigante and

Mascota veins. AUX Colombia disclosed a historical mineral resource estimate in January 2013. In July 2013 AUX Colombia terminated all exploration activities, and the Project was placed on care and maintenance from mid-2013 to the first quarter of 2015.

In January 2012, MDCIH formed a strategic partnership with EBX through a preferred equity investment. In 2013, following the financial difficulties of EBX, MDCIH took ownership of the Project as a redemption on the original investment. In February 2015, MDCIH took ownership of AUX Colombia and on November 6, 2015, changed the subsidiary name to Sociedad Minera de Santander S.A.S. (Minesa), and changed the Project name to Soto Norte.

Between 2015 and 2018 Minesa undertook geochemical and channel sampling, and drilled 95 diamond drillholes totalling 42,498 m. No further exploration specific activities have taken place since 2018. Technical studies have been undertaken as required to support ongoing Project design studies.

The results of the Project diamond drillhole samples have been utilized for historical as well as the current mineral resource and mineral reserve estimate. Historical mineral resources have been estimated with effective dates of November 2010, July 2012, January 2013, February 2016, January 2017, July 2017, May 2019, and January 2021.

In August 2017 the first historical mineral reserve was estimated as part of a prefeasibility study, based on the historical January 2017 mineral resource estimate. In January 2021 the second historical mineral reserve was estimated as part of the 2021 feasibility study, based on the historical May 2019 mineral resource estimate.

None of these historical mineral resource and mineral reserve estimates are current. They should not be relied on and have been superseded by the current mineral resource and mineral reserve estimates disclosed in this technical report.

On April 12, 2022, Aris Mining (formerly Aris Gold Corporation) acquired a 20% joint venture interest in Minesa and became the Project operator. On November 2, 2023, the Project was renamed to PSN. On June 27, 2024, Aris Mining acquired an additional 31% joint venture interest in the Project, increasing its total ownership to 51%, with MDCIH retaining the remaining 49%.

1.4 Geological setting, mineralization, and deposit types

1.4.1 Regional, local, and property geology

Regionally, the Property is located in the western branch of the Eastern Cordillera of the Colombian Andes, where the geology is characterized by the creation of subduction zones and associated magmatism, uplifted blocks, and compressional faulting. The north-northwest trending Santander Massif hosts the Soto Norte mineralization, bound by the Bucaramanga fault to the west and the Socota-Santander fault to the east. The Project geology is related to magmatic events and contact metamorphism between these two faults.

Locally, the Santander Massif at the Project comprises three major geological units, including the Bucaramanga Complex comprising paragneisses, migmatites, amphibolites, quartzite, marbles, and granulates; the Central Santander Plutonic Group comprising intrusive calc-alkaline rocks ranging from tonalites, granodiorites, and leucogranite; and sedimentary rocks.

District scale faulting forms topographic relief and the dominant northeast trending faults, which broadly control the shape of the intrusive complex. The principal faults include the La Baja, Mongora, and Cucutilla faults, which are interpreted to belong to a wider regional structural corridor that acts as one of the controls on mineralization throughout the California – Vetas mining district. Intrusive rock on the north side of the La Baja fault, and gneiss on the south side of the fault, is the host of the Soto Norte mineralization.

Mineralization at the Property is hosted in gneisses of the Bucaramanga Complex and leucogranites of the Santander Plutonic Group, and mostly occur within tectonic-hydrothermal breccia bodies emplaced in a dilatant structural setting.

1.4.2 Mineralization

The faults hosting the parallel Gigante and Mascota mineralization trends represent two linking structures between the principal faults, with the Mascota mineralization hosted by the La Rosa fault zone and the Gigante mineralization hosted by the La Baja fault zone. Mineralization took place during active faulting along these structures. The faults converge at depth and are indicated to join into a single structure. Numerous minor faults are present, some of which are mineralized and have been partially exploited at the surface by artisanal and small scale miners.

Mineralization at the Property comprises parallel anastomosing veins within the fault systems, with variable widths and characteristics. Veins at Mascota have open-space filling textures, with hydrothermal brecciation and brecciated fragments of wall rock. Veining in the Gigante structure is mostly characterized by more compact, less vuggy, and often banded textures and is characterized by more heavily altered wallrock and clay content. Aserradero is a smaller, lower grade deposit located to the southeast of Mascota and Gigante.

Gold and electrum have a strong relationship with fine, crystalline pyrite and occur either free with the gold, adhering to pyrite particles, or encapsulated within the pyrite crystal lattice. Copper sulphides appear to have a partial affinity for pyrite but have much less of an association with gold than pyrite. Silver occurs principally as silver sulphosalts, pyrargyrite, and proustite. Copper occurs principally as enargite and to a lesser extent as bornite, chalcopyrite, chalcocite, and tetrahedrite-tennantite.

The Mascota and Gigante vein trends cover a strike extent of 2.6 km and have been drilled to a depth of approximately 800 m below the surface. The width of the veins is variable and averages between 1 and 3 m. The mineralized structures extend to the surface and are open at depth and along strike, resulting in a high exploration potential for expansion from future underground drilling stations.

1.4.3 Deposit types

The Soto Norte deposit is considered a high-sulphidation epithermal deposit, with gold, silver, and copper occurring mainly in sulphides. The deposit is related to porphyry stocks and dikes that crosscut older sedimentary, igneous, and metamorphic rocks. The hydrothermal source fluids flowed through fault related pathways, generating background propylitic and phyllic alteration of the local rocks during mineralization, followed by silicification and argillic alteration in the centre of the main veins, zoning outward to intermediate argillic and propylitic alteration that formed during the principal stages of mineral deposition. This model has formed the basis of the past exploration plans that have followed the vein trends along strike and down dip.

1.5 Exploration

Ventana began surface and underground geochemical sampling in early 2006, accompanied by ground magnetic and induced polarization (IP) geophysical surveys in the area of Gigante, followed by a ground magnetic and pole-dipole IP resistivity survey. Surface diamond drilling started in August 2006, with 143,568 m completed by the end of March 2011.

Between 2011 and 2013, AUX Colombia completed a further 200,124 m of diamond drilling over a strike length of 2.5 km.

In February 2012, an airborne magnetics and radiometrics survey was completed over Gigante and the surrounding area. A ground magnetics and IP survey was completed in March 2012 in the Gigante area.

The combined ground magnetics data shows strong anomalies with a northeast-southwest trend that coincides with the general trend of the mineralization and geochemical sampling results. The combined IP results show similar trends and indicate that the anomalies continue to the southwest through the Galway and Calvista properties, also owned by Aris Mining, and onwards to the town of California.

AUX Colombia completed mapping and sampling during excavation of the mine portal exploration tunnel and the existing tunnels into Gigante and Mascota.

Between 2015 and 2018 Minesa collected mobile metal ion soil, rock, and channel samples taken from historical mine workings. The combined Project geochemical samples confirm the northeast-southwest trend of gold mineralization subparallel to the La Baja Creek, and higher grade zones on the northeast limits towards the La Bodega area.

95 diamond drillholes were completed, totalling 42,498 m. Twelve geotechnical drillholes were completed to provide

geological and geotechnical information for a previously considered tunnel access.

No other exploration work has taken place since 2018.

1.6 Drilling

Diamond drilling was carried out by a range of different contractors during 2006 to 2018, for a total of 904 holes for 375,235 m. Drillholes were collared at HT diameter (71 millimetres (mm)) or HQ diameter (63.5 mm), then reduced as drilling conditions allowed to NT diameter (58.9 mm) or NQ diameter (47.6 mm). In a few cases the diameter was reduced to BT diameter (40.8 mm) or BQ diameter (36.5 mm).

No drilling for the purposes of mineral resource definition has been completed since 2018.

The drilling grid was first completed at 100 by 100 m spacing and later tightened to 50 by 50 m, with further tightening to 25 by 25 m on shallower areas above 150 m below the surface. At depth the drilling intersections remain relatively wide at 50 to 100 m, which is a function of the steep intersection angles required by the steep topography. Further infill drilling will be completed after underground development is in place to provide a better drilling intersection.

The mineralized structures are open at depth and along strike, with high exploration potential to target the deep structures from underground drilling stations.

As the drillhole intersections through the vein interpretations are used as an input into the mineral resource estimate, the relevancy of the raw drillhole sample assay results are superseded by the mineral resource estimate and are more meaningfully described in the context of the mineral resource estimate.

1.7 Sampling, analysis, and data verification

1.7.1 Sample preparation

The diamond drill core sampling process was similar across Project owners with minimal differences. Once logging was completed, the core was marked for cutting with a diamond bladed core saw, and the right hand side of the half core pieces were placed in plastic sample bags. For broken fragments, half the volume was selected by hand. Intervals with clay or unconsolidated material were split vertically with a knife while still saturated. The sampling intervals and assay results were recorded in the core logs. After cutting, the left hand side of the half core was placed back in the core box for storage. Although the standard sample length was 1.0 m, it was adjusted to align with geological contacts.

1.7.2 Quality assurance and quality control

The quality assurance, quality control (QAQC) submission rate increased over time as the Project owners prioritized data integrity and reliable assay results. QAQC samples from Ventana and AUX Colombia involved the laboratory selecting a pulp reject split of the original sample as directed by the Project’s geologists. Coarse blanks and commercial standard samples were also submitted.

Minesa implemented a rigorous QAQC system to ensure the validity, accuracy, and reliability of the sample assays. Analytical control measures included the use of certified reference materials, coarse and fine blanks, coarse and pulp reject duplicates split by the laboratory, and quarter core samples of the second half of the core sample.

Between 2006 and 2011, the QAQC insertion ratios were less than 10%, but increased to between 11 and 19% between 2012 and 2018, coinciding with Minesa’s ownership of the Project and their procedural improvements. Over the entire sampling history, 37 different standards were submitted with the results showing a low failure rate within acceptable parameters, with close to zero bias in some of the most representative standards. Less than 10% of the results exceeded three standard deviations of the certified value. The cause of these failures cannot be definitively determined but may be related to a mislabelling of the standard. No significant bias was observed. Blanks initially comprised coarse blanks until AUX Colombia began submitting fine blanks. The results of the fine blanks were satisfactory. Duplicate sample submissions increased over time and included coarse and pulp rejects and quarter core samples. Pulp duplicates showed high variability attributed to the presence of coarse gold. Variability was also noted to a lesser extent in the quarter core samples. The variability was not observed in the coarse duplicates, suggesting potential issues with the homogenization or splitting process during the selection of the pulp duplicates.

1.7.3 Security measures

Security measures for sample handling and chain of custody have been similar across Project operators. Ventana's chain of custody was maintained and monitored throughout the process with half cores selected for analysis, bagged, sealed, and then placed in larger bags, which were also sealed. Storage on site was in a locked core shed with 24 hour security until the samples for the entire drillhole were shipped as a single batch for sample preparation. The laboratory verified the security seals and signed off on receipt of the samples.

AUX Colombia included a chain of custody protocol where the names of all persons handling drill core were recorded on forms. Upon receipt of the sample at the laboratory, the sample barcode was scanned.

The Minesa chain of custody security protocol included that a member of the geological staff always accompanied the sample during transportation before handing them over to the laboratory for sample preparation. The samples were then transported by DHL to the analytical laboratory.

1.7.4 Analytical procedures

All of the Project diamond drill core samples have been prepared and analyzed by independent commercial laboratories.

The details of Ventana's drillhole sample preparation methods at the laboratory are unknown, but the samples were analyzed using fire assay for gold and silver as well as a 36 element ultra trace package using hot aqua regia digestion and inductively couple plasma – mass spectrometry (ICP-MS).

AUX Colombia's drill core samples were crushed and pulverized at ALS Chemex Bucaramanga then shipped by air courier to ALS Chemex in Lima for analysis, which held ISO 9001 and 17025 accreditations. The entire sample was crushed, then a 500 g split was collected with a riffle splitter and pulverized. The samples were assayed for silver and gold using fire assay with atomic absorption (AA) finish. Any sample with a gold grade greater than 5 g/t and a silver grade greater than 100 g/t was re-assayed using fire assay with gravimetric finish. 51 trace elements were assayed using aqua regia and ICP-AES/ICP-MS.

Minesa's drill core samples were prepared by drying, crushing, quartering with a Jones splitter, and pulverizing the subsample. The samples were assayed for gold using fire assay with AA finish and any gold assay greater than 100 g/t was re-assayed with gravimetric finish. Silver assays with grades greater than 100 g/t were re-assayed using aqua regia digestion with AAS finish, and samples greater than 1500 g/t were re-assayed using fire assay with gravimetric finish.

1.7.5 Data verification

In 2016, an independent consulting firm conducted a reanalysis of 922 quarter core samples and 1,148 pulp duplicates at an independent check laboratory, including standard controls in both sample types, and conducted a thorough review of the database data collected prior to 2014. The results indicated a low bias in the primary laboratory compared to the check laboratory, although silver assays showed higher variability. The results from the standard samples showed low bias and fell within acceptable limits.

For the current technical report, the qualified person responsible for geology verified the geological data supporting the mineral resource and mineral reserve estimates through the personal inspection and through collaboration with the Project team, including:

- cross validation of the database entries with selected original laboratory certificates;
- reviews of the geological and geographic environment of the Project;
- reviews of the nature and extent of all exploratory work completed by the Project owners, including those relevant to the current mineral resource estimate;
- reviews of mineralized and non-mineralized drill core intersections;

- reviews of standard operating procedures related to drilling, sampling, and analytical processes covering several stages in the sampling and assaying chain from raw samples to prepared assay pulps; logging, re-logging, core sampling processes, analytical QAQC controls, and chain of custody; and bulk density determination methods;
- reviews of sample storage facilities for drill core, coarse rejects, and pulp rejects;
- reviews of database management processes; and
- independent sample checks of drill core and pulp rejects.

Based on the personal inspection and geological database review, the qualified person has found that the drilling, logging, and sampling practices meet acceptable international standards, thus concluding that the sample preparation, safety protocols, and analytical procedures implemented for the Project provide an adequate current basis for the mineral resource estimate. The following were observed:

- the geology and mineralization controls are well understood and appropriately considered during drilling and geological interpretation;
- no material issues were identified in the database;
- the translation of previous drillhole collar coordinates to the current coordinate system and the on site inspection verified their accuracy with no concerns regarding the transformation method;
- survey review identified some anomalous measurements, which were examined and appear to be related to geological factors. Other anomalous measurements were not reviewed in detail but they do not appear to significantly impact the survey. These should be investigated further as the Project work progresses;
- assay results compared with certificates show minimal inconsistencies;
- in the early years of drilling, the QAQC sample insertion rate was limited, but this has been progressively enhanced to meet industry standard protocols;
- although the early drilling campaigns included limited QAQC samples, the overall assessments show no contamination issues with coarse and fine blanks during crushing and pulverization. Standard samples exhibit acceptable accuracy, though a few extreme outliers may be attributed to coding errors. Quarter core and coarse reject duplicates demonstrate good precision while pulp duplicates show lower precision, likely due to issues with homogenization or splitting during the pulverization stage; and
- independent sample checks by the qualified person included the re-assay of 20 pulp samples, which confirm consistency with the original grades. Six quarter core and two coarse reject samples were also re-assayed, showing some differences, generally with lower values. These variations, though not entirely clear, are likely associated with natural gold variability, the smaller sample size of the quarter core compared to the half core original sample, and sampling processes. The mineralization evidence is strong, with no significant indications of bias or errors in sample handling.

In the opinion of Kate Kitchen, the qualified person responsible for this disclosure, the data used for the purpose of estimating the mineral resources and mineral reserves and the development of the economic analyses are sufficiently reliable.

1.8 Mineral processing and metallurgical testing

1.8.1 Testwork history

Several metallurgical testwork programs have been undertaken between 2009 and 2018 to support the metallurgical assumptions utilized for the progressive Project studies, utilizing samples that were representative of the growing mineral resource as it was known at the time of the studies. These studies included processing method trade off studies as well as refinements of the selected operating parameters, as the properties and response of the samples under the testwork conditions were increasingly better understood.

The first two testwork programs considered flowsheets utilizing cyanidation. Testwork was conducted to support a scoping study by Ventana in 2010 that proposed a flowsheet consisting of comminution, gravity separation, and flotation to produce separate copper and pyrite concentrates, intensive cyanide leaching of the gravity concentrate, cyanide leaching of the combined copper cleaning tailings and pyrite concentrates to recover copper and silver via sulphide precipitation and gold through an adsorption – desorption recovery circuit. Testwork was later undertaken to support a scoping study by AUX Colombia in 2012 that proposed a flowsheet consisting of comminution, gravity separation, and flotation to produce a bulk sulphide concentrate, with pressure oxidation of the concentrate followed by solvent extraction and electrowinning to produce copper cathode, and cyanidation for gold recovery.

The remaining testwork programs eliminated the use of cyanidation for environmental reasons. Testwork was conducted to support trade off studies and a prefeasibility study by Minesa in 2017. The initial trade off study selected flotation as the preferred process route, due to capital costs and environmental advantages over flotation followed by pressure oxidation and cyanide leaching. Further trade off studies directed the work towards the production of sequential copper and pyrite gold flotation concentrates.

Testwork was undertaken to refine and parameterize the flowsheet selected during the 2017 prefeasibility study, comprised of comminution and flotation to produce separate copper and pyrite concentrates, and to validate the key metallurgical assumptions, to support a feasibility study undertaken by Minesa in 2021. In 2024, a gravity gold circuit trade off study was conducted based on previous testwork undertaken in 2010, 2017, and 2018 to justify the inclusion of a gravity concentrator in the flowsheet proposed for the current prefeasibility study.

The metallurgical testwork to date has been conducted on a wide range of samples representative of the material expected to be processed over the life of mine. The studies have been conducted to a sufficient quality and extent to support the process flow sheet presented in this prefeasibility study and has been utilized to support the previous feasibility study. The results of this testwork has been estimated into the mineral resource and mineral reserve block model, with the estimated variable used to develop the production schedule and economic analyses.

The level of testwork conducted to date that supports the development of robust process design criteria which has resulted in a flowsheet that recovers the required amount of gold, silver, and copper at saleable grades meets the typical expectations for a prefeasibility level of study. The flowsheet developed to produce a separate copper and pyrite concentrate through sequential flotation is viewed as the most technical and economically viable solution while mitigating risk.

1.8.2 Mineralogical testwork

Mineralogical testwork indicates that the processed ore will have fine to very fine grained gold present as native gold, electrum, and tellurides, mostly locked in other minerals or at the grain boundaries of the minerals, predominantly sulphide minerals. Silver is present in native gold, electrum, and telluride, as well as sulphosalts with antimony, arsenic, and bismuth, and is more correlated with copper. Pyrite is the most abundant sulphide mineral and is significantly coarser than the other sulphides. Enargite is the most abundant copper bearing mineral with the remaining copper distributed between bornite, covellite, chalcocite, and chalcopyrite. With the majority of gold particles associated with sulphide minerals, high recoveries by flotation are expected, and free gold and some of the larger entrained gold can be recovered through gravity. Fully silica encapsulated gold, which accounts for an estimated less than 1% of the mill feed, cannot be recovered through flotation, but some may be caught in a gravity circuit. Roughly half of the gold is associated with copper sulphide minerals and will be captured in the copper concentrate while the remaining gold associated with pyrite and chalcopyrite will be recovered in the pyrite concentrate. Silver is expected to be recovered in greater quantities in the copper concentrate.

1.8.3 Comminution testwork

Comminution testwork shows that the ore is highly variable and moderately hard for semi-autogenous grinding (SAG) and relatively hard for ball milling, compared to other ores. The expectation is that the ore will become increasingly harder at greater depth. The Bond Work Index results indicate that the material becomes abnormally harder at finer sizes, resulting in a higher energy requirement and a coarser grind product from a SAG mill, resulting in reduced throughput. The addition of a pebble crusher to a SAG will increase throughput. The current process flowsheet has made allowance for a SAG mill in closed circuit with a pebble crusher and a ball mill in closed circuit with a cyclone cluster, which greatly mitigates identified risks.

1.8.4 Gravity separation and flotation testwork

Gravity separation and flotation testwork shows that both Mascota and Gigante material respond well to flotation.

The findings of the gravity gold circuit trade off study were that there is gravity recoverable gold present in the Soto Norte ores and that the inclusion of a gravity circuit does not appear to have any adverse impact on the overall recovery of the economic minerals. The main concern is the lack of data from Gigante and the marketing effects for selling copper and pyrite concentrates with a reduced gold content. The benefits of a gravity circuit include the reduction of variability of gold in flotation, lowering the mass pull and producing a higher grade gold-silver concentrate while maintaining the overall recovery of the system, quicker and more pronounced process optimizations, and is unaffected by variations in clay content of the mill feed. The incorporation of a gravity concentrator in the processing flow sheet will not have any significant impact on the downstream circuit and will significantly mitigate the risk of losing coarser gold to the flotation tailings. This could potentially facilitate the recovery of approximately 15% of the gold either in the form of a gold-silver concentrate or as a doré, instead of recovering to the copper concentrate, which is economically more favourable. Other economic benefits include higher gold payabilities from a doré product compared to a concentrate product on the order of between 3.9 and 9.9%, minimal or no penalties, and quicker access to cash flow. The cost benefits, at a high level, outweigh the relatively low capital cost requirements for a gravity circuit and gold safe room, and negligible operating costs, and indicate an increase in revenue of approximately 4 to 10%.

1.8.5 Deleterious elements

Potential deleterious elements that can derive economic penalties include arsenic present in enargite/tetrahedrite that will be recovered reaching levels of up to approximately 3% into the copper concentrate; antimony present in tetrahedrite that may be present in the concentrate in amounts that may incur penalties; zinc mostly present in sphalerite that is present in higher quantities in certain parts of the deposit but has only a minor association with copper sulphides and pyrite; bismuth present in a wide range of minerals; cadmium, and mercury that are present in certain areas of the deposit. Grade control will be required to mitigate penalties.

Flotation testwork has been conducted of sufficient quality to parameterise the circuit including deriving the recoveries of minor elements, including deleterious elements, including the deportment of deleterious elements such as cadmium, mercury, zinc, bismuth, antimony, and arsenic, which may incur penalties although they are not deemed to pose a significant risk to the economic outcomes. There are no known processing factors or deleterious elements that could have a significant effect on the economic extraction of the ore that has not been considered and accounted for in the processing plan and economic model.

1.8.6 Metallurgical recovery

For design and economic analysis purposes, average metallurgical recoveries are estimated at 92.8% for gold, 88.8% for silver, and 92.8% for copper.

1.9 Mineral resource and mineral reserve estimates

1.9.1 Mineral resource estimate

The mineral resource estimate has been tabulated using a cut-off grade of 1.6 g/t Au, based on a gold price of $2,600 per ounce, an overall gold metallurgical recovery of 92.8%, a mining cost of $42 per tonne, a processing cost of $22 per tonne, a general and administration (G&A) cost of $20 per tonne, and an effective 3.7% gold royalty.

The mineral resource estimate is constrained within mineable stope optimizer shapes generated at a 1.6 g/t Au cut-off grade and using a 3.8 m minimum mining width, and is inclusive of material below 1.6 g/t Au within the shapes.

The Soto Norte mineral resource estimate effective August 18, 2025 is shown in Table 1-1.

Table 1-1 Soto Norte mineral resources effective August 18, 2025

Classification	Tonnes (Mt)	Gold grade (g/t)	Silver grade (g/t)	Copper grade (%)	Contained gold (Moz)	Contained silver (Moz)	Contained copper (Mlb)
Measured	3.8	7.99	36.8	0.25	1.0	4.6	21.4
Indicated	35.2	5.29	27.3	0.18	6.0	30.9	137.8
Measured + Indicated	39.0	5.55	28.2	0.19	7.0	35.5	159.2
Inferred	25.1	4.81	24.6	0.13	3.9	19.9	74.5

Notes:
- Totals may not add due to rounding.
- Mineral resources are inclusive of mineral reserves.
- Mineral resources are not mineral reserves and have no demonstrated economic viability.
- A gold price of $2,600 per ounce was used for the mineral resource estimate.
- The mineral resource estimate utilized a gold cut-off grade of 1.6 g/t.
- The mineral resource estimate was constrained within mineable optimizer shapes generated at a cut-off grade of 1.6 g/t Au and using a 3.8 m minimum mining width, and is inclusive of material below 1.6 g/t Au in the shapes.
- The mineral resource estimate was prepared by Kate Kitchen, MAIG of Mining Plus who is a qualified person as that term is defined by NI 43-101.
- There are no known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors or risks that could materially affect the mineral resource estimate or the development of mineral resources.

1.9.2 Mineral reserve estimate

The mineral reserve estimate for the Soto Norte Project effective August 18, 2025 is shown in Table 1-2.

Table 1-2 Soto Norte mineral reserves effective August 18, 2025

Classification	Tonnes (Mt)	Gold grade (g/t)	Silver grade (g/t)	Copper grade (%)	Contained gold (Moz)	Contained silver (Moz)	Contained copper (Mlb)
Proven	2.6	8.78	37.1	0.25	0.7	3.0	14.2
Probable	17.7	6.72	31.4	0.19	3.8	17.9	75.0
Proven + Probable	20.3	7.00	32.1	0.20	4.6	20.9	89.2

Notes:
- Totals may not add due to rounding.
- A gold price of $2,200 per ounce was used for the mineral reserve estimate.
- The mineral reserve estimate was constrained within mineable optimizer shapes and utilized a cut-off grade of 2.0 g/t Au.
- The mineral reserve estimate was prepared under the supervision of and reviewed by Peter Lock, FAusIMM, CP, of Mining Plus, who is a qualified person as that term is defined by NI 43-101.
- Other than as disclosed in this technical report, there are no known mining, metallurgical, infrastructure, permitting, or other relevant factors or risks that could materially affect the mineral reserve estimate or the potential development of the mineral reserves.

1.10 Mining operations

The planned mining method is longitudinal open stoping with waste rock fill, cemented rock fill, or paste fill to maximize reserve extraction while maintaining a safe, modern, and efficient mining environment, and is planned to have a production rate of 2,750 tpd, in line with the processing plant capacity dedicated to the underground mine. The paste fill will utilize

10.2 Mt of process tailings, just under half of the total tailings generated at the process plant. The total annual material movement of ore and waste has been limited to a maximum of 2 million tonnes.

A 5.5 m wide, 5.5 m high, 4 km long central decline from the surface utilizing the existing portal is planned for the main transportation route for personnel and equipment, and extends to the full depth of the mine design over a vertical range of 700 m. Ore and waste rock will be crushed underground then transferred to the surface by a service raise and then conveyed on the rope conveyor to the processing plant. The primary underground infrastructure, including ventilation shafts, material handling systems, and service facilities is strategically positioned along the central decline to support efficient operations.

The mine will be developed over a total strike length of 1.6 km at Mascota and 1.8 km at Gigante. From the central decline, access drives will extend to ore drives or footwall drives. Footwall drives in waste rock will be developed to establish ventilation circuits, for the transport of waste rock fill, and for access to lower grade stopes in the later stages of the mine. Where possible, a parallel ore drive will be developed in ore to access the stopes instead of a footwall drive in waste rock. This will allow access to lower grade stopes past the previously mined and backfilled higher grade stopes and reduce the overall development requirements. The use of parallel footwall or ore drives allows for multiple stoping fronts, ensuring a continuous and systematic extraction process with operational flexibility. Between 5 and 20 active stopes per month are required to sustain a processing rate of 2,750 tpd, depending on the contribution of development ore and the width of each stope. The stope dimensions have a 15 m length, with sub-level intervals ranging from 20 to 25 m in the upper areas of the mine and 30 m in the lower areas. A minimum stope width of 2.5 m was planned. 50% of the stopes are less than 5 m wide, 40% of the stopes are between 5 and 10 m wide, and 10% are greater than 10 m wide. The average stope comprises 6,000 tonnes. The stopes were divided into panels with vertical heights ranging between 90 and 100 m and scheduled using a bottom up mining plan within each panel.

The Project design has considered the requirement to avoid potential groundwater drawdown in the páramo and to protect the aquatic ecosystem of the La Baja Creek. To mitigate these risks, underground grouting is planned to minimize groundwater infiltration into the mine. Advanced cover drilling and grouting during mine development will be undertaken to enable early identification and pre-grouting to seal any water bearing structures to limit groundwater flows into the underground workings.

Any groundwater in the mine will be collected and managed in two separate systems, one for clean ground water and another for water that has come into contact with mining activities. Both streams will undergo water treatment before it is safely returned to the La Baja Creek, if required. During periods of low seasonal flow, treated water from the underground will be used to supplement the La Baja Creek, based on monitoring data.

1.11 Processing and recovery operations

The processing plant is designed to treat plant feed throughputs of a nominal 2,750 tpd and a maximum 3,500 tpd. The processing plant will receive 2,750 tpd of crushed run of mine ore from the Soto Norte underground mining operation, and can receive an additional 750 tpd of mill feed purchased from contract mining partners, to produce three saleable products and one waste product, including a gold concentrate from a gravity gold operation, a copper concentrate from a copper flotation operation, a pyrite concentrate from a pyrite flotation operation, and a tailings product for disposal in the filtered tailings facility and to create paste backfill for the underground mine. For design and economic analysis purposes, average metallurgical recoveries are estimated at 92.8% for gold, 88.8% for silver, and 92.8% for copper.

No cyanide or mercury will be used at the Soto Norte processing facilities. The processing circuit comprises primary ore crushing at the underground mine, ore transport from the mine portal to the process plant on the rope conveyor, receipt of mill feed purchased from contract miners at the process plant, primary grinding in a SAG mill with a supporting pebble crusher, secondary ball mill grinding, an upfront gold gravity recovery circuit to recover up to 15% of the coarse gold-silver particles and to produce a filter cake, and a two stage sequential flotation circuit to recover fine gold, silver, and copper in separate copper and pyrite concentrates.

A target grade of 16% for the copper concentrate has been set on the basis of testwork results as well as considerations for maximizing the gold content in the copper concentrate and reducing penalty element concentrations. Potential penalty elements considered in the economic analysis include arsenic, bismuth, cadmium, antimony, and likely zinc, with payments for contained copper and gold anticipated to be far in excess of any potential penalties. The non-sulphide waste content of the concentrates will be restricted to 10%.

1.12 Infrastructure, permitting, and compliance activities

1.12.1 Mine infrastructure

The underground infrastructure will include:

- a centrally positioned workshop strategically located off the decline and fully equipped for maintaining all underground equipment;
- an explosives magazine strategically located adjacent to the workshop for the storage of ammonium nitrate and fuel oil (ANFO) and separate secure storage compartments for detonators to ensure the strict segregation from other explosives materials. A fully automated fire suppression system will be installed and configured to activate immediately upon detection of smoke. The explosives magazine will be designed, constructed, and operated in compliance with the Colombian explosives regulations;
- a permanent fuel storage bay with the capacity to store 10,000 litres of diesel and 2,000 litres of hydraulic oil and temporary skid mounted fuel storage units positioned on active levels to facilitate efficient refuelling;
- chambers designed for servicing all underground equipment;
- a rock crusher chamber for reducing the ore and waste size;
- an underground rope conveyor system to transport crushed waste and ore rock to the surface;
- a paste fill plant for backfill and ground support operations and a paste fill reticulation system to deliver the paste fill to the stopes;
- wash bays for cleaning mobile equipment;
- lunchrooms and shift supervisor offices located on the primary intake ventilation circuit that can be used for safe firing areas and fresh air bases;
- stores for the supply of spare parts and consumables;
- ore and waste passes to streamline the handling of ore and waste material;
- sumps and pump stations for effective water management and dewatering operations;
- service holes to facilitate the routing of utilities and services; and
- portable or fixed refuge chambers installed at intervals such that the workers are never more than 750 m away from the nearest refuge chamber to take refuge in the event of a mine emergency, and escape ladderways installed in a dedicated central escapeway, to provide a second means of egress in the event of a mine emergency.

1.12.2 Rope conveyor system

The rope conveyor system is a proven, efficient, low footprint, low impact, and state of the art solution for material and cargo transportation. Environmental and community impact benefits include its silent operation, the elimination of road construction and related traffic and dust and exhaust emission issues, and minimal land disturbance restricted to the tower bases. The covered conveyor minimizes dust generation as material remains stationary during transport, and the system allows for straightforward closure and site rehabilitation at the end of its life. Economic and operational benefits include low capital and operating cost requirements, and relatively simple construction logistics. The straightforward design has only a limited number of moving parts, reducing the potential for defects, minimizing inspections and maintenance, and increasing operational availability. The majority of the maintenance can be carried out in a safe environment at the station.

The planned Doppelmayr designed bi-directional rope conveyor system has a nominal capacity of 3,000 tpd and a maximum capacity of 5,500 tpd for transporting ore and waste rock from the mine portal to the processing plant, a nominal capacity of 1,080 tpd and a maximum capacity of 2,400 tpd for transporting tailings from the processing plant to the mine for use as backfill, and a capacity of 140 tpd for the transport of consumables from the processing plant to the mine. The system will be used to carry the high voltage power and fibre optic cables. The automated system will have a length of 7,100 m.

1.12.3 Process support facilities

The process support facilities will include a waste receiving area at the rope conveyor station, dedicated receiving areas for underground ore and mill feed purchased from contract mining partners, the milling circuit, the process plant, services, accommodation, storm water pond, roads, the main power supply, and an existing military base.

The ore and waste from the underground mine will be transported from the mine portal to the processing facilities on the rope conveyor. Waste material will be diverted to a stockpile and then transported by truck for the construction of the filtered tailings storage facilities and for use as engineered fill in terraces.

The milling circuit facilities will include the ball and SAG mills, conveyors, pebble crushers, and space for a gold safe room. The process facilities will include the flotation circuits, reagent storage facilities, water treatment facilities, power supply, a weigh bridge, thickening and filtration facilities, roads, and a tailings pump line. A dedicated, approximately 6 km long route is planned for the tailings line with spillage risk mitigation features including a containment channel and access road for maintenance, and will also be used for an overhead line supplying power from the main substation to the water intake, filter plant, and ancillary equipment.

The service area will include workshop, warehouse, mobile equipment workshop, outdoor storage yard, fuel station, offices, wash bay, diesel storage, solid waste disposal yard, modular sewage treatment facility, access gate, and soccer field.

The accommodation area will include the accommodation blocks to house 170 people, mess hall, entertainment area, gym, clinic and emergency response building, and core sheds.

1.12.4 Filtered tailings storage facilities

Multiple potential tailings storage facility locations have been considered and engineered for the Project over time. Currently, the facility location has shifted from the 2021 feasibility study location and its development is now planned for one of the past alternative sites, in a valley surrounded by steeply sloped mountainous terrain, approximately 3.5 km to the south of the processing facility. The valley side slopes are moderately steep, while the slopes along the valley bottom are shallower. The location is primarily in a grassland area, with areas of dense shrubs and trees present along the upper reaches of the tailings basin.

The Project and the facility site are located in a high seismic region. A site-specific seismic hazard evaluation for the Project has been undertaken and the analyses were utilized for the current prefeasibility study engineering.

Knight Piésold developed a water balance model using climatic and hydrologic data to simulate water transfers and storages. The water balance simulations represent a range of possible flows and volumes over time. A sensitivity analysis was performed to estimate the variation of maximum volumes required to be stored in the contact water collection pond. Hydrologic analyses were undertaken by Knight Piésold to estimate peak flows for the design of hydraulic structures.

Deterministic, limit equilibrium slope stability analyses were also advanced by Knight Piésold on the facility embankment, tailings stack, and contact water collection pond dam under a variety of loading conditions. The findings from the stability assessment were incorporated into the facility design to confirm acceptable slope stability.

Knight Piésold conducted prefeasibility study engineering of the filtered tailings facility and associated contact water collection pond, haul road, and surface water management structures. The civil design components of the facility include the foundation and embankment abutment preparation, embankment, tailings stacking plan, geosynthetic liner system, drainage systems, contact water collection pond, and ancillary infrastructure including roads and surface water management channels. The facility design meets slope stability requirements as guided by the Canadian Dam Association and the International Committee on Large Dams, standards that are internationally recognized as best practice and provide detailed technical design and risk management criteria. The facility has a capacity of 11.8 million tonnes, sufficient to accommodate the current mineral reserve requirements of 10.3 Mt of combined process tails and waste rock. Additional storage capacity may be required in the future, depending on the amount of any future mineral reserves discovered through exploration drilling programs and on the volume of material purchased from contract mining partners.

The facility will comprise a tailings basin formed by natural topography, excavations into existing ground, placement of grading fills, and containment embankment at the downstream end of the tailings basin constructed of structural fill. The entire tailings basin and most of the upstream face of the containment embankment (all of the surfaces that will be in contact with tailings) will be lined with a composite liner comprising geomembrane overlying a layer of compacted low-permeability soil. A temporary tailings storage area is planned for the temporary placement of tailings when rainy field or upset plant conditions prevent the acceptable placement of tailings.

A 2 km long haul road will extend from the existing public road to the facility embankment crest. The haul road is configured to provide containment and act as the downstream retaining embankment for the contact water collection pond. A 2.3 km long facility perimeter access road will run along the facility perimeter to allow access to the facility for operations and maintenance.

The surface water management strategy for the facility is to construct non-contact diversion channels to limit surface water runoff over the tailings and embankment surfaces during the operations and passive closure phases. The primary drainage systems will comprise underliner and overliner drainage systems.

The contact water collection pond will have a minimum storage capacity of 70,000 cubic metres and will be located downstream of the embankment. An emergency spillway is included in the design. The upstream face of the collection pond dam and its pond area will be lined with a composite liner comprising a low permeability subbase overlain by a geomembrane, which will extend to the average maximum monthly pond elevation. Above the maximum monthly pond elevation, the pond will be soil lined. All lined surfaces will be overlain by a protective soil layer and articulated concrete blocks. The water in the collection pond will be pumped to the water treatment plant.

Non-contact stormwater outside of the facility footprint will be routed around the outer perimeter of the facility via two non-contact diversion channels that will discharge downstream of the collection pond to the Suratá River in compliance with the discharge permits. The water diversion channels are designed to divert up-gradient runoff around the facility to limit water from running onto the tailings stack and embankment.

The facility will store filtered tailings from the processing plant and potentially acid generating waste rock from the underground mine. A system of portable conveyors known as grasshoppers will transport the filtered tailings from the filtration plant to the perimeter of the tailings facility, from where they will be loaded in trucks for hauling and deposition to the tailings basin. Truck hauling of the filtered tailings will be utilized when the conveyors are under maintenance.

To support with geotechnical stability and to limit long term settlement of the tailings, the filtered tailings will be placed and compacted. The in place moisture content will likely be less than approximately 14%. The tailings will be placed in horizontal lifts, starting at the upstream toe of the embankment and extending up the valley.

Potentially acid generating waste rock will be hauled to the facility by truck and placed in trenches excavated into the compacted tailings stack, then immediately covered with compacted filtered tailings.

Knight Piésold developed a closure plan to provide a long term solution for a safe, structurally stable, and non-erodible cover for environmental sustainability. The facility closure plan was guided by federal regulations and guidance by the Autoridad Nacional de Licencias Ambientales (National Authority of Environmental Licenses, ANLA), and wherever that guidance was inapplicable, local regulations and or international standards were considered. Chemical and physical stability of the facility will be maintained by careful consideration of directing and managing surface water runoff and by designing a cover system that will effectively control meteoric infiltration into the tailings facility and perform appropriately with little to no maintenance. The tailings basin will be closed by progressively placing a cover system over the tailings as operations proceed. The cover system will comprise an upper revegetated layer or growth media overlying a compacted clay zone which will help prevent wind or water from eroding the tailings and will reduce water infiltration into the tailings.

Knight Piésold developed a closure and post closure monitoring plan consistent with the facility’s current design, including the required geotechnical instrumentation, the installation locations, monitoring, and maintenance.

1.12.5 Water sources and management

The Project has been designed with robust water management and protection as guiding principles. There is sufficient and readily available water within the Project area for all aspects of the operation including mining, processing, the camp, and other activities. Construction water will be sourced under permits from the La Baja Creek and Suratá River, which both have abundant water supply. During operation, the main source of water for the underground mine and its associated infrastructure will be from groundwater and the main source of water for processing will be from the adjacent Suratá River.

The Project plans to reuse, recycle, and return approximately 96.5% of the Project’s total water requirements back to the environment, resulting in a net water use estimated at 3.5%. The majority of the demand will be supplied by the Suratá River, which will supply approximately 2.8 litres per second of net make up water for the processing plant, representing

0.22% of the average flow of the Suratá River at the planned water access point near the village of Suratá, and 0.08% of the average flow of the Suratá River at Bucaramanga, the nearest city located 55 km downstream from the Project.

The Project includes a comprehensive water management plan that incorporates plans and programs tailored to each water application or activity with significant interaction with water to ensure that both the quality and quantity of local water resources are preserved throughout the life of the operation, and all potential impacts related to water use have been thoroughly identified, assessed, and addressed through prevention, mitigation, and compensation measures. The underground mine has been designed to minimize groundwater flows into the underground workings through advanced cover drilling and grouting ahead of mine development to identify and seal any water bearing structures before mining reaches them, greatly reducing potential inflow, and to manage, treat, and if required, safely return any captured water to the environment in compliance with the environmental standards and discharge permits. The process plant has been designed to minimize the use of water and to recycle both process water and the water obtained from the dewatering of the process tailings. The filtered tailings storage facility has been designed to divert rainwater runoff away from the facility and to drain and manage any subsequent dewatering of the tailings and any rainwater falling onto the facility. Continuous monitoring of water flow and quality will be undertaken with defined measures planned if any changes are detected. These strategies will help ensure that the current water flow rates within the ecosystem are maintained.

Water from mine dewatering, seepage collected from the filtered tailings facility, and process water streams will report to water treatment plants where each stream will be treated separately. Treated water from the domestic wastewater and industrial wastewater treatment plants located at the mine will discharge at La Baja Creek, in compliance with the discharge permits. Treated water from the other domestic and industrial wastewater treatment plants will be discharged to the Suratá River, all in compliance with the discharge permits.

Precipitation runoff from clean water areas will be either diverted to the surrounding environment or channeled to a designated location, where it converges with other catchment areas. The collected water will be directed towards the nearest river through controlled drainage pathways, in compliance with the water discharge permits. Grey water run off will be collected separately and diverted to a collection pond. A water treatment plant will then process the water which will then be either pumped to the process plant or safely discharged into the environment in compliance with the water discharge permits.

1.12.6 Power

The estimated power requirements are 13 MW at the process plant and 10 MW at the underground mine, for a total Project requirement of 23 MW. An existing 34.5 kV power line from Bucaramanga supplies a 5 MVA at 34.5 kV capacity substation at the underground mine, which can be used for construction power and as an alternative emergency power source, but is insufficient for operations.

Electrical power to the operations will be supplied from the Palos 115 kV substation in the Bucaramanga area via a 34 km long 115 kV, 45 MVA capacity overhead line leading to a new substation at the process plant, where the voltage will be stepped down to 13.8 kV for reception by the mine’s main distribution substation. The main distribution station will supply power via 13.8 kV cables to the process plant, by 13.8 kV overhead line to the filtered tailings facility and water intake plant, and by a 7 km long 13.8 kV cable installed on the rope conveyor to service the underground operation and the underground substation.

Standby and emergency power supply will be provided by a 3.125 MW diesel generator station at the process plant, a 630 kW diesel generator station at the filtered tailings facility, a 250 kW diesel generator at the water intake plant, a 250 kW diesel generator at the emergency ponds, and by a 2.5 MW diesel generator station at the underground mine.

1.12.7 Environmental factors

The Property is almost entirely located in a mountain ecosystem within a tropical humid climate zone, characterized by forests of medium sized trees of less than 20 m in height. Endemic and conservation status species identified in the Project area of influence include plant species of regional and/or national concern, and six native fish species in the Suratá River. The Project area of potential influence does not contain any sensitive or strategic ecosystems, but it does include priority conservation areas designated by the National Council for Economic and Social Policy. These areas show signs of vegetation degradation and transformation. The majority of the Project is located on land that shows pervasive anthropogenic alteration over time, and observations of wildlife and aviation fauna are almost non-existent. Upon closure, land reclamation and re-establishment of vegetation and soil profiles are not expected to be a challenge. Outside of the Project area, the most notable strategic ecosystem is the Santurbán páramo. Based on the current delimitation of the Santurbán páramo, the Project footprint is located 600 m horizontally from the Santurbán páramo and approximately 350 m in elevation below it. The Colombian government is still in the process of redefining the Santurbán páramo boundaries, and while these distances may change, the Project is outside of the Santurbán páramo boundaries.

The Project’s planned processing plant, filtered tailings facility, and associated water treatment facilities are located within the La Baja Creek catchment and adjacent minor tributaries of the Suratá River. Larger stream flows are sustained by groundwater discharge and are responsive to rainfall events. Groundwater in the area is complex and influenced by geology and structures.

Numerous environmental and social baseline studies and monitoring programs have been undertaken since 2013 and are ongoing as further information is required to support the Project development plans.

The Project’s attention to reducing the environmental effects of the proposed Project embodies local knowledge and the results of options analyses to find environmentally appropriate and cost-effective solutions to de-risking impacts of the Project. The Project has developed a detailed environmental and social management program to guide the Project’s activities during construction, operation, and closure, consistent with the environmental and social impact assessment (ESIA) process outcome. Each plan has a monitoring component and adaptive management process to evaluate the plan effectiveness and inform updates as required, and reporting requirements to regulators, communities, and stakeholders. The plan components are at various stages of development and will be validated by the community of the area of influence. The components cover topics including biotic and abiotic programs, socioeconomic programs, landscape management and control programs, and an emergency response and readiness plan. Community validation of the programs is ongoing, ensuring local input into final designs.

Colombian law requires that a value of not less than 1% of the value of the Project’s capital expenditure and associated development costs must be invested in environmental and/or sustainability related projects. Funds from the 1% investment plan will be managed by the Corporación Autónoma Regional para la Defensa de la Meseta de Bucaramanga (CDMB) based on the programs jointly identified with PSN. The total capital expenditure associated with the development plan includes expenditures that exceed the 1% requirement.

A mining concession holder is liable for environmental remediation and other penalties that may arise as a result of the concession holder’s actions or omissions occurring after the date the concession contract is awarded. Concession holders are not, however, responsible for environmental liabilities associated with historic artisanal or unauthorized workings. However, the CDMB has requested that the Project address the restoration of certain historic workings. PSN is and will continue to keep working with the regulatory authorities to remediate damage wherever possible.

Water quality within the titles is monitored at points that include areas of historical process plants and artisanal mining tunnels. Historic mines have been sealed off as part of a mine closure program, and there is an ongoing program of disassembling process plants and removing contaminants left behind due to past mining and processing activities. These remediation efforts will continue with the development of the Project.

1.12.8 Permitting factors

The Project holds a mining license granted by the ANM in 2018 and amended in 2021 for mining title 095-68 covering the mineral resources and reserves, the planned underground mining infrastructure, and the surface infrastructure at the mine area. The Project holds the licenses it requires for the current phase of the Project, which authorize occupancy of the land, the use of water for drilling, potable water usage, and water treatment and discharge.

The key permissions required to commence Project construction and operation are the approvals of the ESIA to obtain the environmental license, completion of the resettlement obligations as set forth in the environmental license, and the amendment and approval of the existing approved works and construction program (PTO, the Programa de Trabajo y Obras) to reflect the environmental license conditions.

PSN will submit an updated ESIA to the CDMB, outlining the Project's description as contemplated in the current prefeasibility study. These changes will require additional studies including a re-evaluation of environmental and social impacts, and restart of the environmental permitting process and timeframes.

Once approved, the environmental license is valid for the life of the Project, subject to compliance audits by the environmental authority. The license may be modified as the Project evolves.

On January 30, 2024, the Colombian Ministry of Environment issued Decree 044 which allows the Ministry to declare temporary reserve areas in certain parts of Colombia. To establish a temporary reserve area, a resolution must be issued by the Ministry detailing the area that is to be temporarily reserved. The Ministry issued Resolution 221 of 2025, amended by Resolution 239 of 2025, by means of which it declared a TRA in the Soto Norte region. The TRA will be in effect for two years, with a possible two year extension. While the TRA is in force, no new concessions or environmental permits may be granted by the mining or environmental regulators. During this period, the Ministry of Environment and Sustainable Development of Colombia must conduct environmental studies to determine whether to make the reserve area permanent. Notwithstanding the TRA, the Soto Norte Project may continue environmental studies, provided no environmental permit is required. Decree 044 and the TRA may delay licencing of the Soto Norte Project.

Decree 044 and the TRA resolutions are presently being challenged in administrative courts, with actions led by the Colombian Disciplinary Office, artisanal and small mining units, the Colombian Mining Trade Association, and the National trade association.

Additionally, the Administrative Tribunal of Santander issued a ruling in July 2025 in a class action proceeding recognizing the Santurbán páramo as a subject of personal rights and designating the Ministry of Environment as its legal guardian. While there is no direct impact on the Soto Norte Project, environmental licensing proceedings for the Soto Norte Project may be delayed or hindered because the Tribunal ordered that: (i) the Ministry of Environment must actively participate and protect the páramo in any licensing process, including through the use of administrative injunctions (ii) all relevant environmental authorities must identify critical transition areas to the páramo in the Soto Norte region for water protection, and (iii) zoning regulations must exclude mining activities in “buffer zones” in alignment with the 2014 delimitation process. The ruling was appealed.

The Soto Norte Project remains several years away from development. With the completion of this technical report, PSN intends to present a fully redesigned project to the Colombian regulators following the conclusion of the environmental and technical studies currently underway.

1.12.9 Social or community factors

Mining was undertaken by the indigenous Sura people in the California – Vetas mining district in Pre-Columbian times and later by the Spanish who produced gold for two and a half centuries, as well as by English and French companies in the early 1800s and 1900s. Colombia continues to have an active artisanal and small-scale mining sector, with traditional miners across the country engaging in small-scale gold extraction, often in remote regions. This sector plays a significant role in local economies and provides livelihoods for many communities.

The municipalities closest to the Project include California, Suratá, and Matanza, which have a combined population of approximately 11,500. The economy of the province of Soto Norte is based on agriculture, mining, and forestry related activities. The economy of California, located closest to the Project, is dominated by artisanal and small scale mining, and the economies of Suratá and Matanza are dominated by ranching, agriculture, and forestry.

The Project is proactively managing the intersection of communities and the Project elements through its community engagement model. In 2023, the Project launched a new strategy to implement best practices in community engagement across the Soto Norte region. The Project team and community leaders and authorities have collaborated to develop social agreements that enable the joint identification of needs and the evaluation of solutions in a coordinated and structured manner. This enables a shared understanding of the Project's role in driving sustainable development in the region and empowers the communities to decide on resource allocation and to propose projects, initiatives, or services to improve their quality of life. Each shared initiative is designed to prevent and mitigate potential disruptions to the Project's operations, facilitate the acquisition of the necessary permits for land access to ensure the completion of required studies, foster a favourable socio-political environment in the area of influence, supporting the Project's milestones, streamlining the permitting process, and to effectively address environmental challenges.

The community widely regards the Project social agreement as a successful model, and is celebrated for its accomplishments in implementing projects and programs that have positively impacted local communities. The model has achieved community integration, effective collaboration between leaders and residents, community empowerment, and accountability for outcomes through collective efforts. The agreements play a vital role in strengthening relationships by encouraging their members to advocate for the Project within the broader community, and ensure a closer and more effective dialogue channel, enhance positive perceptions of the Project, and foster stakeholder trust, all of which are critical elements in de-risking social support for the Project and instilling confidence in the Project's success.

The strategic engagement model includes the social agreement commissions, a sponsorship plan, social houses, information and socialization forums to deliver timely and transparent information about the Project's operations and activities, and regional and national engagement beyond the area of influence. The sponsorship plan involves assigning Project social team members to specific villages and sectors within the communities of California and Suratá to maintain systematic engagement with families and local leaders, to foster a deeper understanding of the communities, address concerns, gather valuable ideas, and counter misinformation and mitigating factors that could impact social management and the Project's reputation. The social houses in California, Suratá, and Matanza have become central hubs for activities and meeting points, and serve as the primary platform for receiving community requests.

Development of the Project involving the municipalities of California, Suratá, and Matanza will provide a diversity of employment and socioeconomic opportunities to residents. The Project will require skilled mine workers, services, material suppliers, contractors, housing, health, education, and skills training. Collaboration with contract mining partners is an integral component of the Project development. The direct income benefits of the Project will result in opportunities for indirect benefits such as support to local business, career opportunities for young adults, investment in non-mining related enterprises, and traditional agricultural, cultural, and artisan pursuits.

The Project currently employs 53 people, of whom 40 are from local communities and 25 are female. The peak workforce during Project construction is estimated at 2,292, mainly comprised of contractors, plus 90 administrative staff and management. During operations the workforce is estimated at 676 company personnel. The Project is targeting 60% of the workforce to be hired from the local community, 20% from the department of Santander, and 18% from other departments in Colombia. Foreign technical and managerial specialists will eventually make up 2% of the workforce. The socioeconomic benefits of the Project will also affect the broader region including the city of Bucaramanga for employees, suppliers, and contractors.

Aris Mining collaborates with small-scale miners, known as contract mining partners, to create mutually beneficial partnerships that support the host communities. This partnership model includes the formation of formal companies that employ between 25 and 500 people as well as mill feed agreements such as those at Aris Mining's Segovia and Marmato mines that comprise long term contracts to supply mill feed for Aris Mining's processing plants, with payments based on gold content, grade, and the spot gold price. These agreements provide the contract mining partners with access to Aris Mining's technical, operational, and safety expertise as well as working capital financing. Aris Mining provides training programs in health and safety, environmental stewardship, accounting, compliance, and business management. Other benefits for the contract mining partners include access to social security and legal protections, government benefits, financial services, and broader market opportunities.

A suitable area within the Project titles was identified for the contract mining partners and in 2021 PSN entered into a four year subcontract with a group of small scale miners known as Calimineros S.A.S. to perform small mining activities covering 0.51 ha within title 125-68, which is located on the eastern boundary of the main 095-68 title. The Calimineros project and its mining plan were approved by the National Mining Authority, and Calimineros submitted a request to obtain an environmental license, and the approval process is pending. An extension of the subcontract was submitted to the National Mining Authority in the second quarter of 2025.

Additional land acquisition required for the construction and operation of the Project will result in resettlement for some members of the surrounding communities. Based on the Project footprint, 108 properties have been identified that will require either acquisition or an easement, provisionally affecting 198 households. Relocation areas to enable continuity of livelihoods are under evaluation. Where easement cannot be agreed upon, they may be imposed by judicial order. If purchase agreements cannot be mutually agreed, expropriation authorization must be obtained from the National Mining Agency prior to starting the expropriation proceedings. The preferred method for acquiring land at Soto Norte for project development will be to reach agreements with landowners following receipt of the environmental permits

Resettlement has been identified as the most significant impact of the Project and therefore is a key focus of the management programs. A draft resettlement action plan (RAP) has been developed to guide the resettlement process. The RAP will be implemented in compliance with Colombian regulations, subject to approval by the environmental authority, will only commence following issuance of the Project's environmental license.

To mitigate risks of delay to the construction schedule, the Project has developed strategies to work collaboratively with affected households during the land acquisition and resettlement process.

1.13 Capital and operating costs

1.13.1 Capital cost estimates

The capital cost estimates have an accuracy of +/-25%, suitable for the prefeasibility study level. A summary of the estimated initial capital expenditures, including any operating costs incurred during the pre-production period, is shown in Table 1-3 and a summary of the estimated deferred and sustaining capital costs are shown in Table 1-4.

Table 1-3 Estimated initial capital costs

Initial capital expenditure	Amount ($M)
Mining	
Mobile equipment	24.2
Fixed equipment	10.3
Lateral development	10.4
Vertical development	0.8
Pre-production operating costs and first fills	9.8
Mining total	**55.5**
Surface	
Infrastructure	172.4
Rope conveyor	74.8
Process plant	95.7
Owner's, indirects, and first fills	21.5
Engineering, procurement, and construction management (EPCM)	35.4
Replacement capital	-
Surface total	**399.8**
Other costs	
Resettlement and environmental monitoring	72.8
Electricity supply down payment	0.2
Other start-up costs	8.8
Capitalized VAT	34.2
Contingency	54.0
Other Costs Total	**170.0**
Total	**625.2**

Table 1-4 Estimated deferred and sustaining capital costs

Deferred and sustaining capital expenditure	Amount ($M)
Mobile equipment	138.0
Fixed equipment	68.1
Lateral development	117.5
Vertical development	31.2
EPCM	3.6
Replacement costs	4.2
Other costs	1.1
Total	**363.6**

1.13.2 Operating cost estimates

The life of mine operating costs, excluding capitalized operating costs, were estimated for underground mining, surface including processing and the rope conveyor, G&A including other costs, realization, and royalties. Royalties due to the Colombian state include a 4% royalty on 80% of the gold and silver produced and a 5% royalty for copper on 100% of the copper produced. The summary of the estimated life of mine operating costs is shown Table 1-5 and the estimated life of mine unit operating cost estimate is shown in Table 1-6.

Table 1-5 Estimated operating costs

Item	Total life of mine ($M)	Pre-production ($M)	Production ($M)	Post-production ($M)
Mining				
G&A	9.3	0.1	9.2	-
Contracts	21.6	0.4	21.1	-
Mine labour	62.8	0.4	62.4	-
Equipment maintenance and operation	236.2	1.4	234.8	-
Power	125.9	0.3	125.7	-
Diesel	36.5	0.2	36.2	-
Explosives	32.8	0.1	32.7	-
Ground support	266.6	1.4	265.2	-
Drilling consumables	48.0	0.3	47.6	-
Mine services	25.1	0.2	25.0	-
First fills	4.9	4.9	-	-
Mining total	**869.6**	**9.8**	**859.8**	**-**
Processing and surface				
Labour	94.8	-	94.8	-
Reagents	47.3	-	47.3	-
Power	214.8	-	214.8	-
Plant maintenance	49.3	-	49.3	-
Rope conveyor	10.1	-	10.1	-
Plant consumables	8.3	-	8.3	-
Processing and surface total	**424.7**	**-**	**424.7**	**-**
Realization				
Treatment charges	17.0	-	17.0	-
Refining charges	47.5	-	47.5	-
Penalties	81.8	-	81.8	-

Item	Total life of mine ($M)	Pre-production ($M)	Production ($M)	Post-production ($M)
Freight	244.0	-	244.0	-
Realization total	390.3	-	390.3	-
Mine site G&A	**395.1**	**8.8**	**386.3**	-
Environmental management plan	**69.8**	**18.1**	**42.2**	**9.6**
Royalties	**393.3**	**-**	**393.3**	-
Total	**2,542.8**	**36.7**	**2,496.5**	**9.6**

Table 1-6 Estimated life of mine unit operating costs

Unit operating costs	Life of mine $/t ore
Mining	41.70
Processing	20.59
Other (G&A, environmental management plan, etc)	21.24
Treatment, refining, and shipping	18.93
Royalties	19.07
Total	121.54

1.14 Economic analysis

This economic analysis was undertaken to assess and confirm the current mineral reserve estimate disclosed in this technical report, utilizing the production schedule and the capital and operating cost estimates. The economic analysis has been conducted on a post-tax, 100% equity (i.e., no debt financing) basis, in constant dollar terms. Sunk costs, such as exploration and the cost of previous studies, were excluded from the analysis.

The economic viability of the mineral reserves has been evaluated using key economic indicators, including annual and cumulative cash flows, NPV, and IRR. The NPV presented in this technical report should not be interpreted as the definitive value of the Project and must be considered in conjunction with the accompanying sensitivity analysis.

The economic analysis incorporates a statutory corporate income tax rate of 35%. The key economic results are presented on a pre-tax basis to facilitate comparison with other projects in different jurisdictions by removing the effect of local tax regimes, and on an after-tax basis incorporating the applicable tax rates and fiscal terms for the Project, providing a more accurate reflection of the potential economic benefit to the Project owners.

The processing facility has been designed with a 3,500 tpd capacity, but the Project underground mining production schedule has been constrained to 2,750 tpd with the additional 750 tpd dedicated for potential mill feed purchases from contract mining partners. The financial projections in this technical report do not account for revenue, operating costs, or profit margins from the dedicated capacity.

The construction period is scheduled for 13 quarters (3.25 years). The first underground ore production from development activities is planned during the final six months of Project construction, with all material stockpiled until the process plant is commissioned. The pre-production stockpile is scheduled to supplement run of mine ore feed during the first six months of Year 1, supporting the production ramp up.

Plant throughput will ramp up progressively to the 2,750 tpd capacity, reaching steady state operations toward the end of Year 2. Based on the current mineral reserve estimate, the mine life extends to Year 22.

The total life of mine production is shown in Table 1-7 and the total life of mine concentrate production is shown in Table 1-8.

Table 1-7 Total life of mine production

Mining	Units	Total
Waste	kt	5,380
Development ore	kt	3,501
Stope ore	kt	17,119
Total material mined	kt	26,000
Mined gold grade	g/t Au	6.98
Mined silver grade	g/t Ag	32.0
Mined copper grade	% Cu	0.20
Contained mined gold	koz	4,627
Contained mined silver	koz	21,216
Contained mined copper	Mlb	90.6

Table 1-8 Total life of mine concentrate production

Concentrates	Units	Copper concentrates	Pyrite concentrates	Gravity gold concentrates	Total
Mass	DMT[1]	169,184	1,441,402	615	1,611,201
Contained metal in concentrates					
Gold	koz	2,092	2,103	105	4,299
Silver	koz	9,305	9,529	1	18,834
Copper	Mlb	67.4	16.6	Nil	84.0
Concentrate grade					
Gold	g/t	385	45	5,307	
Silver	g/t	1,711	206	32	
Copper	%	18	0.52	Nil	

Note 1. Dry metric tonnes

The financial analysis utilized the following metal price assumptions for the base case:

- Gold: $2,600/oz
- Silver: $29/oz
- Copper: $4.30/lb

These metal prices were selected as being in line with the median of the long term forecasts of a group of banks and financial institutions, as at the end of August 2025.

The results of the economic analysis are summarized in Table 1-9. The economic analysis excludes any contribution from the 750 tpd processing capacity dedicated for contract mining partners, which is intended to support regional formalization initiatives and environmental improvements. The NPV at a range of discount rates is shown in Table 1-10.

Table 1-9 Soto Norte economic evaluation results

Key indicators	Units	Total
Total gold in concentrates life of mine	koz	4,299
Initial life of mine at an owner-mining rate of 2,750 tpd	Years	22
Average annual gold production (years 2 to 10)	koz	263
Average annual gold production (years 1 to 21)	koz	203
Life of mine average cash cost	$/oz Au	345
Life of mine average all in sustaining cost	$/oz Au	534

Key indicators	Units	Total
Average EBITDA (years 2 to 10)	$M	547
Average annual EBITDA (years 1 to 21)	$M	410
Summary cash flow for the life of mine ($M) at $2,600/oz gold price		
Revenue from payable gold sales		10,403
Less: royalties		393
Less: operating costs, net of by-product silver and copper		1,381
Less: sustaining capital		364
Operating margin		8,265
Less: income tax		2,630
After-tax cash flow		5,635
Less initial capital including pre-production costs, VAT, and contingency		625
Less: closure costs		25
Net cash flow		4,985
Pre-tax indicators at $2,600/oz gold price		
NPV at 5% discount rate	$M	4,203
IRR	%	45.8
Payback period (from start of operations)	Years	1.9
After-tax indicators at $2,600 gold price		
NPV at 5% discount rate	$M	2,680
IRR	%	35.4
Payback period (from start of operations)	Years	2.3
After-tax indicators at $3,200/oz gold price		
NPV at 5% discount rate	$M	3,559
IRR	%	42.1
Payback period (from start of operations)	Years	2.0

Table 1-10 Sensitivity of NPV to discount rate

Discount rate	Units	Pre-tax NPV	After-tax NPV
0.0%	$M	7,615	4,985
5.0% (base case)	**$M**	**4,203**	**2,680**
10.0%	$M	2,481	1,519

The sensitivity of the after-tax $NPV_{5\%}$, after-tax IRR, and after-tax payback period to a range of gold prices is shown in Table 1-11.

Table 1-11 Sensitivity of key economic indicators to gold price

Gold price / Indicator	$2,000/oz	$2,200/oz	$2,400/oz	**$2,600/oz Base case**	$2,800/oz	$3,000/oz	$3,200/oz
After-tax $NPV_{5\%}$ ($M)	1,800	2,093	2,387	**2,680**	2,973	3,266	3,559
After-tax IRR (%)	27.7	30.4	33.0	**35.4**	37.8	40.0	42.1
Payback period (years)	2.8	2.6	2.5	**2.3**	2.2	2.1	2.0

The result of the economic analysis indicates that the Project is economically viable under the base case assumptions, based on the current mineral reserve estimate and the assumptions described herein. At a $2,600 per ounce gold price, the after-tax $NPV_{5\%}$ is $2.7 billion, the after-tax IRR is 35.4%, and the payback period is 2.3 years from the start of processing operations.

The analysis excludes any contribution from the 750 tpd processing capacity dedicated for contract mining partners, which is intended to support regional formalization initiatives and environmental improvements. The economic results are not a measure of fair market value.

1.15 Exploration, development, and production

There are no current exploration or production plans for the Project. The Soto Norte Project remains several years away from development. Following the conclusion of the environmental and technical studies at the Soto Norte Project, PSN will submit an updated ESIA to the CDMB, outlining the Project's description as contemplated in the current prefeasibility study. These changes will require additional studies including a re-evaluation of environmental and social impacts, and restart of the environmental permitting process and timeframes.

1.16 Conclusions

This prefeasibility study confirms the technical, economic, and community benefits of the Project. The design prioritizes the protection of the environment, water resources, and the health and wellbeing of the local communities.

The underground mine is designed to produce 2,750 tpd of ore using safe, modern methods. The process plant is designed for 3,500 tpd to accommodate an additional 750 tpd of mill feed purchased from contract mining partners from the local communities. The final products will be copper concentrates, pyrite concentrates, and gravity gold concentrates, and no cyanide or mercury will be required or used. Purchasing mill feed from contract mining partners will eliminate the use of mercury, which is commonly used by local traditional small scale miners, improve water and energy efficiency, ensure tailings are safely managed, and strengthen environmental stewardship.

The peak workforce during Project construction is estimated at 2,292, mainly comprised of contractors, plus 90 administrative staff and management. During operations the workforce is estimated at 676 company personnel. The Project is targeting 60% of the workforce to be hired from the local community, 20% from the department of Santander, and 18% from other departments in Colombia. Foreign technical and managerial specialists will eventually make up 2% of the workforce.

The Project's expected mine life of 22 years is based on the current mineral reserve estimate, and there are opportunities for future mine extension if additional mineral reserves are defined through continued exploration. Over the life of the mine, production is estimated at 4.3 Moz of gold, 18.8 Moz of silver, and 84.0 Mlb of copper. Average annual gold production is expected to be 263 koz between years 2 and 10, and 203 koz between years 1 and 21.

At the base case assumption of $2,600 per ounce of gold, the Project is estimated to contribute $2.6 billion in income taxes and $393 million in royalty payments to the Colombian government. The estimated initial capital expenditure is $625 million. The cumulative after-tax net cash flow is $5.0 billion, including initial capital costs, pre-production costs, sustaining capital costs, closure costs, VAT, and contingency. Cash costs per ounce of gold are estimated at $345 and AISC per ounce of gold are estimated at $534. At the base case assumption of $2,600 per ounce of gold, the Project has an after-tax $NPV_{5\%}$ of $2.7 billion, an IRR of 35.4%, and a pay back period of 2.3 years.

With the prefeasibility study complete, environmental and technical studies will be advanced and PSN intends to submit an updated ESIA to the CDMB reflecting the design described in this study.

1.17 Recommendations

The qualified person responsible for Section 11 notes that aqua regia digest may not fully digest the sample silica matrix and may provide lower assay results than fire assays as used by Ventana and AUX Colombia, and recommends that future drillhole samples are assayed for silver using fire assay methods.

The qualified person responsible for Section 13 recommends the following metallurgical testwork:

- Undertake additional comminution testwork from an increased density of samples representative of the life of mine plan to provide data for more accurate comminution simulations, for an estimated cost of $17,000.
- Undertake settling and filtration testwork with actual site process water and recirculated water, for an estimated cost of $30,000.
- Undertake additional testwork to characterize the gravity recoverable gold present in samples representative of the material in the life of mine plan, for an estimated cost of $6,000.
- Undertake additional locked cycle tests on samples representative of the material in the life of mine plan to characterize the nature of the gravity, copper, and pyrite concentrates, for an estimated cost of $10,000.
- Undertake a more detailed financial analysis of the economic benefits of a gravity circuit and investigate alternative technologies, for an estimated cost of $5,000.
- Undertake an assessment of the logistics and environmental impacts of including a gravity circuit, for an estimated cost of $12,000.

The qualified person responsible for Section 9 and 14 notes that the Project's expected mine life of 22 years is based on the current mineral reserve estimate, and that there are opportunities for future mine extension if additional mineral reserves are defined through continued exploration. There are numerous areas of high grade inferred material within the mineable stope optimizer shapes used to constrain the mineral resource estimate that are located adjacent to the mineral reserve stopes designed around indicated material that could be targeted for exploration. An initial exploration drilling program of 35 drillholes for approximately 12,500 m is recommended to target the highest grade areas of inferred mineral resources, and those located in the upper areas of the mine, comprising 1.2 Mt at 12.50 g/t Au for 482,000 ounces, to potentially convert those volumes to indicated mineral resources, at an estimated cost of $1.3 M.

The qualified person responsible for Section 16 makes the following recommendations:

- Further trade off and mine optimization studies are recommended for the future study stages, including the decline layout, the location of key underground infrastructure, options for the materials handling system, refinements to the blasting methods, and additional work on the ventilation design including optimal sizes, consideration of a ventilation on demand system, and ventilation heat modelling, for a total cost of $210,000.
- Additional refinements related to the underground infrastructure, including capital and operating costs, are recommended for air, water, electrical, ventilation control, egress and refuge chambers, explosive magazine, dewatering, mine services, paste fill plant and delivery, surface to underground cement delivery, and additional paste fill test work, for a total cost of $193,000.
- Additional geotechnical studies are recommended, including a Mathews stability graph check, three dimensional stress modelling, cemented paste fill design strength and strength gain rate tests as well as identifying implementation opportunities, and a trade off study to optimize the management of slimes in the paste fill or in the filtered tailings storage facility, for a total cost of $48,800.

The qualified person responsible for Section 18.5 makes the following recommendations to advance the design of the filtered tailings storage facility to the detailed engineering level, at an estimated cost of $3 million, considering both engineering consulting and contractor costs.

- Complete site investigation work including borehole drilling, test pit excavation, instrumentation installation, in situ testing, and geophysical surveys to inform the next phases of design.
- Advance geotechnical laboratory characterization work on the foundation soils in their in situ state and as structural fill following excavation, moisture conditioning, placement, and compaction.
- Design and size the required drainage works.
- Undertake slope stability analyses to evaluate facility phasing concepts.
- Explore opportunities to deposit tailings in non-horizontal lifts to reduce staging challenges.
- Develop facility safety and operations documents, including the operations, maintenance, and surveillance manual, the emergency response plan, and the trigger action response plan.

- Develop breach and inundation studies on the filtered tailings facility and contact water collection pond for potential damage assessment, risk classification, potential loss of life analysis, and emergency plan design and implementation.
- Complete additional geotechnical analyses as required by the regulators and the Project, such as probabilistic slope stability analyses and or nonlinear dynamic deformation models.
- Advance applicable engineering analyses to levels commensurate with future design phases.
- Develop a groundwater sampling program.
- Develop detailed closure requirements for the facility.
- Develop a cover system that will limit long term infiltration into the facility.
- Further refine grading of the required excavation works, containment embankment, and tailings stack.
- Create a cover revegetation plan.

The qualified person responsible for section 18.6 recommends that a comprehensive hydrological study is undertaken for future studies to refine the stormwater design, assess the long term water balance, and develop a more detailed mitigation strategy. Most of this work is currently underway to support the ESIA application.

The qualified person responsible for Section 20 makes the following recommendations:

- Complete environmental and social baseline surveys, including along the rope conveyor, tailing pipeline route, and the filtered tailings storage watershed, and confirm the absence of fish and amphibians in the filtered tailings facility footprint.
- Conceptualize a reclamation research program to support future progressive and final reclamation and revegetation programs, including collaborative partnering with local greenhouse owners to develop native seedling sources for revegetation.
- Continue efforts to advance contract mining partner agreements and action plans for implementation.
- Prepare a large mine project area poster in each of the social houses to become a focal point for conversations and an opportunity to listen and learn from community members as they relate their issues and experiences.

This work is covered in the ongoing environmental permitting work and has been accounted for within the existing social and environmental budgets.

2 Introduction

2.1 Issuer and purpose of the technical report

This technical report has been prepared for Aris Mining in compliance with the disclosure requirements of NI 43-101 to disclose material updates to the Soto Norte Project (Soto Norte, the Project, or the Property) resulting from updated mineral resource and mineral reserve estimates and the results of a prefeasibility study of a smaller, more efficient development plan for the Project than was previously considered in the 2021 feasibility study.

The effective date of this technical report is August 18, 2025. No new material information has become available between this date and the signature date given on the certificate of the qualified persons. The quality of information, conclusions, and estimates contained in this technical report is based on information available at the time of the effective date and the assumptions, conditions, and qualifications set forth in this technical report. Except for the purposes legislated under Canadian securities law, any other uses of this technical report by any third party is at that party's sole risk. The user of this technical report should ensure that this is the most recent technical report for the Property, as any previous technical reports are no longer current.

Aris Mining is a Canadian mining company with its common shares listed on the Toronto Stock Exchange under the symbol ARIS and the NYSE American LLC under the symbol ARMN.

2.2 Source information

Unless otherwise stated, information, data, and illustrations contained in this technical report or used in its preparation have been provided by the authors for the purpose of this technical report.

2.3 Qualified persons and personal inspections

This technical report was prepared by Kate Kitchen, MAIG, Area Manager Geology, of Mining Plus; Peter Lock, Executive Director and Principal Mining Consultant, FAusIMM of Mining Plus; Jan Eklund, P.E., Process Consultant of LogiProc Pty. Ltd. (LogiProc); Nicholas Sianta, P.E., Geotechnical Engineer of Knight Piésold; and Rolf Schmitt, P.Geo., Technical Director – Geology of ERM Consultants Canada Ltd., each of whom is a qualified person as that term is defined by NI 43-101 and all of whom are independent of Aris Mining for the purpose of NI 43-101. The responsibilities of each qualified person are shown in Table 2-1.

Ms. Kitchen conducted the personal inspection of the Project between June 17 and 19, 2025. During the inspection, Ms. Kitchen reviewed the geological and geographic environment of the Project; the nature and extent of all exploratory work completed at the Project, including those relevant to the current mineral resource estimate; representative mineralized and non-mineralized drill core intercepts through the relevant mining zones; the sample storage facilities for drill core, coarse rejects, and pulp rejects; and database management processes.

Mr. Lock conducted the personal inspection of the Project between June 17 and 19, 2025. Mr. Lock reviewed the proposed locations of the underground operation and the filtered tailings storage facility; representative mineralized and non-mineralized drill core intercepts through the relevant mining zones; the environmental and community setting, and road access and logistics for mining.

Mr. Eklund conducted the personal inspection of the Project between July 15 and 17, 2024. Mr. Eklund reviewed the proposed locations for the underground operation, the process plant, the filtered tailings storage facility, the environmental and community setting, road access and logistics for processing, and water sources in the Project area.

Mr. Sianta has conducted personal inspections of the Project on multiple occasions since Knight Piésold's involvement in the Project began in October of 2020. The three most recent personal inspections were from May 17 to 30, 2022, from November 14 to December 8, 2022, and from April 17 to May 13, 2023. During these inspections, Mr. Sianta performed activities such as the supervision of the geotechnical site investigation works advanced at the tailings storage facility site, inspection and validation of geotechnical laboratories used for the testing of site materials, engineering assessments of the tailings storage facility site regarding geology, topography, and constructability, and office based design work with the Project team.

Mr. Schmitt conducted the personal inspection of the Project on December 9, 2024. During the inspection, Mr. Schmitt reviewed the Project access logistics and road capital and safety improvements; the community and socioeconomic setting, the environmental setting of the mine portal and effluent water treatment plant; the environmental setting of the proposed rope conveyor, processing facilities, and filtered tailings facilities; the steep slope erosion and sediment control rehabilitation and revegetation plot; the streamside metal leaching and acid rock drainage rehabilitation and reclamation; the scope of the distance and elevation difference of the mine footprint to the páramo; the historical anthropogenic landscape alterations; the presence of active and former unauthorized mining operations and worker residences in proximity of the future mine; the Project social houses in the communities of California, Matanza, and Suratá; and the demonstration relocation houses for future resident resettlement.

Table 2-1 Responsibilities of each qualified person

Qualified person	Section responsibility
Kate Kitchen	2: Introduction; 3: Reliance on other experts; 4: Property description and location; 5: Accessibility, climate, local resources, infrastructure, and physiography; 6: History; 7: Geological setting and mineralization; 8: Deposit types; 9: Exploration; 10: Drilling; 11: Sample preparation, analyses, and security; 12.1: Data verification; 14: Mineral resource estimates; 23: Adjacent properties; 24: Other relevant data and information; 27: References; and the relevant summaries of those sections included in 1: Summary; 25: Interpretation and conclusions; and 26: Recommendations.
Peter Lock	2: Introduction; 3: Reliance on other experts; 12.2: Data verification; 15: Mineral reserve estimates; 16: Mining methods; 19: Market studies and contracts; 21: Capital and operating costs related to underground mining; 22: Economic analysis; 24: Other relevant data and information; 27: References; and the relevant summaries of those sections included in 1: Summary; 25: Interpretation and conclusions; and 26: Recommendations.
Jan Eklund	2: Introduction; 12.3: Data verification; 13: Mineral processing and metallurgical testing: 17: Recovery methods; 18: Project infrastructure with the exception of Section 18.5; 21: Capital and operating costs related to processing and surface infrastructure; 24: Other relevant data and information; 27: References; and the relevant summaries of those sections included in 1: Summary; 25: Interpretation and conclusions; and 26: Recommendations.
Nicholas Sianta	2: Introduction; 12.4: Data verification; 18.5: Project infrastructure related to the filtered tailings storage facilities; 24: Other relevant data and information; 27: References; and the relevant summaries of those sections included in 1: Summary; 25: Interpretation and conclusions; and 26: Recommendations.
Rolf Schmitt	2: Introduction; 3: Reliance on other experts; 20: Environmental studies, permitting, and social or community impact; 24: Other relevant data and information; 27: References; and the relevant summaries of those sections included in 1: Summary; 2: Introduction; 25: Interpretation and conclusions; and 26: Recommendations.

2.4 Currencies, units, and coordinate system

Unless stated otherwise in this technical report, all currency amounts are in United States dollars and quantities are in metric units. Project coordinates are in the local reference system Magna Sirgas origin Bogotá. North is up on all plan views.

3 Reliance on other experts

In preparing this report, Mining Plus has relied on information provided by employees of Aris Mining, provided regarding environmental liabilities described in Section 4.5 and regarding the current legal status of permits described in Section 4.6.

Except for the purposes legislated under applicable securities laws, any use of this technical report by any third-party is at that third-party’s risk.

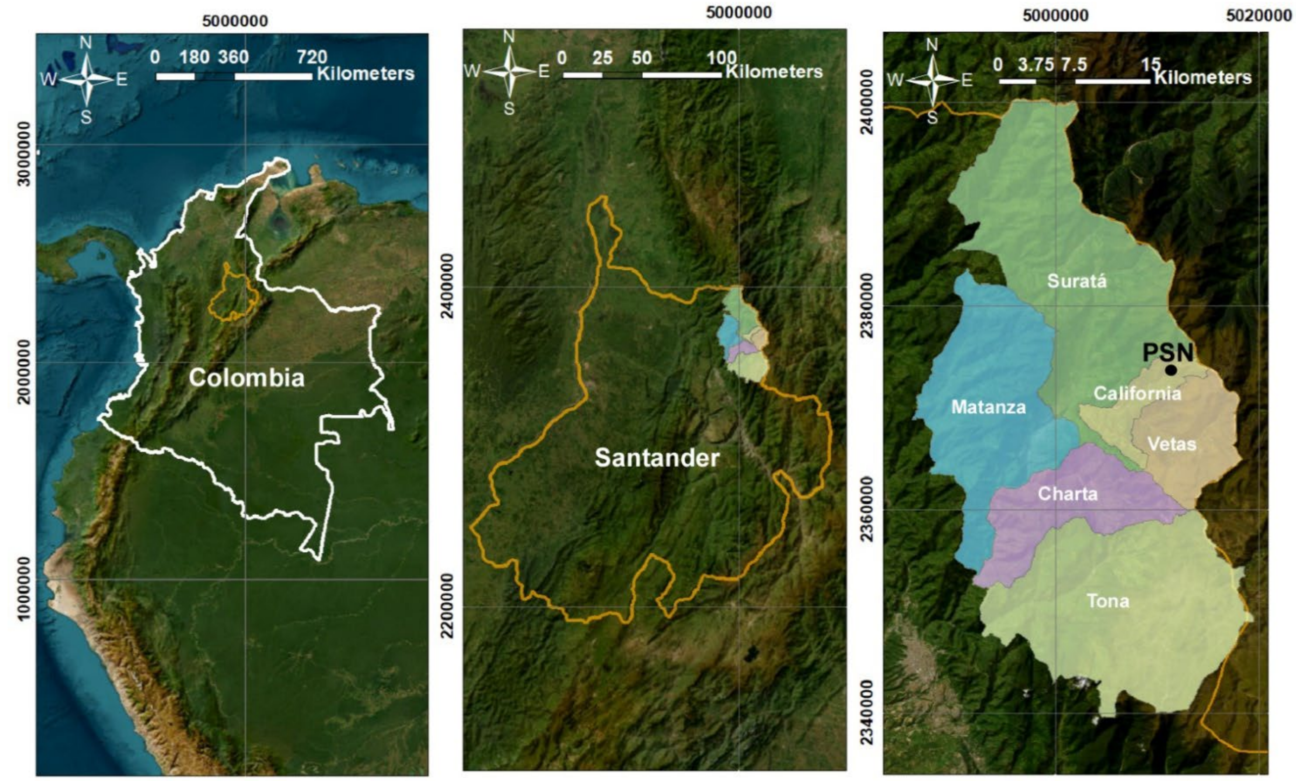
4 Property description and location

4.1 Property location

The Project is located in the California – Vetas mining district, approximately 350 km north of Bogotá, 55 km northeast of the city of Bucaramanga, the capital of Santander Department, and 9 km northeast of the town of California, at 7°22' North and 72°54' West.

A map of the Property location is shown in Figure 4-1.

Figure 4-1 Property location map - source Aris Mining 2025



4.2 Mineral tenure and Aris Mining's interest

There are 20 titles with a total area of 3,225.22 ha associated with the Property, all of which are 100% owned by PSN, with the exception of title 14947, of which PSN owns an option for 80% of the title. PSN is a company existing under the laws of Colombia, of which Aris Mining is the Project operator and holds an indirect 51% of joint venture interest and MDCIH holds the remaining 49% joint venture interest.

Title 095-68 is the key title containing the Property's mineral resources and mineral reserves, the planned underground mining infrastructure, and the surface infrastructure at the mine area. A summary of the titles is provided in Table 4-1 and a plan of the titles is shown in Figure 4-2.

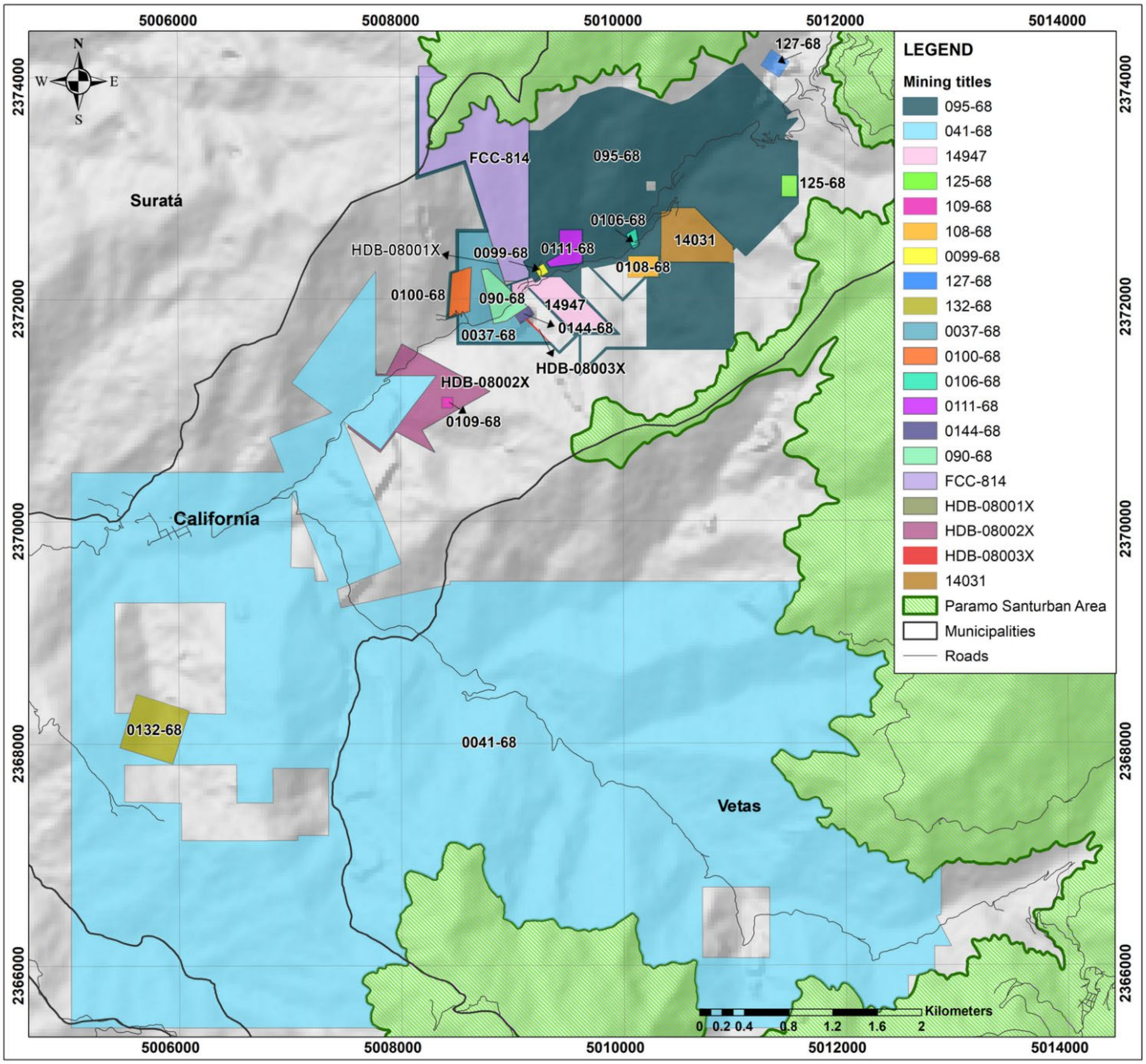
The work required to maintain the titles is dependent on the stage of the contract, such as performing exploration activities during the exploration stage. The title owner is required to comply with the environmental regulations and obtain all the necessary permits during the exploration stage, and the environmental license must be obtained for the construction and mining stages. Economic requirements include paying a surface fee during the exploration and construction stages and paying the royalty and obtaining an environmental mining insurance policy during the mining stage.

Table 4-1 Property mineral title summary

Title number	Title type	Area (ha)	Expiry date
095-68	Concession Contract	379.40	2028-06-08[1]
111-68	Exploitation License	7.70	2021-02-08[2]
125-68	Concession Contract	3.00	2040-05-07
127-68	Exploitation License	3.45	2020-04-18[2]
144-68	Exploitation License	1.30	2018-06-08[2]
FCC-814	Concession Contract	85.00	2037-02-14
HDB-08001X	Concession Contract	0.02	2040-03-10
HDB-08002X	Concession Contract	32.80	2040-03-09
HDB-08003X	Concession Contract	0.20	2040-02-24
14031	Exploitation License	26.7	2013-07-22[2]
100-68	Exploitation License	7.30	2018-05-20[2]
106-68	Exploitation License	1.30	2018-06-07[2]
108-68	Exploitation License	5.70	2018-10-08[2]
037-68	Exploitation License	33.80	2014-08-19[2]
090-68	Exploitation License	10.10	2018-05-20[2]
099-68	Exploitation License	1.00	2018-06-07[2]
14947	Concession Contract	20.95	2049-09-23
109-68	Exploitation License	1.00	2018-10-01[2]
132-68	Exploitation License	25.00	2018-05-27[2]
041-68	Concession Contract	2,579.50	2049-08-25

[1] PSN will request an extension of the title in accordance with the terms of the contract.
[2] The right has been exercised to convert the title from the historical Exploitation License type to the current Concession Contract type, as a result of a change in the law, and the expiry date of the expired titles will be updated when the Concession Contract is granted.

Figure 4-2 Property mineral title plan – source Aris Mining 2025



4.3 Surface rights and legal access

The mining titles at the Project do not provide property or rights over the land, but the right exists to expropriate or impose an easement over the land through administrative and/or judicial proceedings if it is required to develop the Project. The preferred method for acquiring land at Soto Norte for project development will be to reach agreements with landowners following receipt of the environmental permits.

PSN owns land with a total area of 192 ha, of which 10 ha are shared with third parties, and leases a further 1 ha of land to support the activities at the Property.

Additional land purchases will be required for Project development and operation.

4.4 Royalties, agreements, and encumbrances

Royalties due to the Colombian state include a 4% royalty on 80% of the gold and silver produced and a 5% royalty for copper on 100% of the copper produced.

When MDCIH sold the initial 20% joint venture interest in the Project to Aris Mining in April 2022, MDCIH retained a precious metals stream at Soto Norte; however, the stream only becomes effective after 5.7 Moz of gold have been produced. Under the current life of mine plan, Soto Norte is expected to produce approximately 4.3 Moz of gold, and therefore the threshold for stream deliveries is not expected to be reached. As a result, the precious metals stream is considered non-material to the economics of the Project.

To the extent known, there are no other royalties, back-in rights, payments, or other agreements or encumbrances to which the Property is subject.

4.5 Environmental liabilities

Environmental effects have arisen from historical workings and ongoing informal mining activities and processing plants within the Project titles. Monitoring data show impacts on the La Baja Creek, which runs through the Project area. These include discharges from informal mining affected by acid rock drainage and/or metal leaching, and there is erosion and mobilization of sediments near the informal mine workings and processing areas. None of the environmental liabilities resulting from these informal operations conducted after the Project acquired the titles are the legal responsibility of PSN. PSN is responsible for managing environmental effects related to its own activities and for the effects of the previous titleholder activities. PSN is and will continue to keep working with the regulatory authorities to remediate damage wherever possible.

4.6 Permits

Currently, the Project has the licenses it requires for the exploration phase of the Project.

The key permissions required for commencing construction and moving into the production phase are the amendment of the existing approved PTO to reflect changes to the ESIA as a result of this current prefeasibility study, and the approval of the ESIA to obtain the construction permit, followed by approval of the mining permit.

Once submitted to the environmental authority, an ESIA evaluation process is expected to take no more than four months, excluding any request to evaluate additional information, the lifting of bans, and public hearings. The process on average takes between 12 and 18 months. Once approved, the environmental license is valid for the life of the Project, subject to compliance audits by the environmental authority. The license may be modified for changes arising as the Project evolves.

The CDMB is the regional environmental authority responsible for the licensing of mining projects in the department of Santander for those projects that produce less than two million tonnes of ore and waste per year. As the Project is designed to produce less than two million tonnes of ore and waste per year, the CDMB will be the environmental authority responsible for the Project.

Several past ESIA studies and project plans designed to produce greater than two million tonnes of ore and waste per year have been submitted to ANLA, the national regulatory authority responsible for projects that produce more than two million tonnes per year. An ESIA process was submitted by Minesa to ANLA in August 2017 based on previous ESIA studies undertaken in 2013 and 2016, and then withdrawn to update the ESIA with changes to the Project design. The updated ESIA was filed by Minesa with ANLA in February 2019. On October 2, 2020, ANLA issued a writ ordering the closure of the file for the study of the Project’s environmental license, based on the consideration that the information provided in the ESIA was insufficient to continue the environmental assessment process and to issue an opinion on the viability of the Project.

Because ANLA’s decision to close or shelve the file on the Project application is based on a procedural conclusion on the perceived insufficiency of the information submitted, PSN is not barred from resubmitting a new application.

The updates to the Project parameters as a result of the new development plan as disclosed in this technical report will require additional studies including a re-evaluation of the environmental and social impacts, and a re-start of the environmental permitting process and timeframes. Compared to previous permitting submissions, the new development plan takes a new approach backed by strong support from the local communities in the Soto Norte area, including scaled down mining operations and processing facilities, a reduced environmental impact, and new processing options to support local small-scale miners. As demonstrated at Aris Mining's other Colombian operations, partnerships with small scale miners have delivered tangible community and environmental benefits. The proposed partnership with local miners at Soto Norte offers a pragmatic solution to reduce existing harm caused by informal mining activities and supports development of the local communities.

There are no requirements to post performance or reclamation bonds.

4.7 Significant factors and risks

On January 30, 2024, the Colombian Ministry of Environment issued Decree 044 which allows the Ministry to declare temporary reserve areas in certain parts of Colombia. To establish a temporary reserve area, a resolution must be issued by the Ministry detailing the area that is to be temporarily reserved. The Ministry issued Resolution 221 of 2025, amended by Resolution 239 of 2025, by means of which it declared a TRA in the Soto Norte region. The TRA will be in effect for two years, with a possible two year extension. While the TRA is in force, no new concessions or environmental permits may be granted by the mining or environmental regulators. During this period, the Ministry of Environment and Sustainable Development of Colombia must conduct environmental studies to determine whether to make the reserve area permanent. Notwithstanding the TRA, the Soto Norte Project may continue environmental studies, provided no environmental permit is required. Decree 044 and the TRA may delay licencing of the Soto Norte Project.

Decree 044 and the TRA resolutions are presently being challenged in administrative courts, with actions led by the Colombian Disciplinary Office, artisanal and small mining units, the Colombian Mining Trade Association, and the National trade association.

Additionally, the Administrative Tribunal of Santander issued a ruling in July 2025 in a class action proceeding recognizing the Santurbán páramo as a subject of personal rights and designating the Ministry of Environment as its legal guardian. While there is no direct impact on the Soto Norte Project, environmental licensing proceedings for the Soto Norte Project may be delayed or hindered because the Tribunal ordered that: (i) the Ministry of Environment must actively participate and protect the páramo in any licensing process, including through the use of administrative injunctions (ii) all relevant environmental authorities must identify critical transition areas to the páramo in the Soto Norte region for water protection, and (iii) zoning regulations must exclude mining activities in "buffer zones" in alignment with the 2014 delimitation process. The ruling was appealed.

The Soto Norte Project remains several years away from development. With the completion of this technical report, PSN intends to present a fully redesigned project to the Colombian regulators following the conclusion of the environmental and technical studies currently underway.

Except for the risks mentioned in this section, there are no other known significant factors or risks that may affect access, title, or the right or ability to perform ongoing work programs on the Property, including permitting and addressing environmental liabilities, aside from the requirement to obtain future environmental licenses and approvals.

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

5.1 Property access, transport, population centres, and mining personnel

The Project is located in the California – Vetas mining district, approximately 350 km north of Bogotá, 55 km northeast of the city of Bucaramanga, the capital of Santander Department, and 9 km northeast of the town of California. The Project is readily accessed year round by vehicle from Bucaramanga via 54 km of paved and unpaved roads to California and then by 9 km of unpaved road that passes through the centre of the Project. Smaller roads and foot trails provide further access throughout the Property, including 13 km of road connecting the processing plant and the underground mine access area. A map of the Property access is shown in Figure 5-1.

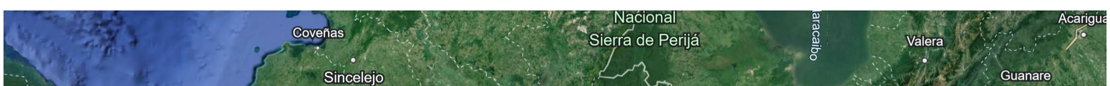
Bucaramanga is readily accessible from Bogotá by 397 km of paved road, and connected to the city of Barrancabermeja, a city with a river port that is the preferred option for transporting concentrates to overseas buyers, via 115 km of paved road. Barrancabermeja is located 650 km on the Magdalena River from the seaport at Cartagena, one of the largest in South America. Bucaramanga is also connected to the city of Santa Marta on the Caribbean coast by 539 km of paved road, which is another potential concentrate transport route.

There are several daily domestic flights from Bogotá and Medellín, where Aris Mining maintains its executive and shared services office, to Bucaramanga, which also offers direct international flights to Panama City and Fort Lauderdale.

Communities in the rural municipalities of California, Suratá, Matanza, Vetas, Charta, and Tona are located within or in the vicinity of the Project area of influence. These municipalities make up the province of Soto Norte and have a total population of approximately 23,000. The Soto Norte region has well developed infrastructure and a local work force familiar with basic mining operations as well as an active small scale mining community. Skilled and unskilled labour are available in the region for the construction and operational phases of the Project, and additional skilled operators and trainers will be required for support.

The region has the majority of all the necessary equipment to support construction and mining activities.

Figure 5-1 Property access map - source Google 2025



5.2 Topography, elevation, vegetation, and climate

The Property is located in a mountainous region with steep river valleys, with topographic relief ranging from 1,620 to 4,200 m above sea level and slope angles within the range of 50 to 75%. Lower lying land is also present with smooth to moderately steep terrain at elevations up to 2,200 m above sea level, where slope angles are within the range of 25 to 50%. Vegetation is light alpine scrub consisting of grasses and shrubs, with significant growth of oak, pine, and eucalyptus trees along watercourses. A good portion of the forest vegetation cover has been altered by agricultural activities.

The climate is cool and humid with an average annual temperature of 18.5°C, varying with elevation and weather conditions. Two rainy seasons generally occur in September to November and March to May. The average annual precipitation is approximately 1,500 mm and ranges from 1,000 and 1,500 mm. Exploration and future mining activities can operate year round.

5.3 Surface rights and surface area

The current Project surface footprint is 433.58 ha, of which 8.34 ha are located within the Project titles. An additional 425.24 ha will need to be acquired following receipt of the environmental permits.

5.4 Power and water

The estimated power demand is 13 MW at the process plant and 10 MW at the underground mine, for a total Project requirement of 23 MW. An existing 34.5 kV power line from Bucaramanga supplies a 5 MVA at 34.5 kV capacity substation at the underground mine, which can be used for construction power and as an alternative emergency power source, but is insufficient for operations.

Electrical power to the operations will be supplied from the Palos 115 kV substation in the Bucaramanga area via a 34 km long 115 kV, 45 MVA line capacity overhead line leading to a new substation at the process plant, where the voltage will be stepped down to 13.8 kV for reception by the mine’s main distribution substation. The main distribution station will supply power via 13.8 kV cables to the process plant, by 13.8 kV overhead line to the filtered tailings facility and water intake plant, and by a 7 km long 13.8 kV cable installed on the rope conveyor to service the underground operation and the underground substation. Supporting infrastructure includes electrical rooms and transformers.

Standby and emergency power supply will be provided by a 3.125 MW diesel generator station at the process plant, a 630 kW diesel generator station at the filtered tailings facility, a 250 kW diesel generator at the water intake plant, a 250 kW diesel generator at the emergency ponds, and by a 2.5 MW diesel generator station at the underground mine.

The Project has been designed with robust water management and protection as guiding principles. There is sufficient and readily available water within the Project area for all aspects of the operation including mining, processing, the camp, and other activities. Construction water will be sourced under permits from the La Baja Creek and Suratá River, which both have abundant water supply. During operation, the main source of water for the underground mine and its associated infrastructure will be from groundwater and the main source of water for processing will be from the adjacent Suratá River.

The Project plans to reuse, recycle, and return approximately 96.5% of the Project’s total water requirements back to the environment, resulting in a net water use estimated at 3.5%. The majority of the demand will be supplied by the Suratá River, which will supply approximately 2.8 litres per second of net make up water for the processing plant, representing 0.22% of the average flow of the Suratá River at the planned water access point near the village of Suratá, and 0.08% of the average flow of the Suratá River at Bucaramanga, the nearest city located 55 km downstream from the Project.

The Project includes a comprehensive water management plan that incorporates plans and programs tailored to each water application or activity with significant interaction with water to ensure that both the quality and quantity of local water resources are preserved throughout the life of the operation, and all potential impacts related to water use have been thoroughly identified, assessed, and addressed through prevention, mitigation, and compensation measures. The underground mine has been designed to minimize groundwater flows into the underground workings through advanced cover drilling and grouting ahead of mine development to identify and seal any water bearing structures before mining reaches them, greatly reducing potential inflow, and to manage, treat, and if required, safely return any captured water to the environment in compliance with the environmental standards and discharge permits. The process plant has been designed to minimize the use of water and to recycle both process water and the water obtained from the dewatering of the process tailings. The filtered tailings storage facility has been designed to divert rainwater runoff away from the facility and to drain and manage any subsequent dewatering of the tailings and any rainwater falling onto the facility. Continuous monitoring of water flow and quality will be undertaken with defined measures planned if any changes are detected. These strategies will help ensure that the current water flow rates within the ecosystem are maintained.

6 History

6.1 Early history and historical production

The California-Vetas mining district was mined by the indigenous Sura people in Pre-Colombian times. The district was reportedly discovered in 1549 by the Spanish, who worked two open pit operations at San Antonio in the La Baja area and at La Perezosa located immediately northeast of the Project. Small scale production continued for two and a half centuries. In the early 1800s and 1900s, English and French companies undertook operations that included a mill and smelter outside the town of California. In 1947, the Anaconda Copper Mining Company had an option on a property that encompassed the current Property and conducted exploration via tunnelling and limited core drilling, and dropped the option.

Artisanal miners holding small tenements at the Property mined at a small scale using haphazard methods, generally by raising and driving over short distances. No production records exist, but an estimated 50,000 to 75,000 tonnes are believed to have been mined from the Property. Approximately 1,500 m of tunnels, drifts, and raises are present on the Property with the most extensive workings developed at the La Bodega mine. Between 2010 and 2012, La Bodega was reported to be producing at a rate of between 10 and 30 tpd. There has been no formal production at the Property.

6.2 Ownership history

In December 2005, Ventana acquired small scale tenements from the artisanal miners and formed the La Bodega project.

In March 2009, EBX began purchasing shares of Ventana, and on May 25, 2011, AUX Canada, an affiliate of EBX, acquired Ventana and changed the Project name from La Bodega to El Gigante. On June 30, 2011, AUX Canada moved from British Columbia to Luxembourg as AUX Acquisitions S.A.R.L. (AUX Acquisitions). In October 2012, AUX Canada's local subsidiary merged with Sociedad Bodega Ventana Baja, consolidating the Project under AUX Colombia.

In December 2012, AUX Colombia acquired the adjacent Calvista Gold Corporation (the Calvista property) and Galway Resources (the Galway property) properties.

In January 2012, MDCIH formed a strategic partnership with EBX through a preferred equity investment. In 2013, following the financial difficulties of EBX, MDCIH took ownership of the Project as a redemption on the original investment. From mid-2013 to early 2015, the Project was placed on care and maintenance.

In February 2015, MDCIH took ownership of AUX Colombia and on November 6, 2015, changed the subsidiary name to Minesa and changed the Project name to Soto Norte.

On April 12, 2022, Aris Mining (formerly Aris Gold Corporation) acquired a 20% joint venture interest in Minesa and became the Project operator. On November 2, 2023, the Project was renamed to PSN. On June 27, 2024, Aris Mining acquired an additional 31% interest in the Project, increasing its total ownership in the joint venture to 51%. MDCIH owns the remaining 49% of the joint venture interest.

6.3 Exploration and development work

6.3.1 Ventana

In December 2005, Ventana began the first modern exploration program comprising geochemical sampling, geophysical surveys, and exploration diamond drilling. Ventana disclosed a historical scoping study in November 2010, and by March 2011, a total of 378 diamond drillholes for 134,078 m had been completed.

6.3.2 AUX Colombia

AUX Colombia drilled 431 diamond drillholes for 198,660 m between 2011 and 2013. AUX Colombia excavated the mine portal between July and October 2012, as well as four drives into the Gigante and Mascota veins. AUX Colombia disclosed a historical mineral resource estimate in January 2013. In July 2013 AUX Colombia terminated all exploration activities, and the Project was placed on care and maintenance from mid-2013 to the first quarter of 2015.

6.3.3 Minesa and Aris Mining

Between 2015 and 2018, Minesa undertook geochemical and channel sampling, and drilled 95 diamond drillholes totalling 42,498 m. No further exploration specific activities have taken place since 2018. Technical studies have been undertaken as required to support ongoing Project design studies.

In December 2021 and October 2022, Geoandina of Bogotá, Colombia undertook geophysical studies to characterize the hydrogeology of potential infrastructure sites.

In January 2022, Geonorth of Bucaramanga and in June 2024, Aerostudies of Medellín, Colombia undertook high resolution aerial photography.

In November 2022, Lettis Consultants International of Concord, United States of America, undertook a site specific seismic hazard evaluation for the Project, and updated the study in April 2024 to consider the new location of the filtered tailings facility.

In January 2023, Geoandina of Bogotá, Colombia undertook geophysical studies at alternate sites for the construction of the tailings storage facility.

In March 2023, CI Ambiental of Bogotá, Colombia undertook geological and geomorphological mapping at 1:10,000 scale to identify the structural features and the recent morpho-dynamic processes present in the Project area of influence. In areas where infrastructure was planned, the mapping was undertaken at a scale of 1:5000, including characterizing the surface material types and rock mass quality. This mapping at both scales was updated in 2024 to consider alternative locations, including the current planned location of the tailings storage facility assumed for this prefeasibility study.

6.4 Historical mineral resource and mineral reserve estimates

The results of the diamond drillhole samples have been utilized for historical as well as the current mineral resource and mineral reserve estimate. Historical mineral resources have been estimated with effective dates of November 2010, July 2012, January 2013, February 2016, January 2017, July 2017, May 2019, and January 2021.

In August 2017 the first historical mineral reserve was estimated as part of a prefeasibility study, based on the historical January 2017 mineral resource estimate. In January 2021 the second historical mineral reserve was estimated as part of the 2021 feasibility study, based on the historical May 2019 mineral resource estimate.

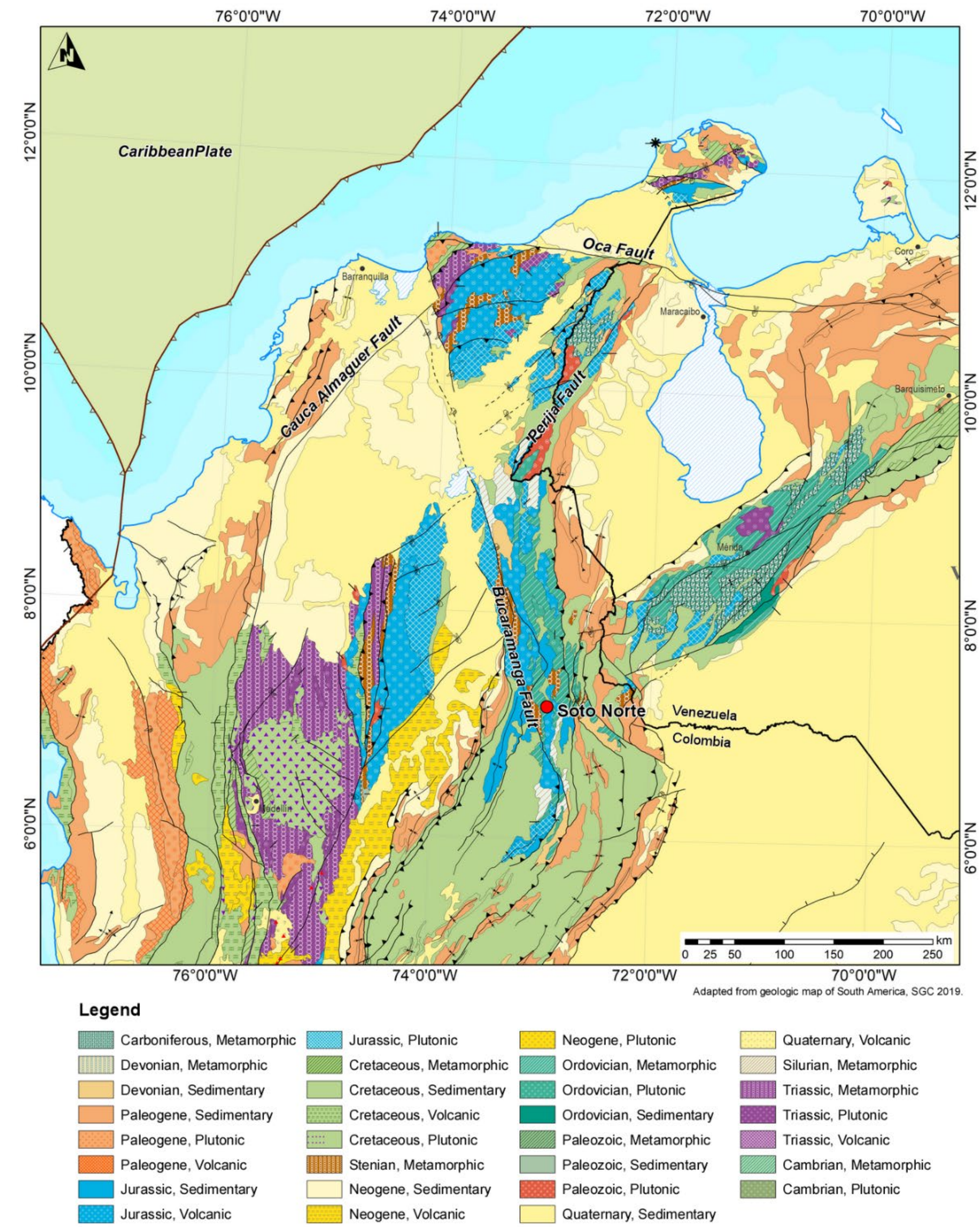
None of these historical mineral resource and mineral reserve estimates are current. They should not be relied on and have been superseded by the current mineral resource and mineral reserve estimates disclosed in this technical report.

7 Geological setting and mineralization

7.1 Regional geology

The Property is located in the Eastern Cordillera, one of three branches of the Colombian Andes, where the geology is characterized by the interaction of the Caribbean, Nazca, and South American tectonic plates which has resulted in the creation of subduction zones and associated magmatism, uplifted blocks, and compressional faulting. The Project is located north of the point of division of the Eastern Cordillera into western and eastern branches. The western branch hosts the north-northwest trending, 50 km wide Santander Massif hosting the Soto Norte mineralization, bound by the Bucaramanga fault to the west and the Socota-Santander fault to the east. The Project geology is related to magmatic events and contact metamorphism between these two faults. A map of the regional geology is shown in Figure 7-1.

Figure 7-1 Map of the regional geology – adapted from the Colombian Geological Service 2025



7.2 Local geology

The Santander Massif at the Project comprises three major geological units, including:

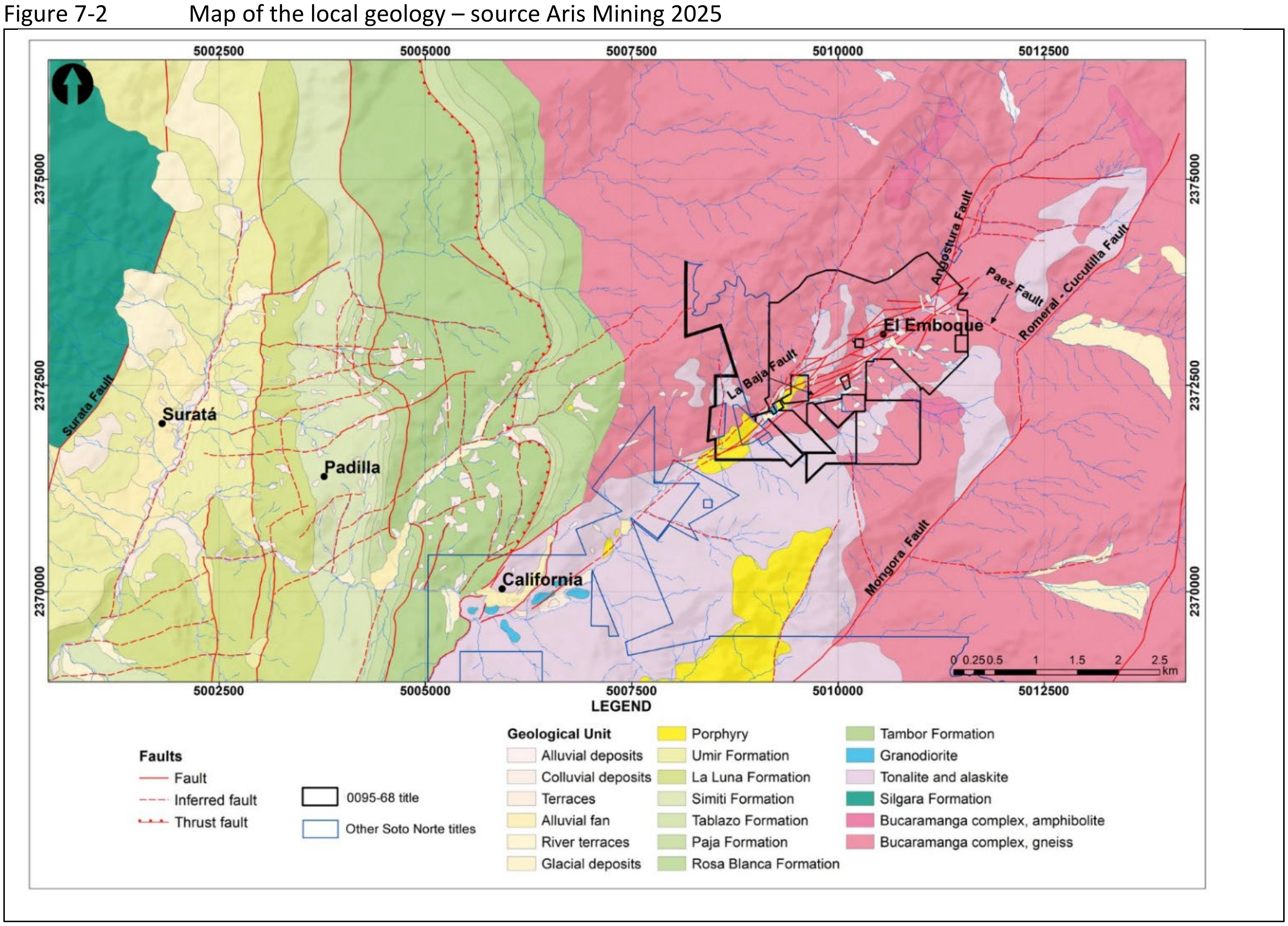
- the Precambrian aged Bucaramanga Complex comprising early Proterozoic aged paragneisses, migmatites, amphibolites, quartzite, marbles, and granulates. Banded gneisses are the most common at the Project and are composed of bands of quartz-feldspar and amphibolite-biotite.
- The late Triassic to early Jurassic aged Central Santander Plutonic Group comprising intrusive calc-alkaline rocks ranging from tonalites, granodiorites, and leucogranite in composition. These granitoids intrude the Bucaramanga gneisses as irregular dikes.
- Mesozoic and Cenozoic aged sedimentary rocks, including Jurassic and Cretaceous aged bedded marine sediments that unconformably overlie the Bucaramanga Complex to the west of the Project area.

District scale faulting forms topographic relief and the dominant northeast trending faults, which broadly control the shape of the intrusive complex. The principal faults include the La Baja, Mongora, and Cucutilla faults, which are interpreted to belong to a wider regional structural corridor that acts as one of the controls on mineralization throughout the California – Vetas mining district. The La Baja fault and sub-parallel branches broadly follow the La Baja Creek and its V-shaped valley. Intrusive rock on the north side of the La Baja fault, and gneiss on the south side of the fault, is the host of the Soto Norte mineralization.

Miocene aged porphyritic dikes crosscut some areas of the district and hydrothermal breccias crosscut the three major geological units. The northeast trending fault systems cut all of the geology and these are cut by continuing, post-mineralization fault activity.

A map of the local geology is shown in Figure 7-2.

Figure 7-2 Map of the local geology – source Aris Mining 2025



7.3 Property geology

Mineralization at the Property is hosted in gneisses of the Bucaramanga Complex and leucogranites of the Santander Plutonic Group, and mostly occur within tectonic-hydrothermal breccia bodies emplaced in a dilatant structural setting.

The faults hosting the parallel Gigante and Mascota mineralization trends represent two linking structures between the principal faults, with the Mascota mineralization hosted by the La Rosa fault zone and the Gigante mineralization hosted by the La Baja fault zone. Mineralization took place during active faulting along these structures, and post-mineralization faulting containing fragments of mineralized veins in broad faults, measuring up to 7 m in width, are noted along the La Rosa and La Baja fault zones.

The La Rosa fault zone varies in dip, ranging from sub vertical to the north-northwest at shallow depths and becoming slightly less steep dipping towards the south-southeast at depth. The La Baja fault zone generally dips steeply to the northwest. The faults converge at depth and are indicated to join into a single structure at elevations below 1,850 m and converge to the southwest and northeast with a broad lens of rock between them that has been cut by numerous minor faults. Some of these minor faults are mineralized and have been partially exploited at the surface by artisanal and small scale miners.

The extent and continuity of the Property mineralized vein systems around gold grades of 0.7 g/t Au as well as the Property drillhole distribution is shown in plan view in Figure 7-3 and on a long section view in Figure 7-4.

Figure 7-3 Plan of the Property mineralized vein systems – source Aris Mining 2025

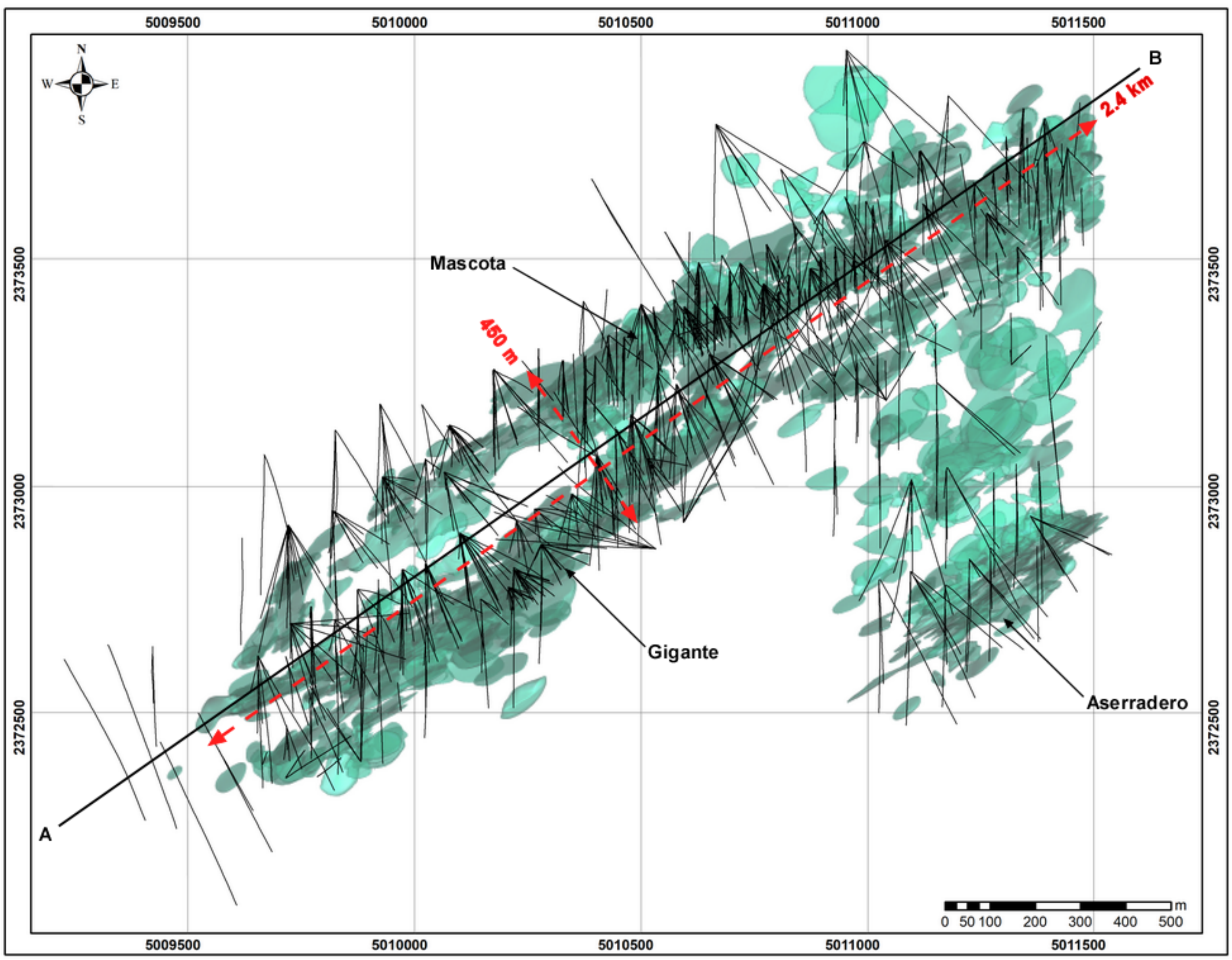
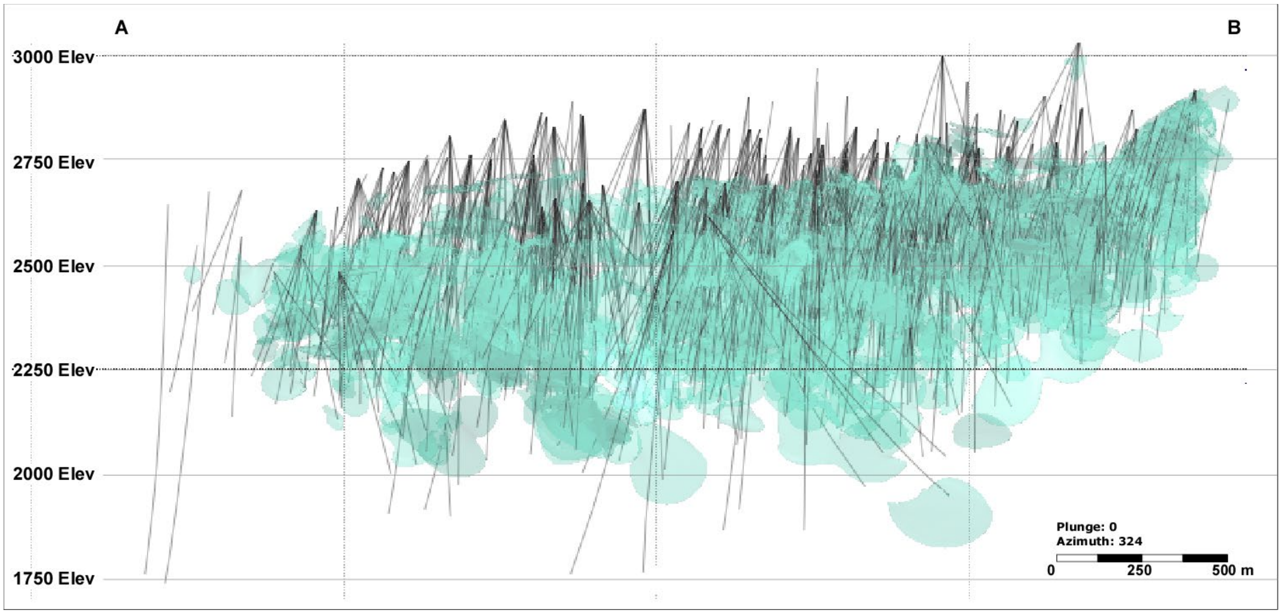


Figure 7-4 Long section of the Property mineralized vein systems – source Aris Mining 2025



7.4 Mineralization and alteration

Mineralization at the Property comprises parallel anastomosing veins within the fault systems, with variable widths and characteristics. Veins at Mascota have open-space filling textures, with hydrothermal brecciation and brecciated fragments of wall rock. Veining in the Gigante structure is mostly characterized by more compact, less vuggy, and often banded textures and is characterized by more heavily altered wallrock and clay content. Aserradero is a smaller, lower grade deposit located to the southeast of Mascota and Gigante.

Gold and electrum have a strong relationship with fine, crystalline pyrite and occur either free with the gold, adhering to pyrite particles, or encapsulated within the pyrite crystal lattice. Copper sulphides appear to have a partial affinity for pyrite but have much less of an association with gold than pyrite. Gold also occasionally occurs within tellurides. Silver occurs as silver sulphosalts, pyrargyrite, and proustite, and sometimes as native silver in the shallower areas. Copper occurs principally as enargite and to a lesser extent as bornite, chalcopyrite, chalcocite, and tetrahedrite-tennantite.

The Mascota and Gigante vein trends cover a strike extent of 2.6 km and have been drilled to a depth of approximately 800 m below the surface. The width of the veins is variable depending on structures, and pinch and swell, and averages between 1 and 3 m. The mineralized structures extend to the surface and are open at depth and along strike, resulting in a high exploration potential for expansion from future underground drilling stations.

Alteration is strongly related to the presence of the faults that host or cut the mineralization, including propylitic alteration that gives way to phyllic alteration dominated by sericite. At distances of 5 to 15 m from the centre of the fault zone, phyllic alteration is replaced by argillic alteration and intensifies particularly on the hangingwall side of much of the Mascota veins. Where veining is strongly developed, particularly where there are a few 1 to 5 m wide veins encapsulated within one wider structure, the wall rock between the veins, and often in the footwall and hangingwall, is dominated by a pervasive silica alteration. Small cavities in the silica alteration and the quartz veins contain white alunite. In some faults the wall rock alteration can be extensive and the fault will often be filled with rock flour that produces a soft clayey gouge. Clay minerals that may influence mineral processing include minor kaolinite which is present in an average concentration of 1 to 2% and illite, which is present in an average concentration of 4 to 6% and is the predominant clay mineral at Gigante. However metallurgical testwork on samples with high clay content well above the average returned positive recovery values, indicating the clay minerals may not influence metallurgical recovery.

Oxidation of the mineralization has an irregular depth and penetrates much deeper around major faults and fractures, and alters pyrite to hematite, goethite, and limonite. Very little of the material in the mine plan is affected by oxidation.

8 Deposit types

The Soto Norte deposit is considered a high-sulphidation epithermal deposit, with gold, silver, and copper occurring mainly in sulphides. The deposit is related to Miocene aged porphyry stocks and dikes that crosscut older sedimentary, igneous, and metamorphic rocks. The hydrothermal source fluids flowed through fault related pathways, generating background propylitic and phyllic alteration of the local rocks during mineralization, followed by silicification and argillic alteration in the centre of the main veins, zoning outward to intermediate argillic and propylitic alteration that formed during the principal stages of mineral deposition.

This model has formed the basis of the past exploration plans that have followed the vein trends along strike and down dip.

9 Exploration

9.1 2006 to 2011 Ventana

Ventana began surface and underground geochemical sampling in early 2006, accompanied by ground magnetic and IP geophysical surveys, followed by a ground magnetic and pole-dipole IP resistivity survey. This work was completed by VDG del Peru, S.A.C. in the area of Gigante. Surface diamond drilling started in August 2006, with 143,568 m completed by the end of March 2011.

None of the Ventana exploration data has been used for the current mineral resource and mineral reserve estimates, except for the diamond drilling.

9.2 2011 to 2013 AUX Colombia

During this period AUX Colombia completed a further 200,124 m of diamond drilling over a strike length of 2.5 km.

In 2012, Serviços Aéreos Industriais of Brazil completed a LIDAR survey covering the entire Project area that was processed into a 1 m contour digital terrain model (DTM). 117 high quality aerial photos were also produced covering the entire California – La Baja valley and adjacent topographic relief, which was used along with the DTM to identify rivers, streams, water sheds, and other geographical features.

In February 2012, an airborne magnetics and radiometrics survey was completed by MPX Geophysics of Toronto, Canada over Gigante and the surrounding area. A ground magnetics and IP survey was completed by Arce Geofisicos of Lima, Peru in March 2012 in the Gigante area.

The combined ground magnetics data shows strong anomalies with a northeast-southwest trend that coincides with the general trend of the mineralization and geochemical sampling results. The combined IP results show similar trends and indicate that the anomalies continue to the southwest through the Galway and Calvista properties, also owned by Aris Mining, and onwards to the town of California.

AUX Colombia completed mapping and sampling during excavation of the mine portal exploration tunnel and the existing tunnels into Gigante and Mascota.

None of the AUX Colombia exploration data has been used for the current mineral resource and mineral reserve estimates, except for the diamond drilling.

9.3 2015 to 2018 Minesa

During this period Minesa collected 80 mobile metal ion soil samples, 22 rock samples, and 134 channel samples taken from historical mine workings. The combined Project geochemical samples confirm the northeast-southwest trend of gold mineralization subparallel to the La Baja Creek, and higher grade zones on the northeast limits towards the La Bodega area.

95 diamond drillholes were completed, totalling 42,498 m. Twelve geotechnical drillholes for a total of 3,061 m were completed to provide geological and geotechnical information for a previously considered tunnel access.

In October 2016 Geoconsult Group Ltd carried out an aerial survey to document unauthorized mining activities along the La Baja Creek and to cover rivers, streams, water sheds, and other geographical features not covered by the 2012 work.

No other exploration work has taken place since 2018. None of the Minesa exploration data has been used for the current mineral resource and mineral reserve estimates, except for the diamond drilling.

9.4 Recommendations

The Project's expected mine life of 22 years is based on the current mineral reserve estimate, and there are opportunities for future mine extension if additional mineral reserves are defined through continued exploration. There are numerous areas of high grade inferred material within the mineable stope optimizer shapes used to constrain the mineral resource estimate that are located adjacent to the mineral reserve stopes designed around indicated material that could be targeted for exploration. An initial exploration drilling program of 35 drillholes for approximately 12,500 m is recommended to target the highest grade areas of inferred mineral resources, and those located in the upper areas of the mine, comprising 1.2 Mt at 12.50 g/t Au for 482,000 ounces, to potentially convert those volumes to indicated mineral resources, at an estimated cost of $1.3 M.

10 Drilling

10.1 Drilling summary

Diamond drilling was carried out by a range of different contractors during 2006 to 2018. No drilling for the purposes of mineral resource definition has been completed since 2018. Due to the steep topography, the drilling contractors utilized machines that could be dismantled and moved either manually, by mule, or by helicopter, as required. A breakdown of the Project diamond drillhole data is provided in Table 10-1.

The drilling grid was first completed at 100 by 100 m spacing and later tightened to 50 by 50 m, with further tightening to 25 by 25 m on shallower areas above 150 m below the surface. At depth the drilling intersections remain relatively wide at 50 to 100 m, which is a function of the steep intersection angles required by the steep topography. Further infill drilling will be completed after underground development is in place to provide a better drilling intersection. The extent of the Property drillhole distribution is shown in plan view in Figure 7-3 and on long section view in Figure 7-4.

The mineralized structures are open at depth and along strike, with high exploration potential to target the deep structures from underground drilling stations.

As the drillhole intersections through the vein interpretations are used as an input into the mineral resource estimate, the relevancy of the raw drillhole sample assay results are superseded by the mineral resource estimate and are more meaningfully described in the context of the mineral resource estimate as disclosed in Section 14.

Table 10-1 Drilling summary

Year	Company	Number of holes	Drilled metres
2006	Ventana	11	2,640
2007	Ventana	41	10,908
2008	Ventana	41	11,083
2009	Ventana	79	30,344
2010	Ventana	176	67,860
2011	Ventana	30	11,243
2011	AUX Colombia	132	52,498
2012	AUX Colombia	277	136,029
2013	AUX Colombia	22	10,132
2016	Minesa	88	38,994
2017	Minesa	6	2,840
2018	Minesa	1	665
Total		904	375,235

10.2 Drilling methods

Drillholes were collared at HT diameter (71 mm) or HQ diameter (63.5 mm), then reduced as drilling conditions allowed to NT diameter (58.9 mm) or NQ diameter (47.6 mm). In a few cases the diameter was reduced to BT diameter (40.8 mm) or BQ diameter (36.5 mm).

The collars of most completed drillholes were marked with a concrete block with the drillhole number and a polyvinylchloride (PVC) plastic pipe was used to mark the location and general orientation of the first few metres of the hole. The platforms were rehabilitated to their original landform after drilling was completed.

The drill core was placed into galvanized steel core boxes with sliding lids then secured with tie straps prior to being transported by mule to the logging facility.

10.2.1 Drillhole collar and downhole surveys

The early drillhole collars were surveyed by a surveying contractor using differential GPS equipment tied into the surface survey grid. Later drillhole collars were tied into the surface triangulation by total station surveying equipment.

Early downhole surveys were measured using Tropari and Pajari single shot instrumentation with readings taken every 25 m downhole. During 2012 and 2013, magnetometer-accelerometer style multi shot instruments from Flexit and Icefield were used, with measurements taken every 3 to 6 m downhole. Drillhole surveying was undertaken by Century Wireline Services between 2016 and 2017 using a Stockholm Precision Tools north seeking gyro with measurements taken at set downhole intervals. A range of geotechnical measurement tools were run at the same time, including acoustic and optic televiewers, density determination, induction – gamma ray measurement of conductivity, resistivity, water flows, and temperature. If the deviation was greater over a given distance than that agreed in the drilling contract, the hole was drilled again or wedged at the cost of the drilling contractor.

10.3 Drilling, sampling, and recovery factors

The drilling, sampling, and recovery factors have varied by operator. The first 30 m of drilling typically has low recovery rates of around 70% due to near surface weathering, but recovery improves below this level. The mineralized zone shows recovery rates greater than 90%.

The qualified person responsible for this section of the technical report is of the opinion that the drilling procedures employed at the Property are reasonable and sufficient for the purpose of estimating mineral resources. The qualified person is unaware of any drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the results included in the drillhole database used for the mineral resource estimate.

11 Sample preparation, analysis, and security

11.1 Introduction

Channel, soil, and drillhole samples have been collected by Ventana, AUX Colombia, and Minesa. The channel and soil samples were collected for exploration target identification for follow up drilling, as well as geological interpretation. None of the channel or soil samples have been utilized in the mineral resource and mineral reserve estimate.

11.2 Channel sampling and security

11.2.1 Ventana

There are 250 channel samples in the database attributed to Ventana, which were taken in 3 m long continuous channels of the walls that crosscut the vein trends. The walls were cleaned prior to cutting the channels with a hammer and chisel with the sample falling onto plastic sheeting and then transferred into a sample bag. The samples were delivered to DHL in Bucaramanga and sent by airfreight to the independent ACME Analytical Laboratories (ACME) in Vancouver, Canada for sample preparation and analysis. No QAQC samples were submitted with the channel samples.

11.2.2 AUX Colombia

AUX Colombia's 339 rock chip samples were transported from the work area via truck to the Project core logging facility. The samples were weighed, barcoded, sealed, placed in larger bags, and sealed with zip ties for transport to the independent ALS Chemex sample preparation laboratory in Bucaramanga. The samples were recorded on a sample form used to track the samples through the sample preparation laboratory. AUX Colombia included a chain of custody protocol with a record of the names of all persons handling the channel samples.

11.2.3 Minesa

Minesa's channel samples were weighed, barcoded, sealed, and placed in larger bags sealed with a zip tie. The sample numbers were recorded on a sample form to track the sample through to the sample preparation laboratory. Minesa had a chain of custody protocol with a record of the names of all persons handling channel samples. Samples were sent via Servientrega, a Colombian courier service, to the independent ALS Colombia laboratory in Medellín for sample preparation. The QAQC protocol for diamond drillhole samples was also followed for the channel samples.

11.3 Soil sampling and security

11.3.1 Ventana

There are 6,627 mobile metal ion soil samples in the database attributed to Ventana. Details of the sampling method and security are unknown, except that the samples were shipped directly to the independent SGS Mineral Services (SGS) in Toronto for sample preparation and analysis.

11.3.2 AUX Colombia

AUX Colombia's soil samples were transported from the work area via truck to the core logging facility. The samples were weighed, sealed, placed in larger bags, then sealed with a zip tie for transport by Servientrega to the independent SGS sample laboratory in Medellín, which was ISO / IEC 17025 accredited for sample crushing and pulverizing. The samples were recorded on a packing form used to track the samples through to the laboratory. No QAQC samples were submitted with the soil samples.

11.3.3 Minesa

Minesa's soil samples were transported from the work area via truck to the core logging facility. The samples were weighed, sealed, placed in larger bags, then sealed with a zip tie for transport by Servientrega to the independent SGS sample laboratory in Medellín. The samples were recorded on a packing form used to track the samples through to the laboratory. QAQC samples were submitted with the soil samples.

11.4 Drillhole sampling, QAQC, and security

11.4.1 Overview

A comprehensive logging protocol included recording lithology, structure, alteration, the presence of sulphides or other mineralization, and geotechnical data, among other features. The work also involved systematically re-logging all previously drilled holes. The logging process was conducted digitally and uploaded to an SQL server, where it was integrated with core photographs, ensuring precise and thorough geological documentation.

The core sampling process was similar across Project owners. Once logging was completed, the core was marked for cutting with a diamond bladed core saw. The right hand side of the half core pieces were placed in plastic sample bags. For broken fragments, half the volume was selected by hand. Intervals with clay or unconsolidated material were split vertically with a knife while still saturated. The sampling intervals and assay results were recorded in the core logs. After cutting, the left hand side of the half core was placed back in the core box for storage, and bar-coded sample tags were placed in the core box at the start of each sampling interval. Although the standard sample length was 1.0 m, it was adjusted to align with geological contacts.

The QAQC submission rate increased over time as the Project owners prioritized data integrity and reliable assay results. QAQC samples from Ventana and AUX Colombia involved the laboratory selecting a pulp reject split of the original sample as directed by the Project's geologists. Coarse blanks comprised unaltered gneiss from a local quarry, and commercial standard samples were also submitted.

Minesa implemented a rigorous QAQC system to ensure the validity, accuracy, and reliability of the sample assays. The system included written protocols for drilling, surveying, sampling, assaying, data management, and database integrity. Analytical control measures included the use of certified reference materials, coarse and fine blanks, coarse and pulp reject duplicates split by the laboratory, and quarter core samples of the second half of the core sample.

Between 2006 and 2011, the QAQC insertion ratios were less than 10%, but increased to between 11 and 19% between 2012 and 2018, coinciding with Minesa's ownership of the Project and their procedural improvements. Over the entire sampling history, 37 different standards were submitted with the results showing a low failure rate within acceptable parameters, with close to zero bias in some of the most representative standards. Less than 10% of the results exceeded three standard deviations of the certified value. The cause of these failures cannot be definitively determined but may be related to a mislabelling of the standard. No significant bias was observed. Blanks initially comprised coarse blanks until AUX Colombia began submitting fine blanks. The results of the fine blanks were satisfactory, while the coarse blanks from channel samples submitted to ACME showed elevated values. Despite this, the failure rate is considered acceptable. Duplicate sample submissions increased over time and included coarse and pulp rejects and quarter core samples. Pulp duplicates submitted to both ACME and ALS showed high variability attributed to the presence of coarse gold. Variability was also noted to a lesser extent in the quarter core samples. The variability was not observed in the coarse duplicates, suggesting potential issues with the homogenization or splitting process during the selection of the pulp duplicates.

In 2016 Geological Consulting Exploration and Mining SpA of Santiago, Chile conducted a reanalysis of 922 quarter core samples and 1,148 pulp duplicates at an independent check laboratory, including standard controls in both sample types, and conducted a thorough review of the database data collected prior to 2014. The results indicated a low bias in the primary laboratory compared to the check laboratory, although silver assays showed higher variability. The results from the standard samples showed low bias and fell within acceptable limits.

11.4.2 Ventana

Ventana's QAQC samples consisted of the laboratory selecting a coarse reject split of the original sample on the instruction of Ventana. Blanks comprised unaltered gneiss from a local quarry. Standard samples were also submitted. The QAQC samples were assayed for gold, silver, and 36 other elements using hot aqua regia digest of a 15 g charge and ICP-MS analysis. Additionally, approximately 15% of the samples were submitted to Inspectorate American Laboratory Inc. (Inspectorate) in Nevada for check assays.

Two independent laboratories, Asomineros and Inspectorate Laboratory, both located in Medellín, were used for sample preparation.

Chain of custody was maintained and monitored throughout the process with half cores selected for analysis, bagged, sealed, and then placed in larger bags, which were also sealed. Storage on site was in a locked core shed with 24 hour security until the samples for the entire drillhole were shipped as a single batch for sample preparation. The laboratory verified the security seals and signed off on receipt of the samples.

11.4.3 AUX Colombia

Upon receipt of AUX Colombia's drill core boxes at the site core logging facility, the boxes were opened and inspected for condition and fit with the downhole metre blocks, and the box numbers were checked against the driller's drilling advance records. A first pass geological log comprised of the main rock types and structures was completed. Measurements of core recovery and rock quality designation (RQD) were made, and the core was photographed. Detailed logs were made of the lithology, structure, alteration, and the presence of sulphides or other mineralization. Density and point load test determinations were also made. The sample interval was marked on the drill core and lines were drawn on the core as a guide for cutting with a diamond bladed rock saw. The standard sample interval was 1 m except for where they were modified due to major lithology changes. After cutting, one half of the core was placed in a plastic sample bag. When the core was too broken for cutting, one half of the volume was selected by hand. Any intervals with clay or other unconsolidated material was split with a knife. Each of the samples were weighed, sealed, placed in a larger bag, then sealed with zip ties for transport to the independent laboratory ALS Chemex in Bucaramanga. The samples were recorded on a sample form used to track the samples through the sample preparation laboratory. AUX Colombia had a chain of custody protocol where the names of all persons handling drill core were recorded on forms. Upon receipt of the sample at the laboratory, the sample barcode was scanned.

11.4.4 Minesa

The drill core was placed by the driller in a galvanized metal core box and two colour lines were marked on the core to indicate the original orientation of the core. The drill core was routinely inspected, with notes taken regarding core recovery or any unusual conditions. Lids were placed on the boxes, secured with straps, and carried down from the collar by mule to a staging area. The core was then transported by pickup truck to the core logging facility by workers supervised by the drilling company. This process was documented on a transport control form when the core boxes were delivered to the core logging facility. Upon receipt of the core at the facility the drill core was inspected and aligned with the two colour lines marked by the drillers. The Minesa chain of custody security protocol included that a member of the geological staff always accompanied the sample during transportation before handing them over to the laboratory for sample preparation. The samples were then transported by DHL to the analytical laboratory.

11.5 Laboratory sample preparation and analytical methods

11.5.1 Ventana

Ventana's channel samples were analyzed by ACME in Vancouver, which held ISO 9001 and 17025 accreditations. The samples were dried at 60°C, the entire sample was crushed to 70% passing 10 mesh, and a 250 g subsample was split and pulverized to 95% passing 150 mesh. A 30 g charge was analyzed for 37 elements by ICP-MS after digestion in hot aqua regia.

Ventana's mobile metal ion soil samples were analyzed by SGS in Toronto, which held ISO-IEC 17025 accreditation. A 50 g charge was analyzed using a partial leach and ultra-trace ICP-MS analysis for gold, silver, and 40 additional elements.

The details of Ventana's drillhole sample preparation methods are unknown, but the samples were analyzed using fire assay for gold and silver and a 36 element ultra trace package using hot aqua regia digestion of a 15 g charge and ICP-MS analysis.

11.5.2 AUX Colombia

The AUX Colombia channel samples were crushed and pulverized and a 30 g charge was assayed by ALS Chemex Bucaramanga for gold and silver using fire assay with AA finish. Samples with a gold grade greater than 5 g/t or silver grade greater than 100 g/t were repeated using gravimetric finish. 51 trace elements were assayed using aqua regia digest and ICP atomic emission spectroscopy (AES) / ICP-MS.

AUX Colombia's soil samples were crushed and pulverized in the SGS Colombia laboratory, which was ISO / IEC 17025 accredited, and then sent to the SGS Lima laboratory for analysis, which had ISO 9001:2015 accreditation. The samples were analyzed using the mobile metal ion (MMI) process and then assayed using ICP-MS.

AUX Colombia's drill core samples were crushed and pulverized at ALS Chemex Bucaramanga then shipped by air courier to ALS Chemex in Lima for analysis, which held ISO 9001 and 17025 accreditations. The entire sample was crushed to better than 70% passing 2 mm, then a 500 g split was collected with a riffle splitter and pulverized to better than 85% passing 75 microns. The samples were assayed for silver and gold using fire assay on a 30 gram charge with AA finish. Any sample with a gold grade greater than 5 g/t and a silver grade greater than 100 g/t was re-assayed using fire assay with gravimetric finish. 51 trace elements were assayed using aqua regia and ICP-AES/ICP-MS.

11.5.3 Minesa

Minesa's channel samples were prepared by ALS Chemex in Medellín, which has ISO 9001:2015 accreditation. The samples were crushed and pulverized and then sent to ALS in Peru for analysis using the same method as drill core samples.

Minesa's soil samples were crushed and pulverized at the SGS Colombia laboratory, which was ISO / IEC 17025 accredited, and then sent to the SGS Lima laboratory for analysis, which had ISO 9001:2015 accreditation. The samples were analyzed using the MMI process.

Minesa's drill core samples were prepared by drying at 105°C, crushing to 70% passing 2 mm, quartering with a Jones splitter to obtain a 250 g subsample, and pulverizing the subsample to 85% passing 75 microns. As of April 2016, the protocol was modified by increasing the subsample weight to 500 g. The samples were assayed for gold using fire assay on a 50 g charge with AA finish and any gold assay greater than 100 g/t was re-assayed with a 50 gram charge with gravimetric finish. Trace elements were assayed using aqua regia digestion followed by ICP-MS. Silver assays with grades greater than 100 g/t were re-assayed using aqua regia digestion with AA finish, and samples greater than 1500 g/t were re-assayed using fire assay on a 50 g charge with gravimetric finish. Other relevant elements such as copper, cadmium, tellurium, lead, and others were re-assayed using an appropriate technique if the assay exceeded the upper detection limit of the original method.

The qualified person notes that aqua regia digest may not fully digest the sample silica matrix and may provide lower assay results than fire assays as used by Ventana and AUX Colombia, and recommends that future drillhole samples are assayed for silver using fire assay methods.

11.6 Bulk density measurements

Bulk density measurements were conducted by geology personnel on drill core at approximately 20 m intervals or more frequently in changes of lithology or alteration. The 10 to 15 cm long samples were dried and then measured using the weight in air, weight in water process, without wax coating. In 2016 Minesa used hairspray as a sealant, and verified one in ten samples at ALS Chemex using paraffin wax coating. No significant differences were observed between the Minesa and ALS Chemex results. The bulk density samples were returned to the core box after geomechanical logging and before sampling for assays.

11.7 Adequacy of the sample preparation, security, and analytical procedures

It is the opinion of the qualified person responsible for this section of the technical report that the sampling, sample preparation and analysis, security, and QAQC protocols at the Project were consistent with generally accepted industry best practices and are suitable for the mineral resource and mineral reserve estimates. The samples were prepared and analyzed by independent certified global laboratories, and there is no relationship between the laboratories and the Project owners. All procedures and analytical assays have been conducted independently and objectively.

The qualified person notes that the use of aqua regia digestion may result in a slight underestimation of certain elements, such as silver, in specific mineralogical contexts. However, this potential underestimation has been accounted for in the resource estimation and is not considered to have a material impact on the results. Therefore, the qualified person's review of the QAQC data has shown that there is no indication of any material bias in the assays, there is no evidence of material sample contamination, and the duplicate samples show the expected variability for the mineralization style.

12 Data verification

12.1 Geology data reviews

The qualified person responsible for geology verified the geological data supporting the mineral resource and mineral reserve estimates through the personal inspection and through collaboration with the Project team, including:

- cross validation of the database entries with selected original laboratory certificates;
- reviews of the geological and geographic environment of the Project;
- reviews of the nature and extent of all exploratory work completed by the Project owners, including those relevant to the current mineral resource estimate;
- reviews of mineralized and non-mineralized drill core intersections;
- reviews of standard operating procedures related to drilling, sampling, and analytical processes covering several stages in the sampling and assaying chain from raw samples to prepared assay pulps; logging, re-logging, core sampling processes, analytical QAQC controls, and chain of custody; and bulk density determination methods;
- reviews of sample storage facilities for drill core, coarse rejects, and pulp rejects;
- reviews of database management processes; and
- independent sample checks of drill core and pulp rejects.

Based on the personal inspection and geological database review, the qualified person has found that the drilling, logging, and sampling practices meet acceptable international standards, thus concluding that the sample preparation, security protocols, and analytical procedures implemented for the Project provide an adequate current basis for the mineral resource estimate. The following were observed:

- the geology and mineralization controls are well understood and appropriately considered during drilling and geological interpretation;
- no material issues were identified in the database;
- the translation of previous drillhole collar coordinates to the current coordinate system and the on site inspection verified their accuracy with no concerns regarding the transformation method;
- survey review identified some anomalous measurements, which were examined and appear to be related to geological factors. Other anomalous measurements were not reviewed in detail but they do not appear to significantly impact the survey. These should be investigated further as the Project work progresses;
- assay results compared with certificates show minimal inconsistencies;
- in the early years of drilling, the QAQC sample insertion rate was limited, but this has been progressively enhanced to meet industry standard protocols;
- although the early drilling campaigns included limited QAQC samples, the overall assessments show no contamination issues with coarse and fine blanks during crushing and pulverization. Standard samples exhibit acceptable accuracy, though a few extreme outliers may be attributed to coding errors. Quarter core and coarse reject duplicates demonstrate good precision while pulp duplicates show lower precision, likely due to issues with homogenization or splitting during the pulverization stage; and
- independent sample checks by the qualified person included the re-assay of 20 pulp samples, which confirm consistency with the original grades. Six quarter core and two coarse reject samples were also re-assayed, showing some differences, generally with lower values. These variations, though not entirely clear, are likely associated with natural gold variability, the smaller sample size of the quarter core compared to the half core original sample, and sampling processes. The mineralization evidence is strong, with no significant indications of bias or errors in sample handling.

In the opinion of the qualified person, the data used for the purpose of estimating the mineral resources and mineral reserves and the development of the economic analyses are sufficiently reliable.

12.2 Mine engineering data reviews

The qualified person verified the mining and mineral reserve factors supporting the mineral resource and mineral reserve estimates through the personal inspection and through reviews of the Project work, including:

- mineral reserve estimation assumptions, including mining recovery and dilution estimates;
- production rates, mine design, equipment selection, schedule, cost estimates, and economic analysis of the life of mine plan;
- geotechnical and hydrological studies; and
- mine power and water requirements.

In the opinion of the qualified person, the assumptions used for the purpose of estimating the mineral resources and mineral reserves and the development of the economic analyses are sufficiently reliable.

12.3 Metallurgy and processing data reviews

The qualified person verified the metallurgical and processing factors supporting the mineral resource and mineral reserve estimates through the site visit and through collaboration with the Project team, including reviews of:

- metallurgical testwork reports supporting the metallurgical recovery estimates and the gold, silver, and copper concentrate production estimates, noting that all testwork was conducted by accredited laboratories, and that in the opinion of the qualified person, the testwork meets best industry practices and the results indicate no evidence of significant bias which could adversely impact on the developed process plan;
- reviewed the location and logistical challenges of the planned infrastructure, including the mine portal, the processing facility, the tailings filter plant, and the filtered tailings facility; and
- reviewed the on-site laboratory and core storage facilities.

In the opinion of the qualified person, the assumptions used for the purpose of estimating the mineral resources and mineral reserves and the development of the economic analyses are sufficiently reliable.

12.4 Filtered tailings storage facility data reviews

As part of Knight Piésold’s prefeasibility study level design of the filtered tailings storage facility, previously generated, site-specific information was reviewed to inform the design of the facility. The primary information reviewed was data generated as part of the geotechnical site investigation campaign performed at the filtered tailings facility site, comprising boreholes, test pits, and geophysical lines, and the topographical survey performed at the site. This is in addition to information shared with Knight Piésold during Knight Piésold’s detailed design of the filtered tailings storage facility at a previously considered site, much of which is applicable to the prefeasibility study design of the current location. For example, unit rates used for the capital expenditure estimate of the filtered tailings storage facility were originally established and verified during the design of the previous facility.

In the opinion of the qualified person, the data and information shared with Knight Piésold to date are sufficiently reliable to inform the prefeasibility study design of the filtered tailings storage facility and develop cost estimates commensurate with the current level of study.

13 Mineral processing and metallurgical testing

13.1 Introduction

Several metallurgical testwork programs have been undertaken between 2009 and 2018 to support the metallurgical assumptions utilized for the progressive Project studies, utilizing samples that were representative of the growing mineral resource as it was known at the time of the studies. These studies included processing method trade off studies as well as refinements of the selected operating parameters, as the properties and response of the samples under the testwork conditions were increasingly better understood.

The following is a summary of the relevant testwork programs and results for samples representative of the material included in the mine plan that have been utilized for the current assumptions in this technical report.

The first two testwork programs considered flowsheets utilizing cyanidation. Testwork was conducted by SGS Lakefield, Canada to support a scoping study by Ventana in 2010 that proposed a flowsheet consisting of comminution, gravity separation, and flotation to produce separate copper and pyrite concentrates, intensive cyanide leaching of the gravity concentrate, cyanide leaching of the combined copper cleaning tailings and pyrite concentrates to recover copper and silver via sulphide precipitation and gold through an adsorption – desorption recovery circuit. Testwork was later undertaken by SGS Lakefield, Canada, JK Tech Pty Ltd of Brisbane, Australia, Citic HIC Australia Ptd Ltd of Silverwater, Australia, and Metso of Belo Horizonte, Brazil to support a scoping study by AUX Colombia in 2012 that proposed a flowsheet consisting of comminution, gravity separation, and flotation to produce a bulk sulphide concentrate, with pressure oxidation of the concentrate followed by solvent extraction and electrowinning to produce copper cathode, and cyanidation for gold recovery.

The remaining testwork programs eliminated the use of cyanidation for environmental reasons. Testwork was conducted by SGS Lakefield, Canada to support trade off studies and a prefeasibility study by Minesa in 2017. The initial trade off study selected flotation as the preferred process route, due to capital costs and environmental advantages over flotation followed by pressure oxidation and cyanide leaching. Further trade off studies directed the work towards the production of sequential copper and pyrite gold flotation concentrates.

Testwork by SGS of Markham and Lakewood, Canada was undertaken to refine and parameterize the flowsheet selected during the 2017 prefeasibility study, comprised of comminution and flotation to produce separate copper and pyrite concentrates, and to validate the key metallurgical assumptions, to support a feasibility study undertaken by Minesa in 2021. In 2024, LogiProc of Sandton, South Africa conducted a gravity gold circuit trade off study based on previous testwork undertaken in 2010, 2017, and 2018 by SGS of Lakewood, Canada, to justify the inclusion of a gravity concentrator in the flowsheet proposed for the current prefeasibility study.

13.2 Samples

The testwork to support the Ventana 2010 study utilized spatially representative samples of the known Mascota deposit at the time, across 700 m of strike and from near surface to a depth of approximately 400 m, including five intervals of partially oxidized material and 16 intervals of sulphide material from 14 drillholes, using a cut-off grade of 2.7 g/t Au. A master composite was prepared from sulphide intervals and five variability composites were prepared from one oxide and four sulphide samples. Two of the lower grade variability samples were blended to form a lower average grade master composite.

The testwork to support the AUX Colombia 2012 study utilized 49 new variability samples, including 20 samples from Mascota, 15 from El Cuatro at Gigante, six from La Bodega at both Mascota and Gigante, three from Gigante, three from Aserradero, and two from Las Mercedes at Gigante. Two composites were created from the Ventana master composite with additional intervals and from 16 drillholes from across the deposit.

The testwork to support the Minesa 2017 study utilized one master composite each from Mascota and Gigante prepared by selecting one drillhole from a 150 m by 150 m grid of the known deposit, using a cut-off grade of 3.0 g/t Au. A total of 24 drillholes from Mascota and 27 drillholes from Gigante were sampled.

The testwork to support the 2021 feasibility study was conducted in 2018, 2019, and 2020.

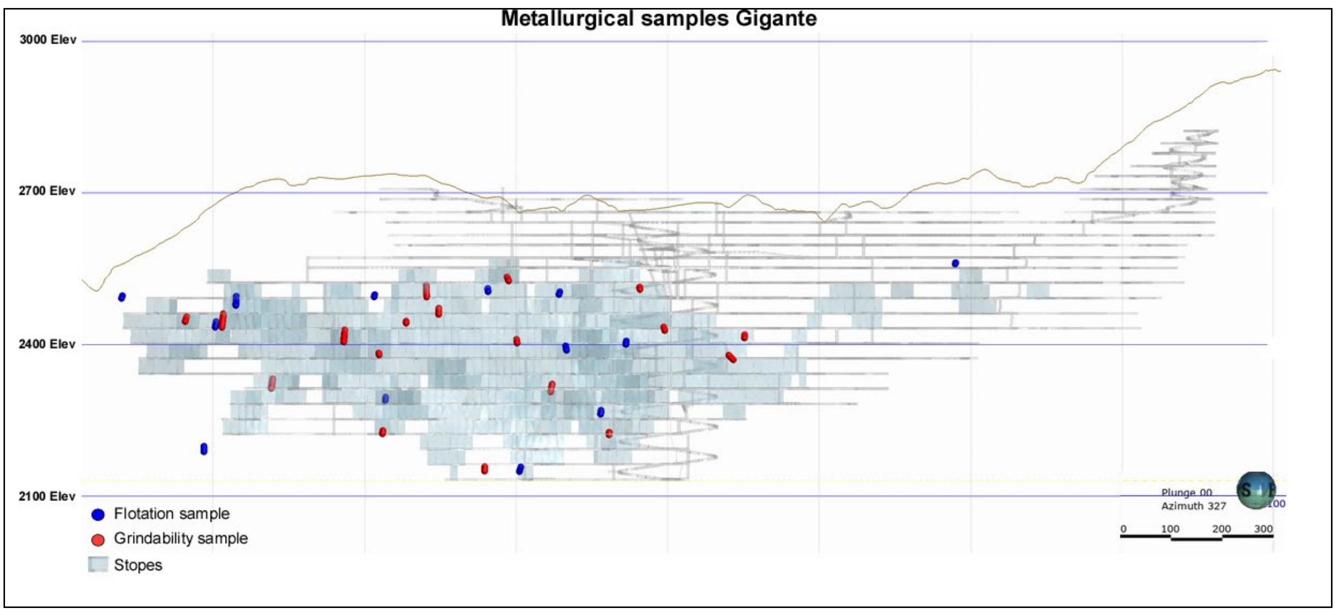
The 2018 testwork utilized a new master composite representing the proposed mining plan for the first three years of operation, comprised of seven drillholes from Mascota and 11 from Gigante, generally at deeper levels in the deposit compared to previous samples, and using a lower cut-off grade criteria.

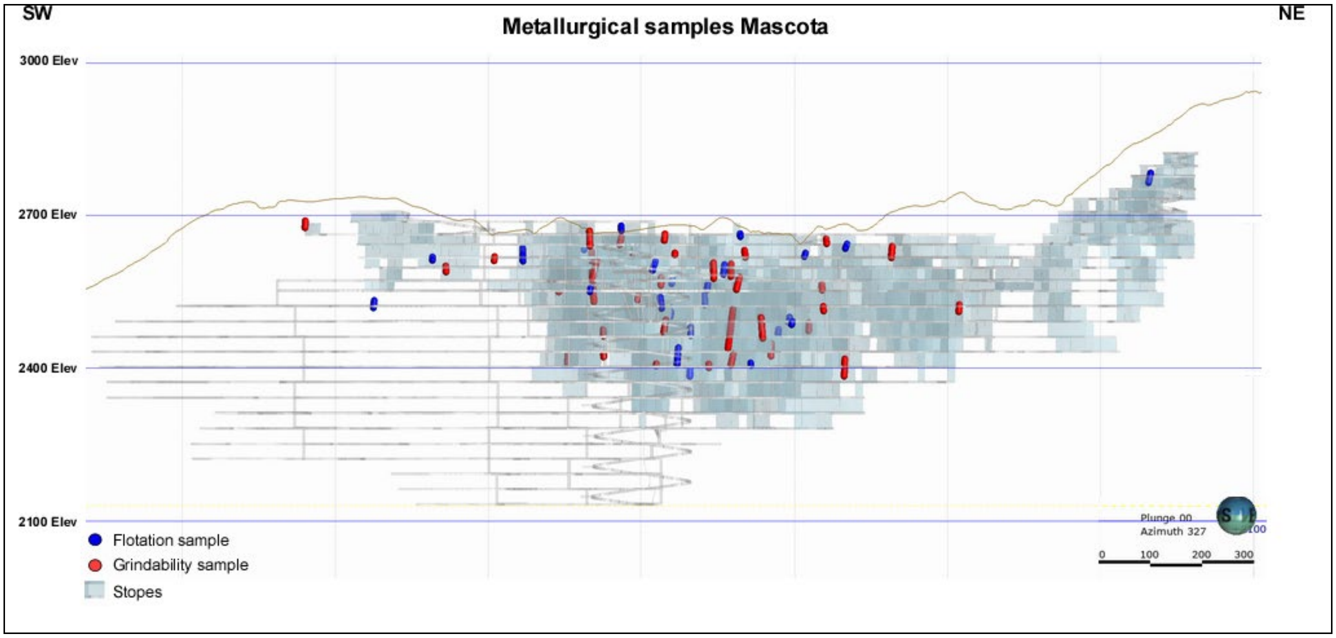
The 2019 testwork used 76 samples for comminution studies and 44 samples for flotation studies. Half of the samples were taken from material scheduled to be mined in the first five years and the remainder from material scheduled for later in the mine life. Samples were taken to ensure equal representation of areas not sampled in previous testwork. A new composite was created to match the average grade of the first five years of mining.

The 2020 testwork utilized 79 sample intervals from Mascota and Gigante for grinding plant throughput assumptions and 46 samples for flotation plant throughput assumptions, following a gap analysis on the information available to date. The process design assumptions were set at 300 tonnes per hour at a grind size of 80% passing 106 microns in a sequential copper and pyrite flotation operation. The copper circuit considered a regrind of rougher concentrate followed by two stages of cleaning in mechanical cells and a final cleaner stage in a column cell. The pyrite circuit considered a regrind of rougher concentrate followed by two stages of cleaning in mechanical cells. All cleaner stage tailings were recycled into previous stages.

For the current prefeasibility study, the 2020 testwork results from samples located within the current mine plan were estimated into the mineral resource and mineral reserve block model to provide recovery and comminution assumptions for the production schedule and economic assessment. These testwork results include 13 flotation samples from Gigante and 24 samples from Mascota, as well as 18 comminution samples from Gigante and 39 samples from Mascota. Long sections of the sample locations relative to the stope designs are provided in Figure 13-1.

Figure 13-1 Long sections of metallurgical testwork sample locations – source Aris Mining, 2025





13.3 Mineralogy and characterization

The seven Ventana 2010 composites and variability samples ranged in grade from 3.31 to 18.5 g/t Au, 11.4 to 108 g/t Ag, and 0.05 to 0.76% Cu. Gold was present in the master composite as native gold and precious metal tellurides, primarily associated with pyrite, with a very small average grain size of five microns. Electrum and silver-rich gold grains were preferentially associated with copper sulphides. The main copper minerals were bornite, chalcocite, chalcopyrite, and copper-arsenic sulphides with minor amounts of covellite and very minor copper-bismuth sulphosalts.

The two AUX Colombia 2012 composites had average grades of 5.81 and 6.56 g/t Au, 0.23 and 0.18% Cu, and 39.9 and 25.9 g/t Ag. Gold was present mainly as native gold and electrum with lesser tellurides. Nearly all of the gold was fine at less than 15 microns and strongly associated with sulphide minerals. Copper minerals were relatively coarse grained and included enargite, chalcopyrite, bornite, chalcocite, and covellite. Enargite was the most abundant copper mineral, containing over 40% of the total copper. Pyrite was the most dominant sulphide mineral at approximately 6% of the mass, with copper sulphides comprising 0.3% of the mass. Sphalerite was present in significant quantities, particularly in some areas of Gigante.

The Minesa 2017 Mascota composite had a grade of 12.40 g/t Au, 50 g/t Ag, and 0.36% Cu, and the Gigante composite had a grade of 7.96 g/t Au, 30 g/t Ag, and 0.26% Cu. The mineral composition was similar to the 2012 composites at approximately 55% quartz, 10% muscovite and clays, 10% pyrite, and 9% alunite. Gold was also strongly associated with sulphide minerals. Enargite was also the most abundant copper mineral at 31 to 34% of the copper in the samples, with the remaining contributed by bornite at 20 to 32%, covellite at 11 to 18%, chalcocite at 5 to 15%, and chalcopyrite at 15 to 17%. Pyrite was also the most significant sulphide at 73 to 75% of the total sulphur content. The Mascota composite had better copper mineral liberation characteristics than the Gigante composite.

The Minesa 2018 composite had a grade of 5.35 g/t Au, 39.7 g/t Ag, and 0.31% Cu. Enargite/tetrahedrite accounted for approximately 28% of the copper; combined bornite, covellite, and chalcocite accounted for 59%; and chalcopyrite accounted for 12%. The 2019 composite had a grade of 5.46 g/t Au, 47 g/t Au, and 0.27% Cu. The Minesa 2020 samples had a wide range of grades at 1.23 to 11.4 g/t Au, 3.5 to 202 g/t Ag, and 0.01 to 2.47% Cu.

In summary the testwork samples indicate that the processed ore will have fine to very fine grained gold, with 90% finer than 15 microns and on average 5 microns, present as native gold, electrum, and tellurides, mostly locked in other minerals or at the grain boundaries of the minerals, predominantly sulphide minerals. Silver is present in native gold, electrum, and telluride, as well as sulphosalts with antimony, arsenic, and bismuth, and is more correlated with copper. Pyrite is the most abundant sulphide mineral and is significantly coarser than the other sulphides. At a sample particle size of 106 microns, over 80% of the pyrite was liberated. Enargite is the most abundant copper bearing mineral with the remaining copper distributed between bornite, covellite, chalcocite, and chalcopyrite. With the majority of gold particles associated with sulphide minerals, high recoveries by flotation are expected, and free gold and some of the larger entrained gold can be recovered through gravity. Fully silica encapsulated gold, which accounts for an estimated less than 1% of the mill feed, cannot be recovered through flotation, but some may be caught in a gravity circuit. Roughly half of the gold is associated with copper sulphide minerals and will be captured in the copper concentrate while the remaining gold associated with pyrite and chalcopyrite will be recovered in the pyrite concentrate. The majority of the copper particles were liberated with the remaining largely associated with pyrite and/or silicates. Regrinding of the copper rougher concentrate may be required to achieve higher final recovery and concentrate grade. The presence of secondary copper minerals in the mill feed will increase the probability of pyrite and sphalerite activation during copper flotation. Silver is also expected to be recovered in greater quantities in the copper concentrate. Arsenic occurred almost exclusively as enargite/tetrahedrite. The type and amount of clay in the expected mill feed is unlikely to cause problems in flotation, settling, or filtration.

13.4 Comminution

The Ventana 2010 master composite was subjected to grinding tests including SMC Test Drop Weight Index, Bond Ball Mill Work Index, Bond Rod Mill Work Index, and Bond Abrasion Test, which characterized the composite as medium hard and highly abrasive.

A subset of the AUX Colombia 2012 variability and composite samples were subjected to the same grinding tests as the Ventana master composite, with similar results to the Ventana testwork.

The Minesa 2017 composites were subjected to the same grinding tests with similar results to the previous tests, with the samples characterized as moderately hard to hard and moderately to highly abrasive.

The Minesa 2019 comminution samples were subjected to SAG grinding power index tests, Bond Work Index test, and SGS Crusher Index tests and the results were estimated into the block model. This testwork characterized the samples as moderately hard for SAG grinding and relatively harder for ball milling than usual, with an average Bond Work Index of 17.6 kilowatt hours per tonne.

The Minesa 2020 comminution samples were subjected to the same tests conducted in 2019, for SAG grinding down to 1.7 mm and ball mill grinding down to 150 microns. The SAG Grinding Power Index results ranged from 38 to 124 minutes with an average of 81 minutes, and the Bond Work Index results ranged from 15 to 21 kilowatt hours per tonne with an average of 17 kilowatt hours per tonne.

This testwork confirmed that the ore is highly variable and moderately hard for SAG grinding and relatively hard for ball milling, compared to other ores. The expectation is that the ore will become increasingly harder at greater depth. The Bond Work Index results indicate that the material becomes abnormally harder at finer sizes, resulting in a higher energy requirement and a coarser grind product from a SAG grinding mill, resulting in reduced throughput. The addition of a pebble crusher to a SAG grinding mill will increase throughput. SAG grinding mills produce flatter particles than a ball mill, resulting in potentially different flotation responses when compared to laboratory testwork. Further, SAG mills are inherently more susceptible to changes in mill feed hardness. The current process flowsheet has made allowance for a SAG grinding mill in closed circuit with a pebble crusher and a ball mill in closed circuit with a cyclone cluster, which greatly mitigates identified risks.

A comminution simulation was conducted by New Concept Projects International of Grand Baie, Mauritius, in September 2024, which resulted in values of 15.9 kWh/t for the Crusher Work Index, 18.8 kWh/t for the Rod Mill Work Index, 19.0 kWh/t for the Ball Mill Work Index, and 0.85 g for the Abrasion Index. These values compare well with the comminution estimates made into the mineral resource and mineral reserve block model prepared for the current prefeasibility study.

The ratio of the Rod Mill Work Index to the Ball Mill Work Index provides a guide for the requirement of a SAG mill operating in closed circuit with a pebble crusher, where ratios greater than 1.3 paired with a Rod Mill Work Index greater than 20 kWh/t requires the inclusion of a pebble crusher. Given the ratio of close to one determined by the comminution simulation, with the Rod Mill Work Index below 20 kWh/t, the SAG and ball mill with pebble crusher configuration is viewed as a robust option, minimizing the generation of ultra fine material and providing the downstream process with a consistent feed.

The simulation indicated that the comminution circuit can handle the nominal proposed feed rate with the SAG mill producing a product of 1500 microns and a ball mill product of 106 microns, assuming a 30% and 250% recirculation for the

SAG mill and ball mill respectively. The simulation highlighted that the desired maximum cannot be achieved with a ceiling value of 152 tonnes per hour. The inclusion of a pebble crusher increases the SAG throughput capacity at a coarser grind size, and is expected to decrease the power requirements.

13.5 Gravity separation and flotation

The Ventana 2010 flowsheet considered the recovery of gold and silver in a gravity concentration circuit followed by leaching of the pyrite flotation concentrate and copper first cleaner-scavenger tailings. The master composite was subjected to gravity separation tests which returned gold recoveries of between 22 and 36% and silver recoveries of 1 to 2%. The assumption used for the economic analysis was that 9.2% of the gold would be present in a gravity concentrate.

Batch rougher flotation tests on the Ventana 2010 master composite were conducted to investigate grind size, collector, pH, and flotation time, initially targeting the production of a bulk sulphide concentrate, and well as copper selective tests. Batch cleaner tests for the production of a copper concentrate followed, producing concentrates of up to 18% copper at a maximum copper recovery of 63%. Copper recoveries for cleaner concentrates greater than 15% copper ranged from 40 to 70%. The highest gold recovery reported for the cleaner tests was 61%. The flotation response was generally insensitive to grind size. Sequential flotation tests of copper followed by pyrite resulted in the best cleaner copper grade of 30% at 56% copper recovery and 52% gold recovery. Tests at lower copper grades of between 19 and 22% copper returned higher copper recoveries of between 70 and 76%. Total gold and silver recovery to both concentrates was between 96 and 97%. The optimum grind size for rougher flotation was determined to be at a P_{80} of 72 microns.

Unoptimized batch rougher-cleaner tests with gravity separation ahead of flotation on the first AUX Colombia 2012 master composite returned variable results, with poor copper cleaner grades and high tailings gold losses in some of the samples. Grind size optimization testwork was conducted at the rougher stage on four samples, which showed decreasing gold tail grades over the range of sizes tested at P_{80} of 90 to 40 microns, with only a marginal increase in gold recovery below a P_{80} of 60 microns. Copper recovery to the copper rougher concentrate also maximized at a P_{80} of 60 microns on three of the four samples. The AUX Colombia 2012 flowsheet considered gravity concentration and intensive leach and a bulk flotation circuit. The gravity separation testwork had highly variable results with the study assuming 8.7% of the gold present in the gravity concentrate.

A locked cycle test was also undertaken during this testwork on the Ventana 2010 master composite, producing a copper cleaner concentrate and a combined pyrite cleaner concentrate and copper cleaner scavenger tail for subsequent cyanide leaching. The copper cleaner concentrate assayed 14.7% copper at a copper recovery of 83.2% and a gold recovery of 66.0%. The total gold recovery, including to the copper cleaner concentrate, the pyrite cleaner concentrate, and the copper cleaner scavenger tail, was 97.2%. A subsequent locked cycle test, which had a gravity separation stage ahead of flotation, produced similar results with a copper cleaner grade of 16.2% copper at a copper recovery of 78.2%, and an overall gold recovery of 97.4%, with 27.4% reporting to the gravity concentrate. A locked cycle test on the lower grade master composite produced a copper concentrate assaying 12.1% copper at a copper recovery of 60.1% and gold recovery of 41.7%. The overall gold recovery was 97.8%, with 9.2% reporting to a gravity concentrate.

A locked cycle test on the first AUX Colombia 2012 master composite produced a copper concentrate assaying 15.3% copper at a copper recovery of 82.9% and a gold recovery of 50.9%, for an overall gold recovery of 96.6%, with 12.5% reporting to the gravity concentrate.

Open circuit rougher-cleaner tests were also conducted on the Ventana 2010 variability samples. Except for one variability sample with very low copper grades, copper concentrate grades ranged from 14 to 24% at copper recoveries ranging from 64 to 82%. Total gold recoveries ranged from 90 to 92% for the oxide and low grade variability samples to between 98 and 99% for the other samples.

An open circuit rougher-cleaner test was conducted on the second AUX Colombia 2012 master composite, which produced a copper concentrate assaying 13.8% copper at a copper recovery of 75.4% and a gold recovery of 48.7%. The gold loss to the pyrite rougher tailings was 4.5%.

The Minesa 2017 flotation testwork program, which considered a flowsheet including sequential flotation producing separating copper and pyrite concentrates, commenced with batch flotation tests, which found that at the target 16% copper concentrate grade, copper recovery for Mascota was 70 to 74% and 68% for Gigante. Gold recovery to the copper concentrate was 40% for Mascota and 35% for Gigante, with a total gold recovery of 95% for Mascota and 89% for Gigante. The flotation performance was largely unaffected by primary grind sizes between 106 and 170 microns for Mascota and 75 to 140 microns for Gigante. Therefore, as of 2017, the final grind size in the cyclone overflow was targeted at a P_{80} of 106 microns. Compared to the previous programs, gold recovery was 2% lower at Mascota and 4% higher at Gigante. Four locked cycle tests were undertaken, with copper concentrate grades ranging from 15.4 to 17.5% copper with gold recoveries ranging from 78.9 to 91.8% and copper recoveries ranging from 73 to 78% copper. A further 25 batch variability tests were undertaken using individual sample intervals, with scattered results that on average matched the copper grade and recovery and gold grade of the locked cycle tests. This testwork showed that maintaining a deeper froth and allowing a longer flotation time reduced regrinding requirements and provided greater flexibility in achieving the target 16% copper concentrate grade. This testwork indicated that, from the testwork conducted to date, gold recovery is insensitive to head grade. Assays were conducted by size distribution with the results from the Mascota composite suggesting that potentially up to 15% of the gold could be recovered by a gravity concentrator. Minesa opted to exclude a gravity concentrator from the 2017 and subsequent study flowsheets on the basis that the incremental benefit gained form gold bullion over gold in concentrate would not justify the additional capital and operating cost requirements.

The Minesa 2018 flotation batch tests confirmed the relationship between grind size and copper recovery noted in previous testwork. A locked cycle test on the 2017 Mascota composite conducted under the optimized flotation conditions of deeper froth and longer cleaner residence times resulted in a significant performance improvement, with a copper concentrate assaying 19.4% copper at a recovery of 81.5% and a total gold recovery of 94.0%. The 2018 composite was tested under the same conditions, resulting in a copper concentrate assaying 16.3% copper at a copper recovery of 86.7% and an overall gold recovery of 95%, with a 10% reduction of non-sulphide gangue in the copper and pyrite concentrates. A review of gold recovery from samples selected from across the deposits indicated that lower gold recoveries are associated with increased proportions of gold locked in silicates and sulphide-silicate composites, rather than with lower gold grades.

The Minesa 2019 flotation tests included a MinnovEx flotation test on each sample to provide rougher flotation parameters, a mineralogical analysis on each sample to link assay grades to the mineralogy to provide rougher stage flotation parameters for the copper and pyrite concentrate streams, and batch rougher and cleaner tests with regrind to determine cleaner stage kinetics. The results were estimated into the block model. The overall recoveries to the combined pyrite and copper concentrates ranged from 89 to 94.5% for gold and 86 to 93% for silver. The copper concentrate grade was fixed at 16% copper, with a recovery of approximately 75% except in lower grade portions of the deposit.

The Minesa 2019 composite was subjected to 20 locked cycle tests to produce 1 kg of copper concentrate and 12 kg of pyrite concentrate. The concentrate assays were 165 g/t Au for the copper concentrate and 36.1 g/t Au for the pyrite concentrate, 2014 g/t Ag for the copper concentrate and 178 g/t Ag for the pyrite concentrate, and 16.9% Cu for the copper concentrate and 0.55% Cu for the pyrite concentrate. The recovery values were estimated into the model and ranged from 89.6 to 94.4% for gold in the combined concentrates, 87 to 91% for silver, and 90.1 to 94.1% for copper.

The Minesa 2020 flotation samples were subjected to sequential copper and pyrite flotation by the Modified Flotation Test to determine the mineral flotation kinetics in rougher flotation. Samples were also subjected to rougher-cleaner tests to determine the kinetics in cleaner flotation after regrinding. The results indicated that the flotation response is very variable in the copper flotation stage, partly due to head grade variation, and that almost all of the metals were recovered in the pyrite flotation stage. With the copper concentrate grade set at 16% copper, the average copper recovery was typically around 75%. Gold recovery to the copper concentrate varied with head grade in the range of 40 to 55% with variable gold grade of about 200 to 400 g/t. With the pyrite concentrate grade fixed at 48% sulphur and about 10% non-sulphide gangue, the average gold recovery was typically around 86% of the remaining gold, with gold grades in the range of 25 to 40 g/t, averaging 33 g/t Au. The recoveries in the combined concentrates ranged from 90 to 94.5% for gold and from 87 to 91% for silver.

A gravity gold circuit trade off study was conducted by PSN in 2024. The 2010 testwork was conducted on the Mascota composite, which indicated the presence of free gold. The 2017 testwork provided insights into the performance of flotation recoveries when combined with a milling grind size that caters for a gravity concentration circuit at a P_{80} of 150 microns. The size fraction assay results for the Mascota composite indicated the potential to recover up to 15% of the gold in a gravity circuit and up to 4.7% for the Gigante composite, with significantly higher gold grades present in the plus 106 micron size fraction. The results of the studies showed that total gold recoveries were the same with or without a gravity circuit, but that a combined gravity and flotation circuit had potential benefits and should be evaluated further. However, the decision had been made to eliminate the gravity circuit in the 2017 study, and no further studies were conducted. The majority of the gold present in the 2018 master composite, at 96.3%, was noted to occur in the minus 106 micron size fraction.

The findings of the trade off study were that there is gravity recoverable gold present in the Soto Norte ores and that the inclusion of a gravity circuit does not appear to have any adverse impact on the overall recovery of the economic minerals. The main concern is the lack of data from Gigante and the marketing effects for selling copper and pyrite concentrates with a reduced gold content. The benefits of a gravity circuit include the reduction of variability of gold in flotation, lowering the mass pull and producing a higher grade gold-silver concentrate while maintaining the overall recovery of the system, quicker and more pronounced process optimizations, and is unaffected by variations in clay content of the mill feed. The incorporation of a gravity concentrator in the processing flow sheet will not have any significant impact on the downstream circuit and will significantly mitigate the risk of losing coarser gold to the flotation tailings. This could potentially facilitate the recovery of up to approximately 15% of the gold either in the form of a gold-silver concentrate or as a doré, instead of recovering to the copper concentrate, which is economically more favourable. Other economic benefits include higher gold payabilities from a doré product compared to a concentrate product on the order of between 3.9 and 9.9%, and minimal or no penalties. The cost benefits, at a high level, outweigh the relatively low capital cost requirements for a gravity circuit and gold safe room and negligible operating costs.

In summary, the findings of these programs were that both Mascota and Gigante material respond well to flotation; varying the grind size between 106 and 170 microns at Mascota and between 75 and 140 microns at Gigante had limited impact on flotation response; gold recovery increased with the recirculation of the pyrite cleaner tailings, pyrite losses in the final tailings were attributed to fine size, poor liberation, and/or insufficient collector dosage which adversely impacts gold recovery; elevated levels of non-sulphide gangue could be addressed by additives and extending cleaner residence time and providing a deeper froth; and that the rougher concentrate required further grinding to 25 microns and the pyrite concentrate to 45 microns for improved gold and copper recovery.

13.6 Thickening and filtration

Thickening and filtration testwork on the Minesa 2017 composites showed optimum thickening rates varying from 0.06 m^2 per tonne per day for the copper concentrate and 0.09 to 0.10 m^2 per tonne per day for the pyrite concentrate and tailings samples. Disc/drum vacuum filtration rates varied from 310 to 850 kg per m^2 per hour for residual moisture contents of 12 to 14%. Pressure filtration rates were up to 1,300 kg per m^2 per hour for residual moisture contents of 12 to 14% for the tailings, 1,800 kg per m^2 per hour for a residual moisture content of 10% for the copper concentrate, and 1,600 per m^2 per hour for a residual moisture content of 8.4% for the pyrite concentrate.

All thickening applications for the copper and pyrite concentrate and tailings responded well with the addition of flocculant. Pressure filtration proved the most successful resulting in higher throughputs and slightly reduced cake residual moistures.

13.7 Deleterious elements

Potential deleterious elements that can derive economic penalties include arsenic present in enargite/tetrahedrite that will recovered reaching levels of up to approximately 3% into the copper concentrate; antimony present in tetrahedrite that may be present in the concentrate in amounts that may incur penalties; zinc mostly present in sphalerite that is present in higher quantities in certain parts of the deposit but has only a minor association with copper sulphides and pyrite; bismuth present in a wide range of minerals; cadmium, and mercury that are present in certain areas of the deposit. Grade control will be required to mitigate penalties.

13.8 Environmental characterization

An acid base accounting test was conducted on the rougher tailings from Ventana's first locked cycle test, which indicated the material to be potentially acid generating, although a net acid generating test indicated that the sample would not generate acid. A strong acid digest elemental analysis indicated that lead was the main element present at an environmentally significant concentration in the sample.

Numerous acid base accounting tests were conducted by AUX Colombia with the majority of the results classified as potentially acid generating and the remainder as uncertain. One strong acid digest elemental analysis indicated that World

Bank controlled parameters were at concentrations within the specified limits while another reported an acidic pH value and copper and zinc concentrations in excess of World Bank limits. Two toxicity characteristic leaching procedure tests reported metal concentrations that were not expected to be of environmental concern. A shake flask extraction test reported all World Bank controlled parameters to be at concentrations within their specified limits. Humidity cell testing consistently reported acidic leachates and concentrations of copper in excess of the World Bank limits. Cumulative depletion rates indicated that the carbonate content of the flotation tailings was exhausted after nine weeks of leaching, and it was expected that acidic drainage would require management.

Testwork conducted on two of Minesa's 2017 rougher flotation samples generated from large cell flotation tests resulted in an acid base accounting test classification of not potentially acid generating, which was confirmed by the net acid generating test. All parameters controlled by the World Bank were within the designated standards in humidity cell testing except for mercury in one sample at El Gigante. The testwork indicated net neutral conditions under certain conditions. A strong acid digest elemental analysis of the humidity cell test residue reported all parameters at concentrations within the World Bank guidelines, and net acid generating testing reported no net acidity.

The results of the humidity cell testwork were included in a numerical model to estimate the metal content of seepage water from the filtered tailings facility, to characterize the required treatment prior to permitted release to the environment or for potential re-use as process plant makeup water.

13.9 Conclusions and recommendations

The metallurgical testwork to date has been conducted on a wide range of samples representative of the material expected to be processed over the life of mine. The studies have been conducted to a sufficient quality and extent to support the process flow sheet presented in this prefeasibility study and has been utilized to support the previous feasibility study. The results of this testwork has been estimated into the mineral resource and mineral reserve block model, with the estimated variable used to develop the production schedule and economic analyses.

Flotation testwork has been conducted of sufficient quality to parameterise the circuit including deriving the recoveries of minor elements, including deleterious elements, including the deportment of deleterious elements such as cadmium, mercury, zinc, bismuth, antimony, and arsenic, which may incur penalties although they are not deemed to pose a significant risk to the economic outcomes. There are no known processing factors or deleterious elements that could have a significant effect on the economic extraction of the ore that have not been considered and accounted for in the processing plan and economic model.

No fatal or significant flaws were detected during the review of the testwork and the current proposed flowsheet. The level of testwork conducted to date that supports the development of a robust process design criteria document which has resulted in a flowsheet that recovers the required amount of gold, silver, and copper at saleable grades meets the typical expectations for a prefeasibility level of study. The flowsheet developed to produce a separate copper and pyrite concentrate through sequential flotation is viewed as the most technical and economically viable solution while mitigating risk.

Compared to the 2021 feasibility study, the main process changes in this current prefeasibility study include a separate receiving and processing facility for contract mining partners; the replacement of a crushing and closed circuit ball mill circuit with a closed circuit SAG and ball mill circuit to improve control and a stable feed to the flotation plant; the inclusion of a gravity circuit after the closed circuit ball mill to mitigate losses of coarse gold in flotation, including a Knelson concentrator for roughing purposes and a shaking table for concentrate polishing; and dispatching the pyrite tailings to the tailings thickener and/or for the purpose of providing paste filling for use in the mine, rerouting to a set of pulse fitters.

Geometallurgical modelling and statistical analysis has highlighted the requirement of additional samples to accurately predict process outcomes.

Overall indicative recoveries to the combined concentrates achieved are forecast to be 92.8% for gold, 88.7% for silver, and 92.8% for copper at a grade of above 16%.

Producing a copper concentrate greater than 16% copper is considered economically viable even with expected entrained deleterious elements. The geometallurgical testwork program represents a significant investment in understanding and quantifying the metallurgical variability in the deposit and has been incorporated in the mineral resource and mineral reserve block model developed for the current prefeasibility study. Any adverse effect on plant performance due to variability could be minimized by adopting a suitable mining schedule and blending the lower grade mill feed with higher grade material.

The following recommendations are made:

- Undertake additional comminution testwork from an increased density of samples representative of the life of mine plan to provide data for more accurate comminution simulations, for an estimated cost of $17,000.
- Undertake settling and filtration testwork with actual site process water and recirculated water, for an estimated cost of $30,000.
- Undertake additional testwork to characterize the gravity recoverable gold present in samples representative of the material in the life of mine plan, for an estimated cost of $6,000.
- Undertake additional locked cycle tests on samples representative of the material in the life of mine plan to characterize the nature of the gravity, copper, and pyrite concentrates, for an estimated cost of $10,000
- Undertake a more detailed financial analysis of the economic benefits of a gravity circuit and investigate alternative technologies, for an estimated cost of $5,000.
- Undertake an assessment of the logistics and environmental impacts of including a gravity circuit, for an estimated cost of $12,000.

14 Mineral resource estimates

14.1 Disclosure

The current mineral resource estimate was prepared by Kate Kitchen, MAIG, Area Manager – Geology of Mining Plus. The mineral resource estimate has been prepared in compliance with the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines of 2019 (CIM, 2019) and reported in compliance with the CIM Definition Standards for Mineral Resources and Mineral Reserves of 2014 (CIM, 2014).

All available drillhole data has been considered for the mineral resource estimate. The mineral resource estimate has been depleted for mining and utilizes a cut-off grade calculation with an effective date of August 18, 2025.

Other than the significant factors and risks described in Section 4.7 of this technical report, there are no other known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors or risks that could materially affect the mineral resource estimate.

14.2 Available data

The mineral resource estimate utilizes 882 diamond drillholes totalling 365,088 m completed between 2006 and 2019. A total of 22 drillholes were excluded from the estimation process, fifteen were removed due to being closely spaced or twinned, and seven were excluded because their orientation was parallel to the mineralized trend. The drillhole spacing ranges from 12.5 by 12.5 m to 100 by 100 m.

14.3 Geological interpretation

Three dimensional models of lithology, weathering, major structures, and mineralization were constructed in Leapfrog 3D geological modelling software utilizing the logged intervals of geological information and sample assays, and constrained to the surface using a digital cartography, aerial light detection and ranging (LIDAR), and a metric camera.

Six lithological units were modelled, with the majority rock unit comprised of gneisses of the Bucaramanga Complex and less than 1% each of leucogranites of the Santander Plutonic Group, breccias, intrusions, porphyry, and overburden.

Three weathering zones were modelled based on oxidation data collected mainly for geomechanics records. The majority of the deposit is fresh rock, with a small amount of transitional and oxide zones. The transitional zone is associated with breccia and follows faults at depth. The oxide zone is thin and does not form a continuous layer.

The interpreted structural model consists of 18 structures and was developed from structural logging and faults coded in the lithology table.

Mineralization at the Project is not strictly controlled by lithology but follows a broader structural trend. Mineralization is present in hydrothermal breccias and veins, as well as extends beyond those features into surrounding host rocks. The geometry of the mineralized domains was interpreted for Mascota, Gigante, and Aserradero based on the distribution of gold grades above a grade threshold, aligned with the structural trends. A high grade interpretation capturing mineralization above 4 g/t of gold were created at Mascota and Gigante, and at all deposits, three sets of interpretations capturing mineralization above 0.7 g/t, 0.22 g/t, and 1.9 g/t Au were developed. Mascota represents 46% of the total volume and is the most economically significant structure, followed by Gigante and Aserradero.

14.4 Statistics, sample compositing, and treatment of extreme grades

Samples were composited to a target length of 1 m, with the residual split evenly over the selected intervals. To address rarely occurring outlier grades, top cutting was reviewed and applied where necessary for each element within each domain. The top cut composite gold grade statistics are shown in Table 14-1.

Table 14-1 Composite statistics

Domain	Number	Maximum (Au g/t)	Top cut mean (Au g/t)	Coefficient of variation
Mascota 1.9 g/t	10,351	462.00	3.36	1.9
Mascota 4 g/t	6,409	2,135.00	12.24	2.2
Gigante 1.9 g/t	4,164	172.00	3.60	1.9
Gigante 4 g/t	1,963	263.90	10.25	1.6
Aserradero 1.9 g/t	1,855	90.00	3.04	1.3

14.5 Block model construction and depletion for mining

A three dimensional block model was constructed covering all the interpreted mineralized zones, constrained within the topographic surface. The selected block size was defined based on the data configuration, using a parent cell size of 10 m east, 5 m north, and 10 m elevation, with sub blocks of 1 m in each direction to better fit the interpretations.

14.6 Composite search and interpolation parameters

Normal scores variograms were generated and models were fitted on directional variogram plots by domain for gold, silver, and copper. The nugget comprised approximately 30% of total variability for gold and for most domains, 90% of the total variability had a range of between 30 and 80 m, averaging 55 m.

Contact analysis was conducted for gold to determine the nature of gold trends across estimation domain boundaries, which established the use of hard boundaries for all domains.

Grade estimates for the purpose of the mineral resource estimate were made for gold, silver, and copper in each domain, utilizing hard boundaries, using ordinary kriging on the 1 m composites. Dynamic anisotropy was utilized to allow the search ellipse to orient itself relative to changes in the strike and dip of the wireframes.

A three search strategy was designed by domain with the first search for the 4 g/t Au domains extending between 58 and 79 m along strike, 50 m down dip, and 10 m across strike, utilizing a minimum of 8 composites and a maximum of 16. The second search extended between 87 and 118 m along strike, 75 m down dip, and 15 m across strike, utilizing a minimum of 8 composites and a maximum of 16. The last search extended a similar distance but required a minimum of 2 composites and a maximum of 8. A restriction of 4 composites per drillholes was utilized for all searches.

Bulk density was estimated from the sample data using inversed distance squared within each of the mineralization domain, using between 2 and 10 samples. Any block not receiving an estimate was coded for the average of each estimation domain.

Grade estimates for the purpose of characterizing concentrate qualities were made for arsenic, bismuth, cadmium, iron, mercury, lead, antimony, sulphur, tellurium, thorium, uranium, and zinc.

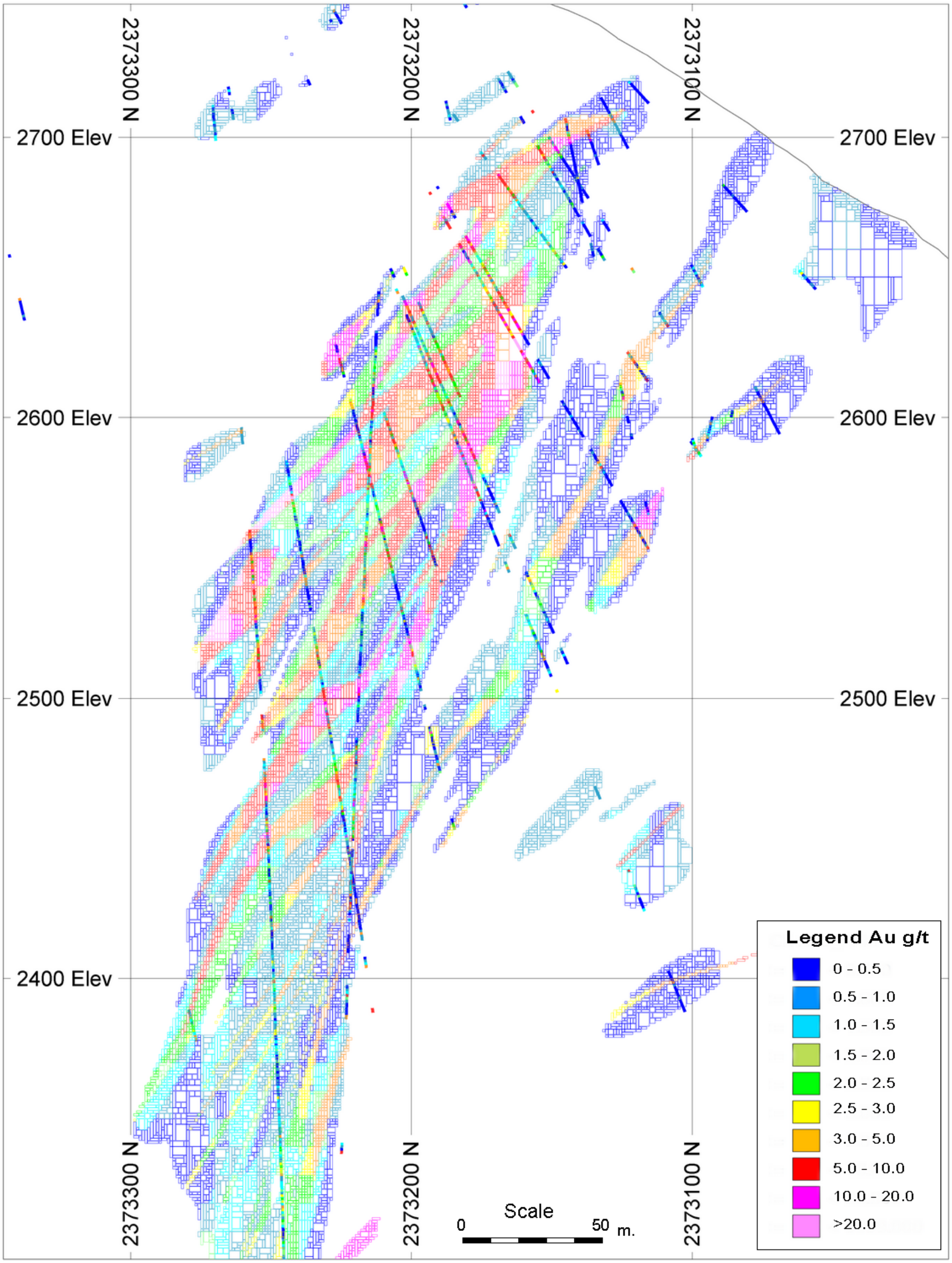
Estimates of metallurgical parameters for the purpose of characterizing processing and recovery parameters based on metallurgical testwork data were made for SAG power index; ball work index; crusher index; and recovery of gold, silver, copper, pyrite, sphalerite, and non sulphide gangue to the pyrite and copper concentrates.

14.7 Estimation validation

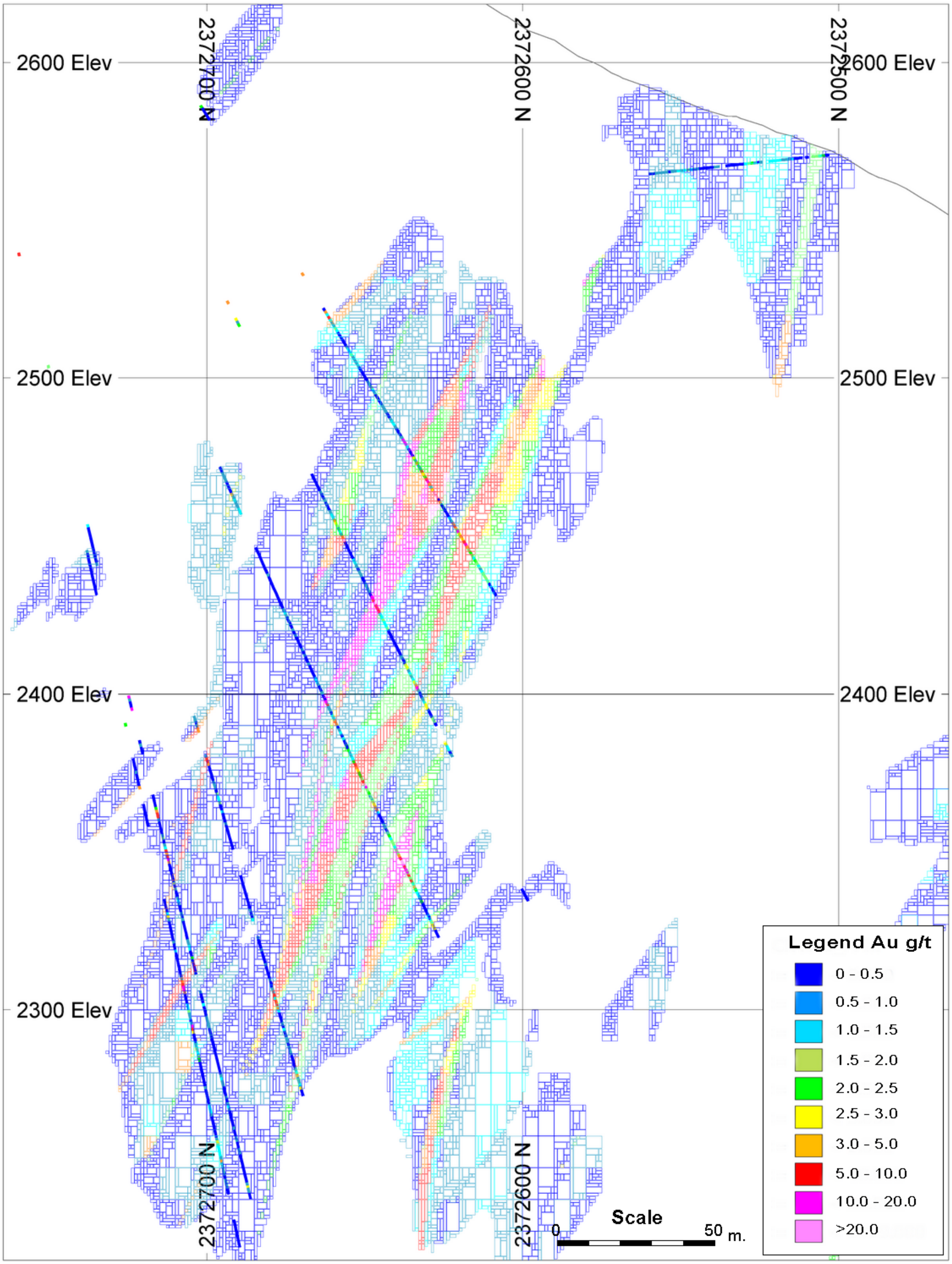
The estimate was validated using standard visual and statistical methods with a good fit of the estimate to the composites. Example cross sections of estimated and input composite gold grades at Mascota and Gigante are shown in Figure 14-1. Global bias was assessed by comparing the means of the estimated and composite grades in each estimation domain, which showed a weighted average variation of -4%. Estimated grade trends were validated by generating swath plots by easting, northing, and elevation. In Mining Plus’s opinion, there is no indication of local bias.

Figure 14-1 Cross sections of estimated and input composite gold grades – Source Aris Mining 2025

Mascota at 501038 mN, ± 12.5 m



Gigante at 5009970 mN, ± 12.5 m



14.8 Mineral resource classification

Mineral resources were classified based on drillhole spacing and the number of estimation search passes required to obtain an estimate. Measured classification was applied where the drill spacing is present on a 20 by 20 m grid and the estimate was made within the first and second pass using a minimum of three drillholes. Indicated classification was applied where the drill spacing is present on a 40 by 40 m grid and the estimate was made within the first and second pass using a minimum of three drillholes. Inferred classification was applied to the remainder of the estimate out to the second pass.

14.9 Cut-off grade and mineral resource constraint

The mineral resource estimate has been tabulated using a cut-off grade of 1.6 g/t Au, based on the assumptions shown in Table 14-2. The mineral resource estimate is constrained within mineable stope optimizer shapes generated at a 1.6 g/t Au cut-off grade and using a 3.8 m minimum mining width, and is inclusive of material below 1.6 g/t Au within the shapes.

Table 14-2 Mineral resource cut-off grade

Metric	Unit	Value
Gold Price	US$/oz	2,600
Gold treatment and refining charges	US$/oz	150
Effective gold royalty	%	3.7
Gold recovery	%	92.8
Gold payability	%	92.6
Mining operating costs	US$/t	42
Processing and surface infrastructure costs	US$/t	22
G&A costs	US$/t	20
Operating profit margin	%	25
Cut-off grade	g/t Au	1.6

14.10 Mineral resource tabulation

The mineral resource estimate has been tabulated using a cut-off grade of 1.6 g/t Au, based on a gold price of $2,600 per ounce, an overall gold metallurgical recovery of 92.8%, a mining cost of $42 per tonne, a processing cost of $22 per tonne, a G&A cost of $20 per tonne, and an effective 3.7% gold royalty.

The mineral resource estimate is constrained within mineable stope optimizer shapes generated at a 1.6 g/t Au cut-off grade and using a 3.8 m minimum mining width, and is inclusive of material below 1.6 g/t Au within the shapes.

The mineral resource estimate for the Soto Norte Project effective August 18, 2025 is shown in Table 14-3.

Table 14-3 Soto Norte mineral resources effective August 18, 2025

Classification	Tonnes (Mt)	Gold grade (g/t)	Silver grade (g/t)	Copper grade (%)	Contained gold (Moz)	Contained silver (Moz)	Contained copper (Mlb)
Measured	3.8	7.99	36.8	0.25	1.0	4.6	21.4
Indicated	35.2	5.29	27.3	0.18	6.0	30.9	137.8
Measured + Indicated	39.0	5.55	28.2	0.19	7.0	35.5	159.2
Inferred	25.1	4.81	24.6	0.13	3.9	19.9	74.5

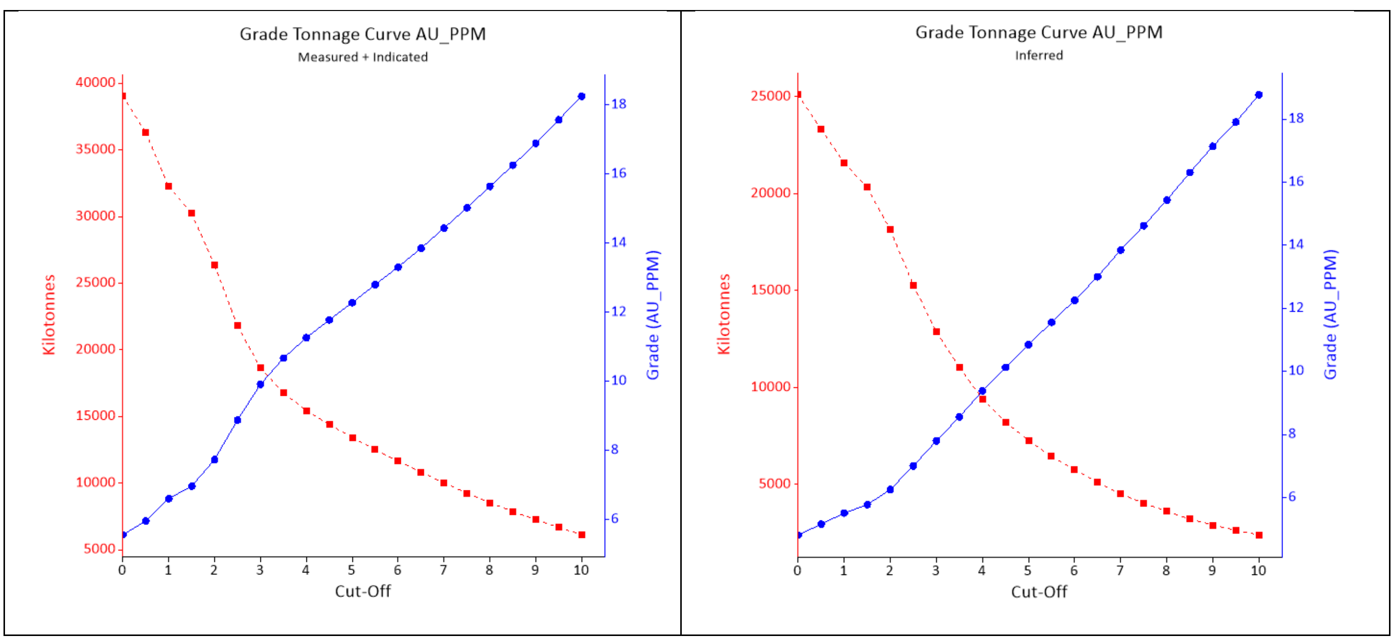
Notes:

- Totals may not add due to rounding.
- Mineral resources are inclusive of mineral reserves.
- Mineral resources are not mineral reserves and have no demonstrated economic viability.
- A gold price of $2,600 per ounce was used for the mineral resource estimate.
- The mineral resource estimate utilized a gold cut-off grade of 1.6 g/t.
- The mineral resource estimate was constrained within mineable optimizer shapes generated at a cut-off grade of 1.6 g/t Au and using a 3.8 m minimum mining width, and is inclusive of material below 1.6 g/t Au in the shapes.
- The mineral resource estimate was prepared by Kate Kitchen, MAIG of Mining Plus who is a qualified person as that term is defined by NI 43-101.
- There are no known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors or risks that could materially affect the mineral resource estimate or the development of mineral resources.

14.11 Mineral resource sensitivity to cut-off grade

Grade tonnage curves showing the sensitivity of the mineral resource estimate to cut-off grade for measured + indicated and inferred material are shown in Figure 14-2.

Figure 14-2 Mineral resource sensitivity to cut-off grade - Source Mining Plus 2025



14.12 Recommendations

The Project's expected mine life of 22 years is based on the current mineral reserve estimate, and there are opportunities for future mine extension if additional mineral reserves are defined through continued exploration. There are numerous areas of high grade inferred material within the mineable stope optimizer shapes used to constrain the mineral resource estimate that are located adjacent to the mineral reserve stopes designed around indicated material that could be targeted for exploration. An initial exploration drilling program of 35 drillholes for approximately 12,500 m is recommended to target the highest grade areas of inferred mineral resources, and those located in the upper areas of the mine, comprising 1.2 Mt at 12.50 g/t Au for 482,000 ounces, to potentially convert those volumes to indicated mineral resources, at an estimated cost of $1.3 M.

15 Mineral reserve estimates

15.1 Disclosure

The current mineral reserve estimated was prepared under the supervision of and reviewed by Peter Lock, FAusIMM, Executive Director and Principal Mining Consultant of Mining Plus. The mineral reserve estimate has been prepared in compliance with the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines of 2019 (CIM, 2019) and reported in compliance with the CIM Definition Standards for Mineral Resources and Mineral Reserves of 2014 (CIM, 2014).

The mineral reserve estimate utilizes a cut-off grade calculation with an effective date of August 18, 2025.

Other than the significant factors and risks described in Section 4.7 of this technical report, there are no other known mining, metallurgical, infrastructure, permitting, or other relevant factors or risks that could materially affect the mineral reserve estimate.

15.2 Method

15.2.1 Cut-off grade

The mineral reserve estimate comprises measured and indicated mineral resources that can be mined economically utilizing a gold cut-off grade based on a mineral reserve gold price assumption and metallurgical recovery and operating cost estimates. A summary of the cut-off grade assumptions for the break-even cut-off grade of 2.0 g/t Au is shown in Table 15-1.

Table 15-1 Mineral reserve cut-off grade

Metric	Unit	Value
Gold Price	US$/oz	2,200
Gold treatment and refining charges	US$/oz	150
Effective gold royalty	%	3.7
Gold recovery	%	92.8
Gold payability	%	92.6
Mining operating costs	US$/t	42
Processing and surface infrastructure costs	US$/t	22
G&A costs	US$/t	20
Operating profit margin	%	25
Cut-off grade	g/t Au	2.0

15.2.2 Stope optimization

Stope optimization based on the selected mining method of longhole open stoping with backfill was performed on the mineral resource blocks coded as measured and indicated using Mineable Shape Optimizer (MSO) software to determine the optimal shape and location of the stopes. In the development ore design, a minor amount of inferred material was included in the mining inventory, representing less than 1.7% of the total ore tonnes. A fixed stope length of 15 m was assumed with stope heights varying between 20 and 25 m in the upper areas of the mine and 30 m in the lower areas. A minimum mining width of 2.5 m was used.

An equivalent linear overbreak slough (ELOS) dilution of 0.65 m was assumed for the hangingwall and footwall sides of each stope in the upper areas of the mine, and 0.75 m was assumed for both sides of the stope in the lower areas. Additional dilution of 5% was applied to secondary stopes to account for overbreak into any previously backfilled adjacent stopes, mining under the cemented sill pillar, or dilution mined from the top of any previously backfilled stopes. No additional dilution was applied to development ore or primary stopes. A minimum pillar with a width of 7 m was maintained between any parallel stopes.

Mined ore recovery factors were applied based on stope width, with 100% ore recovery assumed for stopes less than 5 m wide, 95% for stopes between 5 and 10 m wide, and 90% for stopes greater than 10 m wide.

A crown pillar of 50 m from the surface was excluded from the optimization. Any stopes located within 10 m of any known artisanal mining and any stopes interfering with the designed capital infrastructure were removed from the inventory.

15.3 Mineral reserve tabulation

The mineral reserve estimate for the Soto Norte Project effective August 18, 2025 is shown in Table 15-2.

Table 15-2 Soto Norte mineral reserves effective August 18, 2025

Classification	Tonnes (Mt)	Gold grade (g/t)	Silver grade (g/t)	Copper grade (%)	Contained gold (Moz)	Contained silver (Moz)	Contained copper (Mlb)
Proven	2.6	8.78	37.1	0.25	0.7	3.0	14.2
Probable	17.7	6.72	31.4	0.19	3.8	17.9	75.0
Proven + Probable	20.3	7.00	32.1	0.20	4.6	20.9	89.2

Notes:

- Totals may not add due to rounding.
- A gold price of $2,200 per ounce was used for the mineral reserve estimate.
- The mineral reserve estimate was constrained within mineable optimizer shapes and utilized a cut-off grade of 2.0 g/t Au.
- The mineral reserve estimate was prepared under the supervision of and reviewed by Peter Lock, FAusIMM, CP, of Mining Plus, who is a qualified person as that term is defined by NI 43-101.
- Other than as disclosed in this technical report, there are no known mining, metallurgical, infrastructure, permitting, or other relevant factors or risks that could materially affect the mineral reserve estimate or the development of the mineral reserves.

16 Mining methods

16.1 Introduction

The planned mining method is longitudinal open stoping with backfill, a safe, efficient, and modern mining method, and is planned to have a production rate of 2,750 tpd, in line with the processing plant capacity dedicated to the underground mine. The Project's expected mine life of 22 years is based on the current mineral reserve estimate, and there are opportunities for future mine extension if additional mineral reserves are defined through continued exploration.

16.2 Mine access and general layout

A 4 km long central decline from the surface, utilizing the existing portal, is planned for the main transportation route for personnel and equipment, and extends to the full depth of the mine design over a vertical range of 700 m. Two additional inclines will provide access to the upper levels of Mascota. In the southwest, a 360 m incline provides access to two levels above, and in the northwest, another 1.3 km incline provides access to an additional seven levels. These developments are 5.5 m wide and 5.5 m high, allowing for haul trucks and secondary ventilation ducting in the backs.

Ore and waste will be transferred to the surface by a service raise and then conveyed on the rope conveyor to the processing plant. The primary infrastructure, including ventilation shafts, material handling systems, and service facilities is strategically positioned along the central decline to support efficient operations.

The mine will be developed over a total strike length of 1.6 km at Mascota and 1.8 km at Gigante. From the central decline, access drives will extend to ore drives or footwall drives. Footwall drives will be developed to establish ventilation circuits, to transport waste rock fill, and to access lower grade stopes in the later stages of the mine. Where possible, a parallel ore drive will be developed to access the stopes instead of a footwall drive in waste rock. This will allow access to lower grade stopes past the previously mined and backfilled higher grade stopes and reduce the overall development requirements. The use of parallel footwall or ore drives allows for multiple stoping fronts ensuring a continuous and systematic extraction process with operational flexibility. Waste and ore development drives are planned at 5.5 m wide by 5.5 m high to accommodate a range of equipment types required for mining.

A plan view of the mine layout is shown in Figure 16-1 and a long section is shown in Figure 16-2.

Figure 16-1 Plan view of mine design - Source Aris Mining 2025

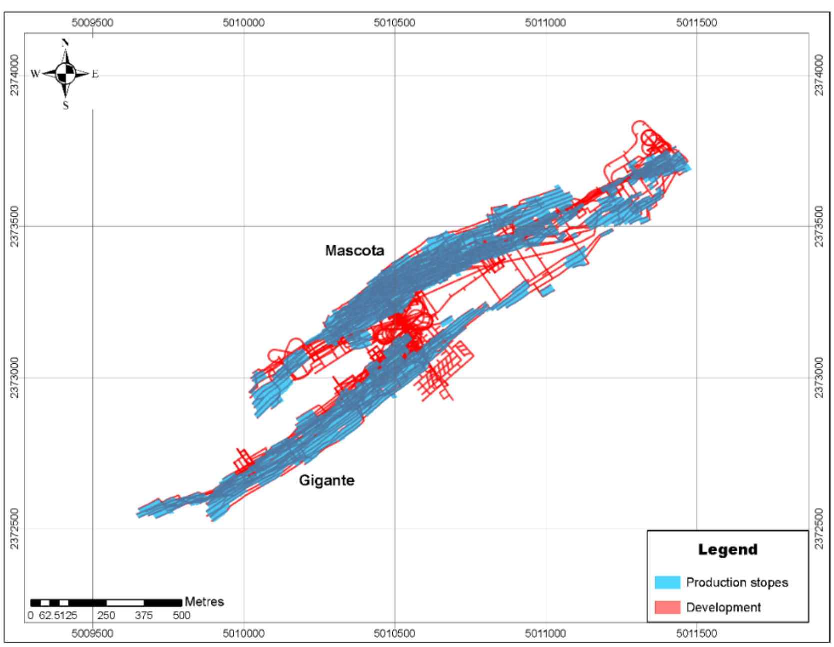
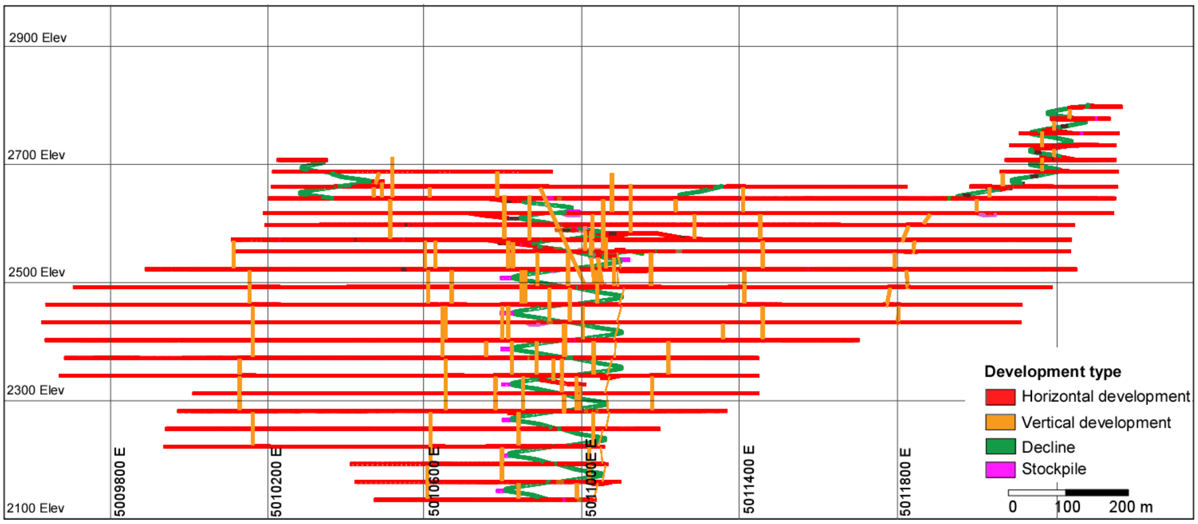


Figure 16-2 Long section of mine design - Source Aris Mining 2025



16.3 Underground infrastructure

The underground infrastructure will include:

- a centrally positioned workshop strategically located off the decline and fully equipped for maintaining all underground equipment;
- an explosives magazine strategically located adjacent to the workshop for the storage of ANFO and separate secure storage compartments for detonators to ensure the strict segregation from other explosives materials. A fully automated fire suppression system will be installed and configured to activate immediately upon detection of smoke. The explosives magazine will be designed, constructed, and operated in compliance with the Colombian explosives regulations;
- a permanent fuel storage bay with the capacity to store 10,000 litres of diesel and 2,000 litres of hydraulic oil and temporary skid mounted fuel storage units positioned on active levels to facilitate efficient refuelling;
- chambers designed for servicing all underground equipment;
- a rock crusher chamber for reducing the ore and waste size;
- an underground inclined rope conveyor system to transport crushed waste and ore rock to the surface;
- a paste fill plant for backfill and ground support operations and a paste fill reticulation system to deliver the paste fill to the stopes. The paste fill plant will receive materials through two separate pipelines routed down the service shaft, including one for tailings slurry and one for binder;
- wash bays for cleaning mobile equipment;
- lunchrooms and shift supervisor offices located on the primary intake ventilation circuit that can be used for safe firing areas and fresh air bases;
- stores for the supply of spare parts and consumables;
- ore and waste passes to streamline the handling of ore and waste material;
- sumps and pump stations for effective water management and dewatering operations;
- service holes to facilitate the routing of utilities and services; and
- portable or fixed refuge chambers installed at intervals such that the workers are never more than 750 m away from the nearest refuge chamber to take refuge in the event of a mine emergency, and escape ladderways installed in a dedicated central escapeway, to provide a second means of egress in the event of a mine emergency.

16.4 Material handling

Prior to the completion of the rope conveyor and the permanent underground material handling system, all underground ore and waste will be trucked in batches to the portal and then to a temporary surface crusher. Both ore and waste will be crushed to a top size of 150 mm. Crushed waste rock will be either sent for further crushing by a secondary crusher at the batch plant and used for concrete and shotcrete production or used as road base material for maintaining the underground roadways. Some of the crushed waste rock material will be used to cover the paste filled stope to create a better surface for underground loaders.

Once the permanent underground crusher is operational, underground trucks will deliver the waste and ore material to the main material transfer drive and dump it into ore and waste passes fitted with a grizzly and rock breaker to break any oversize rocks. The ore and waste will then be fed in batches to a permanent crusher located on the crusher level, and then temporarily stored in waste or ore bins located below the crusher. An underground inclined conveyor system will be used to move the crushed waste and ore rock in batches to the surface loading conveyor for transport to the processing plant on the rope conveyor.

16.5 Hydrology and hydrogeology

16.5.1 Introduction

Hydrological (surface) and hydrogeological (groundwater) considerations for the mine design have drawn on the finding of previous studies supplemented by site field work undertaken in August 2023 and data interpretation of the mineralized breccia zones in October 2023.

The Project design has considered the requirement to avoid potential groundwater drawdown in the páramo and to protect the aquatic ecosystem of the La Baja Creek. To mitigate these risks, underground grouting is planned to minimize groundwater infiltration into the mine. Advanced cover drilling and grouting during mine development will be undertaken to enable early identification and pre-grouting to seal any water bearing structures to limit groundwater flows into the underground workings.

Any groundwater in the mine will be collected and managed in two separate systems: one for clean ground water and another for water that has come into contact with mining activities. Both streams will undergo treatment before it is safely returned to the La Baja Creek, if required. During periods of low seasonal flow, treated water from the underground will be used to supplement the La Baja Creek, based on monitoring data.

This strategy will help ensure that current flow rates within the ecosystem are maintained. Previous study work including geological interpretation, piezometer data, and isotope monitoring indicate that the páramo will not be impacted by mining. Shallow groundwater conditions are present in the páramo due to the presence of low permeability bedrock, and the shallow water is not connected to the deeper groundwater that will flow into the mine. There is a limited hydraulic connection between the páramo and the La Baja Creek valley due to the low permeability bedrock. The vegetation of the páramo is sustained through its reliance of fog and drizzle combined with a low evapotranspiration rate, and the moisture rich organic soils are vertically dissociated from the deeper ground water zones. Additional groundwater modelling has been completed to confirm these findings for the current Project design.

16.5.2 Hydrology

The Project is located within the La Baja Creek catchment area which drains the upland páramo and is fed by the Angostura and Paez tributaries. The La Baja Creek joins the Vetas River which flows into the Suratá River. The Project processing plant, filtered tailings facility, water treatment plant, and supporting infrastructure are planned around minor tributaries of the Suratá River. Local water quality is degraded due to the effects of unregulated small scale mining activities and the lack of sewage treatment plants in the rural area, particularly in the upper regions of the La Baja Creek.

16.5.3 Hydrogeology

Groundwater is recharged by the direct infiltration of rainfall through the soil zone and then into the underlying open structural fabric, although the amount of direct recharge is limited because of the near-surface weathering and healing of fractures, which reduces the near-surface permeability. Near surface groundwater discharge from the Project area occurs to the La Baja Creek.

The majority of the groundwater flow in the Project area occurs through localized shallow flow pathways within the top few hundred metres of the surface, with tens to hundreds of metres of lateral extent. The flow pathways are aligned along the main strike of the geological structures, although the flow paths tend to have limited lateral extent because of the variable nature of the structures, the differences in the alteration of the rock, and the presence of clay gouge within the main structural zones. These gouge bearing faults are closed and are expected to act as low permeability barriers both along the strike and over the extent of the fault structures. As a result, the subsurface groundwater system is considered highly compartmentalized. Open fractures within fault damage zones may produce some local continuity of groundwater flow around the workings and there may be some minor upwelling of warmer groundwater from depth.

The hydrogeological units within the Project setting have been defined through drill core logging and an extensive testing program along with geological mapping and geophysical surveys. The degree of fracturing and hydrothermal alteration of the crystalline rocks is more intense next to the mineralized zone, where a series of main faults and breccias exist. These faults have a low permeability in some cases, but between the main faults, structural movement has produced more intense fracturing in some zones between the faults. In conjunction with the alteration due to mineralization, the reactivation of an earlier set of structures appears to produce parallel zones of increased permeability that are strongly controlled by the strike of the structures. Fracturing is reduced away from the immediate area of the ore mining zone. Country rocks unaffected by faulting are considered to be much more sparsely dissected than the fault zones. The country rock faults are mostly closed and are unlikely to become more open at depth due to the confining pressures.

Towards the páramo the degree of fracturing and the permeability of the rocks decrease significantly. Similarly, fracture intensity and permeability generally decrease with depth. Groundwater flow is therefore expected to be associated with the main structural faulting zones. Groundwater inflow will be controlled through pre-grouting programs ahead of the stoping operation with ongoing updates to the hydrogeological model used to refine the necessary mitigation measures.

Simulated modelling has indicated that dewatering rates of up to 225 litres per second may be possible in the early years of the operation, with the main contribution anticipated in the mine access area. The grouting program has been designed to reduce the groundwater inflow to 70 litres per second in the upper areas of the mine. Potential reductions to river flow will be monitored as the Project proceeds and will be mitigated if necessary through pumping treated water to a point upstream to maintain adequate flow.

16.6 Mine water management

Advanced cover drilling, drain holes, and the systematic grouting program will control ground water inflows. Clean water will be drained from the rock using drain holes and pumped to the surface. Part of this water will be used for the mine needs such as drilling service water, dust suppression, use in the paste fill, batch plant, and other mine services.

A series of sumps and pump stations will be installed throughout the mine to manage the anticipated water inflows and will be installed on dedicated levels as mining progresses deeper. Two separate water systems are planned. A system for clean water from groundwater inflows will be returned to a water treatment plant on the surface prior to being returned to the La Baja Creek. Water that has been in contact with mining activities will be handled separately through a parallel pumping and treatment arrangement. Both clean and contact water systems will utilize a series of dams to facilitate pumping to the higher levels before reporting to the water treatment plants on the surface.

16.7 Mine ventilation

The airflow requirements and ventilation strategy for the Project considers the designed mine layout, the peak diesel equipment fleet, and additional ancillary airflow demands. The design addresses primary airflow requirements and the associated fans necessary to effectively meet operational demands. The ventilation system is designed to comply with Colombian regulations and prioritizes worker health and safety by maintaining air velocity and airflow within prescribed limits.

Diesel engine exhaust is the primary driver of airflow demand in mechanized underground mines, as exhaust emissions contain toxic gases and particulates. To comply with the Colombian regulations, the mine will utilize newer equipment and drills will operate on diesel power at 20% utilization due to intermittent use and the reliance on electric power for drilling activities.

The airflow requirements were estimated for each piece of equipment based on industry standards and the utilization of equipment. A ventilation simulation model was built in VentSim software to simulate the ventilation flows through the development design and identify any areas of insufficient air flow or excessive pressure.

The ventilation system design follows a conventional primary exhaust strategy, with the decline and fresh air raises supplying the levels via a primary fresh air intake. Fresh air will be circulated in the levels either by auxiliary fans or by opening the regulators at the end of the levels for full primary air flow through. All exhaust air is then expelled through regulator controlled return air raises to the primary fans. The ventilation system consists of lateral ventilation drives and vertical raises for air intake, exhaust raises, and inter-level raises for both return and fresh air in each zone.

Four ventilation zones were established covering central decline infrastructure, the Gigante mining zone, and the north and south Mascota mining zones. The system is designed to keep the ventilation zones separate, allowing them to operate independently. Some raises may change from fresh to exhaust air or vice versa over the life of the mine to ensure that the system is operating smoothly and efficiently. The infrastructure such as workshops, paste plant, explosives magazine, stores, and offices are designed to exhaust directly to the surface to ensure that contaminants are not brought into the mine including in the case of a fire or chemical leak in the area.

The ventilation planned is sufficient to support the current equipment fleet and anticipated production activities. Future studies will include a trade off study to determine the optimal sizes for the primary ventilation network, ventilation heat modelling, and a consideration of a ventilation on demand system.

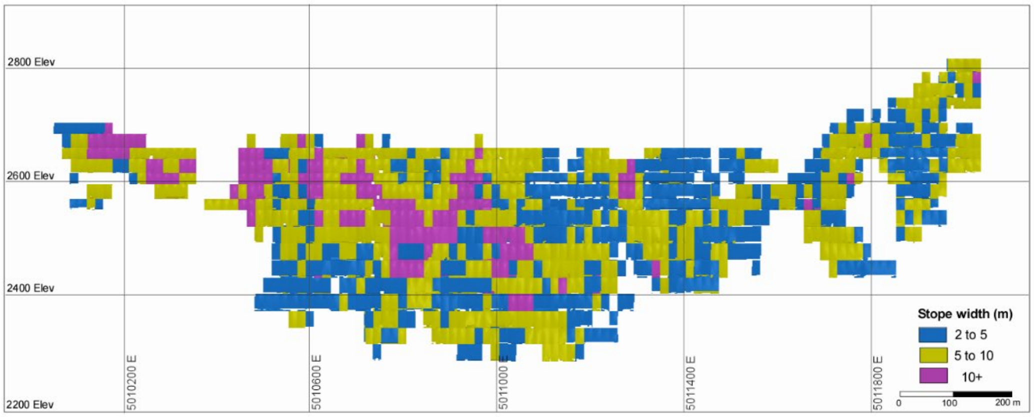
16.8 Mine geotechnical parameters

Geotechnical domain models were generated for Mascota and Gigante to classify the ground as fair or poor. Stope stability was analyzed using the Hudyma pillar stability graph with the Hoek Brown Pillar Strength curves. For stope design, an intact rock strength of 30 Megapascals (MPa) was assigned to the upper domains to account for the presence of higher water infiltration, and 40 MPa was assigned to the deeper domains. Additional geotechnical analysis including updating the geotechnical and stress modelling will take place in future studies to refine the geotechnical domains and confirm stope sizes and support requirements.

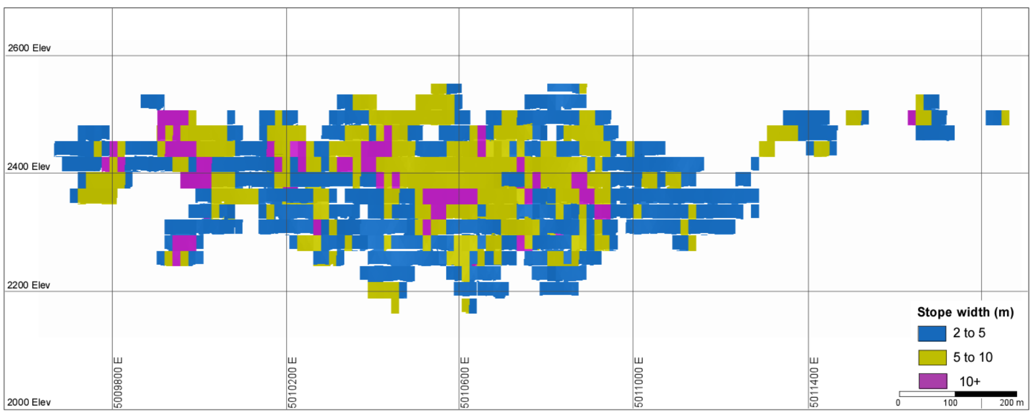
A stope size analysis was conducted using the Mathews modified stability graph method to support the selection of the planned stope dimensions. The selected stope dimensions have a 15 m length, with sub-level intervals ranging from 20 to 25 m in the upper areas of the mine and 30 m in the lower areas. A minimum stope width of 2.5 m was planned. 50% of the stopes are less than 5 m wide, 40% of the stopes are between 5 and 10 m wide, and 10% are greater than 10 m wide. The average stope comprises 6,000 tonnes. Long sections of the stope widths are shown in Figure 16-3.

Figure 16-3 Long sections of stope widths – Source Aris Mining 2025

Mascota



Gigante



Stopes will be mined and filled using either rock fill, cemented rock fill, or paste fill before the next stope in the cycle is mined. Isolated stopes will be left without fill where possible. A temporary rib pillar of the full stope length may be left in place if the mining sequence requires. Between parallel stopes, a waste pillar with a minimum width of 7 m was planned.

A crown pillar extending 50 m from the surface is planned, based on a minimum factor of safety of 2, for long term stability of the slopes above the mine. Additionally, no stopes were planned within 10 m of artisanal mining. Backfill will also provide additional support for any larger or multi-level stopes. For support of the stope brow, straps held by rock bolts will be primarily used to allow safe access for service crews and for mining.

Cable bolting requirements were determined by geotechnical domain and are required for all hangingwall and footwall exposures of multi-level stopes, with additional cable bolting for stope back support requirements as specified in the ground support plan.

16.9 Mining method

Longhole open stoping is the preferred mining method, using waste rock fill, cemented rock fill, or paste fill to maximize reserve extraction while maintaining a safe and efficient mining environment. Temporary rib pillars will be utilized, and sill pillars will be recovered under cured cemented rock fill or paste fill.

The stopes were divided into panels covering multiple stopes over multiple levels, with vertical heights ranging between 90 and 100 m, and scheduled using a bottom up mining plan within each panel.

The longitudinal open stoping mining method was adjusted to suit the contiguity and extent of the stopes and the underlying panel stopes. For stopes that require immediate waste rock backfill for support, the Avoca mining method will be utilized.

With Avoca mining, ore is only drawn on one level, resulting in better productivity, a lower requirement for tele-remote loaders, fewer sill pillars, a lower binder requirement for cemented rock fill, and requires only one slot per group of stopes on a level. Cemented rock fill will be used to form sill pillars at the bottom of the Avoca panels, with rib pillars either side, to contain uncemented rock fill and to allow adjacent blocks to be mined later. Uncemented rock fill can be used in Avoca panels where adjacent stopes are consecutively mined.

Initial mining will target higher grade blocks within the panels, which requires cemented fill ribs to separate the mined blocks from future mining. The remaining stopes in the sequence will be backfilled using waste rock fill, and the last stope in the sequence will be backfilled with cemented rock fill. The first stope in the sequence is a primary stope without dilution from adjacent stopes and the remaining stopes are secondary stopes incurring additional dilution from the fill from adjacent previously backfilled stopes.

Between 5 and 20 active stopes per month are required to sustain a processing rate of 2,750 tpd, depending on the contribution of development ore and the width of each stope.

Development undertaken with a jumbo drilling machine will be used for all lateral development and accounts for 17% of the ore production. Slot raises will be drilled and progressively fired into the slot and stope void. The blasted ore will be removed from the lower level by a conventional loader up to the stope brow and removed by a tele-remote loader when the stope is open past the brow. Once the stope has been fully excavated, a bund wall will be installed at the stope access and the stope void will be backfilled according to the dynamic backfill strategy.

16.10 Backfill

Backfill provides structural support, enhances ground stability, and reduces dilution. The backfilling method will utilize waste rock fill and cemented rock fill in the early stages of the mine, followed by exclusively cemented paste fill in the later years utilizing tailings from the process plant, to improve ground stability and to reduce the surface tailings storage requirements.

Waste rock fill sourced from underground development in waste rock will be used in areas where back fill does not directly impact safety, dilution control, or ore recovery. Waste rock fill can be placed into the stope from the top drive using a dump truck, ejector truck, or a loader, and then pushed over the edge until the fill point is reached.

Cemented rock fill will consist of waste rock mixed with cement and water to form a high strength, load bearing fill to provide stability when mining adjacent stopes to ensure that the surrounding rock remains intact and to minimize the risk of unplanned stope collapse and excessive dilution. The cemented rock fill will be placed in layers to ensure proper settlement and structural integrity, using loaders from adjacent access drives.

Cemented paste fill utilizing tailings from the process plant will be utilized for the majority of the backfill requirements over the life of mine. The paste fill will utilize 10.2 Mt of process tailings, just under half of the total tailings generated at the process plant.

The backfill strategy prioritizes the use of potential acid generating waste rock in the waste rock and cemented rock fill. The remaining potential acid generating waste will be transported to the processing plant and then hauled to the filtered tailings storage facility by truck and placed in trenches excavated into the compacted tailings stack, then immediately covered with compacted filtered tailings. Non acid generating material will be transported to the processing plant and stored in dedicated surface waste rock storage facilities.

A dynamic backfill strategy was developed to reduce binder usage and delay the use of cemented paste fill and considered geotechnical factors such as ground conditions and geotechnical domain, stope location, stope size, availability of access drives, proximity to the crown pillar, the minimum pillar width between stopes, and the required geotechnical support. The longitudinal open stoping method minimizes both vertical and undercut exposure spans and therefore minimizes cemented paste fill strength and binder requirements.

A range of binder content will be used in the cemented paste fill to achieve the required strengths, depending on the stope widths, and the amount of time the binder can cure before mining is schedule to take place below it. Additional strength and strength gain rate tests with blended binders and representative tailings material will be conducted in future studies to optimize the binder content requirements and to identify any implementation opportunities.

The paste mixing plant will be located underground and will utilize tailings conveyed from the processing plant to the rope conveyor unloading station near the mine portal, then transferred through a surface raise to holding tanks prior to transfer to the mixing plant. Binder will be delivered by a dedicated transfer system from the rope conveyor unloading station to binder storage tanks located adjacent to the underground mixing plant.

The paste fill will be delivered and placed in stopes through pumping through designated boreholes and through pipelines extending into the stopes where an upper level development drive is accessible. Where an upper level development drive is unavailable, the paste fill will be pumped into the stope through a system of injection pipes. A paste fill barricade comprised of reinforced shotcrete and steel mesh will be constructed for each stope to contain the paste fill until it cures.

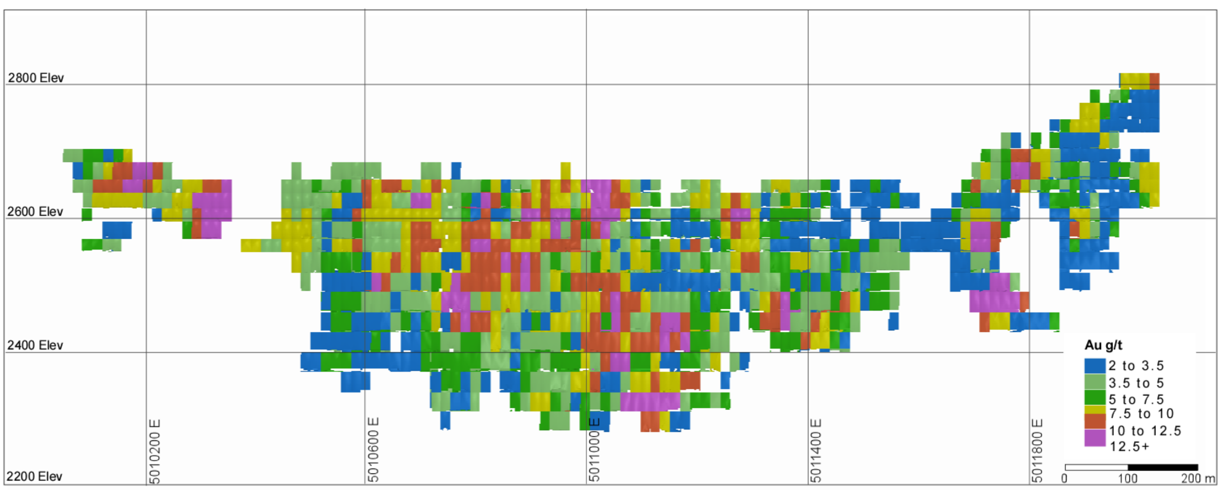
16.11 Mine planning and schedule

Deswik mine planning software was used to develop a sequenced and scheduled plan for mine design development and stope extraction, based on outputs from the stope optimization process and refined through multiple iterations. The schedule has been designed to achieve sustainable production in line with the 2,750 tpd processing plant capacity dedicated to the underground operation and limiting the total annual material movement of ore and waste to a maximum of 2 million tonnes. The ramp up to full production is two years.

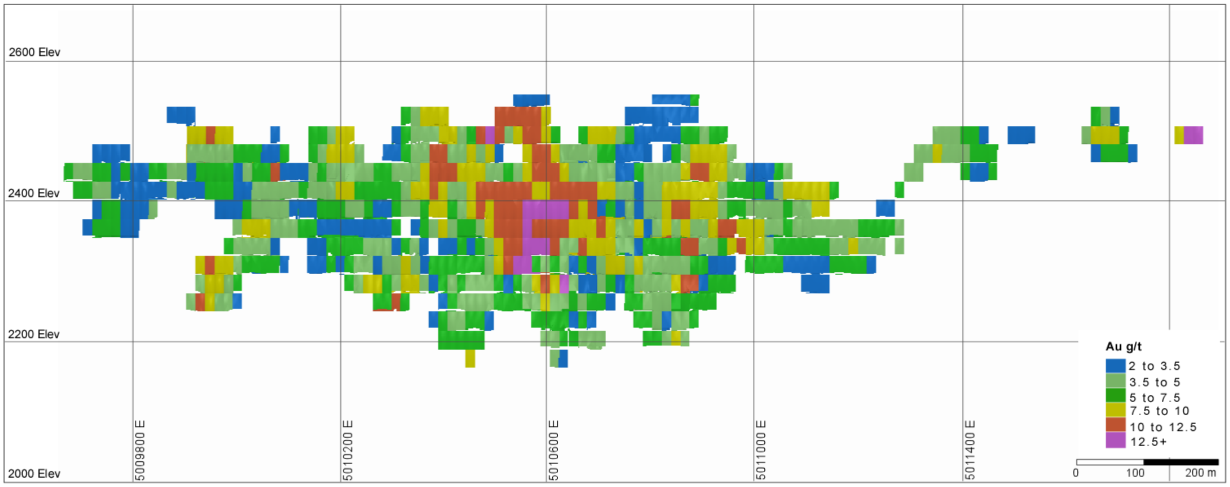
Production priorities were assigned based on gold grades, with the highest grades receiving the highest priority. The schedule was sequentially filled based on a higher grade first principle to achieve the desired production rate. Lower grade stopes are introduced after ten years of mining and scheduled to achieve the required production rate. Longitudinal stope panels were scheduled to be mined on retreat to the stope access. The mined ore schedule includes the dilution and ore recovery assumptions provided in Section 15.2. Long sections of the stope gold grades are shown in Figure 16-4.

Figure 16-4 Long sections of stope gold grades – Source Aris Mining 2025

Mascota



Gigante



Produced gold grades are initially above 8 g/t, gradually decreasing after ten years and averaging 6.98 g/t Au over the life of mine. Annual gold production rates are expected to average approximately 253 koz over the first ten years of production, and 158 koz from years 11 to 21.

The schedule has a high development requirement in the initial seven years and reduces in the following years as the mine is in the steady production stage and requires minimal development to maintain production.

The annual schedule of mine development metres is shown in Table 16-1 and the annual mine production schedule is shown in Table 16-2.

Table 16-1 Annual mine development schedule

Type	Total	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
Lateral ore (m)	51,837	1,192	6,654	5,716	4,167	7,149	6,468	5,850	4,924	3,941	2,236	494	628	921	864	3	268	220	56	85
Lateral waste (m)	68,688	1,788	11,063	11,263	8,855	7,045	7,777	5,350	3,497	1,894	1,727	543	1,779	3,644	1,147	269	547	247	209	43
Vertical (m)	8,767	218	1,086	1,626	1,417	820	745	989	705	153	104	50	82	538	176	29	30	-	-	-

Table 16-2 Annual mine production schedule

Type	Total	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22
Waste (Mt)	5.4	0.1	0.9	0.9	0.7	0.5	0.6	0.4	0.3	0.1	0.1	0.0	0.1	0.3	0.1	0.0	0.0	0.0	0.0	0.0	-	-	-	-
Development ore (Mt)	3.5	0.1	0.5	0.4	0.3	0.5	0.4	0.4	0.3	0.3	0.1	0.0	0.0	0.1	0.1	0.0	0.0	0.0	0.0	0.0	-	-	-	-
Stope ore (Mt)	17.1	-	0.1	0.5	0.7	0.5	0.6	0.6	0.7	0.7	0.9	1.0	1.0	0.9	0.9	1.0	1.0	1.0	1.0	1.0	1.0	1.0	0.9	0.2
Total ore (Mt)	20.6	0.1	0.5	0.9	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	0.9	0.2
Gold grade (g/t)	6.98	8.54	9.80	8.17	9.82	8.74	8.92	9.05	8.75	8.44	10.29	7.64	6.98	5.60	4.70	4.82	5.23	5.31	4.94	4.41	5.44	5.17	5.90	5.58
Silver grade (g/t)	32	44	52	48	47	34	32	33	35	36	38	34	31	30	22	22	25	29	28	29	39	20	21	22
Copper grade (%)	0.20	0.32	0.21	0.20	0.20	0.20	0.20	0.22	0.22	0.22	0.21	0.25	0.23	0.19	0.15	0.16	0.15	0.20	0.18	0.19	0.28	0.16	0.17	0.14
Contained gold (koz)	4,626.7	23.4	160.5	232.9	316.8	281.7	287.9	291.6	282.6	271.8	331.5	246.4	225.4	180.6	151.6	155.4	168.6	171.3	159.4	142.2	175.6	166.9	169.8	32.9
Contained silver (koz)	21,216.1	121	848	1,381	1,511	1,084	1,023	1,061	1,145	1,148	1,220	1,093	1,006	963	720	709	807	926	906	926	1,256	644	591	130
Contained copper (Mlb)	90.6	0.6	2.4	4.0	4.4	4.4	4.5	4.9	4.8	4.9	4.6	5.6	5.1	4.2	3.3	3.5	3.3	4.5	4.0	4.1	6.2	3.4	3.4	0.6

16.12 Mine equipment

Mine equipment was selected based on operating factors such as availability, effective utilization, equipment life, and productivity assumptions. A table of the selected fleet is shown in Table 16-3.

Table 16-3 Mine equipment fleet

Item	Specification
Jumbo drill for development drilling, ground support, and cover drilling	Epiroc Boomer M20 S
Production longhole drill	Epiroc Simba E70 S
Development loader	Epiroc ST18 S
Production loader	Epiroc ST18 S with radio remote control
Truck	Epiroc MT65 S
Charging development	Epiroc Terrah TS100 CHA/CHE
Charging production	Epiroc Terrah TS100 CHA/CHE
Cable bolter	Epiroc Cabletec M10 S
Secondary breakage drill	Epiroc Boomer S1D
Scissor lift platform	EPIROC Terrah TS 100 SCL
Light vehicle	Toyota Hilux
Grader	Caterpillar CAT 12K
Stores truck	Epiroc Terrah TS 100 FL
Lube truck	Epiroc Terrah TS 100 LU
Shotcrete agitator	Normet Ultimec LF 600
Shotcrete sprayer	Normet Spraymec 8100 VC
Personnel carrier	Epiroc Terrah TS 100 PEC
Backfill rehandle loader	Epiroc ST18
Raisebore drill	Robbins 92RHC
Coring drill rig	Boart Longyear LM 75
Concrete pump	Bunker B100
Boxhole raise borer	Epiroc Easer 10S or Rhino 100

The annual schedule of mine equipment is shown in Table 16-4.

Table 16-4 Annual mine equipment schedule

Type	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22
Jumbo	3	6	6	5	5	4	4	3	2	2	1	1	1	1	1	1	1	1	1	1	1	1	1
Production longhole drill	-	1	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	2
Development loader	3	6	6	5	5	4	4	3	2	2	1	1	1	1	1	1	1	1	1	-	-	-	-

Type	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22
Production loader	-	1	2	2	2	2	2	2	2	3	3	3	3	3	3	3	3	3	3	3	3	3	2
Truck	1	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	2
Charging development	1	2	2	2	2	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-	-	-
Charging production	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Cable bolter	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Secondary breakage drill	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Scissor lift platform	3	4	4	4	4	5	5	5	5	5	5	5	5	5	5	5	5	5	5	4	4	4	4
Light vehicle	22	25	25	25	25	25	25	25	25	25	24	25	25	25	25	25	24	24	23	23	23	23	23
Grader	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Stores truck	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Lube truck	1	4	4	4	4	4	4	4	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2
Shotcrete agitator	3	4	4	4	4	4	4	3	2	2	2	2	2	2	1	1	1	1	1	1	1	1	1
Shotcrete sprayer	2	4	4	4	4	3	3	2	2	2	2	2	2	1	1	1	1	1	1	1	1	1	1
Bus – personnel carrier	3	4	4	5	5	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Boxhole raise borer	-	2	2	2	2	1	1	1	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-

16.13 Mine personnel

The requirements for professional and management staff, the workforce, and maintenance personnel for the underground mine were estimated based on the typical levels for the size of the operation, assuming two 12 hour shifts, 24 hours per day, seven days per week. Maintenance, operator, and labour estimates were estimated based on the annual equipment requirements. A contractor workforce is required for raise boring and drill and grouting operations. The annual estimate of mine personnel requirements is shown in Table 16-5.

Table 16-5 Annual mine personnel requirements

Category	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22
Management/professional	32	57	57	57	57	57	57	57	57	57	57	57	57	57	57	57	57	57	57	57	57	57	57
Operational	103	203	212	224	224	215	215	200	185	185	179	179	179	176	173	173	173	173	173	155	155	155	137
Underground support	93	105	105	105	105	105	105	105	105	105	105	105	105	105	105	105	105	105	105	105	105	105	105
Contractor	10	14	14	14	14	14	14	14	10	10	10	10	10	10	10	10	6	6	6	-	-	-	-
Total	238	379	388	400	400	391	391	376	357	357	351	351	351	348	345	345	341	341	341	317	317	317	299

16.14 Recommendations

Further trade off and mine optimization studies are recommended for the future study stages, including the decline layout, the location of key underground infrastructure, options for the materials handling system, refinements to the blasting methods, and additional work on the ventilation design including optimal sizes, consideration of a ventilation on demand system, and ventilation heat modelling, for a total cost of $210,000.

Additional refinements related to the underground infrastructure, including capital and operating costs, are recommended for air, water, electrical, ventilation control, egress and refuge chambers, explosive magazine, dewatering, mine services, paste fill plant and delivery, surface to underground cement delivery, and additional paste fill test work, for a total cost of $193,000.

Additional geotechnical studies are recommended, including a Mathews stability graph check, three dimensional stress modelling, cemented paste fill design strength and strength gain rate tests as well as identifying implementation opportunities, and a trade off study to optimize the management of slimes in the paste fill or in the filtered tailings storage facility, for a total cost of $48,800.

17 Recovery methods

17.1 Introduction

Several metallurgical testwork programs have been undertaken between 2009 and 2018 to support the metallurgical assumptions utilized for the selected processing flow sheet for the current prefeasibility study, utilizing samples representative of the mineral resource and mineral reserve estimates. These studies included processing method trade off studies as well as refinements of the selected operating parameters, as the properties and response of the samples under the testwork conditions became increasingly better understood.

The processing plant is designed to treat plant feed throughputs of a nominal 2,750 tpd and a maximum 3,500 tpd, assuming a 365 day per year operation with two 12 hour shifts and a 91.3% operation availability. The processing plant will receive 2,750 tpd of crushed run of mine ore from the Soto Norte underground mining operation, and can receive 750 tpd of mill feed purchased from contract mining partners, to produce three saleable products and one waste product, including a gold concentrate from a gravity gold operation, a copper concentrate from a copper flotation operation, a pyrite concentrate from a pyrite flotation operation, and a tailings product for disposal in the filtered tailings facility and to create paste backfill for the underground mine.

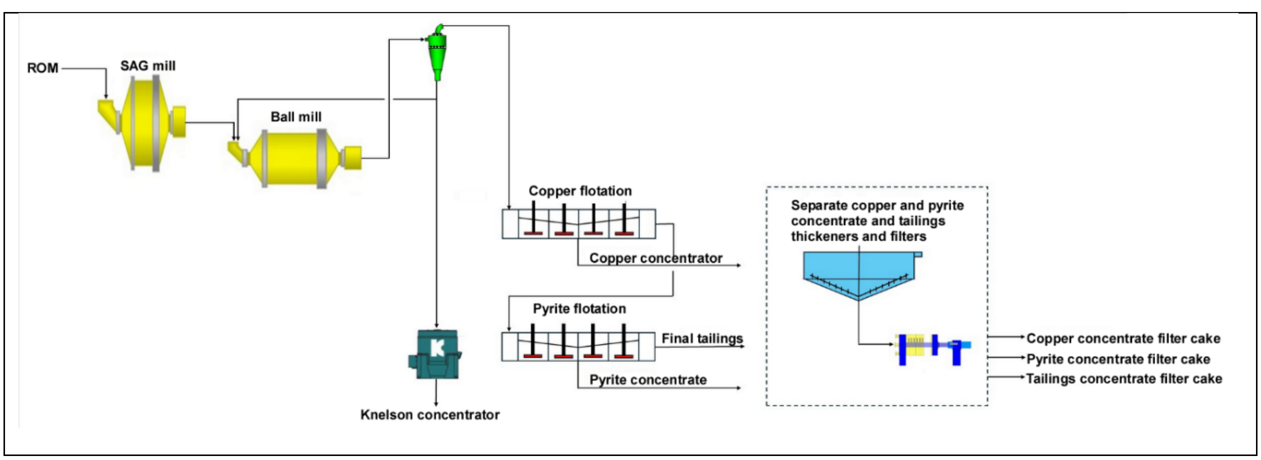
No cyanide or mercury will be used at the Soto Norte processing facilities. The processing circuit comprises primary ore crushing at the underground mine, ore transport from the mine portal to the process plant on the rope conveyor, receipt of mill feed purchased from contract miners at the process plant, primary grinding in a SAG mill with a supporting pebble crusher, secondary ball mill grinding, an upfront gold gravity recovery circuit to recover up to 15% of the coarse gold-silver particles and to produce a filter cake, and a two stage sequential flotation circuit to recover fine gold, silver, and copper in separate copper and pyrite concentrates. The process flow sheet is shown in Figure 17-1.

A target grade of 16% for the copper concentrate has been set on the basis of testwork results as well as considerations for maximizing the gold content in the copper concentrate and reducing penalty element concentrations. Potential penalty elements considered in the economic analysis include arsenic, bismuth, cadmium, antimony, and likely zinc, with payments for contained copper and gold anticipated to be far in excess of any potential penalties. The non-sulphide waste content of the concentrates will be restricted to 10%.

Capital and operating cost estimates for the construction and operation of the processing plant have been considered in the economic analysis, including the requirements for power, water, and process materials.

For design purposes, average metallurgical recoveries are estimated at 92.8% for gold, 88.8% for silver, and 92.8% for copper.

Figure 17-1 Soto Norte processing flow sheet – Source LogiProc 2025



17.2 Process description

17.2.1 Feed preparation and transfer to surface

The primary ore will be crushed to a maximum size of 150 millimetres (mm) underground, then transferred to a surface conveyor bin, and extracted by an apron feeder for transport by the rope conveyor to the process plant. The ore will discharge onto a cleaner conveyor at the process plant fitted with a hammer sampler for metallurgical testwork and a cross belt magnet to remove any tramp metal. Any low quality ore can be directed to a separate stockpile via a diverter chute and blended as required.

Mill feed purchased from contract mining partners will be delivered by truck to a secondary run of mine to an approximately 40 tonne capacity stockpile and transferred by the secondary vibrating feeder onto a feed conveyor fitted with a weightometer and a metal detector for discharge onto an impact crusher. The impact crusher will discharge onto a transfer conveyor fitted with a hammer sampler for head grade analysis and metallurgical accounting.

This arrangement ensures that throughput and metallurgical accounting for both the underground mining ore and the mill feed purchased from contract mining partners are appropriately measured and accounted for, and provides for flexibility in feeding the plant.

The combined run of mine ore will be transferred via the final stockpile feed conveyor and stockpiled at the final 12,000 tonne live capacity run of mine stockpile above the process plant tunnel. Ore will be withdrawn from the final stockpile onto a conveyor using two apron feeders fitted with a cross belt magnet, metal detector, and end belt magnet. The conveyor will discharge into the SAG mill for grinding.

17.2.2 Grinding and gravity concentration

Primary ore grinding will occur in a closed circuit, single stage 1,305 kilowatt SAG mill, with the milled material reporting to a single deck vibrating classifying screen.

SAG oversize material greater than 20 mm will discharge onto a transfer conveyor to a diverter chute into a pebble crusher surge bin, then extracted by a vibrating feeder to feed the pebble crusher. The crushed pebbles will be recycled to the SAG mill for further grinding. A ball loader and ball loader chute will load grinding media into the SAG mill as required.

SAG undersize material less than 20 mm will discharge onto a cyclone feed hopper and pumped to the primary cyclone cluster for classification. The cyclone underflow will report to the cyclone underflow screen for further classification. The screen oversize greater than 2 mm will be recycled to a 2,237 kilowatt ball mill operating in closed circuit with a hydrocyclone cluster for regrinding. The ball mill will receive material from the vibrating screen, raw water, and lime slurry for dilution and pH control and grind the material to a P_{80} of 106 microns. The hydrocyclone overflow will be pumped to the sequential flotation circuit. A ball mill trommel will separate the milled ore from the metal scats which will report directly to a dedicated metal scats bunker.

The screen undersize less than 2 mm reports to a Knelson gravity concentrator circuit and the recovered concentrate will be upgraded on a shaking table then pumped to the gold safe room to a dedicated gold concentrate filter press feed tank, then pumped from the tank to a filter press for dewatering. The filtrate will then be pumped to the cyclone feed hopper. The product cake will be transferred by a bottom feeder and discharged into a product bin for collection. The product bins can then be transported to the port.

The gravity circuit rejects will be recycled back to the ball mill. The undersize material will also be directed back into the ball mill to combine with the oversize material for regrinding. The bulk of the screen underflow slurry reports to the ball mill while the gravity concentrator circuit receives a material load equivalent to the fresh feed of the milling circuit.

17.2.3 Sequential copper and pyrite flotation and regrind

The hydrocyclone overflow from the SAG mill and the tailings from the gravity circuit will be pumped to the copper rougher flotation tank of the sequential copper and pyrite flotation circuits. The slurry from the tank will be pumped into a bank of four forced air rougher flotation cells with the addition of lime slurry, frother, and copper collector reagents to enhance the flotation of the minerals in the circuit, maintain pH, and control performance at each stage. The rougher flotation cells produce a low grade copper concentrate that will be pumped to the copper concentrate regrind circuit for further liberation before being sent to the copper cleaning circuit for further concentration. The tailings from the copper rougher flotation circuit report to the pyrite conditioning tank and feed the pyrite rougher flotation circuit.

The copper rougher concentrate will be reground in open circuit with the hydrocyclone cluster to further grind the copper rougher concentrate to improve the liberation of the minerals in the slurry for increased recoveries, before it reports to the copper cleaner flotation circuit. The copper rougher concentrate, copper recleaner tailings, and lime slurry will be fed into the regrind mill cyclone feed tank and pumped to the copper regrind mill cyclone cluster for classification.

A three stage copper cleaning operation includes a copper cleaner flotation circuit and a copper recleaner flotation circuit, comprised of forced air mechanical flotation cell tanks, and a copper tertiary cleaner flotation circuit with a Jameson flotation cell. The slurry from the regrind mill circuit will flow into three copper cleaner flotation cells, then the tailings will be recirculated back to the copper rougher flotation circuit and the concentrate will be pumped to two copper recleaner flotation cells. The concentrate stream from the copper recleaner flotation circuit will be pumped to the Jameson cell, while the recleaner tailings stream will be returned to the copper regrind mill circuit. The concentrate stream from the Jameson cell will be pumped to the copper concentrate thickener and the tailings stream will be pumped back to the copper recleaner flotation circuit for recleaning.

The copper rougher flotation circuit tailings and the pyrite cleaner tailings will report to the pyrite conditioning tank and then pumped into seven pyrite rougher flotation cells operating in series. The tailings stream from this circuit will be sent to a dedicated pyrite rougher tailings tank which feeds both the tailings thickener and the pulse back filters. The low grade pyrite concentrate from this rougher flotation circuit will be sent for further liberation at the pyrite regrind circuit consisting of a single pyrite regrind mill operating in open circuit with the hydrocyclone cluster. The regrind mill product will be combined with the fine cyclone overflow and pumped to five pyrite cleaner flotation cells. The tailings from this circuit will be recirculated back to the pyrite rougher flotation cells while the concentrate stream from the cleaner cells will report to three pyrite recleaner flotation cells. The pyrite recleaner tailings stream will report to the pyrite regrind mill cyclone fill tank and the pyrite recleaner concentrate stream will report to the pyrite concentrate thickener.

17.2.4 Concentrate dewatering

The copper and pyrite concentrates will be sent to a single dedicated high rate thickener and a horizontal plate pressure filter press. The concentrate thickener overflow will report to the overflow tank and the process water will be recirculated back the flotation circuit. The concentrate underflow stream will be thickened to a density of 55 to 60% solids. The thickener underflow will be pumped to an agitated filter feed tank then pumped to the concentrate filter press and dewatered to produce a filter cake with a 9% moisture content. The recovered filtrate will be collected in the concentrate filtrate tank and pumped back to the concentrate thickener overflow tank for distribution in the flotation circuit. The concentrate filter cake will be discharged into a dedicated bin with bottom screw feeders that draw the concentrate from the bin and discharge it into the allocated product bins. The bins will be placed on a dedicated concentrate conveyor fitted with a weightometer to record the total concentrate produced and transported to the port.

17.2.5 Thickened tailings dewatering and storage

The tailings thickening circuit consists of a tailing thickener feed box and a high rate thickener to thicken the final tailings to a moisture content of 55 to 60% solids in preparation for tailings filtration, and the recovered process water will be recycled back to the process plant. The tailings thickener overflow will be directed to the tailings thickener overflow tank then pumped to the water treatment plant. The thickened underflow stream will be pumped to an agitated tailings transfer tank to ensure continuous suspension of the solids in the slurry. The thickened slurry underflow will be pumped to the tailings filter feed tank and fed to three parallel tailings filter presses for filtration to produce a tailings cake with a 15% moisture content. The filter cake discharges from each filter press onto a filter cake conveyor for delivery to two transfer conveyors onto the grasshopper conveyor, then stacked and compacted at the filtered tailings facility. The filtrate stream from the filter presses will be collected in a tailings filtrate tank, then pumped to a reservoir tank, then passed through a water treatment plant. Some of the excess treated water will be discharged in compliance with discharge permits.

The pyrite rougher flotation tailings will report to the pyrite rougher tailings tank then pumped to the tailings thickener for pressure filtration prior to stacking in the filtered tailings facility or to the paste fill plant for use as backfill in the underground mine. For paste filling, the tailings slurry will be pumped from the pyrite rougher tailings tank to the pulse back filters to reduce the water content of the slurry to the required 65 to 67% solids then pumped onto the rope conveyor for transfer to a discharge tank at the mine portal. The resulting filtrate from the pulse back filters will be sent to the tailings thickener overflow tank.

17.3 Reagents and consumables

Powder lime will be delivered to the site in bulk road tankers and unloaded into a dedicated storage silo at the lime preparation plant, which includes a dust collector and ventilation fan to ensure air quality and containment.

Liquid copper collector will be received in containers and pumped to the copper rougher flotation conditioning tank.

Liquid frother will be received in containers and transferred to a day tank for distribution throughout the copper circuit, excluding tertiary flotation, as well as throughout the pyrite circuit, to generate a stable froth.

Pellet form pyrite collector will be delivered in bags or boxes, and unloaded into the feed bin equipped with a dust filter to capture any airborne dust and a dust filter fan for ventilation. The pyrite collector will be mixed with raw water and pumped to the pyrite rougher and cleaner flotation cells for pyrite mineral collecting.

Pellet form gangue depressant will be delivered in bags and unloaded into a feed bin equipped with a dust filter and filter fan to manage dust control during unloading. The gangue depressant will then be mixed with water for the depression of gangue in the copper cleaner and pyrite cleaner flotation cells.

Liquid pyrite activator will be delivered in bulk road tankers and stored in a storage tank and pumped to the pyrite flotation circuit to serve as an activator.

Bulk bagged flocculants serving as settling aids in the copper and pyrite concentrates and the tailings thickeners will be stored in bins then mixed with raw water and pumped via an inline mixer, where the solution will be further diluted with raw water before reporting to the thickener feed boxes.

17.4 Power, water, and air

The estimated power requirements are 13 MW at the process plant which will be supplied from the Palos 115 kV substation in the Bucaramanga area via a 34 km long 115 kV, 45 MVA capacity overhead line leading to a new substation at the process plant, where the voltage will be stepped down to 13.8 kV for reception by the mine's main distribution substation. The main distribution station will supply power via 13.8 kV cables to the process plant and by 13.8 kV overhead line to the filtered tailings facility and water intake plant.

Standby and emergency power supply will be provided by a 3.125 MW diesel generator station at the process plant, a 630 kW diesel generator station at the filtered tailings facility, a 250 kW diesel generator at the water intake plant, and a 250 kW diesel generator at the emergency ponds.

The Project has been designed with robust water management and protection as guiding principles. There is sufficient and readily available water within the Project area for all aspects of the operation including mining, processing, the camp, and other activities. Construction water will be sourced under permits from the La Baja Creek and Suratá River, which both have abundant water supply. During operation, the main source of water for the underground mine and its associated infrastructure will be from groundwater and the main source of water for processing will be from the adjacent Suratá River.

The Project plans to reuse, recycle, and return approximately 96.5% of the Project's total water requirements back to the environment, resulting in a net water use estimated at 3.5%. The majority of the demand will be supplied by the Suratá River, which will supply approximately 2.8 litres per second of net make up water for the processing plant, representing 0.22% of the average flow of the Suratá River at the planned water access point near the village of Suratá, and 0.08% of the average flow of the Suratá River at Bucaramanga, the nearest city located 55 km downstream from the Project.

The Project includes a comprehensive water management plan that incorporates plans and programs tailored to each water application or activity with significant interaction with water to ensure that both the quality and quantity of local water resources are preserved throughout the life of the operation, and all potential impacts related to water use have been thoroughly identified, assessed, and addressed through prevention, mitigation, and compensation measures. The process plant has been designed to minimize the use of water and to recycle both process water and the water obtained from the dewatering of the process tailings. The filtered tailings storage facility has been designed to divert rainwater runoff away from the facility and to drain and manage any subsequent dewatering of the tailings and any rainwater falling onto the facility. Continuous monitoring of water flow and quality will be undertaken with defined measures planned if any changes are detected. These strategies will help ensure that the current water flow rates within the ecosystem are maintained.

Water from mine dewatering, seepage collected from the filtered tailings facility, and process water streams will report to seven water treatment plants where each stream will be treated separately, including a 0.68 litre per second plant to supply potable water at the mine, a 0.51 litre per second plant to treat camp waste water, a 43.18 litre per second plant for the treatment of waste water from the underground mine and paste plant, a 0.34 litre per second plant to provide potable water at the processing plant, a 34.12 litre per second plant to treat thickener bleed water, a 29.84 litre per second plant to treat the tailings filtration water, and a 0.13 litre per second plant to supply potable water at the filtration plant. Treated water from the domestic wastewater and industrial wastewater treatment plants located at the mine will discharge at La Baja Creek, in compliance with the discharge permits. Treated water from the other domestic and industrial wastewater treatment plants will be discharged to the Suratá River, all in compliance with the discharge permits. The potable water will feed the restrooms and safety showers.

Three low pressure air blowers will provide the flotation cells with air for particle attachment. Three air compressors will provide high pressure air for plant instruments and general service points. The filter press systems will also have dedicated compressor systems for cake blowing and drying.

17.5 Conclusions

Several metallurgical testwork programs have been undertaken between 2009 and 2018 to support the metallurgical assumptions utilized for the selected processing flow sheet for the current prefeasibility study, utilizing samples representative of the mineral resource and mineral reserve estimates. These studies included processing method trade off studies as well as refinements of the selected operating parameters, as the properties and response of the samples under the testwork conditions became increasingly better understood.

The processing plant is designed to treat plant feed throughputs of a nominal 2,750 tpd and a maximum 3,500 tpd. The processing plant will receive 2,750 tpd of crushed run of mine ore from the Soto Norte underground mining operation, and can receive an additional 750 tpd of mill feed purchased from contract mining partners, to produce three saleable products and one waste product, including a gold concentrate from a gravity gold operation, a copper concentrate from a copper flotation operation, a pyrite concentrate from a pyrite flotation operation, and a tailings product for disposal in the filtered tailings facility and to create paste backfill for the underground mine.

No cyanide or mercury will be used at the Soto Norte processing facilities. The processing circuit comprises primary ore crushing at the underground mine, ore transport from the mine portal to the process plant on the rope conveyor, receipt of mill feed purchased from contract miners at the process plant, primary grinding in a SAG mill with a supporting pebble crusher, secondary ball mill grinding, an upfront gold gravity recovery circuit to recover up to 15% of the coarse gold-silver particles and to produce a filter cake, and a two stage sequential flotation circuit to recover fine gold, silver, and copper in separate copper and pyrite concentrates.

The inclusion of a gravity circuit in the flowsheet minimizes the loss of coarser gold or larger gold bearing minerals to the tailings due to flotation dynamic limitations, particularly in the selectively mined mill feed purchased from contract mining partners, which has the potential to contain a disproportionate amount of coarser gold. Additionally, the upfront recovery of gold will result in a reduced mass pull in the downstream flotation circuit, reducing power and reagent requirements, and could facilitate the recovery of up to one third of the gold as a gold-silver concentrate or doré, which is economically more favourable.

A target grade of 16% for the copper concentrate has been set on the basis of testwork results as well as considerations for maximizing the gold content in the copper concentrate and reducing penalty element concentrations. Potential penalty elements considered in the economic analysis include arsenic, bismuth, cadmium, antimony, and likely zinc, with payments for contained copper and gold anticipated to be far in excess of any potential penalties. The non-sulphide waste content of the concentrates will be restricted to 10%.

Capital and operating cost estimates for the construction and operation of the processing plant have been considered in the economic analysis, including the requirements for power, water, and process materials.

For design purposes, average metallurgical recoveries are estimated at 92.8% for gold, 88.8% for silver, and 92.8% for copper.

Key changes in the current prefeasibility study process flow sheet compared to the flow sheet developed for the previous 2021 feasibility study include:

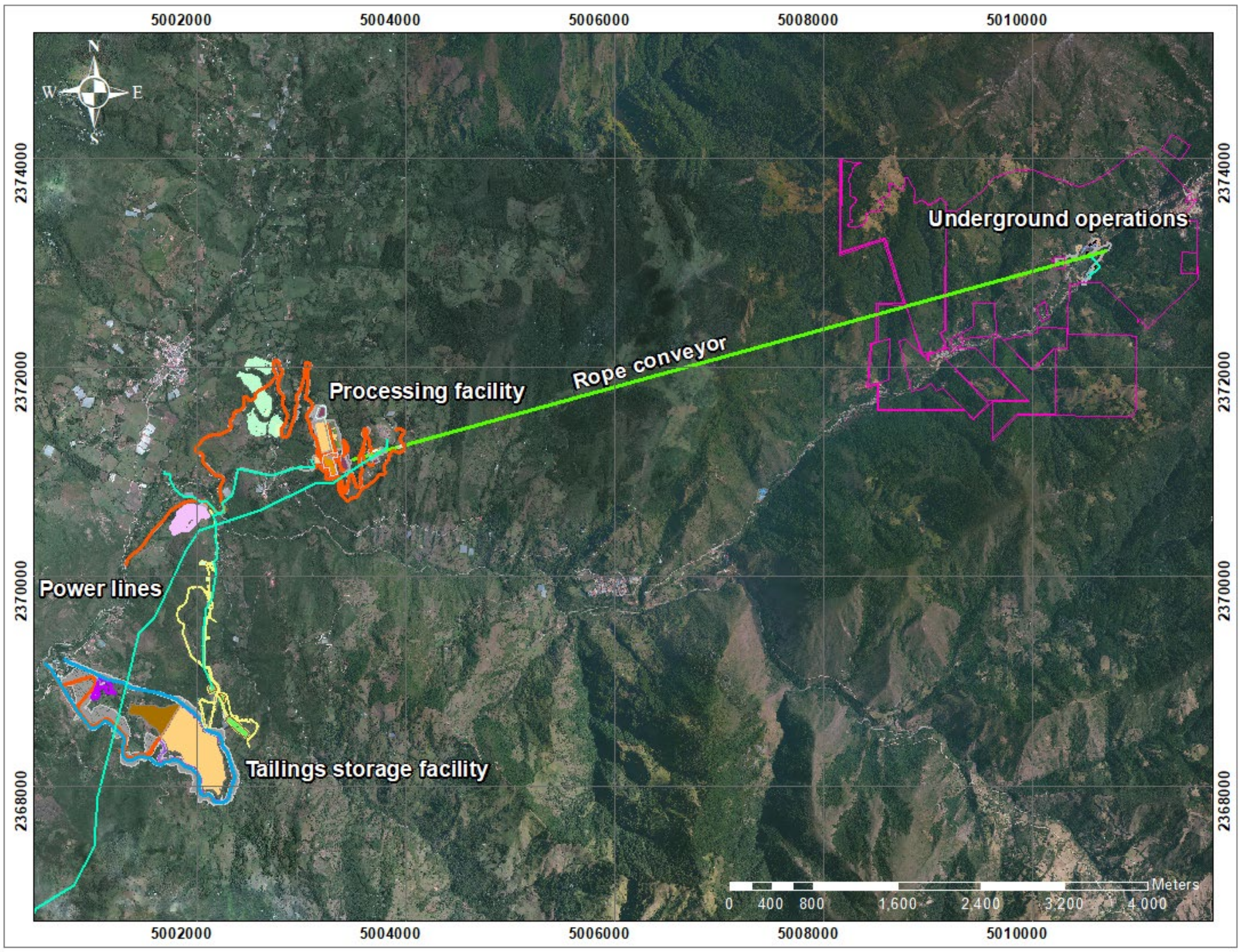
- Throughput reduced from 7,000 tpd to a minimum of 2,750 tpd and a maximum of 3,500 tpd.
- The addition of a separate receiving facility for mill feed purchased from contract mining partners.
- The inclusion of a ball mill to allow for finer grinding for improved gold liberation and recovery, higher throughput and load balancing, a more efficient use of energy, a reduction of overgrinding, better control of the grind size distribution, reduction of wear on the SAG mill liners, reduces the grinding media required in the SAG mill, and increases the efficiency of the SAG mill. The downsize is increased capital and operating costs and higher energy consumption, however, the power requirements estimated in the current prefeasibility study are estimated at 13 MW per annum, compared to the 39 MW estimated in the 2021 feasibility study.
- The inclusion of a gravity gold recovery circuit to recover up to 15% of the gold before it enters to flotation circuit, to reduce the amount of gold locked in the grinding circuit, minimize gold losses in the tails, the reduction of the mass pull in the downstream flotation circuit, reduction of power costs, and economic benefits on payable gold at close to 99%.
- The inclusion of a paste fill plant using the pyrite rougher flotation tailings for use as backfill in the underground mine.

18 Project infrastructure

18.1 Introduction

The Project site infrastructure is divided into four areas, including the underground mining site, the process plant and supporting services, the filtered tailings storage facility, and the site wide areas such as the water intake, roads, tailings pipeline, power line, and rope conveyor. Development of surface infrastructure is constrained by steep terrain, requiring careful planning to minimize the overall footprint. A plan of the planned Project infrastructure is shown in Figure 18-1. Existing infrastructure at the Project includes the mine portal and terrace, roads, a core cutting and logging shed, core storage facilities, offices, accommodation, and a mess hall.

Figure 18-1 Project infrastructure plan – Source Aris Mining 2025



18.2 Surface mine infrastructure

Several terraces have been planned to accommodate the mine infrastructure, and the existing portal terrace will be increased to accommodate the tail end structure of the rope conveyor. The facilities include ventilation shafts, warehouse, wastewater treatment plant, the mine portal, the tail end of the rope conveyor, electrical substation, accommodation blocks, the mess, and the camp administration office.

A new, 1.5 km long, 6 m wide roadway is planned for light vehicle access between the existing portal area and the mine shift change houses and the accommodation units. The road will utilize an existing bridge and will require the construction of two new bridges.

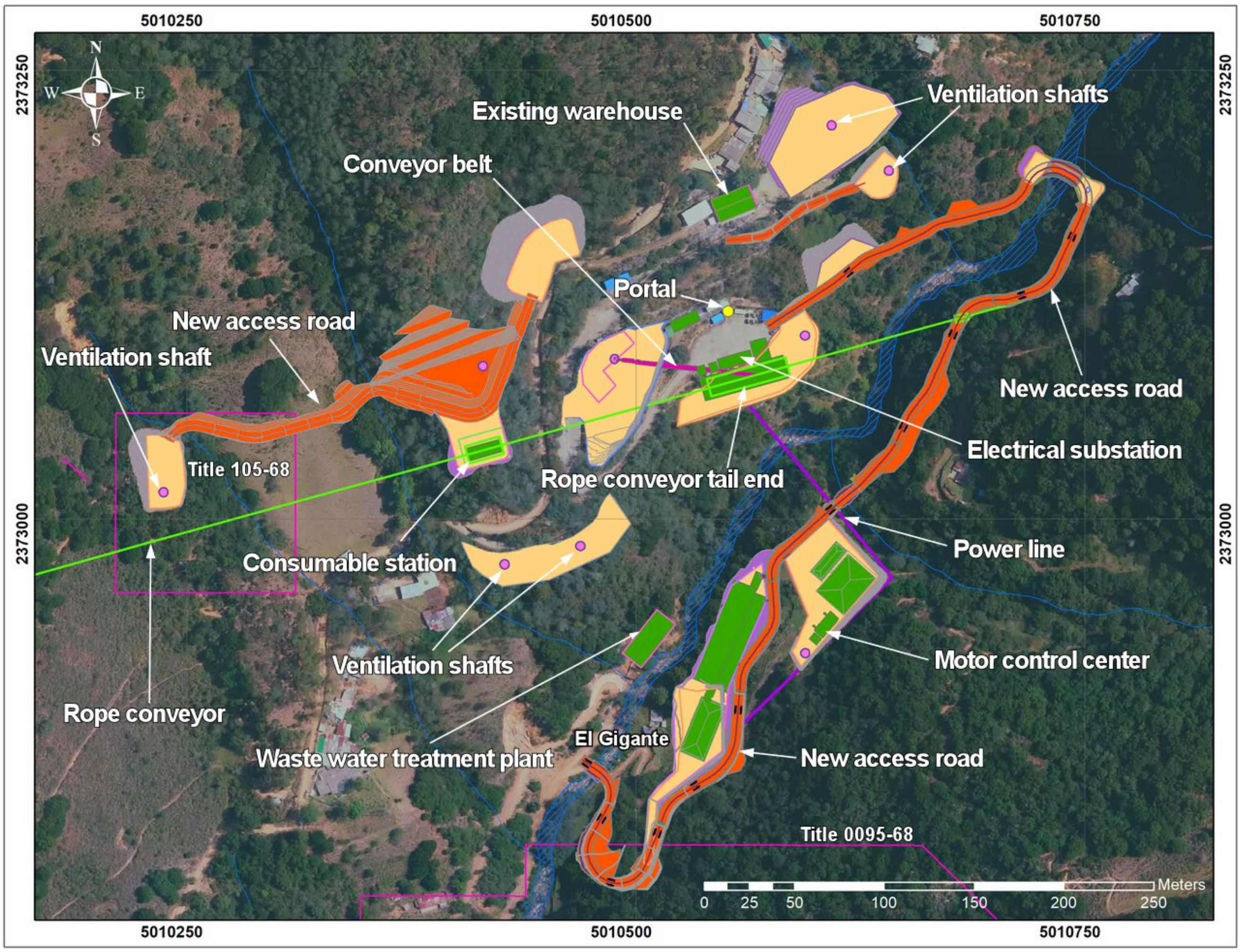
Two new accommodation blocks are planned to house the required personnel, including a block of double rooms that can house 216 people and a block of double rooms that can house 108 people, for a total number of 324 people that can be accommodated on site. The mess facility is located at the top of one of the accommodation blocks and can seat 48 people outdoors and 224 people indoors.

The mine shift change house can accommodate 40 male workers and 15 female workers, and is located close to the accommodation units. The camp administration office is located at the change house.

The power supply to the mine area will be provided via lines installed along the rope conveyor. The power substation, motor control centre, and two transformers will be located at the tail end of the rope conveyor.

A plan of the current and planned infrastructure at the mine area is shown in Figure 18-2.

Figure 18-2 Mine area surface infrastructure plan – Source Aris Mining 2025



18.3 Rope conveyor system

18.3.1 Introduction

The rope conveyor system is a proven, efficient, low footprint, low impact, and state of the art solution for material and cargo transportation. Environmental and community impact benefits include its silent operation, the elimination of road construction and related traffic and dust and exhaust emission issues, and minimal land disturbance restricted to the tower bases. The covered conveyor minimizes dust generation as material remains stationary during transport, and the system allows for straightforward closure and site rehabilitation at the end of its life. Economic and operational benefits include low capital and operating cost requirements, and relatively simple construction logistics. The straightforward design has only a limited number of moving parts, reducing the potential for defects, minimizing inspections and maintenance, and increasing operational availability. The majority of the maintenance can be carried out in a safe environment at the station. A photograph of an example rope conveyor system on similar terrain to the Project is shown in Figure 18-3.

Figure 18-3 Example of a rope conveyor system on similar terrain – source Doppelmayr 2025

18.3.2 Capacity

The Doppelmayr designed bi-directional rope conveyor system was planned for a daily throughput of 3,000 tpd with the capability of moving ore from the underground mine portal to the processing plant, returning tailings from the processing plant back to the underground mine for use in backfilling the mined stopes, the movement of consumables in carriers, and to carry the high voltage power and fibre optic cables. The automated system will have a length of 7,100 m and will cover an elevation difference of 461 m. Dedicated terraces will serve for loading and unloading at the underground mine and processing facilities. A plan of the rope conveyor system route is shown in Figure 18-4 and a long section is shown in Figure 18-5.

The system has a nominal capacity of 3,000 tpd and a maximum capacity of 5,500 tpd for transporting ore and waste rock from the mine portal to the processing plant, a nominal capacity of 1,080 tpd and a maximum capacity of 2,400 tpd for transporting tailings from the processing plant to the mine for use as backfill, and a capacity of 140 tpd for the transport of consumables from the processing plant to the mine.

Figure 18-4 Plan of the rope conveyor system route – source Aris Mining 2025

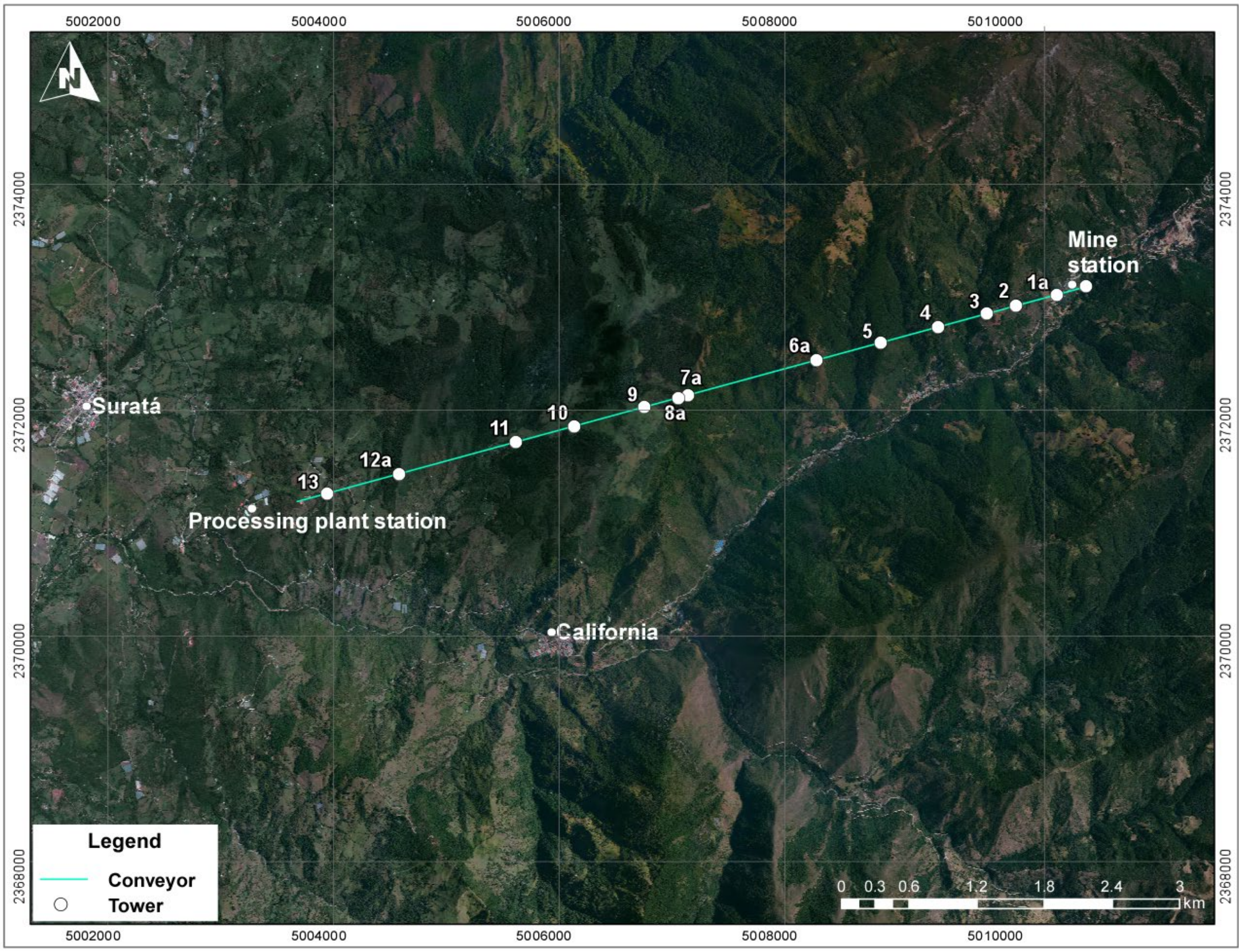
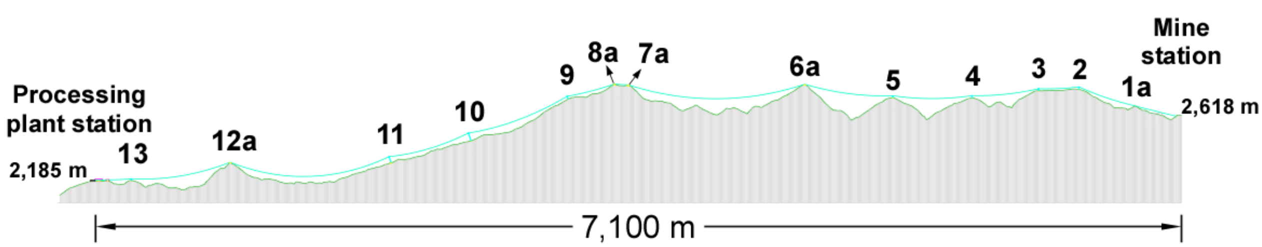


Figure 18-5 Long section of the rope conveyor system route – adapted from Doppelmayr 2025



18.3.3 Design

The line structure is comprised of six fully locked steel wire track ropes, with track rope frames mounted at regular intervals to keep the six ropes in alignment while distributing the forces acting on each rope. The lines are tightly tensioned and anchored at both ends. The running wheels of the conveyor belt travel on the track ropes.

Tubular shaft towers are installed at intervals depending on the topography, as shown in Figure 18-4 and Figure 18-5, in order to provide sufficient clearance between the system and the ground or vegetation, and the track ropes are clamped onto the tower heads.

Material is transported on a continuous, reinforced flat belt comprised of steel cords, with corrugated side walls and transversal cleats. The conveyor belt is driven by drive drums and has two independent mechanical braking systems. The conveyor belt is screwed onto axles arranged at regular intervals to support the conveyor belt, and a running wheel is fitted to each end of an axle. Together the axle and running wheels form a wheel set, and travel on the track ropes, guiding the conveyor belt, and passing through the stations at regular intervals. The running wheels guide the belt along the entire length of the line so that no skewing occurs with crosswinds or irregular load conditions.

18.3.4 Operation

Ore, waste, and tailings will be fed onto the conveyor via a chute. The conveyor operates in a continuous loop, with material transported on both the upper and lower sides of the belt. The top belt has a roof cover.

Consumables will be transported by five 1,800 kg capacity cargo carriers and one 900 kg combined inspection and cargo carrier, connected to the hauling rope. Loading and unloading the first and second carriers in the station with two forklifts on both side of the conveyor takes two minutes. It takes 45 seconds to move the third and fourth carriers to the station and two minutes to unload and unload those carriers, then 45 seconds to move the last fifth and sixth carriers to the station and two minutes to unload and load. Once loaded or unloaded, the travel time from the first to last station is 45 minutes.

18.3.5 Closure

At mine closure, the rope conveyor stations, towers, and conveyor components will be disassembled and removed, along with the foundations and any other structures, followed by landscape restoration. The closure costs have been included in the Project closure cost estimates.

18.3.6 Conclusions

There are no technical risks related to the operation of the rope conveyor as all parameters are within values known to the supplier and are in operation in other locations. The key risks and uncertainties that could affect the Project are with respect to construction costs and timing, including geotechnical conditions at the tower locations, the final design parameters, and accessibility to the tower locations.

Detailed engineering studies prior to construction will provide input for a more detailed construction cost estimate and further minimize risks related to refining the route, foundation requirements, system technical parameters and specification, power requirements, the interfaces with the mine and processing plant, electrical engineering, cost quotations, and the construction schedule.

18.4 Process support facilities

The process support facilities will include a waste receiving area at the rope conveyor station, dedicated receiving areas for underground ore and mill feed purchased from contract mining partners, the milling circuit, the process plant, services, accommodation, storm water pond, roads, the main power supply, and an existing military base. A plan of the process support facilities is shown in Figure 18-6.

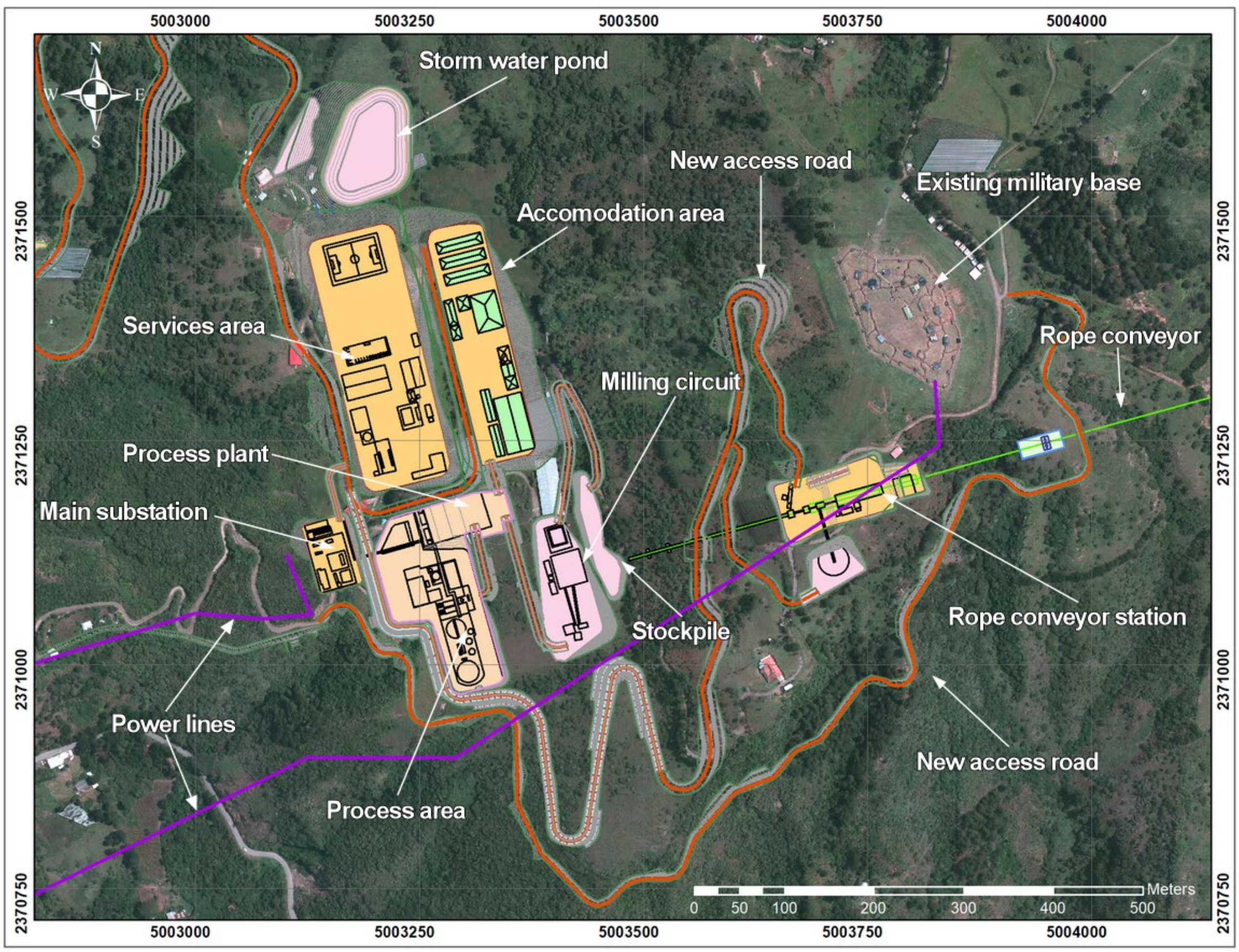
The ore and waste from the underground mine will be transported from the mine portal to the processing facilities on the rope conveyor. Waste material will be diverted to a stockpile and then transported by truck for final placement.

The milling circuit facilities will include the ball and SAG mills, conveyors, pebble crushers, and space for a gold safe room. The process facilities will include the flotation circuits, reagent storage facilities, water treatment facilities, power supply, a weigh bridge, thickening and filtration facilities, roads, and a tailings pump line. A dedicated, approximately 6 km long route is planned for the tailings line with spillage risk mitigation features including a containment channel and access road for maintenance and will also be used for an overhead line supplying power from the main substation to the water intake, filter plant, and ancillary equipment.

The service area will include a workshop, warehouse, mobile equipment workshop, outdoor storage yard, fuel station, offices, wash bay, diesel storage, solid waste disposal yard, modular sewage treatment facility, access gate, and soccer field.

The accommodation area will include the accommodation blocks to house 170 people, mess hall, entertainment area, gym, clinic and emergency response building, and core sheds.

Figure 18-6 Plan of process support facilities – Source Aris Mining 2025



18.5 Filtered tailings storage facilities

18.5.1 Site description

Multiple potential tailings storage facility locations have been considered and engineered for the Project over time, including feasibility study level designs disclosed in the 2021 feasibility study, and scoping level engineering studies of two alternative sites. The current facility is now located at one of the previously considered alternative sites, situated 3.5 km to the south of the processing facility in a valley surrounded by steeply sloped mountainous terrain.

The valley side slopes are moderately steep, with some slopes exceeding 1 vertical to 1 horizontal (1:1). Slopes along the valley bottom are shallower, ranging from 1:3 to 1:4. Elevations range from 1,950 m above sea level at the upper reaches of the facility basin to 1,650 m above sea level near the planned contact water collection pond. The location is primarily in a grassland area, with areas of dense shrubs and trees present along the upper reaches of the tailings basin.

The Project and the facility site are located in a high seismic region. Lettis Consultants International Inc (LCI) completed a site-specific seismic hazard evaluation in November 2022, with an update in April 2024 to evaluate alternative locations for the facility, including the current location. The work by LCI included both probabilistic and deterministic seismic hazard analyses which were utilized for the current prefeasibility study engineering design.

18.5.2 Site investigations and engineering analyses

Site investigations by Ingetec Ingenieros Consultores of Bogotá, Colombia during 2022 and 2023 included boreholes, test pits, geophysical surveys, and laboratory testing on sampled materials. The borehole drilling comprised geotechnical and geological logging, the execution of in situ permeability tests, standard penetration tests, soil and rock sampling, and the installation of vibrating wire piezometers to measure pore pressures. The test pits were excavated for surficial geotechnical and geological logging, sampling of surficial materials, and the execution of in situ density tests. The geophysical survey work included seismic refraction and multichannel analysis of surface waves lines. Samples recovered from boreholes and test pits were subjected to laboratory tests to study the properties of the foundation materials, including index tests, permeability tests, and shear strength tests.

Knight Piésold developed a water balance model using climatic and hydrologic data from historical records, supplemented with synthetic data where required, and mine data for the future operations, to simulate water transfers and storages. The water balance simulations cycle through the future mine operational conditions and the historical data records to create data that represent a range of possible flows and volumes over time. A sensitivity analysis was performed to estimate the variation of maximum volumes required to be stored in the collection pond.

Hydrologic analyses were undertaken by Knight Piésold to estimate peak flows for the design of hydraulic structures such as channels and the collection pond, erosion protection requirements, and the required sizes and geometries of the system components.

Deterministic, limit equilibrium slope stability analyses were also advanced by Knight Piésold on the facility embankment, tailings stack, and contact water collection pond dam under a variety of loading conditions. The findings from the stability assessment were incorporated into the facility design to confirm acceptable slope stability.

18.5.3 Design

Knight Piésold conducted prefeasibility study engineering of the filtered tailings facility and associated contact water collection pond, haul road, and surface water management structures. The civil design components of the facility include the foundation and embankment abutment preparation, embankment, tailings stacking plan, geosynthetic liner system, drainage systems, contact water collection pond, and ancillary infrastructure including roads and surface water management channels. The facility design meets slope stability requirements as guided by the Canadian Dam Association and the International Committee on Large Dams, standards that are internationally recognized as best practice and provide detailed technical design and risk management criteria. The facility has a capacity of 11.8 million tonnes, sufficient to accommodate the current mineral reserve requirements of 10.3 Mt of combined process tails and waste rock. Additional storage capacity may be required in the future, depending on the amount of any future mineral reserves discovered through exploration drilling programs and on the volume of material purchased from contract mining partners.

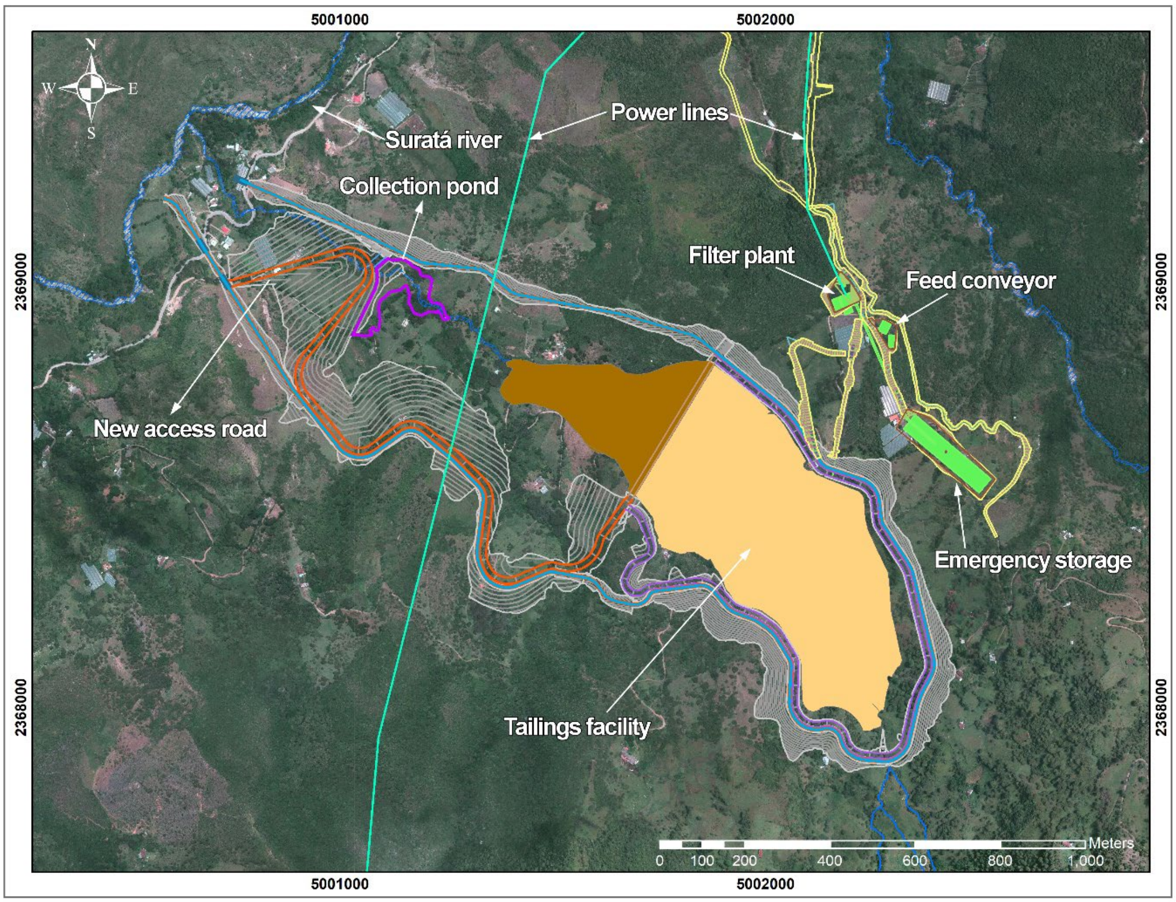
The facility will comprise a tailings basin formed by natural topography, excavations into existing ground, placement of grading fills, and containment embankment at the downstream end of the tailings basin constructed of structural fill. Approximately 10 m of material will be removed from the basin and embankment footprint to facilitate the required tailings storage capacity and to provide competent foundation materials. The entire tailings basin and most of the upstream face of the containment embankment (all of the surfaces that will be in contact with tailings) will be lined with a composite liner comprising geomembrane overlying a layer of compacted low-permeability soil. To facilitate geosynthetic placement, all basin slopes will maintain a maximum 1:2 slope. An additional 10 m excavation is required beneath the downstream embankment toe to satisfy geotechnical slope stability requirements. The composite liner will likely consist of a 2 mm thick, double sided textured linear low density polyethylene geomembrane.

A temporary tailings storage area is planned for the temporary placement of tailings when rainy field or upset plant conditions prevent the acceptable placement of tailings.

A 2 km long haul road will extend from the existing public road to the facility embankment crest. The haul road is configured to provide containment and act as the downstream retaining embankment for the contact water collection pond. A 2.3 km long facility perimeter access road will run along the facility perimeter to allow access to the facility for operations and maintenance.

A plan of the filtered tailings storage facilities is shown in Figure 18-7.

Figure 18-7 Plan of filtered tailings storage facilities – Source Aris Mining 2025



18.5.4 Water management

The surface water management strategy for the facility is to construct non-contact diversion channels to limit surface water runoff over the tailings and embankment surfaces during the operations and passive closure phases. The primary drainage systems will comprise an underliner drainage system, which includes the embankment's internal drainage system, and an overliner drainage system.

The granular underliner drain system will be placed in the excavation of the embankment and tailings basin to aid with dewatering during construction, collecting seepage from the embankment, and limiting upward pressure on the basin lining system during operations. The main underliner drain will be located along the low point of the embankment excavation, with additional drains constructed up the natural drainages in the basin areas. Additional lateral underliner drains may be placed during construction in side valleys as site conditions dictate. The embankment's internal drainage system, anticipated to comprise a chimney and blanket drain to evacuate seepage that penetrates the liner system on the upstream slope of the embankment, will tie into the underliner drains. Seepage and groundwater collected by the underliner drainage system will be discharged to the contact water collection pond.

The overliner drainage system will comprise a continuous blanket drain placed between the composite liner and the filtered tailings throughout the entire tailings basin. This layer will provide drainage for the seepage contact water released as the tailings stack consolidates over time. Furthermore, the blanket drain extends up the basin sidewalls and daylights at the surface such that direct precipitation on the tailings stack (contact water) may run off into the drain. Water captured in the overliner drain will be routed to a sump at the upstream toe of the embankment, where it will enter a series of pipes that pass through the composite liner system, along the foundation of the embankment, and discharge into the contact water collection pond. Concrete encased steel pipes will be located under the embankment for adequate resistance to loading and potential movement, and perforated corrugated polyethylene pipes will be located in the basin area. The overliner drain will also help protect the geomembrane liner from environmental effects and traffic during tailings placement and compaction.

The contact water collection pond will have a minimum storage capacity of 70,000 cubic metres and will be located downstream of the embankment. It will be formed by the facility haul road, which acts as the collection pond's dam, and natural topography. An emergency spillway is included in the design. The upstream face of the collection pond dam and its pond area will be lined with a composite liner comprising a low permeability subbase overlain by a geomembrane, which will extend to the average maximum monthly pond elevation. Above the maximum monthly pond elevation, the pond will be soil lined. All lined surfaces will be overlain by a protective soil layer and articulated concrete blocks such that cleaning equipment may enter to remove sediment accumulated in the pond without risking damage to the pond liner. The water in the collection pond will be pumped to the water treatment plant.

Non-contact stormwater outside of the facility footprint will be routed around the outer perimeter of the facility via two non-contact diversion channels that will discharge downstream of the collection pond to the Suratá River in compliance with the discharge permits. The water diversion channels are designed to divert up-gradient runoff around the facility to limit water from running onto the tailings stack and embankment.

18.5.5 Tailings placement

The facility will store filtered tailings from the processing plant and potentially acid generating waste rock from the underground mine. A system of portable conveyors known as grasshoppers will transport the filtered tailings from the filtration plant to the perimeter of the tailings facility, from where they will be loaded in trucks for hauling and deposition to the tailings basin. Truck hauling of the filtered tailings will be utilized when the conveyors are under maintenance.

To support with geotechnical stability and to limit long term settlement of the tailings, the filtered tailings will be placed and compacted to an in place density of at least 95% of the Standard Proctor maximum dry density. The in place moisture content will likely be less than approximately 14%. The tailings will be placed in horizontal lifts, starting at the upstream toe of the embankment and extending up the valley. At full buildout, the majority of the overall tailings slope will be 1:7.5 from the upstream embankment crest. The slope flattens to an overall slope of 1:20 in the upper reaches of the facility, reflecting underlying changes in the orientation of the valley.

Potentially acid generating waste rock will be hauled to the facility by truck and placed in trenches excavated into the compacted tailings stack, then immediately covered with compacted filtered tailings to ensure long term geochemical stability.

18.5.6 Closure

Knight Piésold developed a closure plan to provide a long term solution for a safe, structurally stable, and non-erodible cover for environmental sustainability. Chemical and physical stability of the facility will be maintained by careful consideration of directing and managing surface water runoff and by designing a cover system that will effectively control meteoric infiltration into the tailings facility and perform appropriately with little to no maintenance.

The facility closure plan for the current prefeasibility study was guided by federal regulations and guidance by ANLA, and wherever that guidance was inapplicable, local regulations and/or international standards were considered. These closure activities will be modified as the facility design is advanced through future engineering phases. The closure plan must be updated every five years to reflect changes in mining activity, and technological, technical, regulatory, economic, social, or environmental developments. Requirements from applicable closure guidelines include developing closure activities, monitoring programs to verify the status of the closure components, and creating a schedule of closure activities.

The tailings basin will be closed by progressively placing a cover system over the tailings as operations proceed. The cover system will comprise an upper revegetated layer or growth media overlying a compacted clay zone which will help prevent wind or water from eroding the tailings and will reduce water infiltration into the tailings. The closure cover will include riprap lined bench channels sized to direct runoff associated with the design storm event. The final soil cover area will be graded to direct precipitation on the cover to the perimeter non-contact channels and to help deter the effects of localized tailings settlement. The downstream slope of the containment embankment will also be progressively closed as construction progresses. Closure design of the embankment slope includes a network of surface drainage channels and benches directing runoff to the non-contact channels to help control concentrated flows and mitigate erosion of the embankment face and closure cover. A revegetation plan will be developed and implemented for all surfaces with closure covers to help anchor the surficial soils and reduce surficial soil losses due to rainfall.

The facility contact water collection pond will be decommissioned when seepage water samples indicate that the water quality meets discharge standards. Tanks may be used to store the seepage water, which will be regularly pumped to a water treatment plant. The overliner drain will be covered at final closure and runoff from the closure cover will be directed to the non-contact channels. The contact water collection channel on the embankment’s downstream face will be extended to bypass the decommissioned collection pond once adequate water quality is achieved.

Service roads around the perimeter of the basin and embankment will be maintained to provide access for monitoring and maintenance activities. Electrical utilities for systems such as pumps and monitoring equipment will also remain functional.

During closure and post closure, monitoring using instrumentation and visual inspections will be conducted to confirm compliance with the facility permits, demonstrate the physical and chemical stability required for closure, and track the site’s progress towards its closure goals. The length of the passive closure period will be defined based on the post closure monitoring period. The monitoring elements will include environmental and physical monitoring such as seepage water quality at the collection pond, revegetation establishment surveys, phreatic surfaces within the tailings facility, potential movement of the tailings facility, and erosion observations and maintenance. Maintenance activities will primarily consist of repairing any erosion damage cracks, or settlement that may form due to tailings consolidation. International best practice indicates that monitoring should continue for at least five years after post closure, and the required length of this period will be defined during future design phases.

An estimate of the closure costs has been included in the economic analysis.

18.5.7 Post closure monitoring

International best practice on the operation and closure of tailings storage facilities indicates that performance monitoring is a critical component to safe management of a facility during its lifecycle, including through post closure activities. Geotechnical instrumentation that monitors items such as pore pressures, phreatic levels, and deformation provides valuable information on a facility’s performance and may provide early warning signs of potential problems. In addition to instrumentation monitoring, visual monitoring and maintenance activities are required to comply with the objectives of the closure design, and may be performed on a routine basis or as needed as the result of visual inspections and monitoring data evaluation.

Knight Piésold developed a post closure monitoring plan consistent with the facility’s current design, including the required geotechnical instrumentation, the installation locations, monitoring, and maintenance. During closure and post closure, monitoring will be carried out to comply with the facility permits, demonstrate the physical and chemical stability required for closure, and track the site’s progress towards its closure objectives. The post closure period is assumed to last a minimum of two years, followed by passive closure. The length of the passive closure period will be a function of the duration of the post closure monitoring, the next land use of the area, and achieving closure and post closure objectives. Monitoring elements will include both environmental and physical monitoring, such as the quality of the infiltration water in the contact water collection pond, topographic surveys to verify vegetation establishment, instrumentation of the facility embankment to evaluate stability, and observations related to erosion and maintenance.

The plan for instrumentation monitoring includes the installation of geotechnical instrumentation in the filtered tailings storage facility and the contact water collection pond dam to monitor both facilities during construction, operations, closure, and post closure. These instruments include open standpipe piezometers, vibrating wire piezometers, and survey prisms.

Open standpipe piezometers will provide a direct measurement of groundwater levels around the facility and the contact water collection pond to develop a contour map of groundwater levels and to evaluate flow entering the underliner drain system. The open standpipe piezometers will also be used to sample groundwater upstream and downstream of the facility for water quality testing to assess potential effects of the filtered tailings storage facility on groundwater quality throughout the life of the facility.

Vibrating wire piezometers will be installed in the filtered tailings storage facility both above and below the geomembrane liner. The piezometers above the geomembrane will evaluate the performance of the overliner drainage system and potential buildup of pore pressure within the tailings stack. The piezometers below the geomembrane will monitor potential pore pressure buildup within the embankment, evaluate the uplift pressure beneath the composite liner system, detect potential seepage through the composite liner, assess the effectiveness of the internal drainage system, and support the development of groundwater level contour mapping. Additional vibrating wire piezometers will be installed within the facility foundation to supplement the existing piezometers in monitoring phreatic conditions within the facility foundation. Vibrating wire piezometers will also be installed in the contact water collection pond dam to monitor the development of pore pressure within the dam during operations and into the closure period.

Survey prisms will be installed on the facility embankment, on the filtered tailings stack, and on the contact water collection pond dam to assess vertical settlement and horizontal displacement including magnitude, direction, and rate of change, to confirm that any displacements do not exceed allowable tolerances.

18.5.8 Recommendations

Engineering of the filtered tailings facility have been advanced to prefeasibility study level. Knight Piésold makes the following recommendations to advance the design of the facility to the detailed engineering level, at an estimated cost of $3 million, considering both engineering consulting and contractor costs:

- Complete site investigation work including borehole drilling, test pit excavation, instrumentation installation, in situ testing, and geophysical surveys to inform the next phases of design.
- Advance geotechnical laboratory characterization work on the foundation soils in their in situ state and as structural fill following excavation, moisture conditioning, placement, and compaction.
- Design and size the required drainage works.
- Undertake slope stability analyses to evaluate facility phasing concepts.
- Explore opportunities to deposit tailings in non-horizontal lifts to reduce staging challenges.
- Develop facility safety and operations documents, including the operations, maintenance, and surveillance manual, the emergency response plan, and the trigger action response plan.
- Develop breach and inundation studies on the filtered tailings facility and contact water collection pond for potential damage assessment, risk classification, potential loss of life analysis, and emergency plan design and implementation.
- Complete additional geotechnical analyses as required by the regulators and the Project, such as probabilistic slope stability analyses and/or nonlinear dynamic deformation models.
- Advance applicable engineering analyses to levels commensurate with future design phases.
- Develop a groundwater sampling program.
- Develop detailed closure requirements for the facility.
- Develop a cover system that will limit long term infiltration into the facility.
- Further refine grading of the required excavation works, containment embankment, and tailings stack.
- Create a cover revegetation plan.

18.6 Stormwater management

18.6.1 Introduction

Average annual precipitation of between 1,000 and 1,500 mm occurs at the Project, which can lead to challenges such as surface runoff, erosion, and sediment transport. Stormwater runoff and preliminary control measures were evaluated to mitigate the potential impacts of rainfall on mining infrastructure and surrounding areas, reduce environmental impacts, and ensure compliance with environmental requirements.

No detailed hydrological assessment was conducted due to limited available data, and therefore the Rational Method was applied as a preliminary approach to estimate peak flows based on available rainfall and drainage area information. The study focussed on the processing plant, portal, and filtered tailings facility areas, which were further divided into smaller catchments based on topography and the natural watercourse flow paths. Two distinct water categories were considered, including clean water and grey water. Clean water refers to runoff from natural soil surfaces, roads, and other areas where there is no risk of contamination. This water remains unpolluted and can be safely managed within the natural drainage system. Grey water includes runoff from areas where contamination is possible, such as locations with process operations, chemical exposure, or oil residues. To prevent environmental contamination, grey water must be collected separately from clean water and treated prior to discharge, ensuring downstream water sources remain protected.

18.6.2 Stormwater management plan

Runoff from clean water areas will be either diverted to the surrounding environment or channeled to a designated location, where it converges with other catchment areas. The collected water will be directed towards the nearest river through controlled drainage pathways, in compliance with the water discharge permits. Grey water run off will be collected separately and diverted to a collection pond. A water treatment plant will then process the water which will then be either pumped to the process plant or safely discharged into the environment in compliance with the water discharge permits.

18.6.3 Stormwater management infrastructure

A range of infrastructure including channels, pipes, weirs, and concrete boxes will be used to transport and manage clean and grey water across the Project. Each catchment area will be evaluated for its specific water flow characteristics, and the infrastructure will be designed to ensure adequate capacity for managing stormwater according to the calculated peak flow and storm duration values. The design of the stormwater system will ensure that water is effectively managed, treated if necessary, and safely directed to its final destination, either through the stormwater network or released to the environment in a controlled manner. The road infrastructure will primarily focus on managing clean water, ensuring that runoff from the road is effectively collected and directed without contamination, ensuring that runoff is managed sustainably, minimizing environmental impacts and ensuring that clean water remains separate from grey water sources.

The channels will be designed to manage runoff from the catchment areas, by directing water to either collection points or natural drainage systems. Piping systems will be employed where open channels are not feasible or where more controlled flow is needed, particularly in areas with limited space or where the infrastructure needs protection. Weirs will regulate water flow, preventing overflow, and direct water towards the appropriate storage or treatment locations. Concrete boxes will function as detention ponds designed to temporarily store water. The ponds will help manage stormwater by allowing it to accumulate, slow down, and gradually release it to the drainage system, reducing the risk of downstream flooding and ensuring that water quality is maintained by keeping clean and grey water separated.

18.6.4 Recommendations

A comprehensive hydrological study is recommended for future studies to refine the stormwater design, assess the long term water balance, and develop a more detailed mitigation strategy. Most of this work is currently underway to support the ESIA application.

18.7 Utilities

18.7.1 Water sources and water treatment facilities

The Project has been designed with robust water management and protection as guiding principles. There is sufficient and readily available water within the Project area for all aspects of the operation including mining, processing, the camp, and other activities. Construction water will be sourced under permits from the La Baja Creek and Suratá River, which both have abundant water supply. During operation, the main source of water for the underground mine and its associated infrastructure will be from groundwater and the main source of water for processing will be from the adjacent Suratá River.

The Project plans to reuse, recycle, and return approximately 96.5% of the Project's total water requirements back to the environment, resulting in a net water use estimated at 3.5%. The majority of the demand will be supplied by the Suratá River, which will supply approximately 2.8 litres per second of net make up water for the processing plant, representing 0.22% of the average flow of the Suratá River at the planned water access point near the village of Suratá, and 0.08% of the average flow of the Suratá River at Bucaramanga, the nearest city located 55 km downstream from the Project.

The Project includes a comprehensive water management plan that incorporates plans and programs tailored to each water application or activity with significant interaction with water to ensure that both the quality and quantity of local water resources are preserved throughout the life of the operation, and all potential impacts related to water use have been thoroughly identified, assessed, and addressed through prevention, mitigation, and compensation measures. The underground mine has been designed to minimize groundwater flows into the underground workings through advanced cover drilling and grouting ahead of mine development to identify and seal any water bearing structures before mining reaches them, greatly reducing potential inflow, and to manage, treat, and if required, safely return any captured water to the environment in compliance with the environmental standards and discharge permits. The process plant has been designed to minimize the use of water and to recycle both process water and the water obtained from the dewatering of the process tailings. The filtered tailings storage facility has been designed to divert rainwater runoff away from the facility and to drain and manage any subsequent dewatering of the tailings and any rainwater falling onto the facility. Continuous monitoring of water flow and quality will be undertaken with defined measures planned if any changes are detected. These strategies will help ensure that the current water flow rates within the ecosystem are maintained.

Water from mine dewatering, seepage collected from the filtered tailings facility, and process water streams will report to seven water treatment plants where each stream will be treated separately, including a 0.68 litre per second plant to supply potable water at the mine, a 0.51 litre per second plant to treat camp waste water, a 43.18 litre per second plant for the treatment of waste water from the underground mine and paste plant, a 0.34 litre per second plant to provide potable water at the processing plant, a 34.12 litre per second plant to treat thickener bleed water, a 29.84 litre per second plant to treat the tailings filtration water, and a 0.13 litre per second plant to supply potable water at the filtration plant. Treated water from the domestic wastewater and industrial wastewater treatment plants located at the mine will discharge at La Baja Creek, in compliance with the discharge permits. Treated water from the other domestic and industrial wastewater treatment plants will be discharged to the Suratá River, all in compliance with the discharge permits. The potable water will feed the restrooms and safety showers.

18.7.2 Power

The estimated power requirements are 13 MW at the process plant and 10 MW at the underground mine, for a total Project requirement of 23 MW. An existing 34.5 kV power line from Bucaramanga supplies a 5 MVA at 34.5 kV capacity substation at the underground mine, which can be used for construction power and as an alternative emergency power source, but is insufficient for operations.

Electrical power to the operations will be supplied from the Palos 115 kV substation in the Bucaramanga area via a 34 km long 115 kV, 45 MVA line capacity overhead line leading to a new substation at the process plant, where the voltage will be stepped down to 13.8 kV for reception by the mine's main distribution substation. The main distribution station will supply power via 13.8 kV cables to the process plant, by 13.8 kV overhead line to the filtered tailings facility and water intake plant, and by a 7 km long 13.8 kV cable installed on the rope conveyor to service the underground operation and the underground substation. Supporting infrastructure includes electrical rooms and transformers.

Standby and emergency power supply will be provided by a 3.125 MW diesel generator station at the process plant, a 630 kW diesel generator station at the filtered tailings facility, a 250 kW diesel generator at the water intake plant, a 250 kW diesel generator at the emergency ponds, and a 2.5 MW diesel generator station at the underground mine.

18.7.3 Fuel

Site wide diesel requirements will be serviced from the terraces at the processing plant site where a 500,000 litre diesel tank will be located. The tank will be fuelled at regular intervals from a local vendor. A diesel bowser truck will be used to transport diesel from the tank to the back up generators. The tank will also connect to the subterranean fuel station tank located near the diesel tank. The 20,000 litre subterranean diesel tank will primarily service plant light vehicles. A localized fuel tank can be placed at the heavy vehicle workshop or closer to the filtered tailings storage facility to service haul trucks.

18.8 Terracing

Terracing will be required to form a flat working area where required, and will be constructed from cut material with structural fill at the front face of the terraces. Where deep cut sections will be required, the cut faces will be stabilized with soil anchors. The fill face of the terrace will be stabilized with a concrete retaining system with geotextiles anchored between fill layers and attaching to the wall face with anchors up to 11 m long. Eleven terraces are planned at the mine, 13 at the processing plant and filtered tailings storage facility, and four at the filter plant.

The current design is based on a generalized soil profiles. During the detailed design stage, more focussed investigations at each relevant site will be conducted to characterize the soil conditions.

18.9 On site roads

The main access road will be from the Puente Panaga site located on the Suratá secondary access road, and will link the processing plant to an existing access road. This road will then connect to the filtered tailings storage facility site and the access to the filter plant and temporary storage areas at the facility. There will be 1.1 km of roads at the mine and accommodation camp, 15.7 km of roads to access the processing plant, filtered tailings storage facility, and water intake terraces, and 11.5 km of roads at the filter plant. The secondary roads are classified as local service roads or community roads that will bypass the processing plant activities. Road widths vary from between 5 and 10 m wide, with the narrower roads located in areas with steep valleys.

Four bridges are planned for the Project, including one for the Vetas River crossing near the processing plant and three located at the mine entrance for local access to the accommodation and support facilities. The bridges will be constructed from pretensioned concrete profiles and designed to counter any seismic activity and to accommodate higher weight loads during the construction process.

18.10 Off site logistics

The Project will produce three gold bearing saleable products, including a gold rich copper concentrate, a gold rich pyrite concentrate, and a gravity recovered gold concentrate, which will be sold to overseas refiners.

The copper and pyrite concentrates will be loaded using conveyors equipped with weightometers and samplers onto lined, 20 foot equivalent containers with a nominal payload of 24 tonnes at the processing plant, then transported with the gravity recovered gold concentrate bins by road to a river port terminal at Barrancabermeja on the Magdalena River, approximately 166 km from Suratá. The containers and bins will be stored then loaded onto river barges for transport approximately 650 km to the seaport at Cartagena, one of the largest in South America, and temporarily stored.

The road haulage operation to Barrancabermeja will be subcontracted to a haulage contractor utilizing the existing road network. The round trip time is approximately 15 hours. Import of equipment and material for Project construction and operation will follow the same route in reverse, or can be transported by road. The use of haulage contractors and existing third party infrastructure including the road, terminal, and river transport, can help minimize Project capital costs and utilize the readily available third party knowledge and resources.

A current river terminal in Barrancabermeja is ready to accept the proposed container traffic, and if necessary, there is an alternative by road. The river terminal facilities include a single harbour crane and a suite of barges and tugs. The river port operator will be responsible for unloading containers from trucks, the storage of containers, and loading to barges, and loading to an ocean going vessel at Cartagena. The most commonly used configuration includes a 1,800 to 4,500 horsepower pusher boat and six to eight barges with a capacity of 1,000 to 1,600 tonnes, holding 36 to 58 containers per barge. The terminal has sufficient capability to handle the Project throughput.

At Cartagena, the copper concentrate containers will be loaded onto a marine vessel and the pyrite concentrate containers will be unloaded to allow for bulk shipping of the concentrates. The final destination for both concentrates is to European or Asian smelters.

19 Market studies and contracts

19.1 Market studies

Several metal traders and market participants were consulted to provide current indicative commercial terms for the three types of concentrates that will be produced at the Project, including gold rich copper and pyrite concentrates and a gravity recovered gold concentrate. The terms received from the various parties were broadly consistent. Sensitivity analysis indicated that using any set of sales terms resulted in only minimal impacts to the economic analysis of the Project. For the purpose of analysis, the final commercial terms were selected from the market participant that was considered to be the most relevant to the Project's circumstances.

19.1.1 Indicative commercial terms for the copper concentrates

The copper concentrates are expected to have a copper grade of approximately 18%, which is below the typical copper content of global trade seaborne concentrates of approximately 22 to 30% Cu. The concentrate is also expected to have a gold grade of approximately 385 g/t and a silver content of approximately 1,711 g/t, which is considered acceptable for marketing as a gold-bearing concentrate, particularly into Chinese markets.

The marketing terms assumed for the economic analysis were:

- Gold – Payability of 96.0%, with a minimum deduction of 1.5 g/t Au; refining charges estimated at $10.00/oz
- Copper – Payability of 96.5%, with a minimum deduction of 1.2% Cu; refining charges estimated at $0.08/lb
- Silver – Payability of 96.0%, with a minimum deduction of 100 g/t Ag; refining charges estimated at $0.05/oz

The overall treatment charge will be $100 per dry tonne of concentrate.

The primary penalty relates to arsenic content, with the copper concentrate containing approximately 1.4% arsenic. The applicable schedule of penalties is: no charge for arsenic content of less than 0.2%, $2.50 per 0.1% for arsenic content of between 0.2 and 0.5%, $5.00 per 0.1% for arsenic content of between 0.5 and 1.0%, and $10.00 per 0.1% for arsenic content greater than 1.0%. Over the life of production, arsenic levels are not expected to exceed 2.1%.

19.1.2 Indicative commercial terms for the pyrite concentrates

The pyrite concentrates are expected to have a gold grade of approximately 45 g/t, a silver grade of approximately 206 g/t, and a copper grade of approximately 0.5%.

The marketing terms assume for the economic analysis were:

- Gold – Payability of 90.0%, with no minimum deduction, refining charges estimated at $6.40/oz
- Silver – Payability of 75.0%, with no minimum deduction; refining charges estimated at $0.68/oz
- Copper – No payability

In some markets, no penalties are applied; however certain smelters may impose a charge of $2.50 per 0.01% for concentrates with a silica content greater than 5%. The silica content of the pyrite concentrate is approximately 8.9%. Accordingly, the financial analysis assumes that a portion of the concentrate will be subject to this silica penalty.

19.1.3 Indicative commercial terms for the gravity recovered gold concentrates

The gravity recovered gold concentrates are expected to have a gold grade of approximately 5,307 g/t and will be produced in much smaller quantities than the pyrite and copper concentrates. They will be sold internationally and are not expected to incur any penalties.

19.2 Contracts

No material contracts are in place or under negotiation. The Project is planning to use contracting companies for the majority of the construction phase of the Project. National contractors will be used where possible for earth moving and basic construction works, and where more specialized skills are required for specific tasks, these will be contracted from the international equipment suppliers and other experienced companies.

During the operational phase of the Project, the large majority of the tasks will be performed by the national Project workforce after having completed sufficient training for their specific roles. More specialized roles will be contracted out, including raise boring and grouting activities. Many support functions such as catering and cleaning will be contracted out to local businesses.

While there has been no negotiation on supply contracts, it is expected that these will be secured for the main consumables of the Project, such as fuel, power, explosives, and cement, in order to guarantee a better price and supply.

19.3 Review and confirmation by the qualified person

The qualified person responsible for this section of the technical report has reviewed the indicative commercial terms and confirms that the results support the assumptions in the technical report.

20 Environmental studies, permitting, and social or community impact

20.1 Environmental setting, studies, and issues

20.1.1 Environmental setting

The Property is almost entirely located in a mountain ecosystem within a tropical humid climate zone, characterized by forests of medium sized trees of less than 20 m in height. Endemic and conservation status species identified in the Project area of influence include plant species of regional and/or national concern, and six native fish species in the Suratá River. The Project area of potential influence does not contain any sensitive or strategic ecosystems, but it does include priority conservation areas designated by the National Council for Economic and Social Policy. These areas show signs of vegetation degradation and transformation. The majority of the Project is located on land that shows pervasive anthropogenic alteration over time, and observations of wildlife and aviation fauna are almost non-existent. Upon closure, land reclamation and re-establishment of vegetation and soil profiles are not expected to be a challenge. Outside of the Project area, the most notable strategic ecosystem is the Santurbán páramo. Based on the current delimitation of the Santurbán páramo, the Project footprint is located 600 m horizontally from the Santurbán páramo and at an elevation approximately 350 m below it. The Colombian government is still in the process of redefining the Santurbán páramo boundaries, and while these distances may change, the Project is outside of the Santurbán páramo boundaries.

The Project's planned processing plant, filtered tailings facility, and associated water treatment facilities are located within the La Baja Creek catchment and adjacent minor tributaries of the Suratá River. Larger stream flows are sustained by groundwater discharge and are responsive to rainfall events. Groundwater in the area is complex and influenced by geology and structures. The regional gneiss has low bulk permeability and deep groundwater levels, often more than 100 m below the surface at topographical highs. Areas of higher permeability are present in fault structures breaking up the rock.

20.1.2 Environmental studies

Numerous environmental and social baseline studies and monitoring programs have been undertaken since 2013 and are ongoing as further information is required to support the Project development plans. These study campaigns include orthophotography to create a digital terrain model, topographic bathymetry to understand the dynamics of the main streams and rivers within the Project's study area for water quality campaigns and hydraulic monitoring, solute transport analysis as a water quality modelling tool, water quality monitoring of the main creeks and rivers within the study area, hydrological analyses to analyze variations in water flow over time, weather station monitoring, air quality monitoring, noise monitoring, soil mapping and geomechanical testing, and flora and fauna characterization.

The Project's attention to reducing the environmental effects of the proposed Project embodies local knowledge and the results of options analyses to find environmentally appropriate and cost-effective solutions to de-risking impacts of the Project.

20.1.3 Environmental management approach

The Project has developed a detailed environmental and social management program to guide the Project's activities during construction, operation, and closure, consistent with the ESIA process outcome. Each plan has a monitoring component and adaptive management process to evaluate the plan effectiveness and inform updates as required, and reporting requirements to regulators, communities, and stakeholders. The plan components are at various stages of development and will be validated by the community of the area of influence. The components cover topics including biotic and abiotic programs, socioeconomic programs, landscape management and control programs, and an emergency response and readiness plan.

Colombian law requires that a value of not less than 1% of the value of the Project's capital expenditure and associated development costs must be invested in environmental and/or sustainability related projects. The total capital expenditure associated with the development plan includes expenditures that exceed the 1% requirement. Funds from the 1% investment plan will be managed by the CDMB based on the programs jointly identified with PSN.

Contractors will receive training on the Project's various environment, community, and health and safety policies and standards and are required to adhere to environmental requirements as stipulated in their contracts.

20.1.4 Historical environmental liabilities

A mining concession holder is liable for environmental remediation and other penalties that may arise as a result of the concession holder's actions and/or omissions occurring after the date the concession contract is awarded. The Colombian legislation does not assign any environmental liability associated with historical artisanal or unauthorized workings to the concession holders, however the CDMB has followed up on the Project for the restoration of historic workings. PSN is and will continue to keep working with the regulatory authorities to remediate damage wherever possible. Water quality within the titles is monitored at points that include areas of historical process plants and artisanal mining tunnels. Historic mines have been sealed off as part of a mine closure program, and there is an ongoing program of disassembling process plants and removing contaminants left behind due to past mining and processing activities. These efforts will continue with the development of the Project.

20.2 Social setting and community requirements

20.2.1 Social setting

Mining was undertaken by the indigenous Sura people in the California – Vetas mining district in Pre-Columbian times and later by the Spanish who produced gold for two and a half centuries, as well as by English and French companies in the early 1800s and 1900s. Colombia continues to have an active artisanal and small-scale mining sector, with traditional miners across the country engaging in small-scale gold extraction, often in remote regions. This sector plays a significant role in local economies and provides livelihoods for many communities.

The municipalities closest to the Project include California, Suratá, and Matanza, which have a combined population of approximately 11,500. The economy of the province of Soto Norte is based on agriculture, mining, and forestry related activities. The economy of California, located closest to the Project, is dominated by artisanal and small scale mining, and the economies of Suratá and Matanza are dominated by ranching, agriculture, and forestry.

The Colombian Ministry of the Interior issued three certifications in 2016, 2017, and 2018 confirming that no Black, Afro-Colombian, Raizal, Palenquero, Indigenous, Roma, or minority communities are registered in the Project area. The certification will be renewed when the final Project footprint is confirmed and the EISA is ready to file.

20.2.2 Community engagement

The Project is proactively managing the intersection of communities and the Project elements through its community engagement model. In 2023, the Project launched a new strategy to implement best practices in community engagement across the Soto Norte region. The Project team and community leaders and authorities have collaborated to develop social agreements that enable the joint identification of needs and the evaluation of solutions in a coordinated and structured manner. This enables a shared understanding of the Project's role in driving sustainable development in the region and empowers the communities to decide on resource allocation and to propose projects, initiatives, or services to improve their quality of life. Each shared initiative is designed to prevent and mitigate potential disruptions to the Project's operations, facilitate the acquisition of the necessary permits for land access to ensure the completion of required studies, foster a favourable socio-political environment in the area of influence, supporting the Project's milestones, streamlining the permitting process, and to effectively address environmental challenges.

More than 500 residents from California, Suratá, and Matanza participated in two sessions per community in 2023 to select development projects by electronic vote to prioritize development, investments, and key projects in each community. The prioritized development areas, and confirmed again in 2024, include road and essential infrastructure, education, health, culture and well being, local economies, strengthening of community action boards, and project management. Seven working commissions in each of the communities were established for each of these priorities, comprised of community volunteers including residents, leaders, business owners, representatives of grassroots organizations, and municipal mayors or their delegates. These commissions analyze prioritized projects, define strategies, manage their execution, monitor investments, and keep the communities regularly informed on progress and outcomes.

The community widely regards the Project social agreement as a successful model, and is celebrated for its accomplishments in implementing projects and programs that have positively impacted local communities. The model has achieved community integration, effective collaboration between leaders and residents, community empowerment, and accountability for outcomes through collective efforts. The agreements play a vital role in strengthening relationships by encouraging their members to advocate for the Project within the broader community, and ensure a closer and more effective dialogue channel, enhance positive perceptions of the Project, and foster stakeholder trust, all of which are critical elements in de-risking social support for the Project and instilling confidence in the Project's success.

The strategic engagement model includes the social agreement commissions, a sponsorship plan, social houses, information and socialization forums to deliver timely and transparent information about the Project's operations and activities, and regional and national engagement beyond the area of influence. The sponsorship plan involves assigning Project social team members to specific villages and sectors within the communities of California and Suratá to maintain systematic engagement with families and local leaders, to foster a deeper understanding of the communities, address concerns, gather valuable ideas, and counter misinformation and mitigating factors that could impact social management and the Project's reputation. The social houses in California, Suratá, and Matanza have become central hubs for activities and meeting points, and serve as the primary platform for receiving community requests.

20.2.3 Community employment, diversity, and socioeconomic opportunities

Development of the Project involving the municipalities of California, Suratá, and Matanza will provide a diversity of employment and socioeconomic opportunities to residents. The Project will require skilled mine workers, services, material suppliers, contractors, housing, health, education, and skills training. Collaboration with contract mining partners is an integral component of the Project development. The direct income benefits of the Project will result in opportunities for indirect benefits such as support to local business, career opportunities for young adults, investment in non-mining related enterprises, and traditional agricultural, cultural, and artisan pursuits.

The Project currently employs 53 people, of whom 40 are from local communities and 25 are female. The peak workforce during Project construction is estimated at 2,292, mainly comprised of contractors, plus 90 administrative staff and management. During operations the workforce is estimated at 676 company personnel. The Project is targeting 60% of the workforce to be hired from the local community, 20% from the department of Santander, and 18% from other departments in Colombia. Foreign technical and managerial specialists will eventually make up 2% of the workforce. The socioeconomic benefits of the Project will also affect the broader region including the city of Bucaramanga for employees, suppliers, and contractors.

The Project has established social houses in the communities which are supporting the socio-economic, employment, training, and diversity aspects of the Project. The Project has contributed to community internet connectivity which will further create diverse economic opportunities for residents and enhance educational opportunities, and has invested in infrastructure and safety improvements on the access roads. The socioeconomic benefits of the Project will also affect the broader region including the city of Bucaramanga for employees, suppliers, and contractors.

20.2.4 Contract mining partners

Aris Mining collaborates with small-scale miners, known as contract mining partners, to create mutually beneficial partnerships that support the host communities. This partnership model includes the formation of formal companies that employ between 25 and 500 people as well as mill feed agreements such as those at Aris Mining's Segovia and Marmato mines that comprise long term contracts to supply mill feed for Aris Mining's processing plants, with payments based on gold content, grade, and the spot gold price. These agreements provide the contract mining partners with access to Aris Mining's technical, operational, and safety expertise as well as working capital financing. Aris Mining provides training programs in health and safety, environmental stewardship, accounting, compliance, and business management. Other benefits for the contract mining partners include access to social security and legal protections, government benefits, financial services, and broader market opportunities.

A suitable area within the Project titles was identified for the contract mining partners and in 2021 PSN entered into a four year subcontract with a group of small scale miners known as Calimineros S.A.S. to perform small mining activities covering 0.51 ha within title 125-68, which is located on the eastern boundary of the main 095-68 title. The Calimineros project and its mining plan were approved by the National Mining Authority, and Calimineros submitted a request to obtain an environmental license, and the approval process is pending. An extension of the subcontract was submitted to the National Mining Authority in the second quarter of 2025.

20.2.5 Land acquisition and resettlement

Additional land acquisition required for the construction and operation of the Project will result in resettlement for some members of the surrounding communities. Based on the Project footprint, 108 properties have been identified that will require either acquisition or an easement, provisionally affecting 198 households. Relocation areas to enable continuity of livelihoods are under evaluation. Where easement cannot be agreed upon, they may be imposed by judicial order. If purchase agreements cannot be mutually agreed, expropriation authorization must be obtained from the National Mining Agency prior to starting the expropriation proceedings. The preferred method for acquiring land at Soto Norte for project development will be to reach agreements with landowners following receipt of the environmental permits.

Resettlement has been identified as the most significant impact of the Project and therefore is a key focus of the management programs. A draft resettlement action plan (RAP) has been developed to guide the resettlement process. The RAP will be implemented in compliance with Colombian regulations, subject to approval by the environmental authority, will only commence following issuance of the Project’s environmental license.

To mitigate risks of delay to the construction schedule, the Project has developed strategies to work collaboratively with affected households during the land acquisition and resettlement process.

20.3 Filtered tailings facility and waste rock management

The planned filtered tailings facility is well-located, with minimal resident impacts and a well constrained catchment area where surface water flows are minimal. The environmental and socioeconomic studies of the areas proposed for these facilities will be concluded and the results will be used to inform the advancement of engineering design and environmental and social mitigations, including the conceptual closure and reclamation plans.

20.4 Site monitoring

The environmental and social management plan has a monitoring component and an adaptive management process to evaluate the plan effectiveness and to inform updates as required, as well as requirements for reporting to regulators, communities, and stakeholders. The monitoring components are in effect during construction, operation, closure, and post closure. The monitoring components of the management plans are at various stages of development and will be validated by the community of the area of influence.

20.5 Water management

Water resources are an important component of the Project development process and has been the subject of extensive and ongoing testwork and analyses, most recently by INERCO Consulting Colombia (INERCO) of Bogotá, Colombia in 2024, whose findings are summarized in this section. To maintain the quality and quantity of water, all impacts associated with the Project works and activities have been identified, assessed, and prioritized. As detailed in the Project’s environmental management plan, measures will be implemented to prevent, control, mitigate, correct, and/or compensate them during operations. The water management plan that will be described in the ESIA and the environmental management plan is coupled with and complies with the requirements of the Ministry of Environment and Sustainable Development. It focuses on compliance with Colombian environmental regulations and international best practices.

The Project has collected and analyzed extensive technical information associated with the Suratá Alto River basin, including among others the Vetas River and La Baja Creek, and related it to the infrastructure and Project’s activities, especially in the underground mine area. These studies include period flow accretion surveys along the main streams; inventories of water users and uses, wells, and water springs; baseline water quality analyses comprising a complete suite of analytical parameters including major ions, heavy metals, and nutrients, and field parameters including temperature, conductivity, and dissolved oxygen content; groundwater level and quality monitoring by means of piezometers, springs and existing limited underground mine workings; hydraulic testing of soil and sub-soil materials; and isotopic analysis on stream water and groundwater; among others. Surface water flow, groundwater flow, and solute transport models were used to characterize potential water resource impacts and to provide input into the Project infrastructure design and water management programs.

The Project gathered and validated meteorological and hydrological information to predict the behaviour of the basins through simulation programs, demonstrating that the groundwater abatement cone in the mine area will not affect the Santurbán páramo, nor will any other activities of the operation. Likewise, the groundwater in the Project area and its surroundings will not be affected in quantity or quality.

The impacts of the underground mine on the surroundings will have their respective water management programs, including the construction of infrastructure dedicated water protection and management, such as advanced cover drilling and a grout injection program to minimize water ingress into the mine and limit the spread of the drawdown cone; water extraction to dewater the mine's operating areas for treatment and reuse in the mine as required with the remainder returned to La Baja Creek according to the permitted discharge criteria and complying with all necessary environmental conditions; drainage and waterproof lining systems for the filtered tailings facility; drains, settlers, hydrocarbon traps, pumping systems, and other conduction lines; and groundwater monitoring systems.

20.5.1 Groundwater management

The Project has developed a comprehensive groundwater management and control program which includes a piezometer monitoring program. Groundwater inflows into the underground mine will be controlled and significantly reduced through a planned, systematic grouting program. Water entering the mine will be directed to a water treatment plant, treated as required, and discharged in compliance with applicable environmental regulations.

Groundwater modelling predicts only a limited zone of drawdown around the mine workings over the life of the operation due to the low permeability of the Bucaramanga Complex gneiss and the presence of low permeability fault zones in the vicinity of the mine footprint.

Flow in the La Baja Creek will be monitored in accordance with standard industry practices and mitigation measures will be developed with the environmental regulator to address potential risks to aquatic ecosystems reliant on the La Baja Creek, particularly during dry seasonal conditions when natural flows may be reduced.

The potential impact of mine dewatering on the ecologically sensitive vegetation of the Santurbán páramo has been investigated in studies and determined to be negligible, most recently confirmed by INERCO in 2024. The Project studies, supported by extensive data and advanced simulation programs, enable accurate predictions of water behavior. These simulations demonstrate that the groundwater drawdown cone from the mining operation will remain confined to the mine area and does not extent to the Santurbán páramo, a critical high-altitude ecosystem. Additionally, INERCO confirmed that no other Project related activities will affect the Santurbán páramo.

Potential risks to community water supplies have been evaluated. Communities within the Project's area of influence, including California, are not expected to be adversely impacted by the mine operations. The Project is committed to ensuring continuity of community water supplies and will provide mitigation for any un-forecasted changes to the availability of groundwater or stream flow.

20.5.2 Extractive wastewater management

Geochemical studies have been completed to develop an understanding of the weathering behaviour of the mine waste, including dry filtered tailings and waste rock, as well as exposed materials in the underground mine, to determine whether contact waters could present a risk to the environment through acid rock drainage and/or metal leaching (ARDML) during operations, closure, and post-closure.

Based on preliminary modelling of the kinetic testing data, the dry filtered tailings will not generate acid during operations if the surface is continually renewed by the deposition of fresh, unoxidized dry filtered tailings or closure material. Any potentially acid generating (PAG) waste rock removed from the mine will be co-disposed in the filtered tailings facility with dry filtered tailings, encapsulated and compacted with non-acid generating material, and covered with low permeability cover, limiting oxidation and release of acid seepage water. The filtered tailings facility seepage modelling is continuing, and it is anticipated that over time, seepage water quality will improve to the point where water treatment is no longer required.

The predicted composition of the underground contact water is expected to remain mildly basic at around pH 8, with metal concentrations below the mine water effluent standards. Where concentrations of any parameters exceed effluent standards, then physical controls will be implemented, such as optimizing the management of waste rock backfill or water treatment. Explosives used underground may result in residues with slightly elevated ammonia and nitrate concentrations, which will be treated prior to discharge under compliance with all of the necessary environmental conditions.

20.5.3 Water supply management

The water supply management system is designed to comply with the terms of reference for mining projects of the Ministry of Environment and Sustainable Development and focuses on compliance with the Colombian environmental regulations as well as alignment with international best practice.

The Project has been designed with robust water management and protection as guiding principles. There is sufficient and readily available water within the Project area for all aspects of the operation including mining, processing, the camp, and other activities. Construction water will be sourced under permits from the La Baja Creek and Suratá River, which both have abundant water supply. During operation, the main source of water for the underground mine and its associated infrastructure will be from groundwater and the main source of water for processing will be from the adjacent Suratá River.

The Project plans to reuse, recycle, and return approximately 96.5% of the Project’s total water requirements back to the environment, resulting in a net water use estimated at 3.5%. The majority of the demand will be supplied by the Suratá River, which will supply approximately 2.8 litres per second of net make up water for the processing plant, representing 0.22% of the average flow of the Suratá River at the planned water access point near the village of Suratá, and 0.08% of the average flow of the Suratá River at Bucaramanga, the nearest city located 55 km downstream from the Project.

The Project includes a comprehensive water management plan that incorporates plans and programs tailored to each water application or activity with significant interaction with water to ensure that both the quality and quantity of local water resources are preserved throughout the life of the operation, and all potential impacts related to water use have been thoroughly identified, assessed, and addressed through prevention, mitigation, and compensation measures. The underground mine has been designed to minimize groundwater flows into the underground workings through advanced cover drilling and grouting ahead of mine development to identify and seal any water bearing structures before mining reaches them, greatly reducing potential inflow, and to manage, treat, and if required, safely return any captured water to the environment in compliance with the environmental standards and discharge permits. The process plant has been designed to minimize the use of water and to recycle both process water and the water obtained from the dewatering of the process tailings. The filtered tailings storage facility has been designed to divert rainwater runoff away from the facility and to drain and manage any subsequent dewatering of the tailings and any rainwater falling onto the facility. Continuous monitoring of water flow and quality will be undertaken with defined measures planned if any changes are detected. These strategies will help ensure that the current water flow rates within the ecosystem are maintained. Water flows will be monitored at various stations to validate the flow predictions.

During closure and post-closure, the water balance indicates a reduction in water supply needs, which can be met internally.

20.6 Permitting requirements

The Project holds a mining license for mining title 095-68, originally granted by the ANM in 2018 and amended in 2021. The title covers the mineral resources and reserves, the planned underground mining infrastructure, and the surface infrastructure in the mine area.

For the current stage of activities, the Project holds licenses authorizing land occupancy, water use for drilling, potable water usage, and water treatment and discharge.

The principal approvals required to commence construction and operations are:

- approval of the ESIA and issuance of the environmental license;

- completion of resettlement obligations as stipulated in the environmental license; and
- amendment and approval of the existing PTO to incorporate the conditions of the environmental license.

PSN will submit an updated ESIA to the regional environmental authority, CDMB, outlining the Project's description as contemplated in the current prefeasibility study. This submission will include additional studies and re-evaluation of environmental and social impacts, thereby restarting the environmental permitting process and associated timeframes.

Once issued, the environmental license is valid for the life of the Project, subject to compliance audits by the environmental authority. The license may be modified as the Project evolves.

There are currently no requirements to post performance or reclamation bonds.

20.7 Mine closure requirements

The Project closure plan outlines the technical, environmental, and social criteria and guidelines that the Project must consider for the closure of the Project components and facilities. The mine closure plan includes end land use objectives, strategies for achieving the objectives, and criteria for measuring successful closure. The mine closure objective is to leave the area in a condition similar to, or improved from, that which existed before the start of mining activity, in order to protect the environment and the health and collective rights of the communities. Community and infrastructure investments over the life of mine will remain as positive social legacies of mining. The mine closure plan development will be consistent with the International Council on Mining and Metal's guidance for involvement of communities and interests affected by mining.

Progressive closure and reclamation of infrastructure during the life of the Project may be undertaken as opportunities arise, and will address all environmental and social aspects and the environmental management plan measures, including soils, water resources, biota, and social management and communications. A temporary closure plan will also be presented in the event the Project has to temporarily stop its operations due to unforeseen circumstances, force majeure, or transitory circumstances of a technical or economic nature.

The final closure plan will include the closure of the Project and facilities that have been used since the beginning of operations, including rehabilitation and/or recovery of areas affected by mine development; dismantling and removal of equipment, materials, and facilities that if not sold, will be treated and disposed of as waste, recycled, or re-purposed; demolition of remaining structures after dismantling, aside from those that contribute to the physical stability of the environment; and maintaining or restoring physical and chemical stability of the site to stable conditions.

The components of the final conceptual mine closure plan include closure and land reclamation of the mining area; dismantling of the processing plant and associated structures and reclamation of the processing footprint; closure of the mine access tunnel and operations terraces, the process plant area, and waste disposal areas; and discussions and consensus agreement between the Project and the municipal mayors and communities in the area of influence to define the possible future use of roads by residents.

Socioeconomic actions in relation to final mine closure will include discussion and consensus agreement with the communities and local authorities to characterize the social situation in the area of influence at the start of closure and conduct a comparative analysis in relation to the start of the Project. An action plan will be established to identify any pending issues that need to be addressed in compliance with the environmental management plan, and develop agreements considering the areas used by the Project for municipal use and those that are privately owned properties with easements.

The post closure plans consider the activities the Project must develop to ensure the physical and chemical stability of the remaining facilities and avoid risks to the environment and safety of people. The post closure plans will be focussed on the maintenance of control structures, the treatment of water from the filtered tailings facility, and monitoring to evaluate the measures. Post closure water treatment costs will be informed by further detailed water quality modelling and updated on a five year cycle.

The closure costs submitted with the new ESIA will meet the International Finance Corporation Performance Standards. The financial model includes closure and post-closure costs to be incurred at the end of the mine life.

20.8 Recommendations

The qualified person responsible for this section of the technical report makes the following recommendations:

- Complete environmental and social baseline surveys, including along the rope conveyor, tailing pipeline route, and the filtered tailings storage watershed, and confirm the absence of fish and amphibians in the filtered tailings facility footprint.
- Conceptualize a reclamation research program to support future progressive and final reclamation and revegetation programs, including collaborative partnering with local greenhouse owners to develop native seedling sources for revegetation.
- Continue efforts to advance contract mining partner agreements and action plans for implementation.
- Install a large Project area poster in each social house to serve as a focal point for discussions with members as they share their issues and experiences.

This work is covered in the ongoing environmental permitting work and has been accounted for within the existing social and environmental budgets.

21 Capital and operating costs

21.1 Capital cost estimate methodology

The estimate of processing and surface infrastructure capital and operating costs was prepared by LogiProc, and the estimate of underground mining capital and operating costs was prepared by Mining Plus. These estimates were utilized to complete the economic analysis.

21.1.1 Processing facilities and surface infrastructure capital costs

The capital cost estimate for the processing facilities and surface infrastructure has been prepared to an accuracy of +/-25%, suitable for the requirements of a prefeasibility study level. The estimates were developed in accordance with the American Association of Cost Engineers Class 3 level, with an expected accuracy of -10% to -20% on the low side and +10% to +30% on the high side. All costs are expressed in Q1 2025 U.S. dollars, using an exchange rate of 1.00 U.S. dollar to 4,200 Colombian pesos. No allowances have been made for future escalation from the base date or changes in the currency exchange rates.

Each element of the estimate was initially developed as a base cost, to which contingency was applied to address anticipated variances between the items in the estimate and the final actual costs. Contingency was assessed as a percentage of direct and indirect costs to reflect potential variances between estimated and actual costs, consistent with the level of confidence required for a prefeasibility study.

Direct costs are mostly quantity based and include all equipment, labour, and materials associated with the physical construction of the permanent equipment. These costs include the process plant, surface infrastructure, utilities, buildings, permanent equipment, mobile equipment, bulk materials, and freight and transport fees. They also include direct and indirect installation costs including direct and indirect labour, construction equipment, small tools and consumables, and contractor temporary facilities, services, and utilities. The estimate allows for the supply the first fills which includes lubrication, oil for the equipment, commissioning spares, and first year spares for the plant. Direct labour rates were derived from current Project and other recent study data, and reflect a workforce composition of local, regional, and Colombian nationals. The total labour hours were derived from first principles and were used to estimate construction workforce levels and the construction camp size.

Indirect costs are the costs associated with the development of the Project that are not related to the permanent facilities, including EPCM services, consultants, temporary construction facilities, catering and lodging, services and utilities, third party engineering services, Owner’s costs, pre-commissioning, and manuals. Contractor indirect costs were derived from first principles and benchmarked against international databases.

Owner’s costs include but are not limited to permitting and main power supply.

Other costs include, but are not limited to, land acquisition, all costs associated with environmental studies, assessment, compensatory measures and remediation of any environmental liabilities, social costs associated with community engagement activities, resettlement costs, closure costs, and Project G&A incurred during the pre-production period, such as health, safety, security, mine site administration, and other shared services and administration.

VAT and import duties were estimated and included in the initial capital expenditure estimate. VAT incurred during the construction period is capitalized and amortized over the applicable depreciation period for tax purposes.

The Project assumes an EPCM approach with Project delivery services estimated based on the total direct costs excluding the mining scope of work. The EPCM cost estimate includes categories for commissioning, engineering and procurement, and construction management. Commissioning and engineering are each 4% and construction management is 8% of the total direct costs for mining, processing, dewatering, and infrastructure utilities and facilities. EPCM costs are expected to include services such as project management, engineering and design, project administration, project document control, cost control, planning, estimating, and scheduling, procurement and logistics, and construction and commissioning management.

Construction rates were obtained by requesting rates from construction companies and benchmarked utilizing global indices data. Contract prices were developed based on the EPCM strategy. Budget quotations were received for all major process plant equipment and some minor equipment, with estimated pricing for other minor items based on recent or current projects and escalated to the base date of the estimate. Indirect costs were developed through factorization and utilizing historical data.

21.1.2 Underground mining infrastructure capital costs

The capital cost estimate for the underground mining infrastructure has an accuracy of +/-25%, suitable for the requirements of a prefeasibility study level. The estimates were developed in accordance with the American Association of Cost Engineers Class 4 level, with an expected accuracy of -15% to -30% on the low side and +20% to +50% on the high side. The estimates are considered to be within the lower ends of these ranges as recent quotes were used for all equipment purchases and several other inputs. All costs are stated in Q2 2025 U.S. dollars, using an exchange rate of 1 U.S. dollar to 4,200 Colombian pesos. No allowance was made in the estimate for escalation from the base date or changes in the currency exchange rates.

Direct costs are considered to be the cost of permanent equipment, materials, and labour associated with the physical construction of the permanent facilities, including direct and indirect installation costs. Freight and transport fees were included as percentages of the budget equipment quote. The estimate allows for the supply of first fills. Direct labour rates were derived from the current Project and other recent study data.

For depreciation purposes, three classes have been established, including 10% for equipment, life of mine using a unit of production method, and 2.5% for other items.

21.1.3 Capital cost summary

A summary of the estimated initial capital expenditures, including any operating costs incurred during the pre-production period, are shown in Table 21-1, the estimated deferred and sustaining capital costs are shown in Table 21-2, the annual estimated initial capital expenditure schedule is shown in Table 21-3, broken down by quarter during the construction period, and the estimated capital expenditure per depreciation class is shown in Table 21-4.

The final closure plan will address the dismantling, disposal, and rehabilitation of all Project facilities and affected areas, ensuring physical and chemical stability is restored across the site. The financial model includes capital cost expenditures related to closure and post-closure costs to be incurred at the end of the mine life.

Table 21-1 Estimated initial capital costs

Initial capital expenditure	Amount ($M)
Mining	
Mobile equipment	24.2
Fixed equipment	10.3
Lateral development	10.4
Vertical development	0.8
Pre-production operating costs and inventory	9.8
Mining total	**55.5**
Surface	
Infrastructure	172.4
Rope conveyor	74.8
Process plant	95.7
Owner's, indirects, and first fills	21.5
EPCM	35.4
Replacement costs	-
Surface total	**399.8**
Other costs	

Initial capital expenditure	Amount ($M)
Resettlement and environmental monitoring	72.8
Electricity supply down payment	0.2
Other start-up costs	8.8
Capitalized VAT	34.2
Contingency	54.0
Other costs total	**170.0**
Total	**625.2**

Table 21-2 Estimated deferred and sustaining capital costs

Deferred and sustaining capital expenditure	Amount ($M)
Mobile equipment	138.0
Fixed equipment	68.1
Lateral development	117.5
Vertical development	31.2
EPCM	3.6
Replacement costs	4.2
Other costs	1.1
Total	**363.6**

Table 21-3 Estimated annual capital expenditure schedule

	Units	Total	-Y4 Q4	-Y3 Q1	-Y3 Q2	-Y3 Q3	-Y3 Q4	-Y2 Q1	-Y2 Q2	-Y2 Q3	-Y2 Q4	-Y1 Q1	-Y1 Q2	-Y1 Q3	-Y1 Q4
Mining	$M	45.7	-	-	-	-	-	-	-	-	-	-	-	37.0	8.7
Surface	$M	399.8	33.6	39.3	46.1	50.5	35.1	22.1	23.0	33.5	31.6	31.8	30.7	16.9	5.5
Other	$M	170.0	17.5	17.9	19.2	20.2	18.2	7.9	7.6	9.2	8.7	8.7	8.9	15.7	10.4
Preproduction operating costs and first fills	$M	9.8	-	-	-	-	-	-	-	-	-	-	4.9	1.6	3.3
Initial total	$M	625.2	51.1	57.1	65.3	70.7	53.3	29.9	30.6	42.8	40.3	40.5	44.6	71.2	27.9

Table 21-4 Estimated capital expenditure per depreciation class

Depreciation class	Capital expenditure ($M)	Annual depreciation rate (%)
Equipment	411	10.00
Life of mine	209	Variable based on unit of production
Other items	393	2.50
Total	1,014	

21.2 Operating cost estimate

21.2.1 Estimate methodology for processing and rope conveyor operating costs

The processing operating costs assume a processing rate of 2,750 tpd for owner mining. The cost estimates use 2025 U.S. dollar estimates. Reagent consumptions were obtained directly from the testwork and process design criteria. Costs for identified items were provided by vendors. Where vendor pricing was unavailable, alternative costing sources were obtained either from LogiProc's internal costing database or from the Project. The onsite labour force structure considered the process operations and maintenance, weighed against similar operations of a similar size and structure. Annual power costs were determined using actual rates from other in country operations.

Variable costs were calculated at the steady throughput rate and include processing plant and rope conveyor power, reagents, plant and rope conveyor maintenance, consumables, and replacement. Fixed costs include plant and rope conveyor labour.

21.2.2 Estimate methodology for underground mining operating costs

The underground mine operating cost estimates use 2025 U.S. dollar estimates as the basis and have been primarily developed based on first principles modelling, and where possible, quotes were sourced for the supply of equipment and consumables. Preference was given to local suppliers for quotations of a similar quality.

The most significant variable costs directly related to the operation are calculated based on the mine and process plant schedules, equipment hours determined through internationally accepted maintenance and availability assumptions, operating consumables using test work data and original equipment manufacturer reference information, productivity rates, and quotations obtained from selected vendors. Equipment maintenance and replacement costs were based on hourly rates, operating and maintenance labour, replacement parts, power, consumables, and other associated services developed from original equipment manufacturer tenders for the supply and maintenance of equipment. The consumption of power, diesel, lubricant, tires, tracks, and ground support costs were based on equipment hours determined using industry standard equipment efficiencies, usage, and production rates.

The estimated mine operating costs include all direct charges attributable to the underground operation for mining both ore and waste rock materials, and the subsequent backfilling of underground voids. The mining activities and corresponding cost estimates comprise mine equipment and labour to support development, production, and materials handling; drilling and grouting operations to control ground water inflows; mine services including but not limited to dewatering, ventilation, and grade control; drilling and blasting costs associated with the mining of ore and development waste; backfilling operations; and the delivery of ore and waste to the underground primary crushing station and underground crushing and transport to the surface rope conveyor.

21.2.3 Estimate methodology for G&A operating costs

G&A expenses were estimated by categories including mine site, corporate, and general other. Mine site G&A costs relate to on site expenses including insurance taxes, software and technology, consultancy and services, travel and transport, leases, and labour, recruitment, and variable compensation. Corporate G&A costs relate to allocated overheads related to off site expenses including consultants, insurance, travel, agreements, training and recruitment, taxes and fees, memberships and associations, statutory and government fees, media plans and events, stakeholder engagement, leases, software and hardware, stationary, utilities, materials, and other services. General other operating costs include security, mine site administration, and other shared services and administration.

Other operating cost categories include the monitoring and management plans related to the environmental management plan, marketing and logistics, and royalties.

21.2.4 Operating cost summary

The life of mine operating costs, excluding capitalized operating costs, were estimated for underground mining, surface including processing and the rope conveyor, G&A including other costs, realization, and royalties. Royalties due to the Colombian state include a 4% royalty on 80% of the gold and silver produced and a 5% royalty for copper on 100% of the copper produced.

The summary of the estimated life of mine operating costs is shown in Table 21-5, the estimated life of mine unit operating cost estimate is shown in Table 21-6, and the estimated annual life of mine operating cost schedule is shown in Table 21-7.

Table 21-5 Estimated operating costs

Item	Total life of mine ($M)	Pre-production ($M)	Production ($M)	Post-production ($M)
Mining				
G&A	9.3	0.1	9.2	-
Contracts	21.6	0.4	21.1	-
Mine labour	62.8	0.4	62.4	-
Equipment maintenance and operation	236.2	1.4	234.8	-
Power	125.9	0.3	125.7	-
Diesel	36.5	0.2	36.2	-
Explosives	32.8	0.1	32.7	-
Ground support	266.6	1.4	265.2	-
Drilling consumables	48.0	0.3	47.6	-
Mine services	25.1	0.1	25.0	-
Pre-production inventory	4.9	4.9	-	-
Mining total	**869.6**	**9.8**	**859.8**	**-**
Processing and surface				
Labour	94.8	-	94.8	-
Reagents	47.3	-	47.3	-
Power	214.8	-	214.8	-
Plant maintenance	49.3	-	49.3	-
Rope conveyor	10.1	-	10.1	-
Plant consumables	8.3	-	8.3	-
Processing and surface total	**424.7**	**-**	**424.7**	**-**
Realization				
Treatment charges	17.0	-	17.0	-
Refining charges	47.5	-	47.5	-
Penalties	81.8	-	81.8	-
Freight	244.0	-	244.0	-
Realization total	390.3	-	390.3	-
Mine site G&A	**395.1**	**8.8**	**386.3**	-
Environmental management plan	**69.8**	**18.1**	**42.2**	**9.6**
Royalties	**393.3**	**-**	**393.3**	-
Total	**2,542.8**	**36.7**	**2,496.5**	**9.6**

Table 21-6 Estimated life of mine unit operating costs

Unit operating costs	Life of mine $/t ore
Mining	41.70
Surface	20.59
Other (G&A, environmental management plan, etc)	21.24
Treatment, refining, and shipping	18.93
Royalties	19.07
Total	**121.5**

Table 21-7 Estimated annual life of mine operating cost schedule

	Units	Total	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27
Mining	$M	**859.8**	28.4	38.1	41.8	49.5	52.6	50.1	47.7	44.0	42.0	39.7	41.5	37.8	39.3	36.1	37.7	36.4	36.6	37.7	38.0	38.0	35.1	11.8	-	-	-	-	-
Surface	$M	**424.7**	12.2	18.3	20.7	20.7	20.7	20.6	20.7	20.6	20.6	20.7	20.7	20.7	20.7	20.7	20.7	20.7	20.7	20.7	20.7	20.7	18.4	3.8	-	-	-	-	-
Other (G&A and environmental management plan)	$M	**438.1**	19.5	19.5	19.5	19.5	19.5	19.5	19.5	19.5	19.5	19.5	19.5	19.5	19.5	19.5	19.5	19.5	19.5	19.5	19.5	19.5	19.5	19.5	1.9	1.9	1.9	1.9	1.9
Treatment, refining and shipping	$M	**390.3**	11.8	17.1	19.8	19.4	19.4	19.8	19.7	19.7	20.2	19.6	18.9	18.5	17.7	18.0	17.8	18.9	18.6	18.5	19.4	17.8	16.4	3.4	-	-	-	-	-
Royalties	$M	**393.3**	15.6	19.9	26.6	23.5	24.0	24.4	23.8	22.9	27.6	21.1	19.3	15.6	13.0	13.3	14.3	14.9	13.9	12.6	15.9	14.1	14.3	2.8	-	-	-	-	-
Total operating costs	$M	**2,506.1**	87.6	112.9	128.4	132.6	136.2	134.4	131.3	126.7	129.9	120.5	119.8	112.0	110.1	107.6	110.0	110.3	109.2	108.9	113.3	110.1	103.7	41.2	1.9	1.9	1.9	1.9	1.9

22 Economic analysis

22.1 Estimate methodology

This economic analysis was undertaken to assess and confirm the current mineral reserve estimate disclosed in this technical report, utilizing the production schedule and the capital and operating cost estimates. The economic analysis has been conducted on a post-tax, 100% equity (i.e., no debt financing) basis, in constant dollar terms. Sunk costs, such as exploration and the cost of previous studies, were excluded from the analysis.

The economic viability of the mineral reserves has been evaluated using key economic indicators, including annual and cumulative cash flows, NPV, and IRR. The NPV presented in this technical report should not be interpreted as the definitive value of the Project and must be considered in conjunction with the accompanying sensitivity analysis.

The key economic results are presented on a pre-tax basis to facilitate comparison with other projects in different jurisdictions by removing the effect of local tax regimes, and on an after-tax basis incorporating the applicable tax rates and fiscal terms for the Project, providing a more accurate reflection of the potential economic benefit to the Project owners.

22.2 Project schedule

The processing facility has been designed with a 3,500 tpd capacity, but the Project underground mining production schedule has been constrained to 2,750 tpd with the additional 750 tpd dedicated for potential mill feed purchases from contract mining partners. The financial projections in this technical report do not account for revenue, operating costs, or profit margins from the dedicated capacity.

The construction period is scheduled for 13 quarters (3.25 years). The first underground ore production from development activities is planned during the final six months of Project construction, with all material stockpiled until the process plant is commissioned. The pre-production stockpile is scheduled to supplement run of mine ore feed during the first six months of Year 1, supporting the production ramp up.

Plant throughput is expected to ramp up progressively to the 2,750 tpd capacity, reaching steady state operations toward the end of Year 2. Based on the current mineral reserve estimate, the mine life extends to Year 22.

The total life of mine production is shown in Table 22-1, the total concentrate production for the life of mine is shown in Table 22-2, the annual tonnes mined over the life of mine are shown in Figure 22-1, the annual processed grade and gold produced in concentrates are shown in Figure 22-2, and the annual life of mine mining and processing schedule is shown in Table 22-3, where Year -1 includes the initial underground ore production from development activities planned during the final six months of construction.

Table 22-1 Total life of mine production

Mining	Units	Total
Waste	kt	5,380
Development ore	kt	3,501
Stope ore	kt	17,119
Total material mined	kt	26,000
Mined gold grade	g/t Au	6.98
Mined silver grade	g/t Ag	32.0
Mined copper grade	% Cu	0.20
Contained mined gold	koz	4,627
Contained mined silver	koz	21,216
Contained mined copper	Mlb	90.6

Table 22-2 Total life of mine concentrate production

Concentrates	Units	Copper concentrates	Pyrite concentrates	Gravity gold concentrates	Total
Mass	DMT	169,184	1,441,402	615	1,611,201
Contained metal in concentrates					
Gold	koz	2,092	2,103	105	4,299
Silver	koz	9,305	9,529	1	18,834
Copper	Mlb	67.4	16.6	Nil	84.0
Concentrate grade					
Gold	g/t	385	45	5,307	
Silver	g/t	1,711	206	32	
Copper	%	18	0.52	Nil	

Figure 22-1 Annual tonnes mined over the life of mine – Source Mining Plus 2025

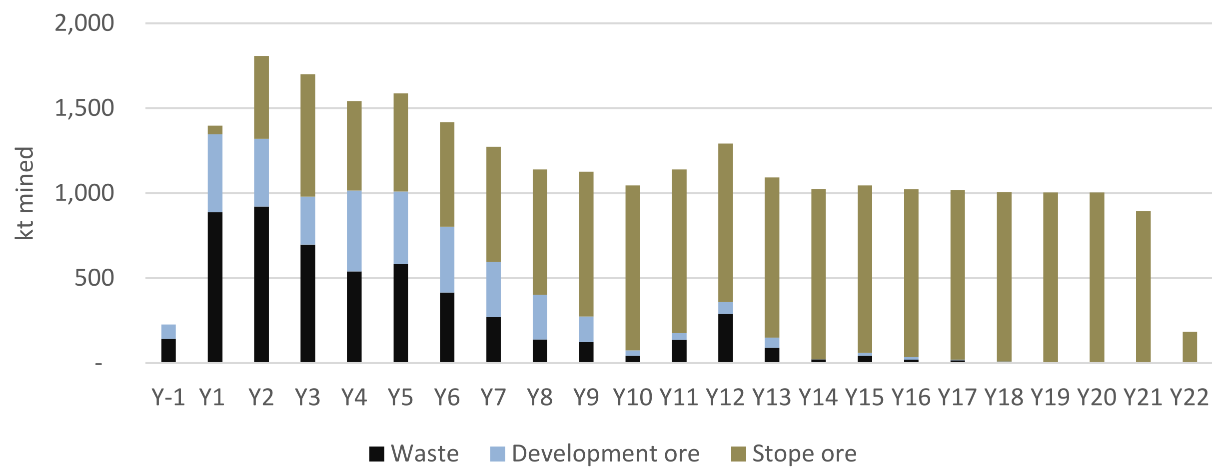


Figure 22-2 Annual processed grade and gold produced in concentrates – Source Mining Plus 2025

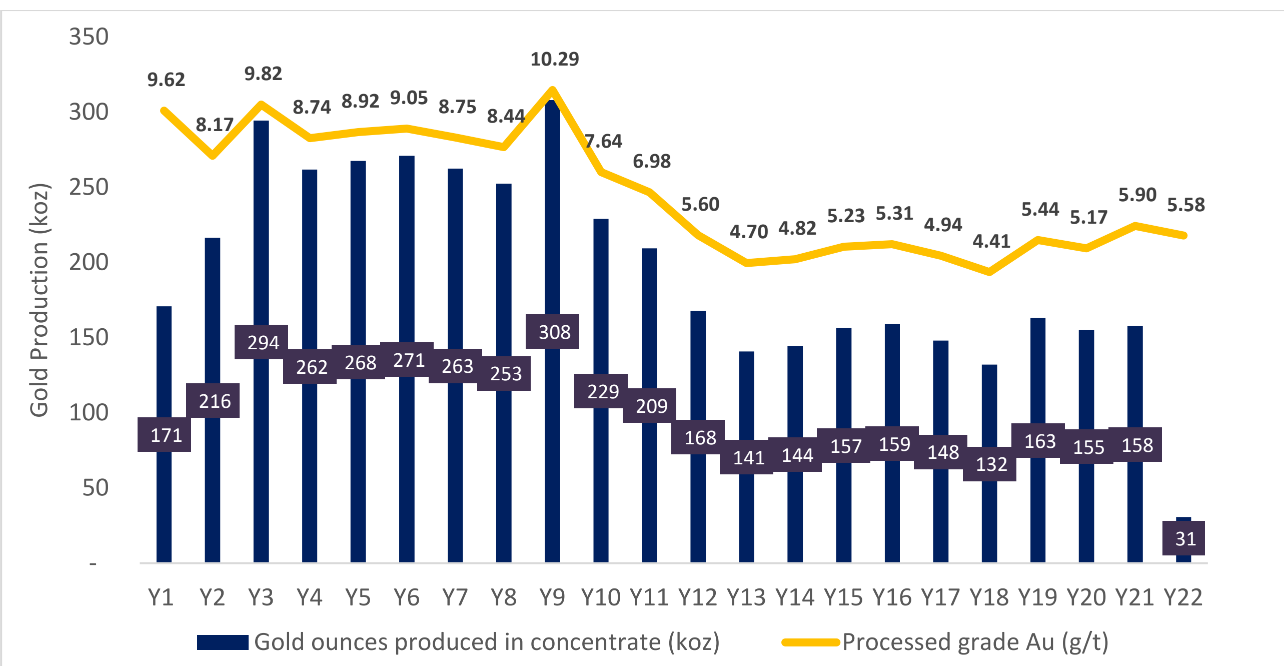


Table 22-3 Annual mining and processing schedule

	Units	LOM	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Y20	Y21	Y22
Mining																									
Waste	kt	5,380	143	888	921	698	540	583	415	270	139	125	42	136	290	89	22	42	20	16	3	-	-	-	-
Development ore	kt	3,501	85	459	400	283	476	427	388	325	263	149	33	41	68	62	0	18	15	4	6	-	-	-	-
Stope ore	kt	17,119	-	50	488	721	527	576	615	678	738	853	970	963	935	942	1,003	986	989	1,000	998	1,004	1,004	895	184
Total mined	kt	26,000	228	1,397	1,808	1,701	1,543	1,586	1,418	1,274	1,140	1,127	1,045	1,140	1,293	1,092	1,026	1,045	1,024	1,019	1,007	1,004	1,004	895	184
Ore grade																									
Gold	g/t	6.98	8.54	9.80	8.17	9.82	8.74	8.92	9.05	8.75	8.44	10.29	7.64	6.98	5.60	4.70	4.82	5.23	5.31	4.94	4.41	5.44	5.17	5.90	5.58
Silver	g/t	32	44	52	48	47	34	32	33	35	36	38	34	31	30	22	22	25	29	28	29	39	20	21	22
Copper	%	0.20	0.32	0.21	0.20	0.20	0.20	0.20	0.22	0.22	0.22	0.21	0.25	0.23	0.19	0.15	0.16	0.15	0.20	0.18	0.19	0.28	0.16	0.17	0.14
Processing																									
Mill throughput	kt	20,620	-	594	887	1,004	1,003	1,004	1,003	1,004	1,001	1,002	1,003	1,004	1,003	1,003	1,003	1,003	1,004	1,004	1,004	1,004	1,004	895	184
Feed grade																									
Gold	g/t	6.98	-	9.6	8.2	9.8	8.7	8.9	9.0	8.8	8.4	10.3	7.6	7.0	5.6	4.7	4.8	5.2	5.3	4.9	4.4	5.4	5.2	5.9	5.6
Silver	g/t	32.0	-	50.7	72.2	79.0	56.7	53.5	55.5	59.9	60.0	63.8	57.2	52.6	50.4	37.7	37.1	42.2	48.4	47.4	48.4	65.7	33.7	30.9	6.8
Copper	%	0.20	-	0.23	0.20	0.20	0.20	0.20	0.22	0.22	0.22	0.21	0.25	0.23	0.19	0.15	0.16	0.15	0.20	0.18	0.19	0.28	0.16	0.17	0.14
Copper concentrate (recovered metal to concentrate)																									
Gold	koz	2,092	-	83	105	143	127	130	132	128	123	150	111	102	82	69	70	76	77	72	64	79	75	77	15
Silver	koz	9,305	-	425	606	663	475	449	465	502	503	535	479	441	422	316	311	354	406	397	406	551	282	259	57
Copper	Mlb	67	-	2	3	3	3	3	4	4	4	3	4	4	3	2	3	2	3	3	3	5	3	3	0
Pyrite concentrate (recovered metal to concentrate)																									
Gold	koz	2,103	-	84	106	144	128	131	133	128	124	151	112	102	82	69	71	77	78	72	65	80	76	77	15
Silver	koz	9,529	-	435	620	678	487	460	477	514	515	548	491	452	433	323	319	362	416	407	416	564	289	265	58
Copper	Mlb	17	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0
Gravity gold concentrate (recovered metal to concentrate)																									
Gold	koz	105	-	4	5	7	6	7	7	6	6	8	6	5	4	3	4	4	4	4	3	4	4	4	1
Silver	koz	1	-	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total metals in concentrates																									
Gold	koz	4,299	-	171	216	294	262	268	271	263	253	308	229	209	168	141	144	157	159	148	132	163	155	158	31
Silver	koz	18,834	-	860	1,226	1,341	962	908	942	1,016	1,019	1,083	970	893	855	639	630	716	822	804	822	1,115	571	524	115
Copper	Mlb	84	-	3	4	4	4	4	5	4	5	4	5	5	4	3	3	3	4	4	4	6	3	3	1

22.3 Taxes and royalties

The economic analysis incorporates a statutory corporate income tax rate of 35%. Depreciation rates are as described in Section 21.2, and royalties are deductible from taxable income.

Under Colombian law, the Project is required to pay the following royalties on gross metal value to the State:

- Gold: 4% royalty on 80% of the contained gold in concentrate
- Silver: 4% royalty on 80% of the contained silver in concentrate
- Copper: 5% royalty on 100% of the contained copper in concentrate

22.4 Marketing assumptions

The marketing terms for the concentrate sales were established based on indicative offers received from multiple metal traders and market participants, each providing slightly different terms. The resulting life of mine average terms are summarized in Table 22-4.

Grade deductions and payability are reflected in gross revenue, while treatment and refining charges, penalties, and freight from mine site to the final international smelter are reported as operating costs.

For revenue recognition in the economic analysis, payment terms were assumed at 10 days post-bill at the port of Cartagena, with final settlement based on smelter assays.

Table 22-4 Life of mine average concentrate marketing terms

Marketing terms	Unit	Copper concentrate	Pyrite concentrate	Gravity gold concentrate
Applied payability				
Gold	%	96	90	96
Silver	%	96	75	96
Copper	%	96.5	0	0
Treatment and refining charges, penalties, and shipping				
Treatment charges	$/DMT	100	0	100
Gold refining charge	$/oz	10	6.4	10
Silver refining charge	$/oz	0.5	0.68	0.5
Copper refining	$/lb	0.08	0	0.08
Penalties	$/DMT	70	49	0
Shipping	$/WMT	154	137	154

22.5 Commodity prices and gross revenue

The financial analysis utilized the following metal price assumptions for the base case:

- Gold: $2,600/oz
- Silver: $29/oz
- Copper: $4.30/lb

These metal prices were selected as being in line with the median of the long term forecasts of a group of banks and financial institutions, as at the end of August 2025.

The annual payable metals in concentrate, the projected payable gold revenue, silver and copper by-product credits, and treatment and refining charges, penalties, and shipping costs are shown in Table 22-5.

Table 22-5 Annual payable metals, gold revenue, by-product credits, and treatment and refining charges, penalties, and shipping costs

	Units	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Y20	Y21	Y22
Copper concentrate (payable metals)																								
Gold	koz	2,008	80	101	138	122	125	127	123	118	144	107	98	78	66	67	73	74	69	62	76	72	74	14
Silver	koz	8,758	407	580	635	448	421	438	475	476	508	452	414	395	293	287	332	378	370	379	523	259	235	53
Copper	Mlb	63	2	3	3	3	3	3	3	3	3	4	4	3	2	2	2	3	3	3	4	2	2	0
Pyrite concentrate (payable metals)																								
Gold	koz	1,893	75	95	130	115	118	119	116	111	136	101	92	74	62	64	69	70	65	58	72	68	69	13
Silver	koz	7,147	326	465	509	365	345	357	386	387	411	368	339	324	243	239	272	312	305	312	423	217	199	44
Gravity recovered gold concentrate (payable metals)																								
Gold	koz	101	4	5	7	6	6	6	6	6	7	5	5	4	3	3	4	4	3	3	4	4	4	1
Silver	koz	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total payable metals																								
Gold	koz	4,001	159	201	274	244	249	252	244	235	287	213	195	156	131	134	146	148	138	123	152	144	147	28
Silver	koz	15,905	733	1,045	1,144	813	766	796	860	863	919	820	753	720	536	526	603	690	675	690	947	476	434	97
Copper	Mlb	63	2	3	3	3	3	3	3	3	3	4	4	3	2	2	2	3	3	3	4	2	2	0
Gold revenue by concentrate																								
Copper concentrate	$M	5,221	207	263	358	318	325	329	319	307	374	278	254	204	171	175	190	193	180	160	198	188	192	37
Pyrite concentrate	$M	4,921	196	248	337	300	306	310	301	289	353	262	240	192	161	165	179	182	170	151	187	177	181	35
Gravity gold concentrate	$M	262	10	13	18	16	16	17	16	15	19	14	13	10	9	9	10	10	9	8	10	9	10	2
Total	$M	10,403	413	524	712	633	647	656	635	611	745	554	507	406	341	349	379	385	358	320	395	375	382	74
Silver by-product credits by concentrate																								
Copper concentrate	$M	254	12	17	18	13	12	13	14	14	15	13	12	11	9	8	10	11	11	11	15	8	7	2
Pyrite concentrate	$M	207	9	13	15	11	10	10	11	11	12	11	10	9	7	7	8	9	9	9	12	6	6	1
Total	$M	461	21	30	33	24	22	23	25	25	27	24	22	21	16	15	17	20	20	20	27	14	13	3
Copper by-product credits																								
Copper concentrate	$M	271	9	12	13	13	13	15	14	15	14	17	15	12	10	10	10	13	12	12	19	10	10	2
Treatment and refining charges and shipping costs by concentrate																								
Copper concentrate	$M	87	3	4	5	5	4	5	5	5	5	5	4	4	3	3	3	4	4	4	5	3	3	1
Pyrite concentrate	$M	302	9	13	15	15	15	15	15	15	15	15	15	15	14	14	15	15	15	14	15	14	13	3
Gravity gold concentrate	$M	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total	$M	390	12	17	20	19	19	20	20	20	20	20	19	19	18	18	18	19	19	18	19	18	16	3

22.6 Economic analysis results

The results of the economic analysis are summarized in Table 22-6. The economic analysis excludes any contribution from the 750 tpd processing capacity dedicated for contract mining partners, which is intended to support regional formalization initiatives and environmental improvements. The NPV at a range of discount rates is shown in Table 22-7 and the annual cash flows are shown in Table 22-8.

Table 22-6 Economic evaluation results

Key indicators	Units	Total
Total gold in concentrates life of mine	koz	4,299
Initial life of mine at an owner-mining rate of 2,750 tpd	Years	22
Average annual gold production (years 2 to 10)	koz	263
Average annual gold production (years 1 to 21)	koz	203
Life of mine average cash cost	$/oz Au	345
Life of mine average all in sustaining cost	$/oz Au	534
Average annual EBITDA (years 2 to 10)	$M	547
Average annual EBITDA (years 1 to 21)	$M	410
Summary cash flow for the life of mine ($M) at $2,600/oz gold price		
Revenue from payable gold sales		10,403
Less: royalties		393
Less: operating costs, net of by-product silver and copper		1,381
Less: sustaining capital		364
Operating margin		8,265
Less: income tax		2,630
After-tax cash flow		5,635
Less initial capital including pre-production costs, VAT, and contingency		625
Less: closure costs		25
Net cash flow		4,985
Pre-tax indicators at $2,600/oz gold price		
NPV at 5% discount rate	$M	4,203
IRR	%	45.8
Payback period (from start of operations)	Years	1.9
After-tax indicators at $2,600 gold price		
NPV at 5% discount rate	$M	2,680
IRR	%	35.4
Payback period (from start of operations)	Years	2.3
After-tax indicators at $3,200/oz gold price		
NPV at 5% discount rate	$M	3,559
IRR	%	42.1
Payback period (from start of operations)	Years	2.0

Table 22-7 Sensitivity of NPV to discount rate

Discount rate	Units	Pre-tax NPV	After-tax NPV
0.0%	$M	7,615	4,985
5.0% (base case)	**$M**	**4,203**	**2,680**
10.0%	$M	2,481	1,519

Table 22-8 Annual after tax cash flow schedule

Parameter	Units	Total	Construction (13 quarters)	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Revenue from payable gold	**$M**	**10,403**	**-**	**413**	**524**	**712**	**633**	**647**	**656**	**635**	**611**	**745**	**554**	**507**	**406**	**341**
Royalties	$M	**(393)**	-	(16)	(20)	(27)	(24)	(24)	(24)	(24)	(23)	(28)	(21)	(19)	(16)	(13)
Net gold revenue	**$M**	**10,010**	**-**	**398**	**504**	**686**	**610**	**623**	**631**	**612**	**588**	**718**	**533**	**487**	**390**	**328**
Mining costs	$M	**(860)**	-	(28)	(38)	(42)	(49)	(53)	(50)	(48)	(44)	(42)	(40)	(41)	(38)	(39)
Processing costs	$M	**(425)**	-	(12)	(18)	(21)	(21)	(21)	(21)	(21)	(21)	(21)	(21)	(21)	(21)	(21)
G&A, environmental monitoring	$M	**(438)**	-	(20)	(20)	(19)	(19)	(19)	(19)	(19)	(19)	(19)	(19)	(19)	(19)	(19)
Treatment and refining costs, shipping	$M	**(390)**	-	(12)	(17)	(20)	(19)	(19)	(20)	(20)	(20)	(20)	(20)	(19)	(19)	(18)
Add: by-product credits	$M	**732**	-	30	42	46	37	36	38	39	40	41	41	37	33	25
Total operating costs (By-product)	**$M**	**(1,381)**	**-**	**(42)**	**(51)**	**(55)**	**(72)**	**(77)**	**(72)**	**(68)**	**(64)**	**(62)**	**(59)**	**(63)**	**(63)**	**(72)**
Working capital movements - operating costs	$M	-	-	(20)	(11)	(15)	4	(1)	(0)	1	1	(7)	10	3	6	4
Sustaining capital	$M	**(364)**	-	(71)	(49)	(44)	(34)	(12)	(10)	(14)	(16)	(16)	(11)	(6)	(13)	(8)
Operating margin	**$M**	**8,265**	**-**	**265**	**393**	**571**	**508**	**533**	**548**	**531**	**509**	**633**	**473**	**421**	**320**	**252**
Income taxes	$M	**(2,630)**	-	(89)	(140)	(198)	(166)	(168)	(172)	(166)	(159)	(203)	(146)	(129)	(102)	(78)
Working capital movements - income tax movements	$M	-	-	22	7	127	(28)	2	3	(5)	(6)	36	(47)	(14)	(21)	(20)
After-tax cash-flow	**$M**	**5,635**	-	197	261	500	314	367	379	360	344	467	280	278	196	154
Initial capital, including VAT and working capital	$M	**(625)**	(625)	-	-	-	-	-	-	-	-	-	-	-	-	-
Closure costs	$M	**(25)**	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Net cash flow	**$M**	**4,985**	(625)	197	261	500	314	367	379	360	344	467	280	278	196	154
Cumulative net cash flow	**$M**	**4,985**	**(625)**	**(428)**	**(167)**	**333**	**647**	**1,013**	**1,393**	**1,753**	**2,097**	**2,564**	**2,844**	**3,123**	**3,318**	**3,472**
Cash cost	$/oz Au	**$345**		*$262*	*$252*	*$203*	*$298*	*$308*	*$287*	*$279*	*$273*	*$216*	*$276*	*$325*	*$404*	*$548*
AISC cost	$/oz Au	**$534**		*$807*	*$593*	*$461*	*$534*	*$455*	*$425*	*$433*	*$439*	*$366*	*$428*	*$457*	*$588*	*$710*

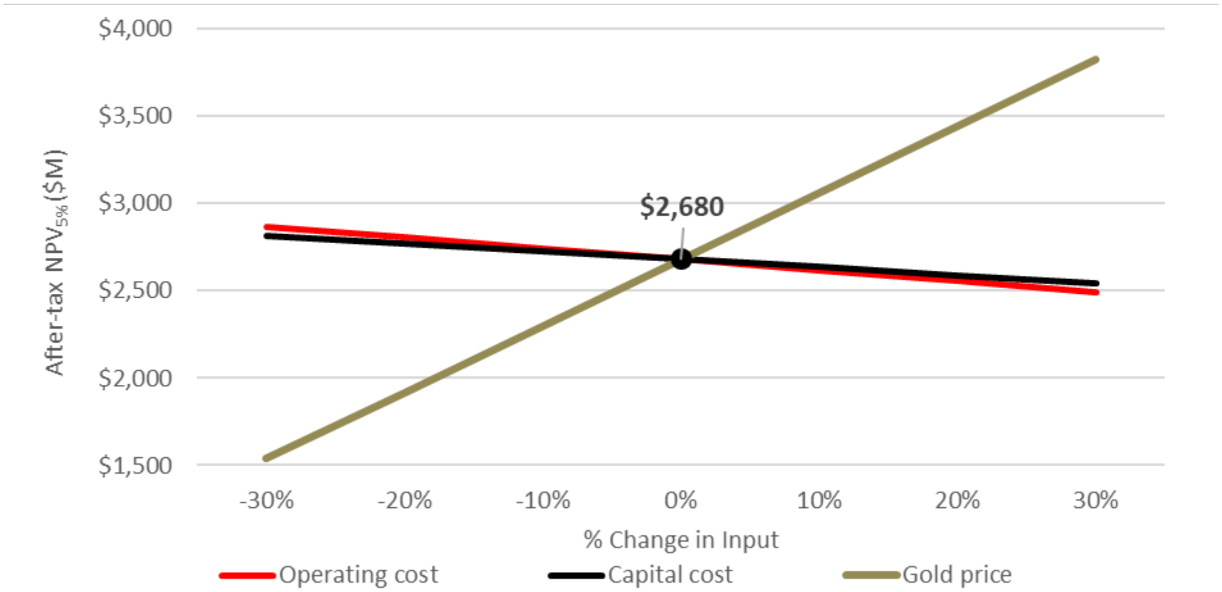
Parameter	Units	Y14	Y15	Y16	Y17	Y18	Y19	Y20	Y21	Y22	Y23	Y24	Y25	Y26	Y27
Revenue from payable gold	**$M**	**349**	**379**	**385**	**358**	**320**	**395**	**375**	**382**	**74**	**-**	**-**	**-**	**-**	**-**
Royalties	$M	(13)	(14)	(15)	(14)	(13)	(16)	(14)	(14)	(3)	-	-	-	-	-
Net gold revenue	**$M**	**336**	**365**	**370**	**345**	**307**	**379**	**361**	**368**	**71**	**-**	**-**	**-**	**-**	**-**
Mining costs	$M	(36)	(38)	(36)	(37)	(38)	(38)	(38)	(35)	(12)	-	-	-	-	-
Processing costs	$M	(21)	(21)	(21)	(21)	(21)	(21)	(21)	(18)	(4)	-	-	-	-	-
G&A, environmental monitoring	$M	(19)	(19)	(19)	(19)	(19)	(19)	(19)	(19)	(19)	(2)	(2)	(2)	(2)	(2)
Treatment and refining costs, shipping	$M	(18)	(18)	(19)	(19)	(18)	(19)	(18)	(16)	(3)	-	-	-	-	-
Add: by-product credits	$M	26	27	33	32	32	46	24	23	5	-	-	-	-	-
Total operating costs (By-product)	**$M**	**(69)**	**(69)**	**(62)**	**(64)**	**(64)**	**(51)**	**(72)**	**(67)**	**(34)**	**(2)**	**(2)**	**(2)**	**(2)**	**(2)**
Working capital movements - operating costs	$M	(1)	(1)	(0)	1	2	(5)	3	4	20	3	-	0	(0)	0
Sustaining capital	$M	(5)	(7)	(18)	(11)	(9)	(4)	(2)	(1)	(1)	-	-	-	-	-
Operating margin	**$M**	**261**	**288**	**290**	**271**	**236**	**319**	**290**	**303**	**57**	**1**	**(2)**	**(2)**	**(2)**	**(2)**
Income taxes	$M	(82)	(92)	(95)	(86)	(74)	(102)	(89)	(93)	-	-	-	-	-	-
Working capital movements - income tax movements	$M	4	8	3	(8)	(10)	24	(11)	4	(75)	4	-	-	-	-
After-tax cash-flow	**$M**	183	204	197	177	152	241	190	214	(18)	5	(2)	(2)	(2)	(2)
Initial capital, including VAT and working capital	$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Closure costs	$M	-	-	-	-	-	-	-	-	-	(12)	(12)	-	-	-
Net cash flow	**$M**	183	204	197	177	152	241	190	214	(18)	(7)	(14)	(2)	(2)	(2)
Cumulative net cash flow	**$M**	**3,654**	**3,858**	**4,056**	**4,233**	**4,386**	**4,626**	**4,817**	**5,031**	**5,013**	**5,005**	**4,991**	**4,989**	**4,987**	**4,985**
Cash cost	$/oz Au	*$510*	*$470*	*$419*	*$462*	*$522*	*$336*	*$499*	*$455*	*$1,191*					
AISC cost	$/oz Au	*$650*	*$618*	*$644*	*$640*	*$696*	*$465*	*$608*	*$563*	*$1,309*					

22.7 Sensitivity analysis

A range of gold prices, operating costs, and capital expenditures were assessed to evaluate the sensitivity of the Project's key economic indicators, including after-tax $NPV_{5\%}$, after-tax IRR, and the after-tax pay-back period.

The sensitivity of the after-tax $NPV_{5\%}$, to +/-30% changes in gold price, operating costs, and capital costs are shown in Figure 22-3, which indicate that the Project is most sensitive to gold price.

Figure 22-3 Sensitivity of $NPV_{5\%}$ to operating and capital costs and gold price – Source Mining Plus 2025



The sensitivity of the after-tax $NPV_{5\%}$, after-tax IRR, and after-tax payback period to a range of gold prices is shown in Table 22-9. The sensitivity of the after-tax NPV5% to operating and capital costs and gold price is shown in Table 22-10.

Table 22-9 Sensitivity of key economic indicators to gold price

Gold price / Indicator	$2,000/oz	$2,200/oz	$2,400/oz	**$2,600/oz Base case**	$2,800/oz	$3,000/oz	$3,200/oz
After-tax $NPV_{5\%}$ ($M)	1,800	2,093	2,387	**2,680**	2,973	3,266	3,559
After-tax IRR (%)	27.7	30.4	33.0	**35.4**	37.8	40.0	42.1
Payback period (years)	2.8	2.6	2.5	**2.3**	2.2	2.1	2.0

Table 22-10 Sensitivity of NPV5% to operating and capital costs and gold price

Change in %	-30%	-20%	-10%	**0%**	+10%	+20%	+30%
Gold Price – impact on After-tax $NPV_{5\%}$ ($M)	1,537	1,918	2,299	**2,680**	3,061	3,442	3,823
Capital Cost – impact on After-tax $NPV_{5\%}$ ($M)	2,815	2,770	2,725	**2,680**	2,635	2,590	2,545
Operating Cost – impact on After-tax $NPV_{5\%}$ ($M)	2,868	2,805	2,743	**2,680**	2,617	2,554	2,491

22.8 Conclusions

The result of the economic analysis indicates that the Project is economically viable under the base case assumptions, based on the current mineral reserve estimate and the assumptions described herein. At a $2,600 per ounce gold price, the after-tax $NPV_{5\%}$ is $2.7 billion, the after-tax IRR is 35.4%, and the payback period is 2.3 years from the start of processing operations.

The analysis excludes any contribution from the 750 tpd processing capacity dedicated for contract mining partners, which is intended to support regional formalization initiatives and environmental improvements. The economic results are not a measure of the Project's fair market value.

23 Adjacent properties

There is no relevant information on adjacent properties to report.

24 Other relevant data and information

There are no other relevant data or information to report.

25 Interpretation and conclusions

This prefeasibility study confirms the technical, economic, and community benefits of the Project. The design prioritizes the protection of the environment, water resources, and the health and wellbeing of the local communities.

The underground mine is designed to produce 2,750 tpd of ore using safe, modern methods. The process plant is designed for 3,500 tpd to accommodate an additional 750 tpd of mill feed purchased from contract mining partners from the local communities. The final products will be copper concentrates, pyrite concentrates, and gravity gold concentrates, and no cyanide or mercury will be required or used. Purchasing mill feed from contract mining partners will eliminate the use of mercury, which is commonly used by local traditional small scale miners, improve water and energy efficiency, ensure tailings are safely managed, and strengthen environmental stewardship.

The peak workforce during Project construction is estimated at 2,292, mainly comprised of contractors, plus 90 administrative staff and management. During operations the workforce is estimated at 676 company personnel. The Project is targeting 60% of the workforce to be hired from the local community, 20% from the department of Santander, and 18% from other departments in Colombia. Foreign technical and managerial specialists will eventually make up 2% of the workforce.

The Project's expected mine life of 22 years is based on the current mineral reserve estimate, and there are opportunities for future mine extension if additional mineral reserves are defined through continued exploration. Over the life of the mine, production is estimated at 4.3 Moz of gold, 18.8 Moz of silver, and 84.0 Mlb of copper. Average annual gold production is expected to be 263 koz between years 2 and 10, and 203 koz between years 1 and 21.

At the base case assumption of $2,600 per ounce of gold, the Project is estimated to contribute $2.6 billion in income taxes and $393 million in royalty payments to the Colombian government. The estimated initial capital expenditure is $625 million. The cumulative after-tax net cash flow is $5.0 billion, including initial capital costs, pre-production costs, sustaining capital costs, closure costs, VAT, and contingency. Cash costs per ounce of gold are estimated at $345 and AISC per ounce of gold are estimated at $534. At the base case assumption of $2,600 per ounce of gold, the Project has an after-tax $NPV_{5\%}$ of $2.7 billion, an IRR of 35.4%, and a pay back period of 2.3 years.

With the prefeasibility study complete, PSN will advance the Project's environmental and technical studies and intends to submit an updated ESIA to the CDMB reflecting the Project revised design as described in this technical report.

26 Recommendations

The qualified person responsible for Section 11 notes that aqua regia digest may not fully digest the sample silica matrix and may provide lower assay results than fire assays as used by Ventana and AUX Colombia, and recommends that future drillhole samples are assayed for silver using fire assay methods.

The qualified person responsible for Section 9 and 14 notes that the Project's expected mine life of 22 years is based on the current mineral reserve estimate, and that there are opportunities for future mine extension if additional mineral reserves are defined through continued exploration. There are numerous areas of high grade inferred material within the mineable stope optimizer shapes used to constrain the mineral resource estimate that are located adjacent to the mineral reserve stopes designed around indicated material that could be targeted for exploration. An initial exploration drilling program targeting those areas in the upper areas of the mine of 35 drillholes for 12,500 m is recommended to potentially convert an additional 1.2 Mt of inferred mineral resources at 12.59 g/t of gold for 482,000 ounces, at an estimated cost of $1.3 M.

The qualified person responsible for Section 13 recommends the following metallurgical testwork:

- Undertake additional comminution testwork from an increased density of samples representative of the life of mine plan to provide data for more accurate comminution simulations, for an estimated cost of $17,000.
- Undertake settling and filtration testwork with actual site process water and recirculated water, for an estimated cost of $30,000.
- Undertake additional testwork to characterize the gravity recoverable gold present in samples representative of the material in the life of mine plan, for an estimated cost of $6,000.
- Undertake additional locked cycle tests on samples representative of the material in the life of mine plan to characterize the nature of the gravity, copper, and pyrite concentrates, for an estimated cost of $10,000
- Undertake a more detailed financial analysis of the economic benefits of a gravity circuit and investigate alternative technologies, for an estimated cost of $5,000.
- Undertake an assessment of the logistics and environmental impacts of including a gravity circuit, for an estimated cost of $12,000.

The qualified person responsible for Section 16 makes the following recommendations:

- Further trade off and mine optimization studies are recommended for the future study stages, including the decline layout, the location of key underground infrastructure, options for the materials handling system, refinements to the blasting methods, and additional work on the ventilation design including optimal sizes, consideration of a ventilation on demand system, and ventilation heat modelling, for a total cost of $210,000.
- Additional refinements related to the underground infrastructure, including capital and operating costs, are recommended for air, water, electrical, ventilation control, egress and refuge chambers, explosive magazine, dewatering, mine services, paste fill plant and delivery, surface to underground cement delivery, and additional paste fill test work, for a total cost of $193,000.
- Additional geotechnical studies are recommended, including a Mathews stability graph check, three dimensional stress modelling, cemented paste fill design strength and strength gain rate tests as well as identifying implementation opportunities, and a trade off study to optimize the management of slimes in the paste fill or in the filtered tailings storage facility, for a total cost of $48,800.

The qualified person responsible for Section 18.5 makes the following recommendations to advance the design of the filtered tailings storage facility to the detailed engineering level, at an estimated cost of $3 million, considering both engineering consulting and contractor costs.

- Complete site investigation work including borehole drilling, test pit excavation, instrumentation installation, in situ testing, and geophysical surveys to inform the next phases of design.
- Advance geotechnical laboratory characterization work on the foundation soils in their in situ state and as structural fill following excavation, moisture conditioning, placement, and compaction.
- Design and size the required drainage works.
- Undertake slope stability analyses to evaluate facility phasing concepts.
- Explore opportunities to deposit tailings in non-horizontal lifts to reduce staging challenges.

- Develop facility safety and operations documents, including the operations, maintenance, and surveillance manual, the emergency response plan, and the trigger action response plan.
- Develop breach and inundation studies on the filtered tailings facility and contact water collection pond for potential damage assessment, risk classification, potential loss of life analysis, and emergency plan design and implementation.
- Complete additional geotechnical analyses as required by the regulators and the Project, such as probabilistic slope stability analyses and/or nonlinear dynamic deformation models.
- Advance applicable engineering analyses to levels commensurate with future design phases.
- Develop a groundwater sampling program.
- Develop detailed closure requirements for the facility.
- Develop a cover system that will limit long term infiltration into the facility.
- Further refine grading of the required excavation works, containment embankment, and tailings stack.
- Create a cover revegetation plan.

The qualified person responsible for section 18.6 recommends that a comprehensive hydrological study is undertaken for future studies to refine the stormwater design, assess the long term water balance, and develop a more detailed mitigation strategy. Most of this work is currently underway to support the ESIA application.

The qualified person responsible for Section 20 makes the following recommendations:

- Complete environmental and social baseline surveys, including along the rope conveyor, tailing pipeline route, and the filtered tailings storage watershed, and confirm the absence of fish and amphibians in the filtered tailings facility footprint.
- Conceptualize a reclamation research program to support future progressive and final reclamation and revegetation programs, including collaborative partnering with local greenhouse owners to develop native seedling sources for revegetation.
- Continue efforts to advance contract mining partner agreements and action plans for implementation.
- Install a large Project area poster in each social house to serve as a focal point for discussions with members as they share their issues and experiences.

This work is covered in the ongoing environmental permitting work and has been accounted for within the existing social and environmental budgets.

27 References

Author and Date	Title
CIM, 2014	CIM Definition Standards for Mineral Resources and Mineral Reserves. Prepared by the CIM Standing Committee on Reserve Definitions, adopted by CIM Council 19 May 2014
CIM, 2019	CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines. Prepared by the CIM Mineral Resource and Mineral Reserve Committee, adopted by the CIM Council on 29 November 2019.

28 Date, signatures, and certificates of Qualified Persons

Certificate of Qualified Person
Kate Kitchen
525 Collins St, Melbourne, Victoria, Australia, 3000

I, Kate Kitchen, MAIG, do hereby certify that:

1. I am employed as Area Manager – Geology with Mining Plus at 525 Collins St, Melbourne, Victoria, Australia, 3000.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report Prefeasibility Study for the Soto Norte Project, Santander, Colombia" with an effective date of August 18, 2025 (the Technical Report).
3. I graduated with a Bachelor of Science in Geology from the University of Melbourne in 2001. I have practiced my profession as a geologist since my graduation in 2001. I am a member in good standing of the Australian Institute of Geoscientists.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* (NI 43 101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "qualified person" for the purposes of NI 43-101.
5. I visited the Soto Norte Project from June 17 to 19, 2025 (three days).
6. I am responsible for information related to Sections 2 through 11, 12.1, Sections 14, 23, 24, and 27, and the relevant summaries of those sections in Sections 1, 25, and 26 of the Technical Report.
7. I am independent of Aris Mining Corporation as described in section 1.5 of NI 43-101.
8. I have not had any prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 3rd day of September, 2025

(signed) Kate Kitchen

Certificate of Qualified Person
Peter Lock
1 George Wiencke Drive, Perth, Western Australia, Australia, 6105

I, Peter Lock, FAusIMM, do hereby certify that:

1. I am employed as Executive Director and Principal Mining Consultant with Mining Plus at 1 George Wiencke Drive, Perth, Western Australia, Australia, 6105.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report Prefeasibility Study for the Soto Norte Project, Santander, Colombia" with an effective date of August 18, 2025 (the Technical Report).
3. I graduated with a Bachelor of Engineering in Mining Engineering at the Western Australian School of Mines in 1997. I have practiced my profession as a mining engineer since my graduation in 1997. I am a Fellow in good standing of the Australasian Institute of Mining and Metallurgy.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* (NI 43 101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "qualified person" for the purposes of NI 43-101.
5. I visited the Soto Norte Project from June 17 to 19, 2025 (three days).
6. I am responsible for information related to Sections 2, 3, 12.2, 15, 16, 19, 21 related to mining costs, 22, 24, and 27, and the relevant summaries of those sections in Sections 1, 25, and 26 of the Technical Report.
7. I am independent of Aris Mining Corporation as described in section 1.5 of NI 43-101.
8. I have not had any prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 3rd day of September, 2025

(signed) Peter Lock

Certificate of Qualified Person
Jan Eklund
Block E, Lonehill Office Park, 3 Lone Close, Lonehill, Sandton, 2062, South Africa

I, Jan Eklund, P.Eng., do hereby certify that:

1. I am employed as a Process Consultant of LogiProc Pty. Ltd. at Block E, Lonehill Office Park, 3 Lone Close, Lonehill, Sandton, 2062 South Africa.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report Prefeasibility Study for the Soto Norte Project, Santander, Colombia" with an effective date of August 18, 2025 (the Technical Report).
3. I graduated with a Bachelor of Science degree in Metallurgical Engineering from the Witwatersrand University in Johannesburg, South Africa, in 1988. I have practiced my profession as an extractive metallurgist since my graduation in 1989. I am a member in good standing with the South African Institute of Mining and Metallurgy.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* (NI 43 101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "qualified person" for the purposes of NI 43-101.
5. I visited the Soto Norte Project from July 15 to 17, 2024 (three days).
6. I am responsible for information related to Sections 2, 12.3, 13, 17, 18 with the exception of Section 18.5, 21 related to processing and surface infrastructure costs, 24, and 27, and the relevant summaries of those sections in Sections 1, 25, and 26 of the Technical Report.
7. I am independent of Aris Mining Corporation as described in section 1.5 of NI 43-101.
8. I have not had any prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 3rd day of September, 2025

(signed) Jan Eklund

Certificate of Qualified Person

Nicholas Sianta
1999 Broadway, Suite 900, 80202, Denver, Colorado, USA

I, Nicholas Sianta, P. E., do hereby certify that:

1. I am a licensed geotechnical engineering consultant in the United States employed by Knight Piésold and Co. in Denver, Colorado, USA at 1999 Broadway, Suite 900, 80202.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report Prefeasibility Study for the Soto Norte Project, Santander, Colombia" with an effective date of August 18, 2025 (the Technical Report).
3. I obtained an Bachelor of Science, Civil Engineering from Colorado State University in 2015 and a Masters of Science, Civil Engineering from the Georgia Institute of Technology in 2018. I am a registered Professional Engineer in the state of Colorado, United States as recognized by the Colorado Department of Regulatory Agencies. I have practiced geotechnical engineering since my graduation in 2015.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* (NI 43 101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "qualified person" for the purposes of NI 43-101.
5. I visited the Soto Norte Project on three separate occasions, with my most recent visit occurring April 17 to May 13, 2023 (26 days).
6. I am responsible for information related to the tailings storage facility presented in Sections 2, 12.4, 18.5, 24, and 27, and the relevant summaries of those sections in Sections 1, 25, and 26 of the Technical Report.
7. I am independent of Aris Mining Corporation as described in section 1.5 of NI 43-101.
8. I have been involved with the Property since Knight Piésold began providing consulting services to the Project in October, 2020.
9. I have read NI 43-101 and Form 43-101F and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 3rd day of September, 2025

(signed) Nicholas Sianta

Certificate of Qualified Person
Rolf Schmitt
1000-1100 Melville Street, Vancouver, V6E 4A6, British Columbia, Canada

I, Rolf Schmitt, P.Geo., do hereby certify that:

1. I am a Technical Consulting Director employed by ERM Consultants Canada Ltd. in Vancouver, British Columbia, Canada at 1000-1100 Melville Street, V6E 4A6.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report Prefeasibility Study for the Soto Norte Project, Santander, Colombia" with an effective date of August 18, 2025 (the Technical Report).
3. I obtained an Honours Bachelor of Science, Economic Geology from the University of British Columbia in 1977; a Masters of Science, Regional Planning from the University of British Columbia in 1985; and a Masters of Science, Exploration Geochemistry from the University of Ottawa in 1993. I am a registered Professional Geoscientist with Engineers and Geoscientists British Columbia and a registered Professional Geoscientist with the Nunavut Association of Professional Engineers and Geoscientists. I have practiced as a geologist since my graduation in 1977.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* (NI 43 101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "qualified person" for the purposes of NI 43-101.
5. I visited the Soto Norte Project on December 9, 2024 (one day).
6. I am responsible for Sections 2, 3, 20, 24, and 27, and the relevant summaries of those sections in Sections 1, 25, and 26 of the Technical Report.
7. I am independent of Aris Mining Corporation as described in section 1.5 of NI 43-101.
8. I have been involved with the Property since my contract for consulting services with Aris Mining commenced on July 9, 2024.
9. I have read NI 43-101 and Form 43-101F and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 3rd day of September, 2025

(signed) Rolf Schmitt