GRAMALOTE PROJECT

COLOMBIA

NI 43-101 Technical Report

PREPARED FOR
B2Gold Corp.

QUALIFIED PERSONS
Mr. Stephen Jensen, P.Geo.
Mr. Peter Montano, P.E.
Mr. John Rajala, P.E.
Mr. Ken Jones, P.E.

EFFECTIVE DATE
July 14, 2025

CERTIFICATE OF QUALIFIED PERSON

I, Stephen Jensen, P.Geo., am employed as the Exploration Manager, Americas, with B2Gold Corp. ("B2Gold"), which has its head offices at 666 Burrard St #3400, Vancouver, BC V6C 2X8, Canada.

This certificate applies to the technical report titled "Gramalote Project, Colombia, NI 43-101 Technical Report", that has an effective date of July 14, 2025 (the "Technical Report").

I am a registered member of the Engineers and Geoscientists of British Columbia (#20213) and of the Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (#L5853). I graduated from the University of British Columbia with a Bachelor of Science, Geology degree in 1987.

I have practiced my profession for 38 years. In this time, I have been directly involved in generating and managing exploration activities including the collection, supervision and review of geological, mineralization, exploration and drilling data; geological models; sampling, sample preparation, assaying and other resource-estimation related analyses; assessment of quality assurance-quality control data and databases; and supervision of Mineral Resource estimates.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects ("NI 43–101").

I visited the Gramalote Project on a continuous basis since June 2008 with the most recent visit on July 4, 2025, a visit duration of one day.

I am responsible for Sections 1.1, 1.2, 1.3, 1.4, 1.5, 1.6, 1.7, 1.8, 1.10, 1.11. 1.23. 1.24; Sections 2.1, 2.2, 2.3, 2.4.1; 2.5, 2.6, 2.7; Section 3; Section 4; Section 5; Section 6; Section 7; Section 8; Section 9; Section 10; Section 11; Sections 12.1, 12.2, 12.3.1; Section 14; Section 23; Sections 25.1, 25.2, 25.3, 25.4, 25.6, 25.16.2; Section 26, and Section 27 of the Technical Report.

I am not independent of B2Gold as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Gramalote Project since 2008.

I have read NI 43–101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument.

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

Dated: August 27, 2025

(Signed) "Stephen Jensen"

Stephen Jensen, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

I, Peter Montano, P.E., am employed as the Vice President of Projects with B2Gold Corp. ("B2Gold"), which has its head offices at 666 Burrard St #3400, Vancouver, BC V6C 2X8, Canada.

This certificate applies to the technical report titled "Gramalote Project, Colombia, NI 43-101 Technical Report", that has an effective date of July 14, 2025 (the "Technical Report").

I am a registered Professional Engineer (#42745, Colorado, USA). I graduated from the Colorado School of Mines in 2004 with a B.Sc. in engineering and a B.Sc. in economics.

I have been directly involved in the design, construction, and operation of gold mines in Nicaragua, Namibia, Mali and have participated in and contributed to projects and studies of gold and coal projects in Venezuela, El Salvador, Australia, and the Philippines. I have participated in long-term and strategic mine planning, Mineral Reserve estimation, and economic analyses of mining projects and mining operations, including from development to closure.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I visited the Gramalote Project most recently October 24, 2024, a duration of one day.

I am responsible for Sections 1.1, 1.2, 1.8, 1.12, 1.13, 1.14, 1.16, 1.18, 1.19 (excepting process), 1.20 (excepting process), 1.21, 1.22, 1.24, 1.25; Sections 2.1, 2.2, 2.3, 2.4.2, 2.5, 2.6; Section 3; Section 5; Section 12.3.2; Section 15; Section 16; Section 18; Section 19; Sections 21.1, 21.2.1, 21.2.2, 21.2.3, 21.2.4, 21.2.6, 21.2.7, 21.2.8, 21.3.1, 21.3.2, 21.3.4, 21.3.5, 21.4; Section 22; Section 24; Sections 25.1, 25.7, 25.8, 25.10, 25.12, 25.13 (excepting process), 25.14 (excepting process), 25.15, 25.16.1, 25.17; Section 26; and Section 27 of the Technical Report.

I am not independent of B2Gold as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Gramalote Project since 2020.

I have read NI 43–101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument.

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

Dated: August 27, 2025

(Signed) “Peter Montano”

Peter Montano, P.E.

CERTIFICATE OF QUALIFIED PERSON

I, John Rajala, P.E., am employed as the Vice President, Metallurgy with B2Gold Corp. ("B2Gold"), which has its head offices at 666 Burrard St #3400, Vancouver, BC V6C 2X8, Canada.

This certificate applies to the technical report titled "Gramalote Project, Colombia, NI 43-101 Technical Report", that has an effective date of July 14, 2025 (the "Technical Report").

I am a registered professional engineer in the state of Washington (No. 43299) and have a B.Sc. and M.Sc. in Metallurgical Engineering from Michigan Technological University (1976) and the University of Nevada – Mackay School of Mines (1981), respectively. I received a M.E. in Mining Engineering from the University of Arizona in 2022.

I have practiced my profession for 47 years, during which I have been directly involved in the operations and management of mineral processing plants for gold and base metals, and in process plant design and commissioning of projects located in Africa, Asia, North, Central and South America.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* ("NI 43–101").

I visited the Gramalote Project most recently from October 4-6, 2011, a duration of three days.

I am responsible for Sections 1.1, 1.2, 1.8, 1.9, 1.15, 1.19 (process costs only), 1.20 (process costs only), 1.25; Sections 2.1, 2.2, 2.3, 2.4.3; Section 12.3.3; Section 13; Section 17; Sections 21.2.5, 21.3.3; Sections 25.1, 25.5, 25.9, 25.13 (process costs only), 25.14 (process costs only); Section 26; and Section 27 of the Technical Report.

I am not independent of B2Gold as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Gramalote Project since 2011. I co-authored a voluntarily-filed technical report on the Gramalote Project as follows:

- Garagan, T., Pemberton, K., Rajala, J., and Jones, K., 2020: Gramalote Project, Colombia, NI 43-101 Technical Report: report prepared for B2Gold, effective date 31 December, 2019.

I have read NI 43–101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument.

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

Dated: August 27, 2025

(Signed) "John Rajala"

John Rajala, P.E.

CERTIFICATE OF QUALIFIED PERSON

I, Ken Jones, P.E., am employed as the Director, Sustainability, with B2Gold Corp. ("B2Gold"), which has its head offices at 666 Burrard St #3400, Vancouver, BC V6C 2X8, Canada.

This certificate applies to the technical report titled "Gramalote Project, Colombia, NI 43-101 Technical Report", that has an effective date of July 14, 2025 (the "Technical Report").

I am a registered Professional Engineer (#42718, Colorado, USA). I graduated from the University of Iowa in 2001 with a B. Sc. in Chemical Engineering.

I have practiced my profession for over 20 years. I have developed, conducted and/or directed environmental and social studies including baseline investigations; materials geochemical characterization; hydrologic, air and noise modeling; closure planning and costing; and environmental and social impact assessment for hard rock mining projects in over a dozen countries in North and South America, Africa and Asia. I have developed, implemented and maintained programs for engineering and administrative compliance regarding international environmental, health and safety regulations and best practices at gold projects in Canada, Nicaragua, Namibia, the Philippines, and Mali.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I visited the Gramalote Project most recently from September 29 to October 7, 2021, a duration of nine days.

I am responsible for Sections 1.1, 1.2, 1.8, 1.17, 1.25; Sections 2.1, 2.2, 2.3, 2.4.4; Sections 4.9, 4.10, 4.11, 4.12; Section 12.3.4; Section 20; Sections 25.1, 25.11, Section 26; and Section 27 of the Technical Report.

I am not independent of B2Gold as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Gramalote Project since 2019. I co-authored a voluntarily filed technical report on the Gramalote Project as follows:

- Garagan, T., Pemberton, K., Rajala, J., and Jones, K., 2020: Gramalote Project, Colombia, NI 43-101 Technical Report: report prepared for B2Gold, effective date 31 December, 2019.

I have read NI 43–101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument.

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

Dated: August 27, 2025

(Signed) “Ken Jones”

Ken Jones, P.E.

CAUTIONARY NOTE REGARDING FORWARD-LOOKING INFORMATION

This National Instrument 43-101 Technical Report (the Report) contains “forward-looking information” and “forward-looking statements” (collectively, forward-looking statements) within the meaning of applicable Canadian and United States securities legislation, including, but not limited to, B2Gold Corp.’s (B2Gold) objectives, strategies, intentions and expectations; projections; outlook; guidance; forecasts; estimates; and other statements regarding future or estimated financial and operational performance, gold production and sales, revenues and cash flows, and capital costs and operating costs, including projected cash operating costs, and budgets; statements regarding future or estimated mine life, metal price assumptions, estimated mineralized material grades, gold recovery rates, stripping ratios, throughput, processing; statements regarding anticipated exploration, drilling, development, construction, permitting and other activities or achievements of B2Gold; the results of and estimates in the 2025 Feasibility Study, including the production and life-of-mine estimates, capital cost and operating cost estimates, the financial projections, estimates and results, and other results of the economic analyses contained therein; the potential to convert existing Inferred Mineral Resources to the Indicated category; the timing to complete detailed designs and source equipment; the potential to develop Gramalote as an open-pit gold mine and any construction or other decision in respect thereof; the anticipated effect of external factors on revenue, such as commodity prices, exchange rates and metal price assumptions, estimation of Mineral Reserves and Mineral Resources, mine life projections, reclamation costs, economic outlook; tailings dam and storage facilities, the maintenance or provision of required infrastructure and information technology systems, government regulation of mining operations and the entering into of major contracts required for development and/or operations; potential environmental, physical, social and economic impacts and plans, measures, and requirements to address such impacts, environmental considerations and closing and reclamation planning; stakeholder engagement; legal proceedings on mining area “corridors”; receipt and timing of additional required permits, modifications, and authorizations, including with respect to surface rights; expectations regarding community relations and social licence to operate including with respect to resettlement of individuals due to project development and artisanal and small miners working in the Project area. All statements in this Technical Report that address events or developments that B2Gold expects to occur in the future are forward-looking statements. Forward-looking statements are statements that are not historical facts and are generally, although not always, identified by words such as “expect”, “plan”, “anticipate”, “project”, “target”, “potential”, “schedule”, “forecast”, “budget”, “estimate”, “intend” or “believe” and similar expressions or their negative connotations, or that events or conditions “will”, “would”, “may”, “could”, “should” or “might” occur. All such forward-looking statements are based on the opinions and estimates of B2Gold’s management as of the date such statements are made. All of the forward-looking statements in this Report are qualified by this cautionary note.

Forward-looking statements are not, and cannot be, a guarantee of future results or events. Forward-looking statements are based on, among other things, opinions, assumptions, estimates and analyses that, while considered reasonable at the date the forward-looking statements is provided, inherently are subject to significant risks, uncertainties, contingencies and other factors that may cause actual results and events to be materially different from those expressed or implied by the forward-looking statements. The material factors or assumptions that B2Gold identified and were applied by B2Gold in drawing conclusions or making forecasts or projections set out in the forward-looking statements include, but are not limited to: the factors identified in Sections 1.12, 14.13 and 25.6 of this Report, which may affect the Mineral Resource estimate; the metallurgical recovery assumptions identified in Section 13 of this Report; the factors related to the 2025 Feasibility Study identified in Sections 1.19, 1.20, 21.3 and 22.1 of this Report; the project risks and opportunities identified in Section 1.22 of this Report; dilution, ore loss and mining recovery assumptions; assumptions regarding stockpiles; the success of mining, processing, exploration and development activities; the accuracy of geological, mining and metallurgical estimates; geotechnical assumptions; assumptions used in the Mineral Resource estimate; anticipated commodity prices and the costs of production; no significant unanticipated operational or technical difficulties; the execution of B2Gold’s business and growth strategies, including the success of B2Gold’s strategic investments and initiatives; the availability of additional financing, if needed; the availability of personnel for exploration, development, and operational projects and ongoing employee relations; acquisition of any required surface rights; maintaining good relations with the communities surrounding the Project; no significant unanticipated events or changes relating to regulatory, environmental, resettlement and social licence matters, health and safety matters; diminishing quantities or grades of reserves; increased costs, delays, suspensions, and technical challenges associated with the construction of capital projects; geotechnical and hydrogeological considerations during mining being different from what was assumed; no contests over title to B2Gold’s properties; no significant litigation; certain tax matters; and no significant and continuing adverse changes in general economic conditions or conditions in the financial markets (including commodity prices and foreign exchange rates).

The risks, uncertainties, contingencies and other factors that may cause actual results to differ materially from those expressed or implied by the forward-looking statements may include, but are not limited to, risks generally associated with the mining industry, such as economic factors (including future commodity prices, currency fluctuations, energy prices and general cost escalation), uncertainties related to the continued development and operation of the Project, the speculative nature of mineral exploration and development; changes to production, exploitation and exploration successes and other estimates; changes to legislation, taxation laws, policies and practices in Colombia, risks and uncertainties associated with political and economic instability in Colombia; fluctuations in the price and availability of infrastructure, energy, and other commodities; the impact of inflation; compliance with government regulations, including anti-bribery and corruption laws, environmental, health and safety regulations and internal control over financial reporting; damage to B2Gold's reputation due to actual or perceived occurrence of any number of events, including negative publicity with respect to environmental matters or dealings with community groups; the failure to obtain required licenses, permits, approvals or clearances from government authorities, including environmental permits, on a timely basis or at all; the potential for conflict with small scale miners; dependence on key personnel and employee relations; risks related to political or social unrest or change; operational risks and hazards, including unanticipated environmental, industrial and geological events and developments and the inability to insure against all risks; failure of plant, equipment, processes, transportation and other infrastructure to operate as anticipated; the availability of natural gas to generate electrical power, and the resulting power rates used in the operating cost estimates and financial analysis; compliance with government and environmental regulations, including permitting requirements and anti-bribery legislation; volatile financial markets that may affect B2Gold's ability to obtain additional financing on acceptable terms; the failure to obtain required approvals or clearances from government authorities on a timely basis; uncertainties related to the geology, continuity, grade and estimates of Mineral Resources, and the potential for variations in grade and recovery rates; uncertain costs of reclamation activities, and the final outcome thereof; changes to applicable laws or regulations, including with respect to interest rates and tax rates; tax refunds; hedging transactions; as well as other factors identified and as described in more detail under the heading "Risk Factors" in B2Gold's most recent Annual Information Form and B2Gold's other filings with Canadian securities regulators and the U.S. Securities and Exchange Commission, which may be viewed at www.sedar.com and www.sec.gov, respectively. The list is not exhaustive of the factors that may affect B2Gold's forward-looking statements. Accordingly, no assurance can be given that any events anticipated by the forward-looking statements will transpire or occur, or if any of them do, what benefits or liabilities B2Gold will derive therefrom. B2Gold's forward looking statements reflect current expectations regarding future events and operating performance and speak only as of the date hereof and B2Gold does not assume any obligation to update forward-looking statements if circumstances or management's beliefs, expectations or opinions should change other than as required by applicable law. For the reasons set forth above, undue reliance should not be placed on forward-looking statements.

CONTENTS

TABLES

FIGURES

1.0 SUMMARY

1.1 Introduction

Mr. Stephen Jensen, P.Geo., Mr. Peter Montano, P.E., Mr. John Rajala, P.E., and Mr. Ken Jones, P.E., prepared a National Instrument 43-101 Technical Report (the Report) on the Gramalote Gold Project (the Gramalote Project or the Project) for B2Gold Corp. (B2Gold). The Project is located within the department of Antioquia in northwestern Colombia.

Gramalote Limited is registered in Colombia as Gramalote Colombia Limited (Gramalote Colombia) and is the operating entity of the Project. Gramalote Limited is a wholly-owned subsidiary of B2Gold.

1.2 Terms of Reference

The Report was prepared to support the news release entitled "B2Gold Announces Positive Feasibility Study Results for the Gramalote Project" filed on July 14, 2025.

The Report includes Mineral Resource estimates for the Gramalote Ridge, Trinidad and Monjas West deposits and Mineral Reserve estimates for the Gramalote Ridge deposit.

Units used in the Report are metric units unless otherwise noted. Monetary units are in United States dollars (US$) unless otherwise stated. The Report uses Canadian English. Mineral Resources are reported in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (May 2014; the 2014 CIM Definition Standards).

In early Project reports, the Gramalote Ridge deposit was also referred to as the Gramalote Central deposit. The current preferred nomenclature is Gramalote Ridge.

1.3 Project Setting

The Project is located approximately 200 km directly northwest of the Colombian capital of Bogota, a distance of approximately 408 km by road, and 100 km northeast of Medellin, the regional capital of the Department of Antioquia.

The Project site is accessible via a well-maintained paved road from Medellin (approximately 2 hours by car) and Bogota (approximately 8–10 hours by car). Both Medellin and Bogota are served by daily international flights.

The area surrounding the Project has mildly tropical climate, supporting the planned year-round mining operations.

The Project site is located approximately 55 km from Puerto Berrio, a distance of approximately 73 km by road. From Puerto Berrio, direct water access is available to Barranquilla, a major ocean port on the Caribbean coast.

Providencia, the town nearest to the Project site, is a historic mining supply center administered by the Municipality of San Roque, located 12 km southwest of the Gramalote Project. An inactive freight and passenger railway line, along with active high-tension electricity lines, pass within 1 km of the Project area. A 2.3 MW hydroelectric plant, owned by B2Gold, is currently generating electricity at Guacas Creek within the Gramalote Project area. The nearest high voltage substation suitable for grid connection is located approximately 30 km from the Project site.

Several small towns near the Project site, with a long history of ranching and small-scale mining, can provide readily available labor. Personnel, equipment, and contractors to support exploration, construction, and production activities are available in Medellin.

The Project is located along the southern margin of the topographic depression formed by the Nus River valley. Elevations in the Project area range from 800–1,500 m above sea level (masl), while elevations over the Antioquia plateau are generally between 2,300–2,500 masl. Natural bedrock outcrops are rare. Evidence of small-scale, artisanal gold mining from both alluvial and hard-rock sources is evident throughout the Nus River valley region. Outside of artisanal mining areas, the region is predominately covered by grass pasture and cropland, with limited and scattered patches of natural vegetation. Agricultural activities in the Nus River valley are centered around cattle ranching and the cultivation of sugar cane, raw sugar (panela), and tropical fruit production.

1.4 Mineral Tenure, Surface Rights, Water Rights, Royalties and Agreements

B2Gold, through Gramalote Colombia, holds 11,008.26 ha in three registered concession contracts, namely integrated mining title 14292, totalling 8,720.71 ha (referred to as the Gramalote Ridge title), concession title 4894, totalling 2,277.77 ha (referred to as the Trinidad title) and concession title QHQ-16081, totalling 9.78 ha. In addition, there is an application for a mineral title, LJC-08012, which has a total area of 94.14 ha.

To extend exploration activities at concession title 4894, Gramalote Colombia submitted an integration request between concession title 4894 and concession title QHQ-16081 in August 2023. If the mining authority approves the integration, Gramalote Colombia will be able to advance exploration in these areas.

B2Gold has commenced the acquisition of surface rights to support mining operations, with purchases having been completed on the majority of the area required. Based on the data and studies conducted, Gramalote Colombia has established a strategy to acquire the properties that are still to be acquired where necessary for the construction and operation stages.

Guacas Creek will provide the water for potable use. Gramalote Colombia currently holds a 40 L/s extraction permit. Water will be treated through a conventional 16 L/s water treatment plant. The Project is located in a high rainfall area, receiving approximately 2.4 m of precipitation annually. The water management system is designed to handle a positive water balance, ensuring sufficient supply for all process water requirements.

Once in production, State royalties on the gold and silver are 4% of the gross metal value at the plant site (as per Article 16 of Law 141 in 1994). However, gross metal value is determined by using 80% of the spot prices for gold and silver on the London Metals Exchange, so the effective royalty rate is 3.2%. These State royalties are independent of national, departmental, and municipal taxes.

1.5 Geology and Mineralization

The Gramalote Ridge, Trinidad and Monjas West deposits are examples of structurally-controlled, intrusive-related gold deposits.

The Project is located in the northern portion of Colombia's Central Cordillera. The terrane primarily comprises a metamorphic basement complex and the Antioquia Batholith. In the general Project area, the main compositional types within the Antioquia Batholith are tonalite and granodiorite, with subordinate monzonite and gabbro. Alaskitic, felsitic, and andesitic dikes have intruded the complex.

Gramalote Ridge is situated between two west–northwest-trending macro-scale curved lineaments that splay off the Palestina fault to the east, and transect the Antioquia Batholith, termed the Nus River and the El Socorro lineaments. Differential movement along the Nus and El Socorro lineaments is thought to have generated north–northwest, north–south and northeast-striking tensional dilation within the tonalite, reflected in the formation of stockwork style sheeted quartz and quartz carbonate veinlets.

The Gramalote Ridge deposit has dimensions of 1,300 x 1,500 x 700 m. The mineralization has been drill tested to a depth of about 750 m. Mineralization at Gramalote Ridge is hosted in uniform medium-grained tonalite with minor amounts of granodiorite and aplite dykes. Alteration is structurally-controlled and occurs as both broad zones and narrow selvedges around veins. Mineralized zones vary in width from 10–150 m in true width with vertical to sub-vertical dips to the south–southeast. The deposit remains open at depth and along strike. The primary mineralization consists of pyrite and chalcopyrite. Secondary minerals include rutile and a titaniferous mineral. Free gold occurs as argentiferous gold coeval with several tellurides and bismuth sulphosalt minerals.

The Trinidad deposit has dimensions of approximately 1,500 x 500 m. Mineralization has been drill tested to a depth of about 300 m. The deposit remains open at depth and along strike, especially to the west. Mineralization is hosted in similar rock types to those that host the Gramalote Ridge deposit. Alteration is similar to that described for Gramalote Ridge. Mineralized zones at Trinidad are 10–80 m thick, and the zones periodically coalesce both along strike and down-dip. Mineralization is associated with stockwork veinlets and alteration along their margins.

The Monjas West deposit measures approximately 1,000 x 500 m and has been drill tested to 400 m depth. The deposit remains open at depth and along strike, especially to the west. Alteration and mineralization styles are similar to that of Trinidad and Gramalote Ridge, with gold correlating with bismuth. Mineralized zones are 10–80 m thick, and the zones periodically coalesce both along strike and down-dip.

A number of additional prospects have been identified that warrant additional exploration examination.

1.6 History

The Gramalote area has a long history of artisanal gold mining, likely dating from Pre-Colombian times to the present day. Historical production was dominated by hydraulic techniques. The miners worked residually-enriched colluvium and mineralized in situ saprolite in the surface oxide zone around Gramalote Ridge, as well as alluvial deposits.

Prior to B2Gold's Project interest, exploration was conducted by Metallica Resources, Inc., Gridiron Exploration Ltd., Placer Dome Exploration Inc., Industrias Peñoles (a Grupo Nus affiliate), and Sociedad Kedahda S.A. (now AngloGold Ashanti Colombia S.A. (AngloGold Ashanti)). These companies completed property reviews, as well as surficial sampling and mapping of the Gramalote Ridge area. AngloGold Ashanti conducted exploration between 2003–2007, including stream sediment, soil, grab, chip, channel and panel sampling, geological and structural mapping of Gramalote Ridge, trial ground geophysical surveys, core drilling, metallurgical test work, and resource estimation.

B2Gold obtained its interest in the Project in 2008. Since then, work has included geological and structural mapping, rock and soil sampling, geophysics, core and reverse circulation (RC) drilling, engineering investigations, resource estimation, metallurgical test work, environmental and social baseline studies and data collection, as well as mining studies.

1.7 Drilling and Sampling

Drilling completed on the Project as of May 21, 2025 includes core and RC drilling, totalling 1,408 drill holes (approximately 269,049 m). No drilling has been conducted since that time.

The resource drilling cut-off date for Gramalote Ridge was March 9, 2022. There are 907 drill holes (approximately 193,126 m) and two adit samples (481 m) supporting the Mineral Resource estimate for Gramalote Ridge. The resource drilling cut-off date for Trinidad is July 14, 2025, and there are 165 holes (approximately 29,137 m) supporting the Mineral Resource estimate for Trinidad. The resource drilling cut-off date for Monjas West was January 4, 2023, and there are 55 holes (approximately 20,416 m) supporting the Mineral Resource estimate for Monjas West.

In addition to exploration, infill, and step out drilling, holes were completed for metallurgical, geotechnical, hydrological, and condemnation purposes.

Initial core logging was originally recorded on paper log sheets with pre-set logging parameters including lithology, alteration (dominant, subordinate and trace), types of veinlets, sulphide mineralization, and comments. Currently, B2Gold drilling data is entered directly into Excel spreadsheets and subsequently imported into a project-specific Access database. Geotechnical logging consists of rock quality designation (RQD), core recovery and rock strength. In addition, magnetic susceptibility and specific gravity are measured. Digital photographs were taken of all drill core for documentation purposes.

Core recoveries are generally excellent with an overall project average of 96.8%.

During the 2006–2007 program, drill hole collars were surveyed using a high-precision differential global positioning system (GPS) instrument. Since 2007, drill hole collars are located on the drill pads using total station survey instruments.

Depending on the drill campaign, ground conditions, and the purpose of the drill hole, down-hole surveys could be taken at 3 m, 6 m, 7 m, 12 m, 20 m, 30 m, 50 m, or 100 m intervals. Instruments used included Pajari, Reflex Maxibor II, E-Z track, Icefield, and Gyro Master tools.

RC samples are taken from 2 m runs. Core samples across all programs aimed at a nominal 2 m length, but could be shorter depending on instances where the sampling observed breaks in alteration, mineralization intensities, and lithology.

On average, the true width of the mineralization at Gramalote Ridge, Trinidad, and Monjas West deposits ranges from about is about 70–80% of the downhole drilled length but varies depending on local orientation of the mineralized zones and the drill hole.

Specific gravity or density determinations were collected from 2008 onwards, using the Archimedes or wax immersion method. Additionally, pycnometer measurements were collected during the initial drilling campaigns.

Sample preparation facilities have included the following laboratories: ALS Bogota (prior to November 2012); ALS Bucaramanga (December 2012); ALS Medellin (January 2013 to November 2018); Bureau Veritas Medellin (primary preparation laboratory from 2019–

Report Effective Date); SGS Medellin (secondary preparation laboratory from November 2021–January 2022); ALS Medellin (secondary preparation laboratory used in April, Sept–October 2020, and February 2022). ALS holds ISO 17025 accreditation for the preparation facilities in Colombia and the analytical facilities in Lima, Peru. Bureau Veritas holds ISO 9001:2015 accreditation for the preparation facilities in Colombia. SGS Colombia holds accreditation ISO/IEC 17021-1:2015 for the preparation facilities in Colombia.

ALS Lima was the primary analytical laboratory prior to 2019. The laboratory was accredited under INDECOPI prior to 2010 and has held ISO 17025 accreditations for selected analytical techniques since 2010. ALS Lima has been used as the check laboratory for Bureau Veritas Lima and SGS Medellin originals since 2019.

Bureau Veritas Lima has served as the primary analytical laboratory since 2019. Bureau Veritas Lima holds accreditations for selected analytical techniques, including ISO 17025, ISO 9001:2015, ISO 14001:2015, and OSHAS 18001:2007. Bureau Veritas Lima was also used as the check laboratory on original AngloGold Ashanti ALS Lima assays from 2018 and September 2020.

SGS Medellin was used for primary analyses between November 2021 and January 2022. SGS Medellin was also used as an umpire laboratory (on ALS Lima original analyses) prior to 2019. The laboratory currently holds the following certifications for selected analytical techniques: ISO 9001:2015, ISO 45001:2018, and ISO 14001:2015.

In 2020, as a result of Covid-related delays in Lima, both ALS Medellin and BV Medellin submitted some pulps to their respective Vancouver laboratories. ALS Vancouver holds ISO/IEC 17025:2017 accreditation since 2005, Bureau Veritas Vancouver holds ISO/IEC 17025:2017 since 2011. The Vancouver laboratories used the same gold–silver analytical techniques as their Peruvian equivalent but the coding on the silver determination was different (discussed below).

ACME Santiago was used as a check laboratory by B2Gold in early 2009 for 2008 ALS Lima originals. Accreditations held by ACME Santiago at the time are not known.

All laboratories used were independent of the then Project operator.

Sample preparation procedures varied depending on the laboratory utilized. All laboratories dried the samples, then crushed to either 70% passing 2 mm, 85% passing 2 mm, or 90% passing 2 mm, and finally pulverized to either >85% passing 75 µm or >90% passing 106 µm.

Analytical methods included:

- Gold: Analysis by fire assay and atomic absorption spectrometry (AAS); repeat assays for samples grading >10 g/t Au using a gravimetric finish;

- Silver: Four acid digestion and inductively coupled plasma (ICP) mass spectrometry (MS), ICP emission spectroscopy (ES) or ICP optical emission spectroscopy;
- Multi-element: Four-acid digest followed by ICP-MS for a 48-element suite; ICP-MS or ICP-ES to obtain a 59-element suite.

The quality assurance and quality control (QA/QC) program has included submission of certified reference materials (CRMs or standards), blanks, and coarse reject and pulp duplicate samples.

Sample security has not historically been monitored. Sample collection from drill point to laboratory relies upon the fact that samples are either always attended to, or stored in the locked on-site preparation facility, or stored in a secure area prior to shipment to the laboratory. Chain-of-custody procedures consist of sending sample submittal forms to the laboratory with sample shipments to ensure that all samples are received by the laboratory.

Drill core, pulps, and rejects are stored at the Project site in purpose-built storage facilities and are under full time supervision.

1.8 Data Verification

Internal data verification by B2Gold staff includes checking of data through a set of scripts that displays any inconsistent data related to the Project logging rules. From 2020 onwards, senior geologists have periodically reviewed the database subset used for Mineral Resource estimation, information consistency, consistency in use of designated codes, and data completeness. A detailed data validation was conducted in 2022 in support of the updated Gramalote Ridge model. Where errors or omissions were noted, these were corrected as required.

External verification included laboratory audits, and review of available QA/QC and supporting data in preparation of technical reports on the Project. No material data issues were noted as a result of these reviews.

The Qualified Persons (QPs) have verified the data in their areas of expertise and concluded that the Project data and database are acceptable for use in Mineral Resource estimation, and can be used to support conceptual mine planning.

1.9 Metallurgical Testwork

Metallurgical testwork facilities involved in the initial testwork included SGS Lakefield in Lakefield, Ontario, Canada; the SGS analytical laboratory in Santiago, Chile; ALS analytical laboratories in Santiago, Chile, and Perth, Australia; FLSmidth in Salt Lake City, Utah; Jenike and Johanson Limited (Jenike and Johanson) in Toronto, Canada; ADP Holdings (Lycopodium) in Perth, Australia, Metso Outotec Corporation (Metso Outotec) in Pennsylvania, USA; Julius Kruttschnitt Mineral Research Centre (JKMRC) in Brisbane, Australia; and the Cooperative Research Centre for Optimising Resource Extraction (CRC-ORE) in Brisbane, Australia. Laboratories and testwork facilities used were independent of B2Gold. Metallurgical laboratories are not typically accredited for testwork other than chemical analyses. Testwork completed in the early stages was used to refine the approaches taken for the feasibility study (2025 Feasibility Study).

The majority of the testwork program supporting the 2025 Feasibility Study was conducted by SGS Lakefield, with the remainder either completed at SGS Lakefield under supervision of a consultant or specialist, or at another laboratory. The program included testwork to establish:

- Comminution characteristics of the ore, using one comminution master composite and nine comminution variability samples;
- Materials handling properties for two composites;
- Mineralogy for the metallurgical master composite;
- Comprehensive head analysis of one metallurgical master composite and ten metallurgical variability samples;
- Gravity recoverable gold content for the metallurgical master composite, along with bulk gravity separation test prior to flotation;
- Optimum grind size for flotation for one metallurgical master composite;
- Optimum regrind size for flotation concentrate leach using metallurgical master composite;
- Optimum leach residence time for flotation concentrate leach using metallurgical master composite;
- Gravity, gravity concentrate leach, flotation and flotation concentrate leach performance for ten metallurgical variability samples using optimized process conditions;
- Cyanide destruction residence time and reagent doses using concentrate leach tailings produced from the metallurgical master composite;
- Engineering data, including testing of oxygen uptake, slurry rheology, carbon kinetics, regrind specific power and thickening characteristics on the products from the metallurgical master composite.

The Gramalote Ridge deposit is characterised as medium to hard competency and moderately soft to moderately hard in terms of Bond ball mill work index. The ore is considered suitable for a high-pressure grinding rolls (HPGR)–ball mill circuit and is the most energy efficient option for the competency of this ore. The mineralization is considered moderately abrasive to abrasive.

A gravity separation circuit was recommended based on testwork conducted by SGS Lakefield and modelling by FLSmidth.

Gramalote Ridge ore is amenable to gold recovery via froth flotation at a coarse grind size of P_{80} 250 µm. A flotation time (laboratory) of 20 minutes and a simple potassium amyl xanthate (PAX) and frother reagent scheme will recover approximately 3–4% of the mass and 93–98% of the gold in the flotation feed. A rougher/scavenger only flowsheet is optimum, and no cleaning is required. The flotation flowsheet design would also allow the flexibility of operation as a rougher-only circuit.

A regrind size of approximately P_{80} 20 µm is required to adequately liberate the gold prior to concentrate leach.

The concentrate leach conditions identified as achieving high gold extraction are: six hours of pre-aeration followed by 30 hours of cyanide leach at 35% solids (w/w), pH 10.5 to 11 maintained with lime, 1 kg/t lead nitrate, 2 g/L NaCN, and dissolved oxygen levels of between 20–25 mg/L.

Based on the very low levels of preg-robbing elements and very good adsorption properties, a carbon-in-pulp (CIP) circuit was recommended.

The concentrate leach tailings responded well to cyanide destruction using the SO_2/air method. The required cyanide destruction residence time is 240 minutes.

The flotation tailings have a specific thickener throughput rate of 0.035 m^2/t/day. The flotation tailings thickener will require 20 g/t of flocculant to achieve an underflow density of 55–60% (w/w).

The flotation concentrate and pre-leach thickeners have been sized on the basis of a 2 m/h rise rate due to froth concerns. The thickeners will require 50 and 55 g/t of flocculant addition, respectively. Thickener underflow densities will be 50% and 45% (w/w), respectively.

Test results indicate that overall gold recoveries in the range of 92–98% and overall silver recoveries in the range of 27–60% could be expected from the Gramalote Ridge ore when using the selected flowsheet. The average gold and silver recoveries for all the variability samples were 95.5% and 46.3% respectively, while for the master composite, recoveries were 95.7% and 54.1% respectively.

There is a logarithmic relationship between gold head grade and overall gold recovery (including gravity, gravity concentrate leach, flotation and flotation concentrate leach). There is also a logarithmic relationship between silver head grade and overall silver recovery.

The logarithmic relationship between overall gold recovery and the calculated gold head grade is shown by the following relationship:

- Overall Gold Recovery (%) = 1.7873 * ln (Gold Head Grade, g/t) + 95.526

At a gold head grade of 2.5 g/t Au, the overall gold recovery is forecast to average 97.2%.

Typically, a plant recovery discount of up to 1.0% is applied to account for soluble gold losses, fine carbon losses and plant problems which may impact on the overall recovery. However, the metallurgical testwork has not considered the additional gold that will be recovered by further leaching of the intensive cyanidation residue in the concentrate leach circuit. It is planned to introduce this residue to the regrind circuit ahead of concentrate leach. As such, no recovery discount has been applied, as it is considered that the laboratory results are suitably conservative and further optimisation may be possible.

The silver recovery relationship is shown by the following equation:

- Overall Silver Recovery (%) = 19.951 * ln (Silver Head Grade, g/t) + 37.707

At a silver head grade of 1.9 g/t Ag, the overall silver recovery is forecast to average 50.1%.

Similarly, no discount will be applied for silver.

There are no deleterious elements from the processing perspective.

1.10 Mineral Resource Estimation

The Mineral Resource model for Gramalote Ridge was built in 2022, based on drilling data collected up to March 9, 2022. Nine drill holes (1,795 m) post-date the resource estimate database cut-off. These holes were drilled for condemnation/infrastructure purposes and are generally outside of the Mineral Resource areas. As of the Report Effective Date, there has been no additional drilling at Gramalote Ridge. The Mineral Resource model for Trinidad was built in 2025, based on drilling data collected up to July 14, 2025. As of the Report Effective Date, there has been no additional drilling at Trinidad. The Mineral Resource model for Monjas West was built in 2023, based on drilling data collected up to January 4, 2023. As of the Report Effective Date, there has been no additional drilling at Monjas West. The models were constrained using pit shell assumptions provided on August 6, 2025, and the Mineral Resource estimates have an effective date of August 6, 2025.

At Gramalote Ridge, Trinidad and Monjas West, low-grade (LG) domains were created at a nominal cut-off grade of 0.1 g/t Au. In addition, at Gramalote Ridge a high-grade (HG) domain was created at a nominal 0.5 g/t Au cut-off. Consideration was given to minimum thickness, logged veining, sulphide, and structural models.

At Gramalote Ridge, bulk densities were estimated to the block model using inverse distance weighting to the second power (ID2) interpolation. The mineralized zones and weathering domains were used as boundaries to this estimate. At Trinidad and Monjas West, bulk densities were applied to the block model-based average of specific gravity samples by mineralized zone and weathering domains.

Capping levels were identified in each domain/sub-domain using distribution (probability) plots, deciles, and spatial observation of high-grade assays. Assays were capped prior to compositing.

Down-hole composites of 4 m length were created for Gramalote Ridge and Monjas West. Trinidad compositing was on 2 m intervals.

Variograms were modeled on 4 m capped composites at Gramalote Ridge and Monjas West to evaluate spatial continuity of gold mineralization and for use in gold grade estimation checks. Variograms were run on 2 m capped composites at Trinidad to evaluate spatial continuity and trends to gold mineralization and for use in gold grade estimation checks. Model variogram nuggets were adjusted downwards for ordinary kriging interpolation to better reconcile with change-of-support distributions.

At Gramalote Ridge, mineralization domain wireframes were coded to 15 x 5 x 10 m blocks with subcells down to 5 x 0.2 x 0.5 m along domain/weathering boundaries. The LG and HG mineralization domains served as a hard boundary relative to the waste domain for grade estimation. Due to the variable nature of grades at the HG/LG contact, 2 m of material uphole and downhole of a contact were shared with the adjacent domain. Search orientations were controlled by Datamine's dynamic anisotropic search. Composites were shared across the oxide (saprolite)/fresh boundary for estimation. Gold and silver grades were estimated using inverse distance weighting to the third power (ID3), ordinary kriging (OK), and nearest neighbour (NN) algorithms into parent-sized blocks. Mineral Resources are reported from the OK estimate.

Mineralization domain wireframes at Trinidad were coded to 15 x 5 x 10 m blocks with subcells down to 3 x 1 x 1 m along domain boundaries. The LG mineralization domain served as a hard boundary for grade estimation. Using 2 m capped composites, gold and silver grades were estimated into 15 x 5 x 10 m blocks. Search orientations were controlled by Datamine's dynamic anisotropic search. Composites were shared across the oxide (saprolite)/fresh boundary for estimation. Gold and silver grades were estimated using ID2, OK, and NN interpolations into parent-sized blocks. Mineral Resources are reported from the ID2 estimate.

The block model estimates were checked using comparison of different declustering (NN and cell declustering) methods; visual comparison of block grades to composites on cross sections and levels; comparison of global block statistics from different estimation techniques; swath plots to review potential local biases in the estimates; and global change of support comparisons.

Drill hole spacing for resource classification at Gramalote Ridge is as follows:

- No Measured classified;
- Indicated: 50–60 m drill hole spacing;
- Inferred: 100–120 m drill hole spacing.

Drill hole spacing for resource classification at Trinidad is as follows:

- No Measured or Indicated classified;
- Inferred: 100–120 m drill hole spacing or any drill hole within 30–40 m.

Drill hole spacing for resource classification at Monjas West is as follows:

- No Measured or Indicated classified;
- Inferred: 100–120 m drill hole spacing or any drill hole within 30 m.

Mineral Resources for Gramalote Ridge, Trinidad, and Monjas West considered potentially amenable to open pit mining methods were constrained within conceptual Lerchs–Grossmann pit shells. The calculated break-even cut-off grades are 0.13 g/t Au for oxide and range between 0.16–0.18 g/t Au for sulphide. Mineral Resources potentially amenable to open pit mining from all deposits are stated above a cut-off of 0.13 g/t Au for oxide and above 0.16 g/t Au for sulphide.

Constraints to the Gramalote Ridge pit optimization were applied around some of the planned infrastructure, specifically the plant, main haul road, and the Guacas River diversion.

1.11 Mineral Resource Statement

Indicated Mineral Resources are reported in Table 1-1, inclusive of those Indicated Mineral Resources converted to Probable Mineral Reserves. Mineral Resources that have not been converted to Mineral Reserves do not have demonstrated economic viability. Inferred Mineral Resources are provided in Table 1-2.

The QP for the estimate is Mr. Stephen Jensen, P.Geo., B2Gold's Exploration Manager for the Americas.

Table 1-1: Indicated Mineral Resource Statement

Area	Tonnes (x 1,000)	Gold Grade (g/t Au)	Contained Gold Ounces (x 1,000)	Silver Grade (g/t Ag)	Contained Silver Ounces (x 1,000)
Gramalote Ridge Oxide	4,908	0.51	81	2.12	335
Gramalote Ridge Sulphide	151,501	0.71	3,443	0.88	4,307
Total Indicated Mineral Resources	***156,409***	***0.70***	***3,524***	***0.92***	***4,642***

Table 1-2: Inferred Mineral Resource Statement

Area	Tonnes (x 1,000)	Gold Grade (g/t Au)	Contained Gold Ounces (x 1,000)	Silver Grade (g/t Ag)	Contained Silver Ounces (x 1,000)
Gramalote Ridge Oxide	730	0.43	10	0.85	20
Trinidad Oxide	9,070	0.43	124	1.05	305
Monjas West Oxide	2,285	0.53	39	0.77	56
Subtotal Oxide Inferred	*12,085*	*0.45*	*173*	*0.98*	*381*
Gramalote Ridge Sulphide	9,666	0.53	164	0.81	251
Trinidad Sulphide	80,090	0.48	1,244	0.53	1,361
Monjas West Sulphide	21,118	0.63	430	0.40	274
Subtotal Sulphide Inferred	*110,873*	*0.52*	*1,839*	*0.53*	*1,886*
Total Inferred Mineral Resources	***122,958***	***0.51***	***2,012***	***0.57***	***2,267***

Notes to accompany Mineral Resource Tables:

1. Mineral Resources have been classified using the 2014 CIM Definition Standards.
2. Mineral Resources are reported inclusive of those Mineral Resources that were converted to Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
3. The QP for the resource estimate is Stephen Jensen, P.Geo., B2Gold's Exploration Manager for the Americas.
4. Mineral Resources are reported on a 100% project basis. The Gramalote Ridge, Trinidad and Monjas West estimates have an effective date of August 6, 2025.
5. Mineral Resources for Gramalote Ridge assume metallurgical recoveries of 84% for oxide and 92.7–97.6% for sulphide, and operating cost estimates of an average mining cost of US$2.59/t mined, processing cost of US$6.13/t processed for oxide and US$9.74/t processed for sulphide, general and administrative cost of US$2.10/t processed and selling cost of $82.84/oz of Au produced.
6. Mineral Resources for Trinidad assume metallurgical recoveries of 81.7% for oxide and 90.9% for sulphide, and operating cost estimates of an average mining cost of US$2.41/t mined, processing cost of US$6.13/t processed for oxide and US$9.74/t processed for sulphide, general and administrative cost of US$2.10/t processed and selling cost of $82.84/oz of Au produced.
7. Mineral Resources for Monjas West assume metallurgical recoveries of 81.7% for oxide and 87.6% for sulphide, and operating cost estimates of an average mining cost of US$2.58/t mined, processing cost of US$6.28/t processed for oxide and US$9.89/t processed for sulphide, general and administrative cost of US$2.10/t processed and selling cost of $82.84/oz of Au produced.

8. Mineral Resources for Gramalote Ridge, Trinidad, and Monjas West are reported at cut-offs of 0.13 g/t Au for oxide and 0.16 g/t Au for sulphide.
9. All tonnage, grade and contained metal content estimates have been rounded; rounding may result in apparent summation differences between tonnes, grade, and contained metal content.

Factors that may affect the Mineral Resource estimate include: metal price and exchange rate assumptions; changes to the assumptions used to generate the gold grade cut-off grade; changes in local interpretations of mineralization geometry and continuity of mineralized zones; changes to geological and mineralization shape and geological and grade continuity assumptions; density and domain assignments; changes to geotechnical, mining and metallurgical recovery assumptions; changes to the input and design parameter assumptions that pertain to the conceptual pit constraining the estimates; and assumptions as to the continued ability to access the site, retain mineral and surface rights titles, maintain environment and other regulatory permits, and maintain the social license to explore and obtain the social license to operate.

1.12 Mineral Reserve Estimation

Mineral Reserves were converted from Indicated Mineral Resources for the Gramalote Ridge deposit only. Inferred Mineral Resources were set to waste.

The mine plan assumes open pit mining using conventional mining methods and equipment.

Pit optimizations were completed using Geovia Whittle pit optimisation software. To define the internal and final pit phases, 86 optimizations were completed, starting at US$525/oz Au and running to US$3,500/oz Au. The US$1,750/oz Au pit shell was selected as the design basis for the ultimate pit. A smaller Phase 1 was defined using the US$768/oz Au shell, allowing for a lower strip ratio and early access to higher-grade material to support rapid payback.

Only two mining phases were selected in total. Phase 1 targets medium and high-grade zones that extend from the pit center toward the eastern pit wall. This phase is designed to deliver higher-grade ore during the initial years of operation, while allowing sufficient time for the Phase 2 ultimate pit to be developed.

For Mineral Reserve reporting, an applied cut-off grade of 0.40 g/t Au is used. Only sulphide material will be processed in the mine plan. Oxide material is not included in the Mineral Reserves.

1.13 Mineral Reserve Statement

The Mineral Reserve estimate for the Project reported within the Gramalote Ridge ultimate pit design is presented in Table 1-3.

Table 1-3: Mineral Reserves Statement

Area	Tonnes (x 1,000)	Gold Grade (g/t Au)	Contained Gold Ounces (x 1,000)
Gramalote Ridge Open Pit	76,700	0.96	2,360
Total Probable Reserves	***76,700***	***0.96***	***2,360***

Notes to accompany Mineral Reserves table:

1. Mineral Reserves have been classified using the CIM Definition Standards, are reported at the point of delivery to the process plant and have an effective date of April 1, 2025.
2. Mineral Reserves are reported on a 100% project basis.
3. The QP for the Mineral Reserve estimate is Mr. Peter Montano, P.E., Vice President, Projects, B2Gold.
4. Mineral Reserves are based on a conventional open pit mining method, gold price of $1,750 per ounce, metallurgical recovery averaging 95.6%, selling costs of $60.00 per ounce including royalties, average mining cost of $2.70 per tonne mined, average processing cost of $8.50 per tonne processed, and average site general costs of $3.80 per tonne processed.
5. Reserve model dilution and ore loss was applied through whole block averaging such that at a 0.40 g/t Au cut-off there is a 1.2% increase in tonnes, a 4.6% reduction in grade, and 3.5% reduction in ounces when compared to the Mineral Resource model.
6. Mineral Reserves are reported above a cut-off grade of 0.40 g/t Au.
7. All tonnage, grade and contained metal content estimates have been rounded; rounding may result in apparent summation differences between tonnes, grade, and contained metal content.

The QP for the estimate is Mr. Peter Montano, P.E., B2Gold's Vice President, Projects. The estimate has an effective date of April 1, 2025.

Factors that may affect the Mineral Reserve estimates include: changes to the gold price assumptions; changes to pit slope and geotechnical assumptions; unforeseen dilution or ore loss; changes to hydrogeological and pit dewatering assumptions; changes to inputs to capital and operating cost estimates; changes to operating cost assumptions used in the constraining pit shell; delays or changes to the resettlement and relocation plans; changes to pit designs from those currently envisaged; changes in mining or milling productivity assumptions; and changes to modifying factor assumptions, including environmental, permitting and social licence to operate.

1.14 Mining Methods

The mining operations will use conventional open pit mining methods and equipment, using owner-operator mining equipment and labour.

Geotechnical designs are based on information collected from several years of surface mapping programs, geotechnical logging data, photo-logging observations, oriented core data, acoustic televiewer data, and results from laboratory testing to assess the intact rock strength. Bench heights are planned to be 20 m. The inter-ramp angles will vary from about 53–60° in fresh rock and will be approximately 37° in overburden and weathered rock.

Hydraulic field tests were completed to define the hydraulic characteristics of the various hydrogeological units across the Project site. These included piezometer measurements, constant discharge tests, falling head tests, and Lugeon/packer hydraulic conductivity tests. Groundwater seepage is expected at the contact of oxide and fresh rock and will be managed with a 25 m berm at this level in the pit to direct groundwater. In-pit water will be managed with mobile diesel pumps and sumps. Estimated pumping rates will range from 100–400 m^3/hour, depending on seasonal conditions.

The calculated cut-off grade for sulphide material was 0.24 g/t Au; however, an elevated cut-off grade of 0.40 g/t Au was applied for mine planning purposes.

The Gramalote Ridge deposit is planned to be mined in two phases:

- Pit Phase 1 is scheduled to begin in the second year of construction and continue through year 4 of operations, reaching a depth of 370 m. At the end of mining, pit Phase 1 will be approximately 900 m long, 500 m wide, and 370 m deep. This pit phase design contains 102 Mt, or 32% of the total LOM tonnage;
- Pit Phase 2 is planned to commence in year 3 of operations and progress to an ultimate depth of 510 m. At the end of mining, pit Phase 2 will be approximately 1,300 m long, 750 m wide, and 510 m deep. This pit phase design contains 217 Mt, or 68% of the total LOM tonnage.

Pit Phase 1 was designed to avoid the need for a diversion of Guacas Creek, which is advantageous for production planning and capital costs. However, the Guacas Creek diversion will be required before commencing pit Phase 2.

The main ramps were designed to 30 m width for two-way access, including allowances for drainage and berms and have a maximum 10% gradient. The dual ramp system will begin at an elevation of 660 m and extend up to the pit exit to be located around 820 m elevation. A double-ramp system supports >75% of the total mined ounces. Below 660 m elevation, a single ramp system is planned, consisting of a 30-m-wide ramp with a 10% slope, extending to 500 m elevation.

Pre-stripping is estimated at 13 Mt for the period prior to the first year of production. Initial ore will be stockpiled at the ROM area and later at El Balzal area. A portion of waste material moved during will be used for the construction of mine platforms and haul roads.

A maximum extraction rate of 35 Mt/year is projected, with an 11-year of mine life including the pre-stripping period. The overall strip ratio is forecast at 3.1:1 (waste:ore). Mining operations will stockpile lower-grade material when applicable, which will be processed at the end of the mine life. The first phase of the pit will be mined early in the LOM to increase the plant feed grade during the initial years of production and to maintain a relatively steady gold production profile over the LOM.

The mine planning strategy includes the use of four stockpiles:

- Fresh ore high-grade: ≥2.1 g/t Au;
- Fresh ore medium grade: 1.3–2.1 g/t Au;
- Fresh ore low-grade: 0.6–1.3 g/t Au;
- Fresh ore very low-grade: 0.4–0.6 g/t Au.

Mineralization grading 0.25–0.4 g/t Au, which is between the calculated cut-off and elevated cut-off for mine planning, is planned to be placed in the San Antonio WRF area.

The WRSF was designed to store approximately 240 Mt of waste, with additional capacity to accommodate future expansion if required. Waste rock that is classified as potentially acid generating (PAG) will be saprolite-encapsulated.

A peak fleet of four excavators, 32 haul trucks and various support equipment will be required to support the LOM plan.

1.15 Recovery Methods

The process plant design is based on a robust metallurgical flowsheet designed for optimum recovery with minimum operating costs. The flowsheet is based upon unit operations that are well proven in industry.

The key project and ore-specific criteria assumed for the plant design include:

- 6 Mt/a of ore;
- Major unit operations and equipment will be sized with a 20% design margin, with the exception of flotation concentrate leach stream equipment;
- Flotation concentrate leach stream equipment will be sized for 5% mass pull without further design margin because typical flotation concentrate mass pull is expected to be 3%;
- Process plant availability of 92% supported by crushed ore storage, stand-by equipment in critical areas and grid power supply;
- Sufficient automated plant control to minimise the need for continuous operator interface and allow manual override and control if and when required.

The plant design commences with single-stage primary crushing with a gyratory crusher to produce a crushed product size of 80% passing (P_{80}) 150 mm. The crushed ore stockpile with a nominal 21,000 t live capacity to provide 15 hours of operation at design plant throughput. During extended periods of up to three days for primary crusher equipment maintenance, ore from the dead part of the stockpile will be reclaimed by an excavator or dozer to continue feeding the downstream secondary screening and crushing circuit.

Crushed ore from the stockpile will be reclaimed by apron feeders positioned under the stockpile to feed the downstream circuit.

Closed circuit secondary crushing using duty/stand-by cone crushers will produce a crushed product size P_{80} of approximately 27 mm. Feed size preparation for a secondary crushed product is required for a grinding efficient HPGR-ball mill circuit as compared to a standard SAG mill circuit. An HPGR will be in closed circuit with wet sizing screens, with undersize slurry reporting to the milling circuit via a cyclone feed pumpbox. A ball mill in closed circuit with hydrocyclones will produce a P_{80} grind size of 250 µm. The cyclone overflow stream will flow by gravity to the flotation circuit.

A gravity gold recovery circuit was included based on the metallurgical testwork results and a modelling optimization study. The circuit will contain two centrifugal gravity concentrators and an intensive cyanidation leach reactor.

The flotation circuit will have five flotation cells arranged in series and will use a simple collector (PAX) and frother reagent addition scheme. The flotation cells can be operated in rougher only, or a rougher/scavenger flowsheet, in the event that higher concentrate grades or lower mass recoveries need to be targeted. However, it is anticipated that the rougher-only flowsheet will be the primary operating method.

The flotation concentrate will be pumped to a flotation concentrate thickener ahead of a regrind circuit. The thickener provides a buffer between the primary grinding and flotation circuits and the regrind mill. It will also provide control of the water balance around the regrind circuit and help to break down froth that may otherwise be an issue in the regrind circuit.

The regrind circuit will include a trash screen to remove any trash or rubbish prior to milling. The regrind mill will be in closed circuit with hydrocyclones. The cyclone overflow product size of P_{80} 20 µm is expected to be achieved with a low cyclone overflow slurry density of 15% solids (w/w) to promote better classification efficiency. The cyclone overflow stream will flow by gravity to the pre-leach thickener.

A pre-leach thickener for increasing the slurry density of the finely ground flotation concentrate stream to the leach circuit will be required to minimise slurry tank volume requirements and reduce overall reagent consumption.

The pre-aeration and leach circuit will consist of six tanks for 6 hours of pre-aeration and 30 hours of leaching residence time at design plant throughput. Slaked lime slurry will be added to the pre-aeration and leach tanks for pH adjustment. Sodium cyanide solution is added to start the gold leaching process in the presence of elevated dissolved oxygen levels and lead nitrate solution.

A carbon-in-pulp (CIP) circuit will consist of six stages of carbon adsorption for recovery of gold dissolved in the leaching circuit. A Pressure Zadra elution circuit will be used to recover gold to doré. The circuit will include an acid wash column to remove inorganic foulants from the carbon with hydrochloric acid.

A carbon regeneration kiln is included to remove organic foulants including any residual flotation circuit reagents, from the carbon with heat.

A cyanide destruction circuit using SO_2 and air will be used to reduce the weakly-acid dissociable cyanide concentration in the flotation concentrate leach tailings discharge stream to an environmentally acceptable level.

A flotation tailings thickener is included in the design to increase slurry density for water recovery prior to tailings discharge to the tailings management facility (TMF).

The power demand for the processing plant will be provided by the national grid supply. The plant will require about 238,649,287 kW/a.

The process plant will use process water, leach water, reclaim water, fresh water, treated water, gland water, and potable water. Make-up water requirements for process water and leach water will be pumped from the reclaim water pond, which will receive water recovered from the tailings decant pumps at the TMF. Water from the thickeners overflow streams will be recycled within the process plant to reduce the external water requirement. Approximately 1,200 m^3/h of decant return water will be recycled from the TMF to the process plant. Another approximately 56 m^3/h of fresh water will be required to make-up the water consumption for the process plant. Fresh water will be sourced from the Guacas Creek.

The major reagents to be used within the process plant will include slaked lime, PAX collector, frother, sodium cyanide, lead nitrate, caustic soda, hydrochloric acid, sodium metabisulphite, copper sulphate pentahydrate, flocculant, anti-scalant, fluxes, and diesel fuel.

1.16 Project Infrastructure

Surface infrastructure to support operations as envisaged in the 2025 Feasibility Study includes:

- One open pit; to be mined in two phases;
- Waste rock storage facility, stockpiles, and water diversions, including Guacas Creek diversion;
- Process plant: grinding, flotation and leaching facilities, management and engineering offices, change house, workshop, warehouse, and electrical grid switchgear substation;

- Mine facilities: heavy equipment workshop, warehouse and laydown areas, fuel storage, fuel distribution bays, wash bay, tire bay, welding shop, assay laboratory, change house, cafeteria, explosive magazine, explosive transference plant, and concrete batch plant;
- Internal mine roads, including haul roads;
- A community bypass road to the Cristales village;
- Mine entrance and camp accommodation;
- Sediment control ponds;
- Quarry borrow pits;
- Water intake on Guacas Creek and associated water treatment plant;
- Tailings management facility;
- Utilities including potable and waste water systems, fire protection, communications, and natural gas.

The TMF will be located in the Palestina Valley, and is designed to store 76.7 Mt of tailings over a 13-year processing period. The valley can accommodate a larger facility if required for any potential future mine expansion or mine life extension. A larger-capacity TMF of 220 Mt capacity was approved in 2016, and that approval is part of the current Environmental Impact Assessment (EIA). The TMF will be the primary reservoir, and will have the capacity to manage the probable maximum flood at all times.

The TMF will store both flotation and leach circuit cyanide destruction tailing streams, which will be separately handled and deposited. The tailings material contains a high percentage of a sand-forming coarse fraction, which is planned to be placed directly downstream of the starter dam, creating a large sand dam.

The Project will have a positive water balance that will require discharge due to the large volumes of surface run-off that will be collected primarily in the TMF. Water quality models currently indicate that water discharges will meet all Colombian regulatory standards. Water control reservoirs, seepage and sedimentation ponds will be located in different catchment areas to manage sediment discharges, storm events and contact water, including from the open pit, TMF, and WRSFs.

Fresh water will be sourced from Guacas Creek, and will form the potable water supply.

A diversion channel is required for Guacas Creek as the later stages of the open pit will impact the creek channel. Water will be diverted from the current creek course via a large embankment, into the new channel.

The planned new approximately 200-person accommodations camp will be located adjacent the administration offices, and will support construction and operations. The existing camp, 20 minutes from the Project area, will provide additional accommodations flexibility, with a 70-bed capacity. A significant portion of the construction workforce is expected to commute from nearby communities, thereby maximizing local opportunities for lodging and transportation services.

Power is planned to come via pipeline from gas-based self-generation, leveraging off the Project proximity of 2 km to the main gas pipeline that supplies the Medellín region. The 2025 Feasibility Study assumes that a take-or-pay agreement will be negotiated with the pipeline owner for the electricity supply. The agreement envisages that the pipeline owner will provide the pipeline to site, and construct the on-site power station.

The power plant will consist of four gas-powered engines that will have a total installed capacity of about 44 MW, sufficient to meet peak Project demands of about 30 MW. Three engines will be in operation, with one on stand-by. The plant will include load-shedding abilities.

1.17 Environmental, Permitting and Social Considerations

1.17.1 Environmental Considerations

The Terms of Reference for the Project EIA were issued by the Autoridad Nacional de Licencias Ambientales (ANLA) in 2012. The EIA was submitted in early 2015, and the Environmental License was granted later that year. A Modified EIA (MEIA) was completed in 2019, and further modifications to the Environmental License were approved in 2019 and 2024. An updated MEIA reflecting the 2025 Feasibility Study is planned for submission.

Baseline data collection began in 2011 and continues to be updated to support the MEIA, with the support of environmental and social specialists. Studies include geology, soils, water, biodiversity, climate, and socio-economic factors.

The Environmental License establishes specific management and monitoring programs to prevent, mitigate, or correct environmental and social impacts. These programs cover all phases of the Project’s life of mine (LOM), including planning, construction, operation, and closure.

B2Gold is also implementing compensation programs to address biodiversity loss, impacts to woody vegetation and threatened species, and changes in land use. These programs include conservation actions such as creating or expanding protected areas, establishing voluntary conservation agreements, and undertaking ecological restoration. Water management programs are in place to preserve watersheds and support sustainable water use.

1.17.2 Closure and Reclamation Planning

ANLA has approved a conceptual closure plan (Closure Plan) for the Project under the current Environmental License that includes a strategy to effectively and progressively rehabilitate areas that have been affected by Project activities. The conceptual Closure Plan will be modified and updated periodically throughout the Project life to reflect significant changes in the design, construction, operation, and closure phases.

B2Gold has estimated the cost of environmental reclamation and closure liabilities for the Project at approximately US$51.2 million. In accordance with the Colombian Mining Code (Law 685 of 2001), a company is required to provide financial guarantees to ensure the Closure Plan can be fully implemented, to cover the costs associated with environmental remediation, and to meet applicable social obligations.

1.17.3 Permitting Considerations

B2Gold was granted an Environmental Licence through Resolutions 1514 (2015), 309 (2016), and 00782 (2019). This license sets out the conditions and requirements that must be met to support Project development, operation, and closure. B2Gold will submit an updated MEIA for approval to reflect Project changes in accordance with Colombian legislation.

B2Gold holds a number of other key permits, however, several additional permits and authorizations will be required for the Project, including:

- Authorization for acquisition and use of explosives;
- Certificates for controlled chemical substances and products;
- Sanitary authorization for potable water supply;
- Construction licenses;
- Technical Regulations for Electrical Installations (RETIE);
- Relocation of existing powerlines.

1.17.4 Social Considerations

The Project's socio-economic baseline studies, conducted as part of the 2015 EIA and 2019 MEIA and updated through 2025, assessed potential impacts on communities within the Project's direct and indirect areas of influence. Community engagement was carried out throughout the EIA and MEIA processes, including over 100 stakeholder meetings, and will continue as part of the updated MEIA. Management and monitoring programs have been developed to prevent, mitigate, and address potential social impacts, covering areas such as community participation, health, education, cultural heritage, land access, and institutional support.

Project development requires land acquisition and the physical or economic resettlement of affected households. B2Gold has prepared a Resettlement Action Plan (RAP) in compliance with Colombian legislation, Environmental License obligations, and international standards. In total, 327 households have been identified as impacted, with 225 to be physically relocated and 102 economically displaced. As of the Report Effective Date, 98% of households have signed individual agreements, with ongoing engagement for the remaining households. B2Gold has acquired land, completed housing and infrastructure designs, and is implementing livelihood restoration, community integration, and public service measures, with quarterly reporting to ANLA until completion.

Artisanal and small-scale mining (ASM) is a traditional economic activity in the area, but poses environmental and social risks such as land and water degradation, child labour, and informal employment. B2Gold has monitored ASM since 2011, with approximately 500 miners active in 2025, half of whom operate within the Project footprint. To mitigate risks and support communities, B2Gold has formalized 73 mining production units under 22 mining associations, benefitting over 180 miners and creating more than 560 formal jobs with social security. Ongoing initiatives include formalization support, association development, EIA preparation assistance, and technical training, with expansion to new areas being considered.

1.18 Markets and Contracts

No market studies have been completed. The doré that will be produced by the mine is readily marketable. Shipping and refining costs are estimated to be US$3.00/oz for gold and US$1.34/oz for silver.

Commodity prices used in Mineral Resource and Reserve estimates are set by B2Gold at a corporate level. The gold price provided for Mineral Reserve estimation is US$1,750/oz, and US$2,500/oz was used for Mineral Resource estimation. The financial model assumes a long-term gold price of US$2,500/oz.

Quotes for major contracts were received to support the 2025 Feasibility Study. Major contracts will include blasting explosives and accessories through Indumil (the only option for explosives under Colombian law), fuel provision, power provision, tire services, contacts related to infrastructure construction, and other mining and processing consumable contracts as needed. Contracts will be negotiated and renewed as necessary with terms expected to align with industry standards and comparable to similar contracts previously executed byB2Gold.

1.19 Capital Cost Estimates

The LOM plan assumes owner-operated mine operations with contractor support in select areas. The construction period is assumed to be two years, scheduled as 2027 and 2028 in this Report, pending permitting.

Capital and operating costs are estimated in either Colombian Pesos or United States dollars depending on the category and are converted at exchange rate of 4,200 COP:US$ where applicable.

Capital costs consist largely of processing facilities, site infrastructure, mining equipment and rebuilds, and resettlement and land purchases. Capital costs are split into:

- Sustaining capital: costs that support the existing LOM plan;
- Non-sustaining capital: costs are for long-term structures or external project that do not necessarily depend on the mine plan. Non-sustaining capital includes camp, roads, initial mining equipment fleet, mine services area and workshops, process plant, tailings storage facility, resettlement, and other major infrastructure development required for mining and processing operations.

A construction capital cost of US$740.1 M is estimated prior to commercial production. A post-construction capital cost of US$67.7 M is estimated after commercial production has begun.

Total mining fleet and infrastructure capital is estimated to be US$140.5 M over the LOM. After construction, US$149.5 M of capitalized waste is included in mining capital costs. The total mining capital cost after construction is estimated to be US$289.9 M over the LOM.

Process capital costs, after construction, are estimated at US$46.5 M over the LOM.

Total site general capital costs are estimated to be US$6.0 M over the LOM.

The total reclamation and closure capital cost is estimated at US$51.2 M.

The overall capital cost estimate is summarized in Table 1-4 and is estimated at US$1,236 M.

1.20 Operating Cost Estimates

Department costs are estimated independently. Some departments are treated as distributable costs, such as power generation and heavy equipment maintenance, and are allocated to other departments based on usage.

Mine operating costs are estimated at US$2.71/t rock mined including capitalized waste.

Table 1-4: Capital Cost Estimate Summary

Area	Sub-Area	Units	Value
Non-sustaining capital	Mining and related infrastructure	$ M	87
	Waste pre-strip and ore stockpile	$ M	42
	Processing	$ M	313
	Site general infrastructure	$ M	186
	Resettlement and land purchase	$ M	32
	Contingency	$ M	81
	Sub-total construction (all site development prior to commercial production)	*$ M*	*740*
	Post-production construction (all site development after commercial production starts)	$ M	68
	Sub-total non-sustaining capital	*$ M*	*808*
Sustaining capital	Mining - capitalized waste	$ M	149
	Mining - excluding capitalized waste	$ M	140
	Processing	$ M	46
	Site general	$ M	6
	Contingency	$ M	34
	Sub-total sustaining capital	*$ M*	*376*
Closure capital	Closure costs	$ M	51
Total All Capital Costs		***$ M***	***1,236***

Note: Totals may not sum due to rounding.

Stockpile and ore rehandle costs are included with the processing costs. The total process operating cost is estimated to be US$8.16/t milled over the life of processing operations.

The total site general operating cost is estimated at US$3.64/t processed over the life of operations.

The estimated LOM plan operating costs are presented in Table 1-5 and Table 1-6.

Table 1-5: LOM Operating Cost Totals

Area	Units	Value
Mining costs	$ M	823
Capitalized waste	$ M	-149
Processing	$ M	626
Site general	$ M	279
Power plant	$ M	0.4
Change in stockpiled ore	$ M	23
Silver sales/credit	$ M	-29
Dore transportation, security, insurance	$ M	3
Refinery charge	$ M	7
Total operating costs	***$ M***	**1,583**

Notes:

1. Distributable costs are included in the area totals. Totals may not sum due to rounding.

2. Operating cost totals are reported from production years only and exclude the estimated construction period.

Table 1-6: LOM Unit Operating Costs (Ore Processed)

Area	Ore Processed (US$/t)	Gold Produced (US$/oz Au)
Mining (all areas)	11.27	382.58
Processing	8.16	276.95
Site general	3.64	123.63
Total	***23.08***	***783.16***

Note: Mining costs are US$2.71/t mined for mining including capitalized waste. Processing costs include stockpile rehandle and ore haulage where applicable. Totals may not sum due to rounding.

Operating costs total US$1,583 M over the LOM. Mining costs will average US$11.27/t ore processed, process costs will average US$8.16/t ore processed and site general costs will average US$3.64/t ore processed, for an overall ore processed cost of US$23.08/t.

LOM plan operating cost estimates total US$783.16/oz Au produced, or US$23.08/t processed.

1.21 Economic Analysis

1.21.1 Forward Looking Information Statement

Identification of information that is forward-looking is included in the statement at the front of this Report.

1.21.2 Methodology Used

The financial model that supports the mineral reserve declaration in a standalone model that calculates annual cash flows based on scheduled ore production, assumed processing recoveries, metal sale prices, exchange rate of 4,200 COP/US$, projected operating and capital costs, and estimated taxes.

The financial analysis is based on an after-tax discount rate of 5%. All costs and prices are in unescalated real dollars.

All costs are based on the forecast costs for the proposed Gramalote Ridge Mine. Revenue is calculated from the recoverable metals and long-term metal prices described in Section 19.2, and exchange rate forecasts.

1.21.3 Assumptions

Royalites average 3.2% over the LOM.

The economic analysis is based on 100% equity financing and is reported on a 100% project ownership basis. The economic analysis assumes constant prices with no inflationary adjustments, and a long-term gold price of US$2,500/oz.

In Colombia, gold miners pay a royalty to the State of 4% on the “mine-head” value of extracted gold, which is calculated at 80% of the international gold price, resulting in an effective royalty rate of about 3.2% of the market value. The international gold price used as a benchmark is based on daily spot prices published by recognized financial sources such as the London Bullion Market Association (LBMA) or other widely accepted international commodities exchanges.

Additionally, gold mining companies are subject to a 35% corporate income tax, and while royalties were initially non-deductible under Colombia's 2022 tax reform, a 2024 Constitutional Court ruling restored their deductibility, allowing miners to subtract royalty payments from their taxable income.

1.21.4 Economic Analysis

The valuation date is January 1, 2027, which is modelled to be the date of a construction decision, pending permitting. A discount rate of 5% is used. The after-tax project NPV is US$941 M. The internal rate of return is calculated to be 22.4%, with a payback period of 3.4 years after production start.

A summary of the financial results is provided in Table 1-7.

1.21.5 Sensitivity Analysis

Changes in metal price, gold grade, capital costs and operating cost assumptions were tested using a range of 25% above and below the base case values. Results are shown in Figure 1-1. The sensitivity to gold grade is not shown as it mirrored the gold price sensitivity within this sensitivity range.

The Project is most sensitive to changes in the gold price and grade, and less sensitive to changes in operating costs and capital costs, which exhibit similar sensitivity.

1.22 Risks

Permit timing assumes that all key permits will be obtained by 2027 to support Project construction. Delays in permits being granted may affect the assumed dates in this Report for construction and operations.

Resettlement is assumed to be completed as the Project progresses. Delays in the resettlement process may affect the assumed dates in this Report for construction and operations.

The Project design basis incorporates the regulatory requirements that were current at the Report Effective Date. Updates or changes to the regulations after the Report Effective Date may require additional controls to be implemented.

The continued economic uncertainty with the current inflation and tariffs could affect the assumptions related to capital and operating cost estimates and therefore affect the economic analysis in this Report.

The formalization of the ASM activities, identification of designated ASM locations, and the management of potential ASM-related impacts in the Project area are important due to their potential effects on site access and development, local security, and receiving environment.

1.23 Opportunities

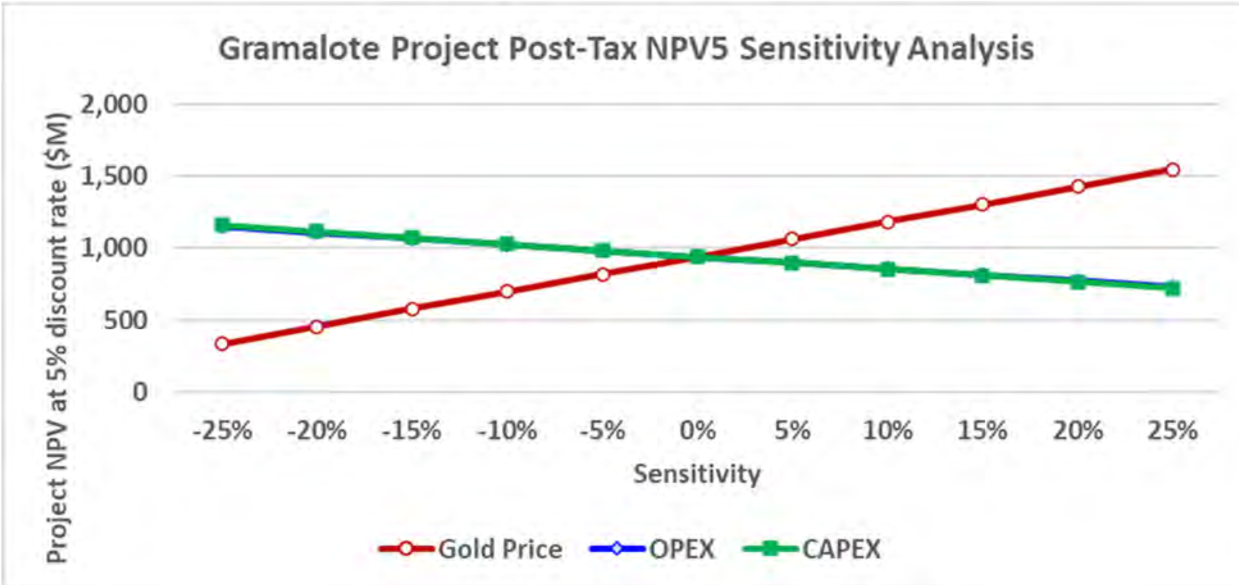
The Mineral Resources at the Trinidad and Monjas deposits represent upside opportunity if the Indicated Mineral Resources can be converted to Mineral Reserves and included in an updated mine plan with additional studies.

Table 1-7: Key Financial Metrics

Item	Units	Value
Production Profile		
Contained gold ounces processed	Moz	2.36
Gold recovery	%	95.7
Average gold grade	g/t	0.96
Gold ounces produced	Moz	2.26
Average annual gold production	koz/yr	177
Mine life	Years	11.0
Mill life	Years	13.0
Ore tonnes processed	Mt	76.7
Waste material mined	Mt	241
Waste to ore strip ratio	Waste: ore	3.1
Project economics - $2,500 /oz project average gold price		
Non-sustaining capital - construction	$ M	740
Non-sustaining capital - post construction	$ M	68
Sustaining capital	$ M	376
External projects, reclamation, and closure capital	$ M	51
Gross gold revenue	$ M	5,650
Net cash flow	$ M	1,630
$NPV_{5.0\%}$	$ M	941
IRR	%	22.4
Payback period after production start	years	3.4
Unit Operating Costs		
LOM cash operating costs (mining, processing, and site general)	$/oz Au	700
LOM AISC	$/oz Au	985
Average LOM mining cost	$/t rock mined	2.71
Average LOM processing cost	$/t processed	8.16

NPV = net present value; IRR = internal rate of return; AISC = all in sustaining costs. AISC per ounce is the sum of cash operating costs, royalties and production taxes, capital expenditures that are sustaining in nature, corporate general and administrative costs, community relations expenditures, all divided by the total gold ounces sold to arrive at a per ounce figure. AISC is both $/oz sold and $/oz produced as there is no timing delay because ounces are produced and sold in the same period.

Figure 1-1: Sensitivity Analysis



Note: Figure prepared by B2Gold, 2025. Opex – operating cost estimate; Capex = capital cost estimate.

There is upside potential for the estimates if mineralization that is currently classified as Inferred can be upgraded to higher-confidence Mineral Resource categories.

The numerous undrilled prospects discussed in Section 9 represent upside exploration potential. Drilling is required to test that potential.

1.24 Interpretation and Conclusions

Under the assumptions in this Report, the Project shows a positive cash flow over the LOM and supports the Mineral Reserve estimates. The mine plan is achievable under the set of assumptions and parameters used.

1.25 Recommendations

As the 2025 Feasibility Study shows a project with positive economics, and a construction decision is at the discretion and purview of B2Gold's Board of Directors, the QPs have no meaningful recommendations to make.

2.0 INTRODUCTION

2.1 Introduction

Mr. Stephen Jensen, P.Geo., Mr. Peter Montano, P.E., Mr. John Rajala, P.E. and Mr. Ken Jones, P.E., prepared a NI 43-101 Technical Report (the Report) on the Gramalote Gold Project (the Project) for B2Gold Corp. (B2Gold). The Project is located within the department of Antioquia in northwestern Colombia (Figure 2-1).

Gramalote Limited is registered in Colombia as Gramalote Colombia Limited (Gramalote Colombia) and is the operating entity of the Project. Gramalote Limited is a wholly-owned subsidiary of B2Gold.

2.2 Terms of Reference

The Report was prepared to support the news release entitled "B2Gold Announces Positive Feasibility Study Results for the Gramalote Project" filed on July 14, 2025.

The Report includes Mineral Resource estimates for the Gramalote Ridge, Trinidad, and Monjas West deposits and Mineral Reserve estimates for the Gramalote Ridge deposit.

Units used in the report are metric units unless otherwise noted. Monetary units are in United States dollars (US$) unless otherwise stated. The Report uses Canadian English. Mineral Resources are reported in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (May 2014; the 2014 CIM Definition Standards).

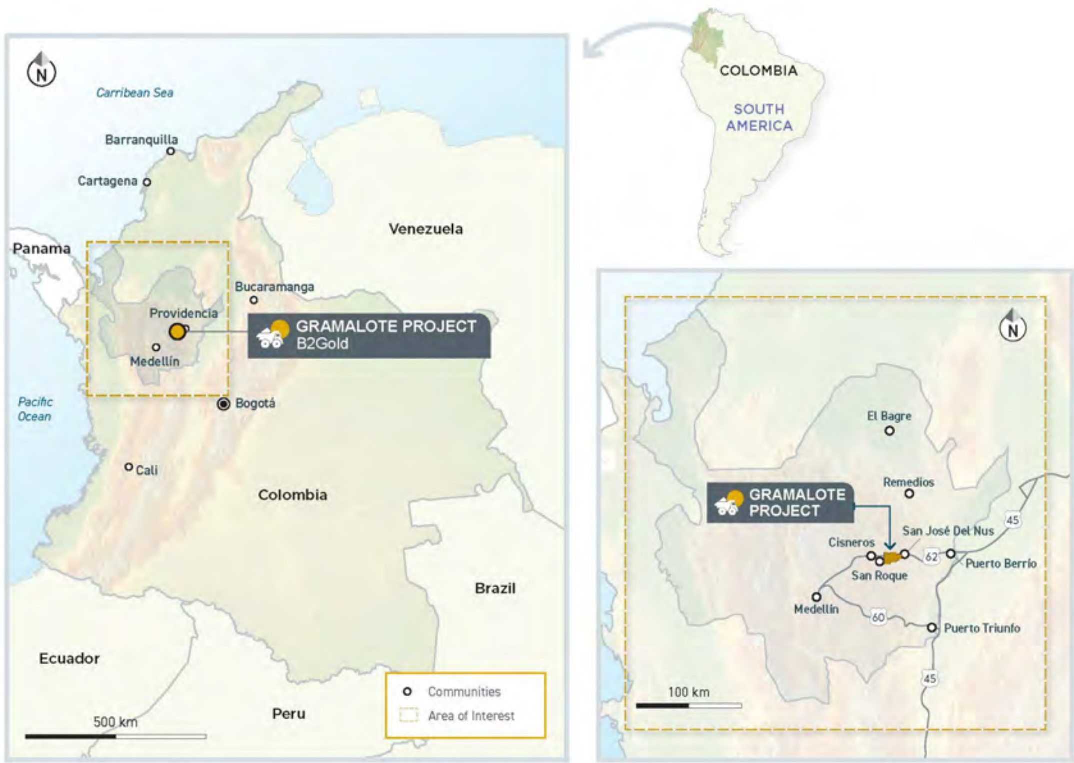
In early Project reports, the Gramalote Ridge deposit was also referred to as the Gramalote Central deposit. The current preferred nomenclature is Gramalote Ridge.

2.3 Qualified Persons

The following B2Gold employees serve as the QPs for this Report as defined in National Instrument 43-101, Standards of Disclosure for Mineral Projects, and in compliance with Form 43-101F1:

- Mr. Stephen Jensen, P.Geo., Exploration Manager, Americas;
- Mr. Peter Montano, P.E., Vice President, Projects;
- Mr. John Rajala, P.E., Vice President, Metallurgy;
- Mr. Ken Jones, P.E., Director, Sustainability.

Figure 2-1: Location Plan



Note: Figure prepared by B2Gold, 2025.

2.4 Site Visits and Scope of Personal Inspection

2.4.1 Mr. Stephen Jensen

Mr. Stephen Jensen has visited the Gramalote Project site on a continuous basis since June 2008 with the most recent visit on July 4, 2025. During site visits he observed active drill sites, confirmed exploration progress, and reviewed logging and sampling procedures. Discussions on geology, mineralization, and resources were held with B2Gold, Gramalote Colombia and during the period of the joint venture with AngloGold Ashanti Colombia S.A. (AngloGold Ashanti), with AngloGold Ashanti staff.

2.4.2 Mr. Peter Montano

Mr. Peter Montano visited the Gramalote Project site on October 24, 2024. While on site, he inspected proposed infrastructure locations, including the open pit, process plant, stockpiles, El Balzal waste rock storage facility (WRSF), and tailings storage facility (TSF).

2.4.3 Mr. John Rajala

Mr. John Rajala visited the Gramalote Project site from October 4-6, 2011. Mr. Rajala viewed core in the core shed and toured the site, focusing on the general topography as well as potential infrastructure sites.

2.4.4 Mr. Ken Jones

Mr. Ken Jones visited the Gramalote Project site from August 19-20, 2019 and again from September 29 to October 7, 2021.

During the 2021 site visit, Mr. Jones observed the main proposed infrastructure sites including the process plant, TMF and pit, visited artisanal mining sites within the project tenement, and inspected site installations present at La Mayoria at that time. During his time in Colombia working with the Gramalote Sustainability team he also discussed the Colombian regulatory framework, the project permitting history including obligations of the Environmental License, and reviewed existing and proposed environmental and social monitoring programs.

2.5 Effective Dates

There are a number of effective dates pertinent to the Report, as follows:

- Database close-out date for estimation:
  - Gramalote Ridge: March 9, 2022;
  - Trinidad: July 14, 2025;
  - Monjas West: January 4, 2023;
- Date of Mineral Resource models:
  - Gramalote Ridge: May 11, 2022;
  - Trinidad: August 6, 2025;
  - Monjas West: February 7, 2023;
- Date of Mineral Resource estimates: August 6, 2025;
- Date of Mineral Reserve estimate: April 1, 2025;
- Date of the economic analysis supporting the Mineral Reserves: July 14, 2025.

The overall Report Effective Date is July 14, 2025.

2.6 Information Sources and References

Reports and documents listed in Section 27 of this Report were used to support preparation of the Report. Additional information was provided by B2Gold and Gramalote Colombia personnel as requested.

2.7 Previous Technical Reports

B2Gold has filed the following technical reports on the Project:

- Gorham, J.P., 2008, Updated Report on the Gramalote Property, Department of Antioquia, Colombia, 43-101 Technical Report prepared for B2Gold Corp., June 2008.
- Meister, S.N., 2009: Gramalote Ridge Project, Department of Antioquia, Colombia, NI 43-101 Technical Report: report prepared for B2Gold, effective date 27 February 2009;
- Hulse, D.E., 2012: NI 43-101 Technical Report on Resources, Gramalote Project, Providencia, Colombia: Report prepared by Gustavson Associates for B2Gold, effective date 1 June, 2012;
- Hulse, D.E., Sobering, G., Newton, M.C. III, Malhotra, D., and Daviess, F., 2014: NI 43-101 Preliminary Economic Assessment, Gramalote Project, Northwest Colombia: report prepared by Gustavson Associates for B2Gold, effective date 1 February, 2014;
- Garagan, T., Pemberton, K., Rajala, J., and Jones, K., 2019: Gramalote Project, Colombia, NI 43-101 Technical Report: report prepared for B2Gold, effective date 31 December 2019.

3.0 RELIANCE ON OTHER EXPERTS

This section is not relevant to this Report.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Introduction

The Project is located approximately 200 km directly northwest of the Colombian capital of Bogota, a distance of approximately 408 km by road, and 100 km northeast of Medellin, the regional capital of the Department of Antioquia.

The geographic center of the Project is located at UTM Zone 18N (WGS84): 509,180 East; 720,330 North (6°31' N, 74°55' W).

4.2 Property and Title in Colombia

Information in this subsection is summarized from Ruiz (2019), Zapata and Villada (2016), and the Colombian National Mining Agency (2019).

4.2.1 Mineral Title

The regulatory framework for mining activities in Colombia is outlined in the country's constitution and Law 685/2001, the Mining Code. The Ministry of Mining and Energy issues government policies in relation to mining sector management. Sub-departments include the National Mining Agency (NMA), which is in charge of mineral titles, the Colombian Geological Service, which conducts scientific research, and the Mining and Energy Planning Unit, which addresses policies and development.

The Colombian State owns mineral resources in the soil and subsoil. Mining exploration and exploitation rights are granted to private parties (including foreigners) through mining concessions registered in the National Mining Registry.

The only mineral title under the 2001 Mining Act is a Mining Concession, which is granted for a 30-year term. There is no automatic extension of term; and any extensions must be negotiated with the NMA. The NMA may impose additional terms and conditions on the extended licence, or may refuse extension grant.

Mineral titles granted under the 1988 Mining Act include contribution agreements, exploration licences, exploitation licences, and concession agreements, which have different terms, rights and obligations to the Mining Concessions under the 2001 Mining Act. Unless a request has been made to govern a particular licence under the 2001 Mining Act, these 1988 Mining Act tenures remain in force, and are governed by the 1988 Mining Act.

Mining activities are divided into three phases for the purposes of Mining Concession administration: Exploration, construction and installation, and exploitation.

Mining is considered to be a public-interest activity, which in practice gives the mining title-holder the ability to request expropriation over properties needed for mining project development. The mining title-holder also has the ability to request imposition easements over properties located outside or inside the area covered by the mining title. Easements are generally established for the same granted term as the Mining Concession.

Annual surface fees are payable during the exploration, and construction and assembly phases. These are calculated based on the Mining Concession area, and current minimum monthly legal wages. Additional fees can apply to mining titles granted under the 1988 Mining Code, or as negotiated with the NMA with concession term extensions.

Indigenous and African-Colombian communities have a preferential right to obtain mining concessions over mining deposits located in areas in which Indigenous and African-Colombian communities live.

4.2.2 Work and Construction Plan (Plan de Trabajos y Obras)

The Work and Construction Plan (Plan de Trabajos y Obras or PTO) is a mandatory technical and operational document that the mining titleholder must prepare and present to the NMA prior to starting any mining activity within the area granted under a mining concession contract. The mining concession contract is the legal instrument that gives a company the right to explore and exploit mineral resources in a defined area. The PTO describes the technical, environmental, safety, social, and production plans for the exploitation phase, and must be reviewed and approved by the NMA. Without an approved PTO, a mining titleholder cannot legally begin extraction activities.

4.2.3 Mining Management Integral System (ANNA Minera)

In November 2019, the NMA launched a new mining cadastral system, the Mining Management Integral System (ANNA Minera), which changed the Colombian Mining Cadaster. The system uses grid cell references for all mining claims, and eliminates the apparent gaps, slivers, fractions, and errors in claim boundary locations ("corridors") that can appear between adjacent claims under the former system. The Gramalote Project claims have been transferred to the new system.

4.2.4 Surface Rights

Surface rights are independent of mining rights. Companies are expected to acquire necessary mining easements, direct surface rights, and conduct surface agreements in support of planned mining operations.

Approximately 40% of all land in Colombia has registered property titles. All land in Colombia must be registered with the Land Registry Office, and each plot has a registration number kept at the cadaster of the Land Registry Office.

4.2.5 Royalties

Once in production, State royalties on the gold and silver are 4% of the gross metal value at the plant site (as per Article 16 of Law 141 in 1994). However, gross metal value is determined by using 80% of the spot prices for gold and silver on the London Metals Exchange, so the effective royalty rate is 3.2%.

These State royalties are independent of national, departmental, and municipal taxes.

4.2.6 Environmental

The Ministry of Environment and Sustainable Development, National Authority for Environmental Licences (ANLA), and regional environmental authorities are responsible for environmental permits for mining construction and exploitation phases, and for any other environmental permits required during the mining title term.

Projects and activities that may severely affect natural resources require environmental authorization in the form of an environmental licence. Depending on the mining project size, the Environmental License will be granted either by the Environmental Licence Agency or by a Regional Environmental Authority, which are independent and autonomous organizations working in the different regions of the country and are in charge of enforcing environmental regulations on mining projects within their zones.

Under Decree 1076 of 2015, an environmental licence is not currently required during the exploration phase, but is required during the construction, installation, and exploitation phases.

An EIA is required under Law 685 of 2001, Law 99 of 1993 and 1076 of 2015, and must meet the provisions of the General Methodology for Presentation of Environmental Assessments and the specific Terms of Reference. There is a provision to amend the EIA through a Modification of the Environmental License (MEIA).

A mining titleholder is required to obtain a mining and environmental insurance policy, which must be in force during the entire project period. The insurance requirement is that 5% of the planned annual exploration expenditures during the exploration phase, and 5% of the planned investment for the assembly and construction phase must be covered. The exploitation phase requires a policy that covers 10% of the total amount when the estimated annual production is multiplied by the mine pit price of the extracted mineral, as established by the Colombian Government. The insurance policy has to be in place for the duration of the Mining Concession term, any term of extension, and for a further three years post termination of any such concession.

There are no specific regulations relating to mine closure and remediation. Closure and remediation obligations are set out in the environmental licence and on a case-by-case basis, depending on the type of mine, mineral and location.

4.3 Project Ownership

In July 2005, Sociedad Kedahda S.A. (Kedahda; (now AngloGold Ashanti Colombia S.A), entered into an option agreement with the Colombian-based Grupo Nus to earn a 75% interest in the Gramalote Project. In October 2007, B2Gold purchased the rights to the Grupo Nus option agreement including the remaining 25% interest, held by Grupo Nus, in the Gramalote Project.

On May 15, 2008, B2Gold entered into an agreement with AngloGold Ashanti, whereby AngloGold Ashanti transferred to B2Gold a 2% interest in the Gramalote JV and assigned to B2Gold other rights relating to Gramalote Colombia, including AngloGold Ashanti's right to acquire an additional 24% interest, so that B2Gold then held a 51% interest in the Gramalote JV (AngloGold Ashanti retaining 49%) and accepted responsibility for management of exploration of the Gramalote Project. In addition, AngloGold Ashanti transferred its interests in additional claims contiguous to the original Gramalote property to the Gramalote JV.

In 2010, AngloGold Ashanti agreed with B2Gold to amend the Shareholder's Agreement. Under the amended terms, AngloGold Ashanti regained its 51% interest in the JV and became manager (operator) of the Project while B2Gold retained a 49% interest.

In 2017, Gramalote Colombia and AngloGold Ashanti completed a detailed Project assessment. Based on marginal Project economics reported to B2Gold by Gramalote Colombia and AngloGold Ashanti, B2Gold elected to only fund US$5.0 M of the 2018 Gramalote Project expenditures, resulting in a reduced interest in the Project to 48.3%.

In late 2018 and early 2019, new resource interpretations and economic analysis indicated that the Project had the potential to become more robust. On December 23, 2019, B2Gold entered into an amended and restated agreement with AngloGold Ashanti with respect to the Gramalote Project such that the Project became a 50:50 joint venture. On January 1, 2020, B2Gold assumed the role of the operator of the Gramalote Project.

On October 5, 2023, B2Gold acquired the remaining 50% interest of the Gramalote Project owned by AngloGold Ashanti, resulting in B2Gold owning 100% of the Gramalote Project.

4.4 Mineral Tenure

B2Gold, through Gramalote Colombia, holds 11,008.26 ha in three registered concession contracts, namely integrated mining title 14292, totalling 8,720.71 ha (referred to as the Gramalote Ridge title), concession title 4894, totalling 2,277.77 ha (referred to as the Trinidad title) and concession title QHQ-16081, totalling 9.78 ha. In addition, there is an application for mineral title, LJC-08012, which has a total area of 94.14 ha.

A summary of the mineral tenure is provided in Table 4-1 and a location plan for the granted mineral titles and the mining title application is provided in Figure 4-1.

4.4.1 Integrated Mining Title 14292

Gramalote Ridge title is integrated mining title number 14292 consists of 11 registered concession contracts totaling 8,720.71 ha.

A Sole Exploration and Exploitation Program application was filed with the Governor's Office of Antioquia to integrate mining titles 6194B, 2042, 14292, 6263, 6185B, 6054, 6032, ICQ-0800631X, 7153, 5917, IFC-08021 into a single title. Resolution No. 040497, dated May 2, 2012, approved the grouping of the concessions into single Mining concession agreement, under mining title number 14292, and title was transferred to Gramalote Colombia.

From a mining legal standpoint, mining title number 14292 is currently in the construction phase and has a PTO approved by Resolution S201500356632 dated December 22, 2015, and Resolution No. 2016060082224 dated October 21, 2016, which were issued by the Secretary of Mines of Antioquia. An Environmental License was granted by the ANLA in accordance with Resolution 1514 dated November 15, 2015, and Resolution 309 dated March 29, 2016, for advanced exploration, construction and commissioning, exploitation and closure activities.

According to the PTO Modification, approved in 2018, the Construction Stage will commence when the resettlement obligation is complete.

Under Resolution 1447 dated August 18, 2021, ANLA authorized construction activities in concomitance with the resettlement process, but subject to compliance with the required resettlement of the affected communities.

4.4.2 Mining Title 4894

Trinidad title is mining title number 4894 has an area of 2,277.77 ha.

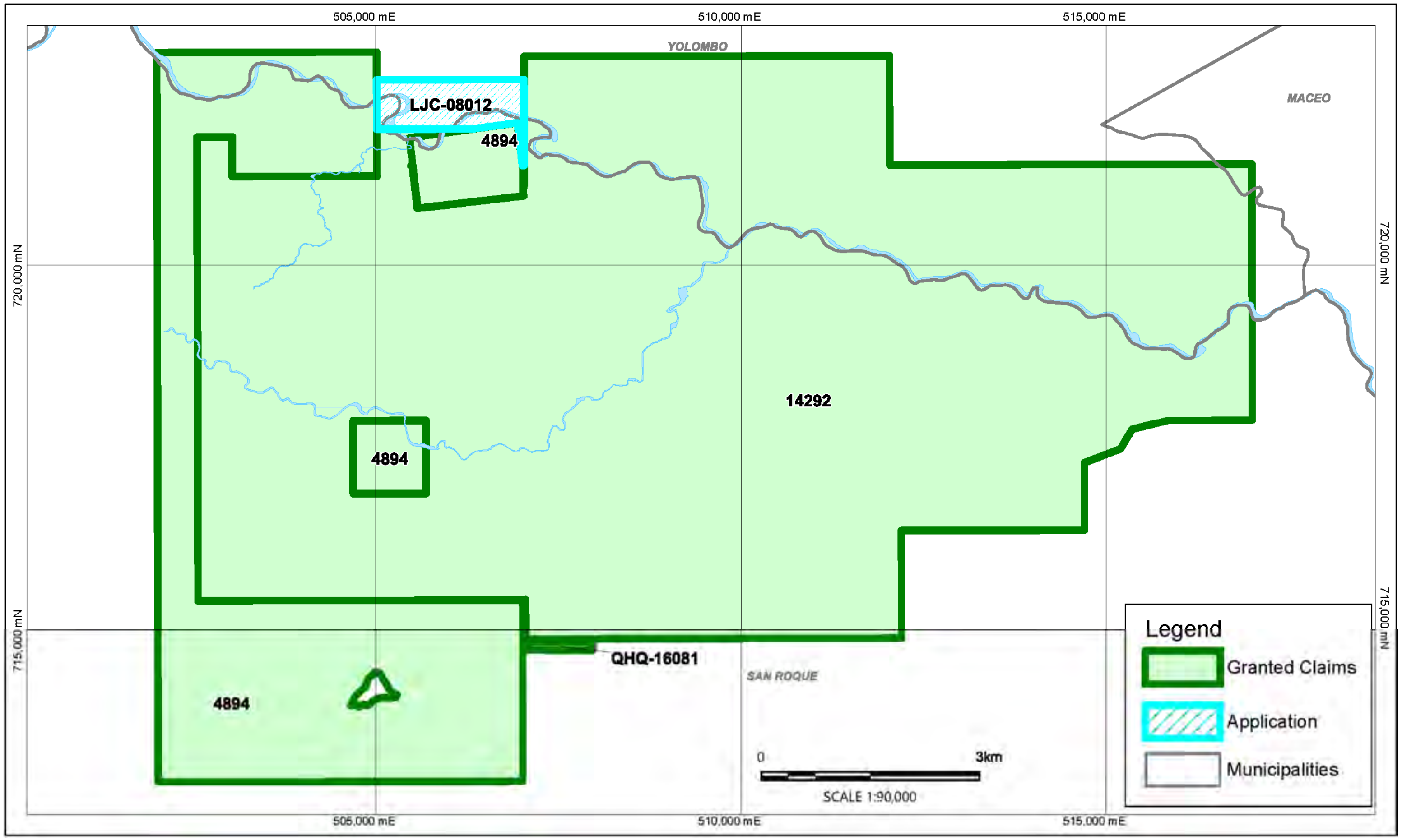
Mining Title 4894 was originally an exploration license and was granted for five years according to Resolution No. 12423 dated December 30, 1999. According to Resolution No. 2016060072784 dated August 11, 2016, the Ministry of Mines of Antioquia authorized the execution of a mining concession contract under Law 685 of 2001 and ordered the preparation of the concession contract in terms of technical concepts 1239565 dated May 13, 2016, and 1242953 dated August 4, 2016.

The concession contract was signed by the Antioquia Governor on May 22, 2019, and registered on June 12, 2019.

Table 4-1: Mineral Tenure Table

Tenement	Registration date (dd/mm/yr)	Registration Number	Title Type	Title Holder	Tenement Expiration Date (dd/mm/yr)	Tenement Area RNM (Ha)	Comments
14292 (Gramalote Ridge)	3-Apr-13	GAGB-07	Integrated Concession Contract	Gramalote Colombia Limited	2-Apr-43	8,720.7095	Mining title in force
4894 (Trinidad)	12-Jun-19	HDMG-04	Concession Contract	Gramalote Colombia Limited	11-Jun-44	2,277.7721	Mining title in force
QHQ-16081	4-Aug-22	QHQ-16081	Concession Contract	Gramalote Colombia Limited	3-Aug-52	9.7817	Mining title in force
LJC-08012	Application	NA	Application	Gramalote Colombia Limited	—	94.1378	Application in technical evaluation

Figure 4-1: Granted Mineral Tenure Location Plan



Note: Figure prepared by B2Gold, 2025. The small excision in the lowermost portion of mining title 4894 represents a portion of land that is under a Land Restitution Process, whereby a judge has ruled that this specific piece of land is not available to a party until a court process is resolved.

To extend exploration activities at concession title 4894, Gramalote Colombia submitted an integration request between concession title 4894 and concession title QHQ-16081 in August 2023. The mining authority approved the integration in June 2025, however, the Integrated Contract still needs to be signed: a process that may take between 6 to 12 months.

4.4.3 Mining Title QHQ-16081

Mining title QHQ-16081 has an area of 9.78 ha.

On July 27, 2022, the Governor's Office of Antioquia granted a mining concession over the former area covered by application QHQ-16081. The registration was executed on August 4, 2022.

In respect of duration of tenure, Colombian mining law states that the exploration phase begins as soon as the concession contract is registered in the National Mining Registry. The total period for the concession contract (exploration, construction, and exploitation) is 30 years, which may be renewed for an additional 20-year period.

Under Colombian mining law, producing mines are subject to a federal royalty of 4% of the gross value of gold and silver production. Gross metal value is determined by using 80% of the spot prices for gold and silver on the London Metals Exchange, so the effective royalty rate is 3.2% market value.

The mining title is part of the integration application discussed in Section 4.4.2.

4.5 Surface Rights

Gramalote Colombia has commenced the acquisition of surface rights to support mining operations, with purchases having been completed on the majority of the area required.

From 2011 to 2013, a land acquisition process (first acquisition round) was undertaken, and the acquisition of 34 properties was negotiated. Because of conditions in the zone where land was being purchased (e.g. affected by violence, absence of formality in land titles, unsettled succession processes, and landowners who did not exercise sovereignty over their property, among other reasons), Gramalote Colombia, after conducting the pertinent previous studies, decided to negotiate the purchase of seven properties with landholders who had to acquire the property through a judicial process to then transfer it to Gramalote Colombia.

Based on the data and studies conducted, Gramalote Colombia has established a strategy to acquire those properties that are still to be acquired where they are necessary for the construction and operation stages. The negotiation process for land acquisition can be conducted directly by Gramalote Colombia or through an expert consultant, within a framework of transparency subject to the land acquisition procedure.

4.6 Water Rights

Guacas Creek will provide the water for potable use. Gramalote Colombia currently holds a 40 L/s extraction permit. Water will be treated through a conventional 16 L/s water treatment plant.

The Project is located in a high rainfall area, receiving approximately 2.4 m of precipitation annually, and the water management system will deal with a positive water balance ensuring sufficient supply for all process water requirements.

4.7 Royalties and Encumbrances

4.7.1 Royalties

Royalties are discussed in Section 4.2.4.

4.7.2 Encumbrances

As of the Report Effective Date, a legal proceeding that was filed by a third party, Natalia Nohaba Vallejo, against the administrative act issued by the Mining Authority (Antioquia Secretary of Mines) is in progress. The Mining Authority (Antioquia Secretary of Mines) did not grant the mining rights that were requested by third parties over some "corridors" existing within the area of the integrated mining title 14292. The "corridors" arise from defects in the Colombian Mining Cadaster graphics.

Gramalote Colombia requested to become a party to the judicial process and was accepted as such on August 28, 2019. Gramalote Colombia will, therefore, be able to support the legal and technical thesis stated by the Mining Authority (Antioquia Secretary of Mines) throughout the proceeding.

In Gramalote Colombia's assessment, the risk of potential impact to the Project is low. If the legal precedent is maintained, the status quo of the mining area should be maintained as well. The "corridors" or graphic defects of the Colombian Mining Cadaster are not considered to be free areas. In the event that the interpretation does consider the "corridors" to be free areas, Gramalote Colombia would have a first right in time as it is the first applicant.

4.8 Permitting Considerations

Permitting considerations for operations are discussed in Section 20.

Environmental permits for water use and tree cutting are required during the drilling phase of exploration. Current water use permits are valid. Tree cutting permits are authorized for the area within the granted environmental licence for the Gramalote Project. Drilling in the Trinidad area was undertaken only in those areas where tree-cutting was not required. Future drill programs may require tree-cutting permits.

No special bonds are required, since all concession contracts require an insurance policy with coverage for environmental and mining liabilities.

4.9 Environmental Considerations

Environmental considerations for operations are discussed in Section 20. There are active artisanal mining groups within the Project tenure; these are also discussed in Section 20.

Current environmental liabilities consist of the exploration camp, access and drill roads, and drill pads.

4.10 Social License Considerations

Social licence considerations for operations are discussed in Section 20.

4.11 Comment on Property Description and Location

To the extent known to the QP, there are no other significant factors and risks that may affect access, title, or the right or ability to perform work on the Project that have not been discussed in this Report.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 Accessibility

The Project is located in the Nus River valley, a transportation corridor of paved highways connecting the Colombian capital city of Bogota and the city of Medellin to the Magdalena River at Puerto Berrio. The Project site is accessible via a well-maintained paved road from Medellin (approximately 2 hours by car) and Bogota (approximately 8–10 hours by car).

Additional information on Project access considerations and available ports is provided in Section 18.2.

Both Medellin and Bogota are served by daily international flights,

5.2 Climate

The area surrounding the Project has mildly tropical climate, with daytime temperatures averaging about 24°C throughout the year. Yearly rainfall averages about 200 cm and falls mostly during two rainy seasons extending from March to May and from September to December.

Future mining operations are planned to be conducted on a year-round basis.

5.3 Local Resources and Infrastructure

The Project site is located approximately 55 km from Puerto Berrio, a distance of approximately 73 km by road. From Puerto Berrio, direct water access is available to Barranquilla, a major ocean port on the Caribbean coast.

Providencia, the town nearest to the Project site, is a historic mining supply centre, administered by the Municipality of San Roque, located 12 km southwest of the Gramalote Project. Gravel roads connect the small town and farm population to the Nus River valley infrastructure.

An inactive freight and passenger railway line, along with active high-tension electricity lines pass within 1 km of the Project area. Regionally, the central Antioquia Department is a hub of large-scale hydro-electricity generation. A 2.3 MW hydroelectric plant, owned by Gramalote Colombia, is currently generating electricity at the Guacas Creek, within the Gramalote Project area. The nearest high voltage substation suitable for grid connection is located approximately 30 km from the Project site.

Several small towns near the Project site, with a long history of ranching and small-scale mining, can provide readily available labor. Local labor is not trained in modern exploration and mining techniques, so training and importation of some expatriates will likely be necessary. Personnel, equipment, and contractors to support exploration, construction, and production are available in Medellin.

Additional information on proposed infrastructure is provided in Section 18.

5.4 Physiography

The Project is located along the southern margin of the topographic depression formed by the Nus River valley, a major west–northwest-trending structural depression within the generally planar south- and east-sloping Antioquia plateau of Colombia's Central Cordillera. The Nus River flows from the central region of the Antioquia plateau into the Magdalena basin to the east. Two other main drainage systems are found within the proposed Gramalote Ridge Mine site area: The Palestina Valley, and the Guacas Creek, in addition to the Nus River.

Topography along the Nus River valley is undulating with locally steep and incised areas. Elevations in the Project area range from 800–1,500 masl, while elevations over the Antioquia plateau are generally between 2,300–2,500 masl.

Natural bedrock outcrops are rare. Tropical weathered latosol profiles are common and average 15 m thick in undisturbed areas.

Small-scale, artisanal gold mining from both alluvial and hard-rock sources is evident throughout the region of the Nus River valley.

Outside of artisanal mining areas, the region is predominately covered by grass pasture and cropland with limited and scattered patches of natural vegetation dominated by lush, low-growth Andean forest, mostly preserved along the courses and headwaters of drainages.

Agricultural activities in the Nus River valley are centered around cattle ranching and the cultivation of sugar cane, raw sugar (panela), and tropical fruit production.

5.5 Seismicity

A probabilistic assessment of the seismicity likely in the Project area was conducted as part of the MEIA. Records used were sourced from the National Seismological Network, covering the period 1993–2003, and from the historical catalogue of Colombia compiled by the Seismological Information Processing Center of the National University of Medellin, covering the period 1656–1993.

The average peak acceleration for a return period of 500 years is predicted to be 0.14 *g* in the Project area. This gravitational acceleration estimate represents a low to intermediate level of seismic risk.

5.6 Sufficiency of Surface Rights

There is sufficient surface area for the open pits, waste rock storage facilities, plant, tailings storage facilities, associated infrastructure and other operational requirements for the planned life-of-mine and mine plan discussed in this Report.

Gramalote Colombia has commenced the acquisition of surface rights to support mining operations (see also discussion in Section 20), with purchases having been completed on the majority of the area required. Purchases include the 2.2 MW hydroelectric power station located in the base of the waterfall of the Guacas Creek, which was secured in 2021.

6.0 HISTORY

6.1 Exploration History

The Gramalote area has a long history of artisanal gold mining, likely dating from Pre-Colombian times to the present day. Historical production was dominated by hydraulic techniques, and by the early 1900s many operations were producing gold throughout the Nus River valley, including at Gramalote Ridge, Guacharacas, La Trinidad, Cisneros and El Limon. The miners worked residually-enriched colluvium and mineralized in situ saprolite in the surface oxide zone around Gramalote Ridge, as well as alluvial deposits.

Modern exploration activities are summarized in Table 6-1. The table also provides information on the internal studies completed by the JV.

6.2 Production

There has been no formal production from the Project area.

There are no records available as to the historical artisanal production. Under the formalization process sponsored by B2Gold, about 11,000 oz gold has been produced by artisanal miners.

Table 6-1: Exploration History

Year	Operator	Note
16th to late 20th centuries		Artisanal mining activities.
Pre-2005	Aristizabal family	Aristizabal family held ownership over part of the current Project area that is now held under mining title 14292.
1995	Metallica Resources, Inc. (MRI)	Executed a preliminary exploration agreement with Sergio Aristizabal. Completed surficial sampling and mapping of the Gramalote Ridge area.
1997	Gridiron Exploration Ltd	Brief evaluation studies.
1999	Placer Dome Exploration Inc.	
2000	Industrias Peñoles (a Grupo Nus affiliate)	
2003	Sociedad Kedahda (AngloGold Ashanti)	Reviewed property area.
2003–2007	AngloGold Ashanti	Consolidated the mineral exploration tenement and formally entered into a joint venture agreement in 2005 with Grupo Nus. Completed detailed topographic surveying over Gramalote Ridge and the surrounding area. Surface sampling including 277 stream sediment samples, regional soil sampling on 50–200-m centers (2,853 samples), infill soil sampling on 50 x 100 m centers over Gramalote Ridge (491 samples); grab, chip, channel and panel sampling of out cropping mineralization around Gramalote Ridge (266 samples); regional reconnaissance rock chip sampling (1,384 samples). Detailed geological and structural mapping of Gramalote Ridge. Trial, in-house ground-based magnetometer survey over Gramalote Ridge with a total of 116 line-km surveyed in 59 lines. Petrographic thin and polished section and microprobe study for gold distribution. Excavation and complete channel and bulk mineral sampling of a 240-m long tunnel (adit), with a 2 x 2 m cross sectional area, in fresh (unweathered) rock, drifting into the northeast flank of Gramalote Ridge. Drilled a total of 43 core holes (12,312 m). Two phases of preliminary metallurgical test work on mineralized sulphide-bearing materials collected from the underground tunnel and drill core from various localities around Gramalote Ridge. Initial Mineral Resource estimate.
2008–2009	B2Gold	Completed surface structural mapping and rock sampling of numerous vein orientations in the main Gramalote Ridge area. Re-mapped and re-sampled the adit. Core drilling program consisting of 89 holes (29,87.70 m) including 21,548.90 m (65 holes) in Gramalote Ridge and 7019.26 m (20 holes) in Trinidad. Gramalote Ridge Mineral Resource estimate updated in 2009.

Year	Operator	Note
2010–2011	AngloGold Ashanti	Evaluation studies, preliminary engineering investigations, metallurgical test sample drilling (2,811.36 m in 10 core holes) 72 core drill holes (25,798.64 m) in Gramalote Ridge, Trinidad, Monjas West, Monjas East, Limon and Topacio exploration drilling.
2012		Resource and exploration drilling of 14,019.65 m in 26 core holes in Gramalote Ridge, Monjas West and La Maria. Mineral Resource estimate updated in 2012. Completion of internal evaluation study (2012 evaluation study); did not meet AngloGold Ashanti internal hurdles. Second internal evaluation study update in late 2012 (2012 evaluation study update).
2013		Total of 29,140.87 m (128 core holes) in Gramalote Ridge, Monjas West and Limon. From 2011-2013 completed 11,682.50 m in 77 core holes for infrastructure condemnation and 5,125 m in 13 core holes for Gramalote Ridge metallurgy. Internal evaluation study further updated (2013 evaluation study). AngloGold Ashanti internal hurdle rates still not met.
2014		Completed 7,160.13 m in 14 core holes in Gramalote Ridge. Completed a preliminary economic assessment (PEA; 2014 PEA). Study shows positive economics under the assumptions and interpretations in the study.
2015–2016		Completed 211 core holes (3,482.64 m) in Gramalote Ridge saprolite area. Mineral Resource estimate updated. Internal evaluation study update (2016 evaluation study). Drilled 7,039.34 m (28 core holes) in exploration targets, several of which are outside current property area.
2017		Completed 14 core holes (3,320.09 m) in Gramalote Ridge. Updated Mineral Resource estimate. Completed an internal mining study, referred to as a 'pre-feasibility study'.
2018		In Gramalote Ridge completed two core holes (1,589.61 m) and 105 RC drill holes (8,484 m) as well as 1928.98 m in 10 exploration holes at outside targets.
2019	B2Gold	Prepared updated resource model and completed a new resource estimate. Undertook a second PEA (2020 PEA). Initiated Gramalote Ridge drill program to support potential conversion of Inferred Mineral Resources to Indicated.
2020–2022		Completed 78,638 m (233 core drill holes) for Gramalote Ridge resource conversion drill program. Total of 3,417.26 m in 117 core holes for feasibility study-related infrastructure. Completed a feasibility study. In August 2022, JV partners determined that the Gramalote Project as envisaged in the 2022 feasibility study did not meet investment thresholds for project development.
2023–2025		B2Gold updated feasibility study for smaller size project scenario. Completed exploration sampling (rock chip, trenching) and infrastructure location-related drilling.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Project is located in the northern portion of Colombia's Central Cordillera. The terrane is primarily comprised of a metamorphic basement complex and the Antioquia Batholith (Figure 7-1). The Cajamarca–Valdivia basement terrane (Cediel and Ciceres, 2000 and Cediel et al, 2003) consists of early Paleozoic metamorphic rocks; as well as ophiolitic, oceanic, volcanic, and intrusive rocks.

During the Cretaceous Period, the Paleozoic basement complex and Mesozoic sequences were intruded by the Antioquia Batholith. This multi-phase, calc-alkaline, I-type intrusive complex predominantly consists of tonalite with localized diorite and granodiorite phases, and is exposed over an area >7,800 km^2.

To the west, the Cajamarca–Valdivia basement is bounded by the Romeral fault-and-suture system, a generally north-striking and dextral transcurrent mega-structure that records the emplacement of additional allochthonous oceanic terranes along the northern Andean margin throughout the late Mesozoic and Cenozoic Periods. Major lineaments are noted within the batholith, especially in its eastern sector (Figure 7-2). These lineaments are generally west–northwest through northwest-trending and record rotation and sinistral shearing events. Subordinate zones of north–south extension and northeast-striking shear are related to the regional sinistral shear patterns and appear to provide an important controlling mechanism for the focusing of late magmato-hydrothermal activity and gold (± Cu, Mo, Pb, Zn, Ag) mineralization in various sectors of the batholith.

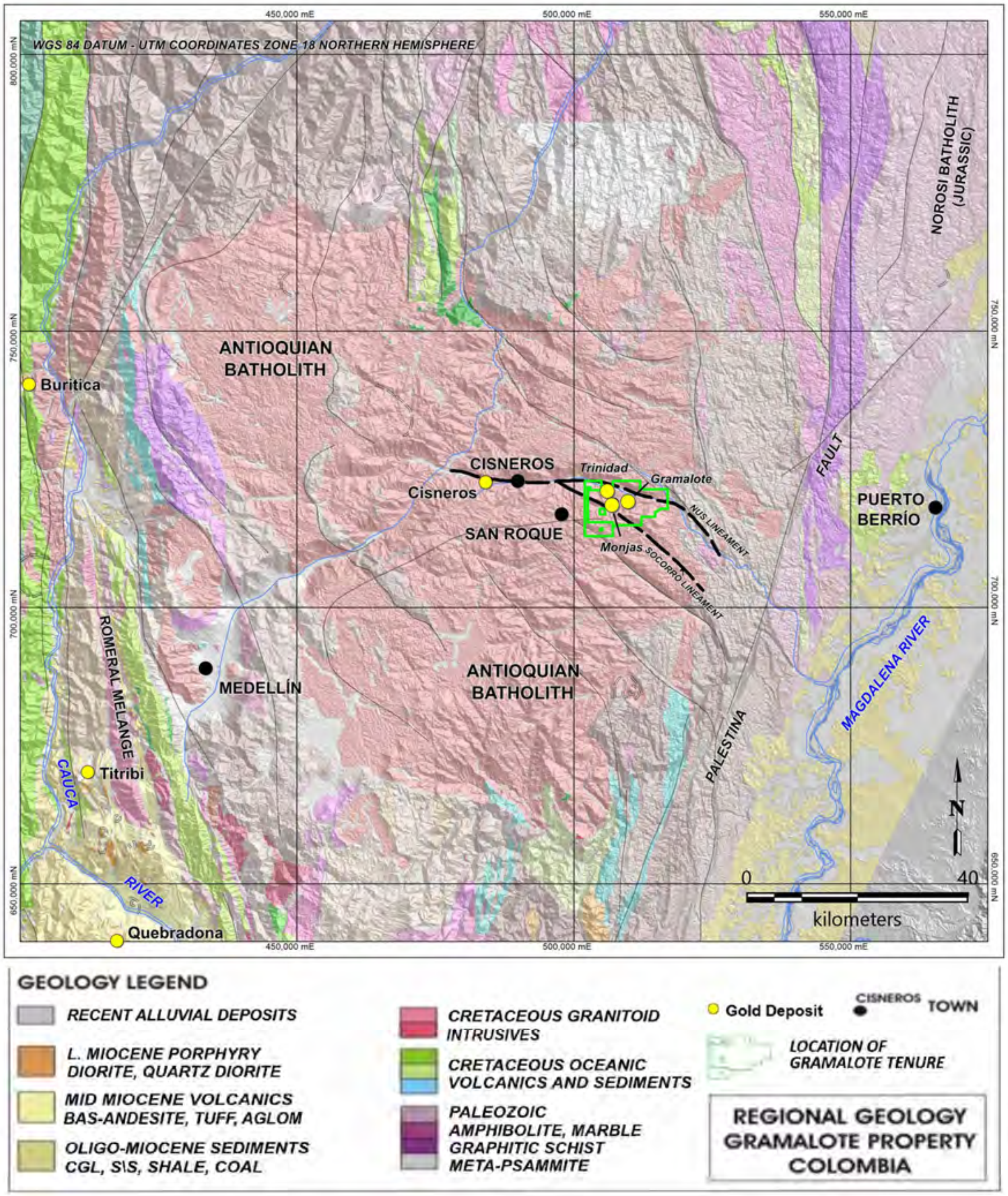
7.2 Project Geology

7.2.1 Lithologies

In the general Project area, the main compositional types within the Antioquia Batholith are tonalite and granodiorite, with subordinate monzonite and gabbro. Alaskitic, felsitic and andesitic dikes have intruded the complex. The major rock types are summarized in Table 7-1. A Project geology map is included as Figure 7-3.

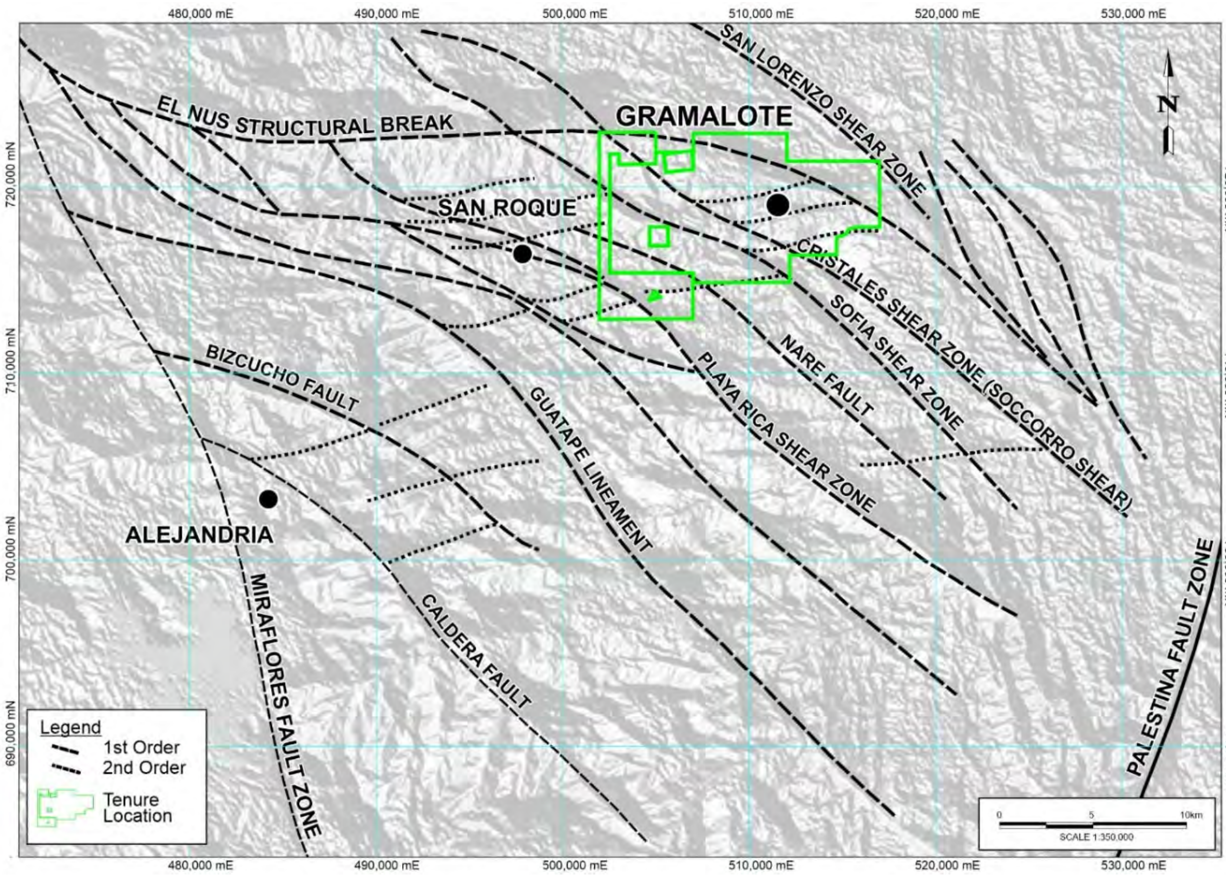
Age dating conducted by AngloGold Ashanti suggest the gold mineralization is associated with the last stages of batholith crystallization at 59.2 ± 1.2 to 60.7 ± 1.0 Ma.

Figure 7-1: Regional Geology



Note: Figure prepared by B2Gold, 2025. B2Gold does not have an interest in projects shown on figure other than the Gramalote deposits.

Figure 7-2: Regional Structural Setting

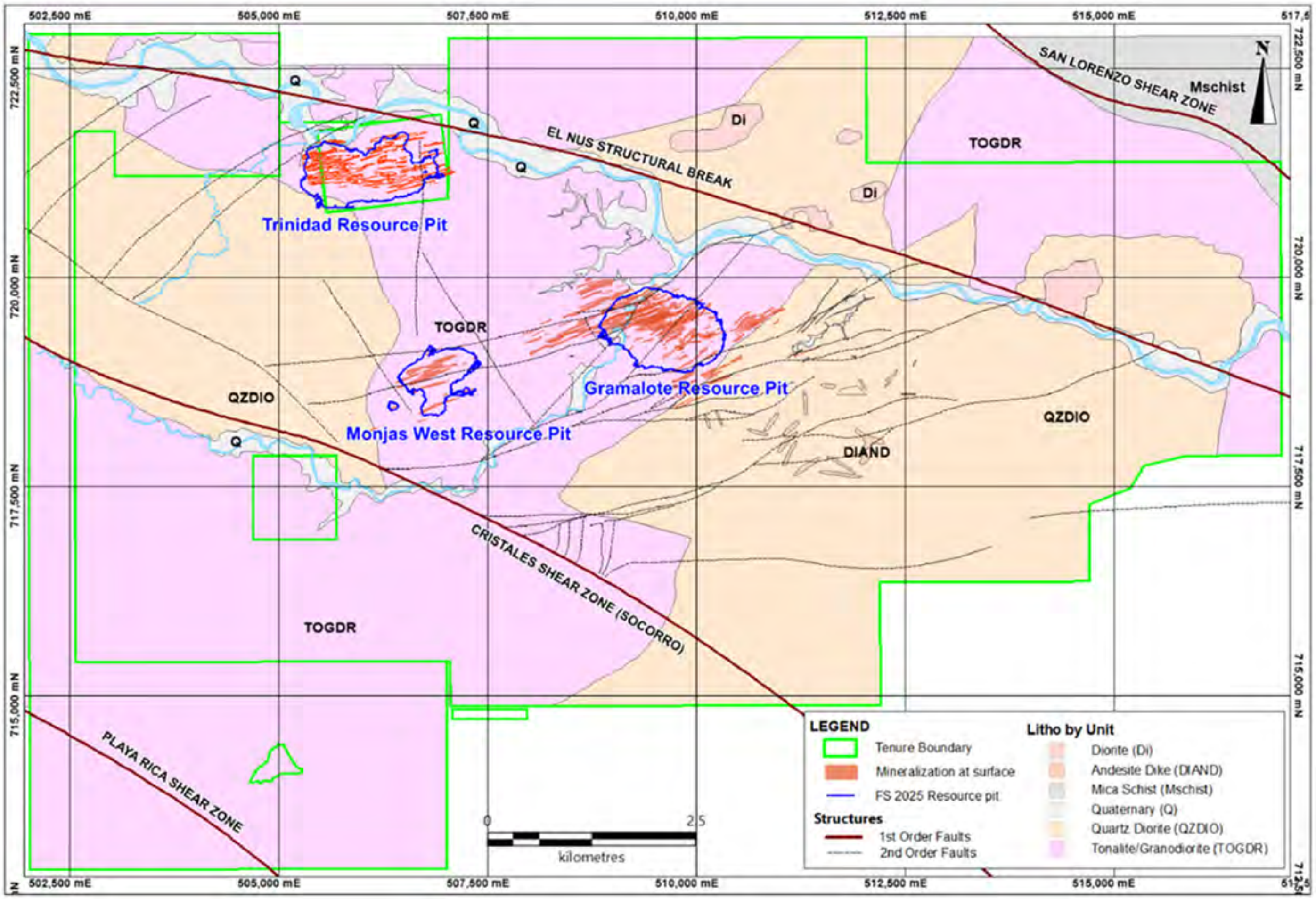


Note: Figure modified by B2Gold, 2025, after Valencia, 2007.

Table 7-1: Major Rock Types

Unit	Note
Tonalite, granodiorite	Consist mainly of plagioclase crystals, quartz and mafic minerals (biotite and hornblende).
Quartz diorite	Quartz content of 10–20%, predominating over calcic plagioclase and mafic minerals, with variable content (hornblende and/or biotite). Mafic minerals can be altered to chlorite.
Fine-grained porphyritic diorite	Porphyritic igneous textures, plagioclase and quartz phenocrystals 0.5–2.0 mm in diameter, embedded in an aphanitic matrix. Matrix also contains mafic minerals such as hornblende and mica (biotite).
Microdiorite dike	Similar to the porphyritic diorite but crystal size <0.5 mm. Recognized in adjacent areas to Gramalote Ridge. Forms centimetric dikes that are occasionally affected by hydrothermal events where in contact with the host tonalite/granodiorite.
Monzogranite	Essentially consists of biotite and plagioclase. Includes medium to large equi-grained pink K-feldspar, and reddish quartz.
Aplites and pegmatites	Aplites consist mainly of alkali feldspar, quartz and mica (biotite locally). Typically, aphanitic texture, varying in size from mm to cm scale. Pegmatites have similar constituents but are coarser.

Figure 7-3: Project Geology Map



Note: Figure prepared by B2Gold, 2025. Gramalote = Gramalote Ridge

7.2.2 Structure

The structural setting is dominated by two north–south-trending fault zones, the Palestina Fault Zone to the east and the Miraflores Fault Zone to the west. These are linked by 'horse tail' regional extensional structures with the Nus River structural break to the north and the Cristales Shear Zone to the south. Duplex shear zones have developed between these two regional structures as second order structures with subordinate third and fourth order tension gashes developing between the second order structures.

Gramalote Ridge is situated between two west–northwest-trending macro-scale curved lineaments that splay off the Palestina fault to the east, and transect the Antioquia Batholith, termed the Nus River and the El Socorro lineaments.

Differential movement along the Nus and El Socorro lineaments is thought to have generated north–northwest, north–south and northeast-striking tensional dilation within the tonalite, reflected in the formation of stockwork style sheeted quartz and quartz–carbonate veinlets.

7.3 Deposit Descriptions

7.3.1 Gramalote Ridge

Dimensions

The Gramalote Ridge deposit has dimensions of 1,300 x 1,500 x 700 m. The mineralization has been drill tested to a depth of about 865 m.

Lithologies

Mineralization at Gramalote Ridge is hosted in a uniform medium-grained tonalite–granodiorite body within the regional quartz diorite and diorite that makes up most the Antioquia batholith. Aplitic and pegmatitic dykes are common within the tonalite, as well as minor andesitic dykes. Internal facies changes are also common within the tonalitic body, which has a transitional intrusive contact towards the south with the quartz diorite hosting the Limon and La Maria veins.

Alteration

Alteration is structurally-controlled and occurs as both broad zones and narrow selvedges around veins. At Gramalote Ridge, vein selvedges range from a few millimetres up to as much as 10 cm.

The broader alteration zones mimic the vein selvedges and progress from an outer quartz–sericite zone to inner carbonate and potassic zones. A minor supergene argillic alteration zone is identified at the surface. The gradual destruction of magnetite from fresh rock to the most intensely altered rock allows the alteration to be 'mapped' to some degree using the magnetic susceptibility of the rock.

Three, diffuse, overlapping alteration zones are typically identified moving from fresh rock to mineralized veins over a range of generally <20 cm:

- A distal incipient sericite (alteration of plagioclase to a fine mica) zone with partially oxidized magnetite and biotite altered to chlorite;
- An intermediate extensive sericitization and incipient carbonate zone with pyrite replacing magnetite, and incipient carbonatization;
- A proximal zone with no magnetite, ±K-feldspar, muscovite fully replacing chlorite, and increased carbonatization.

Weathering

Mineralization in saprolite and saprock (oxide) is weathered insitu and does not show evidence of significant transport or displacement. Weathered mineralization is limited to the areas directly above fresh mineralization and represents a small percentage of the mineralization. Saprolite/saprock thickness is variable from 5–55 m, with an average thickness of 18 m for Gramalote Ridge overall, and about 14 m thick above mineralization.

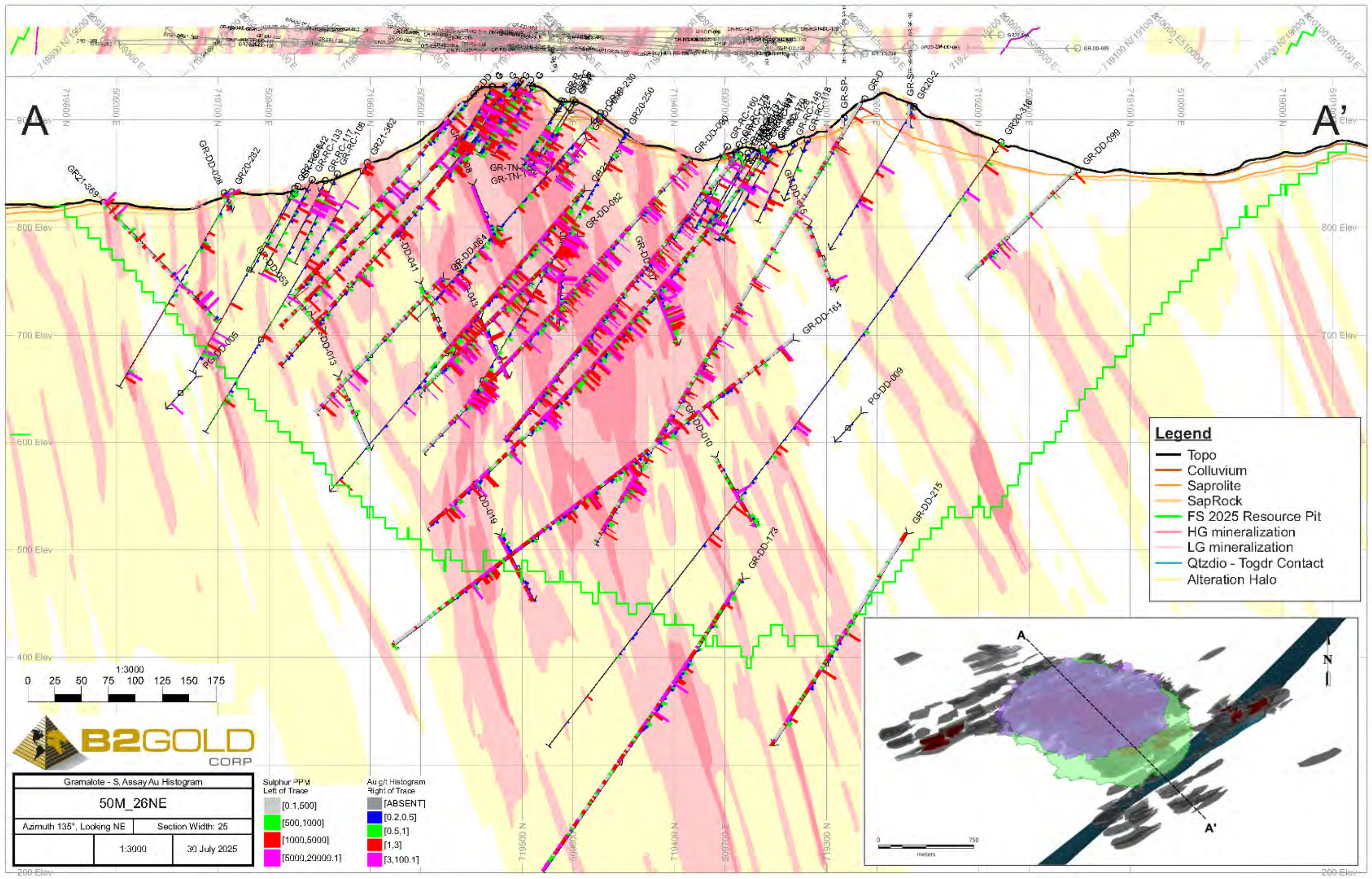
Mineralization

Gramalote Ridge mineralization occurs within several zones that periodically coalesce both along strike and down-dip. Zones vary in width from 10–150 m in true width with vertical to sub-vertical dips to the south–southeast. These east–northeast-trending mineralized corridors are made of a subtle to moderate increase in vein density, coalescing alteration selvedges, and sulphide percent, with internal veinlets trending in three main directions (northwest, north–south and northeast).

The deposit remains open at depth.

Mineralization is vein hosted, either in sheeted veinlets or in local stockworks, and is structurally-controlled. The mineralized zones form magnetic lows in the magnetic maps, due to the alteration halos associated with the mineralized veinlets. Figure 7-4 shows a cross-section through the Gramalote Ridge deposit.

Figure 7-4: Cross-Section, Gramalote Ridge Deposit



Note: Figure prepared by B2Gold, 2025.

The following paragenetic stages are identified and associated with vein and alteration types:

- Quartz–carbonate veinlets with weak potassic alteration halos and fine-grained pyrite;
- Quartz–calcite–pyrite or chalcopyrite–gold veinlets; related to well-developed K-feldspar or carbonate alteration. Associated with elevated gold grades. Gold occurs in fractures in pyrite together with chalcopyrite. Sulphides can include molybdenite and, more rarely, sphalerite;
- Pyrite–white mica veinlets with minor calcite or quartz. Associated with moderate to strongly developed white mica alteration in host rocks. Rarely mineralized;
- Coarse grained pyrite veinlets and lenses. Commonly associated with the highest grade gold values. Preferentially located at edges or centerlines of quartz–carbonate veinlets.

The vein type 1–6 are summarized, from youngest to oldest, in Table 7-2. Cross cutting relations can be very variable, and most vein types, especially Types 1–4 may also be synchronous. Anomalous gold mineralization is associated with Type 1, Type 2 and Type 5 vein types, with Type 1 and Type 2 being the most prevalent. Type 7 veins are associated with the intrusive fabric, not the mineralizing systems.

The silver to gold ratio is approximately 1:1. Free gold occurs as argentiferous gold coeval with several tellurides and bismuth sulphosalt minerals.

The primary sulphide mineralization consists of pyrite and chalcopyrite.

7.3.2 Trinidad

Dimensions

The Trinidad deposit has dimensions of approximately 1,500 x 500 m. Mineralization has been drill tested to a depth of about 521 m.

Lithologies

The Trinidad mineralization is hosted in similar rock types to those described for Gramalote Ridge.

The host tonalite–granodiorite is weakly magnetic and mineralization/alteration is magnetically destructive, so the mineralized zones show on magnetic maps as a magnetic low. The magnetic low associated with Trinidad supports a strike orientation of ~075–080°, and this agrees with the apparent continuity of mineralization in closer-spaced drill holes. These drill holes also support a steep (~80–85°) dip of the mineralization to the north.

Table 7-2: Vein Types

Stage	Composition	Note
Type 7	Chlorite ± epidote	Variable amount of carbonate and occasionally fine pyrite. Commonly associated with the main intrusive fabric, rather than the mineralizing system.
Type 6	Carbonate	Carbonate veins with no sulphides.
Type 5	Coarse pyrite + quartz	Veinlets with more pyrite than quartz. Irregular edges with variable amount of chalcopyrite. The pyrite is euhedral, coarse to medium grained, in comparison with the grey, very fine-grained pyrite found in Type 3 veins.
Type 4	Quartz + molybdenite ± pyrite	Bands of bluish-grey colour with abundant molybdenite. Variable shapes and generally irregular edges. Contain variable amount of pyrite and chalcopyrite. Typically cm thickness.
Type 3	Fine pyrite	Veinlets composed almost exclusively of pyrite with variable amount of quartz. Form brecciated braided veinlets. Dark grey colour due to pyrite content.
Type 2	Quartz + carbonate + pyrite + chalcopyrite	Typically show irregular edges, have a lensoid shape when centimetric thickness. Contain variable amount of pyrite and chalcopyrite. The carbonate is typically siderite or ankerite. Mainly associated with quartz–sericite alteration halos.
Type 1	Quartz ± pyrite	May have chalcopyrite. Well-defined edges and generally straight shapes with tabular aspect. More quartz than sulphur content and generally associated with potassic alteration halos.

Alteration

Alteration is similar to that described for Gramalote Ridge.

Weathering

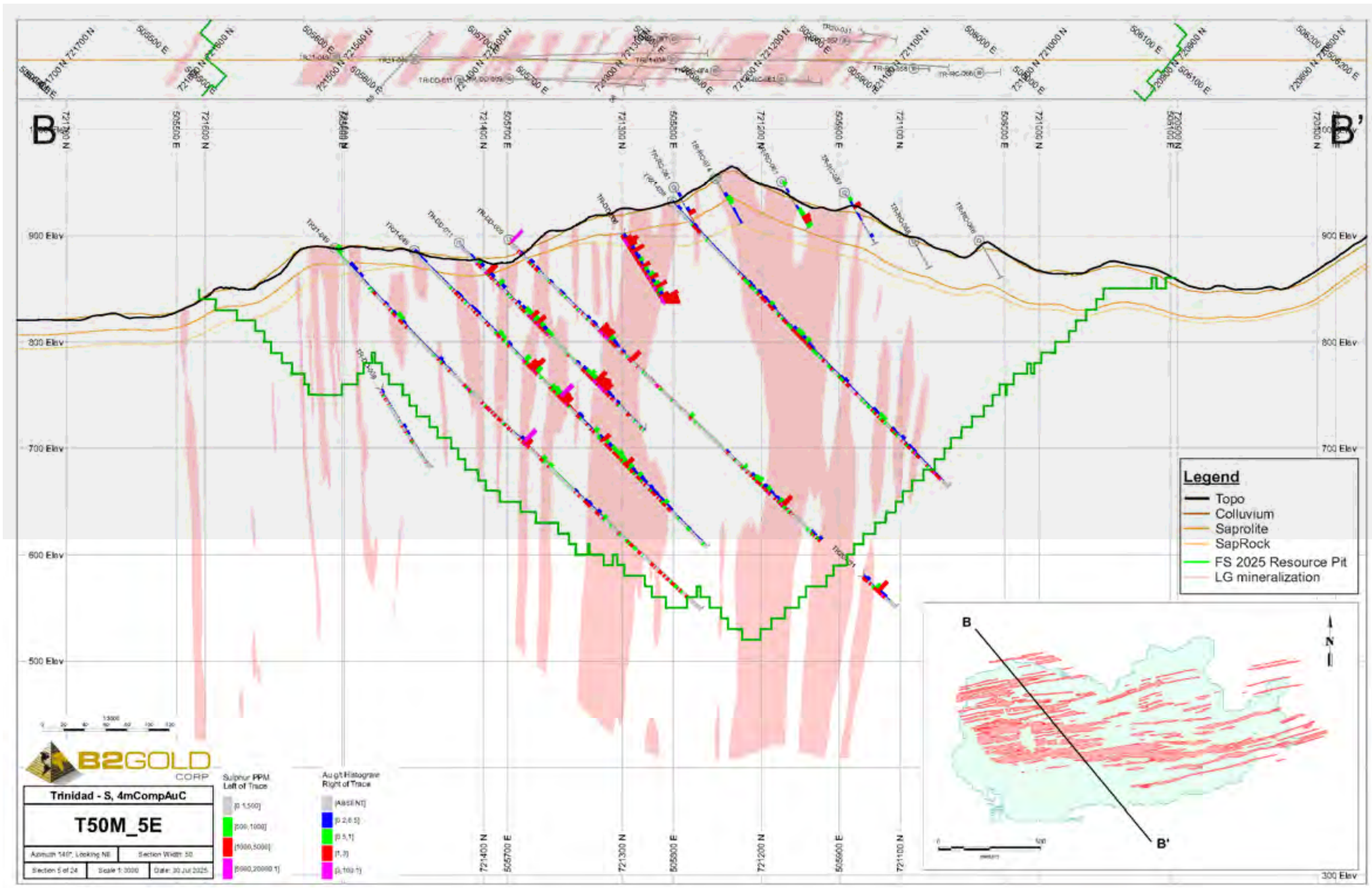
Saprolite/saprock thickness is variable, with an average thickness of 31 m overall, and about 33 m thick over the mineralization.

Mineralization

Mineralized zones at Trinidad are 10–80 m thick and are continuous along strike for up to 1,500 m and down dip for >500 m. As at Gramalote Ridge, such zones periodically coalesce both along strike and down-dip. The deposit remains open at depth and along strike, especially to the west. A cross-section through the Trinidad deposit is provided as Figure 7-5.

Mineralization is associated with stockwork veinlets and alteration along their margins. Veinlets have been classified into seven types (refer to Table 7-2), and gold mineralization is associated with Type 1 (quartz with fine pyrite), Type 2 (quartz with iron carbonate) and Type 5 (quartz with granular pyrite) veins.

Figure 7-5: Cross-Section, Trinidad Deposit



Note: Figure prepared by B2Gold, 2025.

7.3.3 Monjas West

Dimensions

The Monjas West deposit has dimensions of approximately 1,000 x 500 m. Mineralization has been drill tested to a depth of 400 m.

Lithologies

The Monjas West mineralization is hosted in three rock types comprising tonalite/leucotonalite, porphyritic dacite and andesite porphyry.

The host tonalite is weakly magnetic and mineralization/alteration is magnetically destructive, so the mineralized zones appear on magnetic maps as magnetic lows located slightly north of the deposit. The magnetic low supports a strike orientation of ~080°, and this agrees with the apparent continuity of mineralization to south of the geophysical anomaly in closer-spaced drill holes. These drill holes also support a steep (~80°–85°) dip of the mineralization to the south.

Alteration

Alteration is similar to that described for Gramalote Ridge and Trinidad. Quartz–sericite predominates over potassic alteration, and carbonate–sericite is punctual.

Weathering

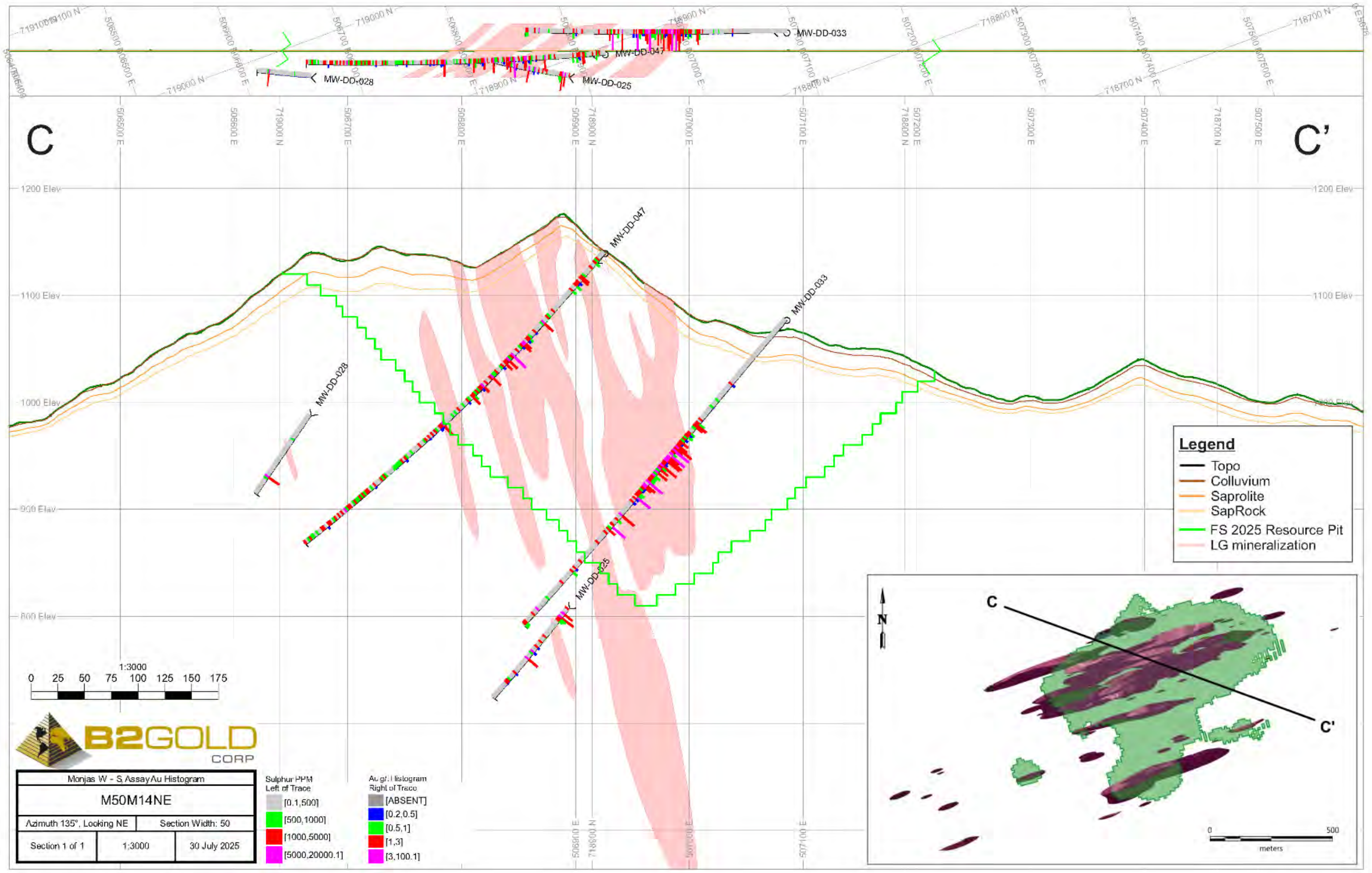
Saprolite/saprock thickness is variable, with an average thickness of 25 m overall, and about 27 m thick over the mineralization.

Mineralization

Mineralized zones at Monjas West are 10–80 m thick and are continuous along strike for up to 700 m and down dip for >350 m. As at Gramalote Ridge, such zones periodically coalesce both along strike and down-dip. The deposit remains open at depth and along strike, especially to the west. A cross-section through the Monjas West deposit is provided as Figure 7-6.

Mineralization is associated with stockwork veinlets and alteration along their margins. Veinlets have been classified into seven types (refer to Table 7-2), and gold mineralization is associated with Type 1 (quartz with fine pyrite), Type 2 (quartz with iron carbonate) and Type 5 (quartz with granular pyrite) veins.

Figure 7-6: Cross-Section, Monjas West Deposit



Note: Figure prepared by B2Gold, 2025.

7.4 Prospects/Exploration Targets

Prospects are discussed in Section 9.

7.5 Comments on Geological Setting and Mineralization

The understanding of the Project geology and mineralization is sufficient to support Mineral Resource and Mineral Reserve estimation and mine planning.

8.0 DEPOSIT TYPES

8.1 Deposit Model

The Gramalote Ridge, Trinidad, and Monjas West deposits are examples of an intrusive-related gold deposits.

Intrusive-related gold deposits have many synonyms, including Intrusion-related gold systems, gold porphyries, and plutonic-related gold quartz veins. The deposit model that follows is abstracted from Lefebure and Hart (2005).

Features of such deposits include:

- Commonly found in continental margin sedimentary assemblages which have been intruded by plutons behind continental margin arcs. Typically developed late in orogeny or post-collisional settings;
- Can be of any age, although they are best known in Paleozoic to Mesozoic rocks;
- Host rocks are granitic intrusions and variably metamorphosed sedimentary rocks. Associated volcanic rocks are rare. The granitoid rocks are lithologically variable, but typically granodiorite, quartz monzonite to granite. Most intrusions have some degree of lithological variation that appear as multiple phases that can include monzonite, monzogranite, albite granites, alkali syenite and syenite. The more differentiated phases commonly contain feldspar and quartz and less than 5% mafic minerals. Medium-to coarse-grained intrusions are commonly equigranular, but can contain megacrysts of potassium feldspar or porphyritic phenocrysts of quartz, plagioclase, or biotite. Biotite is common, hornblende is only locally observed, pyroxene is rare, and muscovite along with tourmaline are common in more highly fractionated phases, aplites or pegmatites. The intrusions have a reduced primary oxidation state;
- Mineralization is strongly structurally controlled and spatially related to highly differentiated granitoid intrusions. Mineralization is commonly hosted by, or close to, the most evolved phase of the intrusion;
- Sheeted veins are planar and often parallel to regional structures. The veins are generally extensional with no offset of walls, although some vein systems may also include shear-hosted veins;
- Relatively restricted alteration zones which are most obvious as narrow alteration selvages along the veins. The alteration generally consists of the same non-sulphide minerals as occur in the veins, typically albite, potassium feldspar, biotite, sericite, carbonate (dolomite) and minor pyrite. Pervasive alteration, dominated by sericite, only occurs in association with the better-grade ore zones;

- Gold mineralization hosted by millimetre to metre-wide quartz veins in equigranular to porphyritic granitic intrusions and adjacent hornfelsed country rock. The veins form parallel arrays (sheeted) and less typically, weakly-developed stockworks; the density of the veins and veinlets is a critical element for defining ore. Native gold occurs associated with minor pyrite, arsenopyrite, pyrrhotite, scheelite, bismuth and telluride minerals;
- Sulphide mineral content is generally <3% and can be <1%. A number of deposits/intrusions have late and/or peripheral arsenopyrite, stibnite or galena veins. Native gold, sometimes visible, occurs with associated minor pyrite, arsenopyrite, loellingite, pyrrhotite, variable amounts of scheelite or more rarely wolframite, and sometimes molybdenite, bismuthinite, native bismuth, maldonite, tellurobismuthinite, bismite, tellurides, tetradymite, galena and chalcopyrite.

8.2 Comments on Deposit Types

The deposits show several features associated with intrusive-related gold deposits, including:

- Alteration and mineralization are structurally controlled, restricted to small haloes along veins, sheeted veins and stockwork arrays;
- Sulphides content <5%;
- Some evidence indicates that the host rock is directly related to fluids evolved from the cooling pluton, including pegmatite, aplite and K-feldspar alteration (Rodríguez, 2009).

Mineralization at the Gramalote Project is an unusual example of the deposit type in that the Gramalote Ridge, Trinidad, and Monjas West host intrusion has a different oxidation state to that of a typical host intrusion, which is reduced. This is reflected in the scarcity of ilmenite, and the common presence of primary magnetite.

The QP is of the opinion that exploration programs that use an intrusive-related gold model are applicable to the Project area.

9.0 EXPLORATION

9.1 Grids and Surveys

All coordinates are referenced to the UTM84-18N projection.

Until 2020, the topographic model was based on mapping at a 1:2000 scale, using photogrammetric restitution of panchromatic aerial photographs at 1:10,000. Access points and drainage systems were captured, and a digital elevation model with contours at 2 m intervals was built. An orthophoto with a 0.3 cm pixel resolution supports identification of 1 m-size objects.

A semi-controlled mosaic with a 0.8 cm pixel was generated from aerial colour photographs at a 1:12,000 scale. The aerial photographs and the semi-controlled mosaics cover an area of 62,000 ha. These data were complemented and controlled with 16 geodetic control points surveyed with dual frequency millimetre accuracy GPS equipment, providing up to centimetre-level accuracy.

In January 2020, a 125 km^2 light detection and ranging (LiDAR) survey was completed by Geoscan Ingeniería (Geoscan), based in Bogota. The system used was an ALIS-560 with a laser pulse rate up to 400 kHz and an effective measurement rate of 150 measurements per second with an exploration angle of +30/-30 (60) degrees. The flight elevation was 500 m above the terrain (terrain highest elevation), with overlap of 100% between lines, and <10 cm RMS precision (with control points). A PPK active GPS was used for a control, with bases located 50 km apart. Two georeferenced vertices for control and 12 bases covered the zone. High resolution DTM and DSM models with vertical precision of 5 cm were delivered in two projection systems: WGS84 and MAGNA-SIRGAS. Geoscan submitted 576 LAS files. With these, the Gramalote Topographer generated a digital terrain model using the LAS cloud and Global Mapper V21 software, and an overview verification was completed.

Drill collar topographic surveys rely on geodetic control points or monuments to assure accuracies of <10 cm. In January 2020 several control points were installed in the Mirador, Bateita, Balsal, and Trinidad areas to supplement control points previously installed by AngloGold Ashanti. The location of these monuments was carried out using high-tech GNSS differential equipment from Topcon Ref. GR-5n with base and rover receiving antennas, as well as the Topcon Tools software for post-processing.

9.2 Geochemistry

Surface geochemical programs were conducted by AngloGold Ashanti in the period 2003–2007, and consisted of stream sediment, reconnaissance and infill soil sampling, and selective and continuous grab, chip, channel and panel sampling.

B2Gold completed stream geochemical analysis, prospecting, and surface rock sampling on a regional basis, together with infill soil geochemistry and surface trenching programs over prospective areas in 2008–2009. Sampling was used to define, then refine, areas that warranted drill testing. A number of the geochemical anomalies remain to be followed-up.

When B2Gold resumed Project management in 2020, the prospecting program recommenced, and B2Gold completed surface mapping and trenching programs over prospective areas as well on the extensions of known deposits. A total of 829 rock chip samples were collected during 2020–2024, and 1,261 trenches excavated for a total of 23,510.12 m. Sample locations are shown in Figure 9-1 and results are provided in Figure 9-2. A number of the geochemical anomalies remain to be followed-up.

9.3 Geophysics

With respect to exploration activities in the Gramalote Project area, the geophysical data have been superseded by drilling data. Information in the following sub-sections is provided for completeness only.

9.3.1 Airborne Geophysical Surveys

Two airborne geophysical studies were completed in the Gramalote Project area.

In 2009, a 490 line-km airborne magnetometer survey (north–south lines) was completed by AngloGold Ashanti at the request of B2Gold.

In 2012, MPX completed a helicopter-borne magnetometer radiometric survey for Gramalote Colombia. A total of about 3,000 line-km were flown over an area of 35,000 ha, covering the tenement holdings at the time on 200 m spaced north–south lines.

9.3.2 Ground Geophysical Surveys

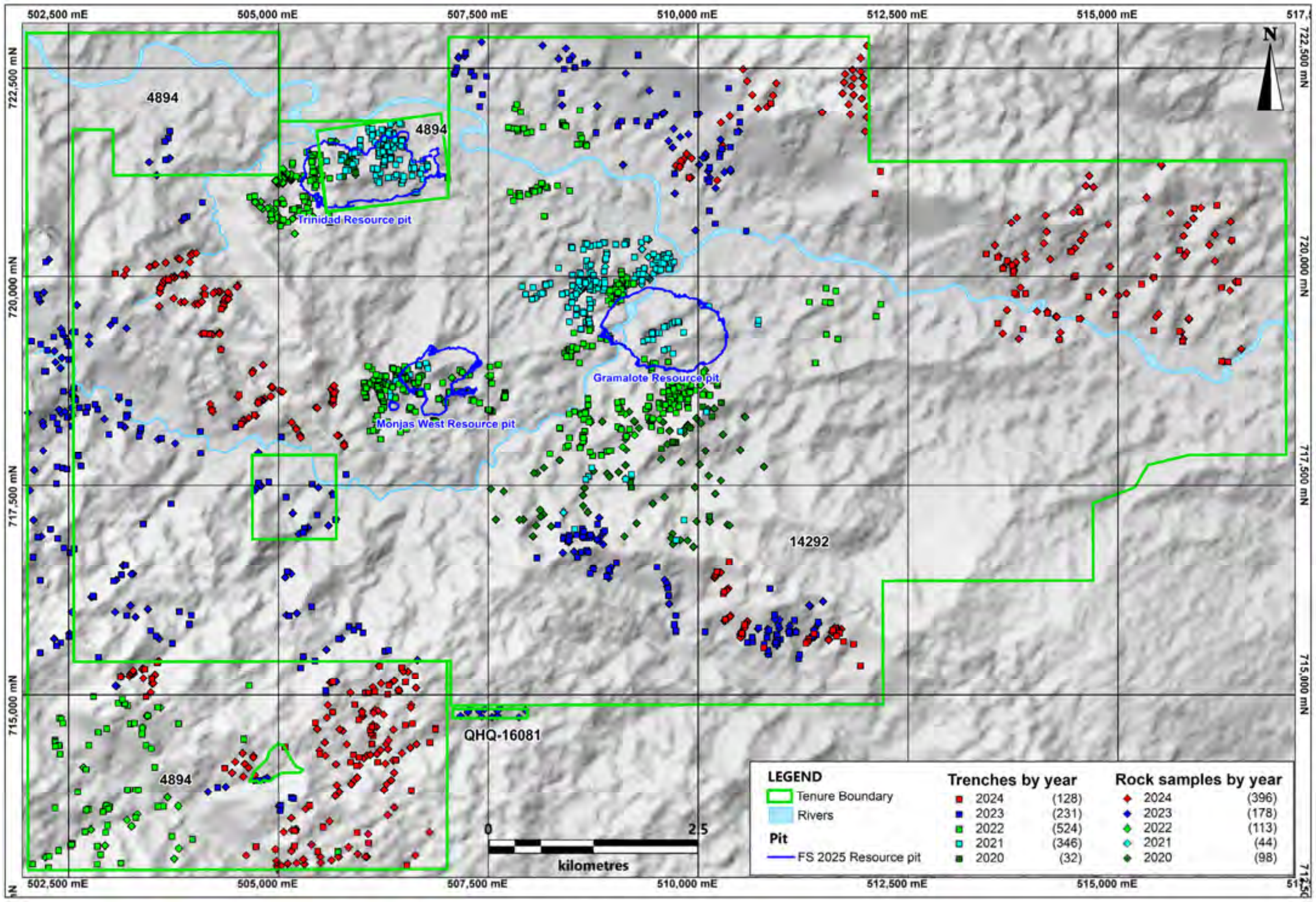
During 2003–2007, AngloGold Ashanti completed a trial, in-house ground-based magnetometer survey over Gramalote Ridge and some of the outlying prospects. The program was to test the possibility of using magnetic field information for exploration purposes at a regional level. A total of 116 line-km was surveyed in 59 lines.

9.4 Petrology, Mineralogy, and Research Studies

Petrographic thin and polished section and microprobe study of prepared core samples, oriented towards gold distribution, were completed.

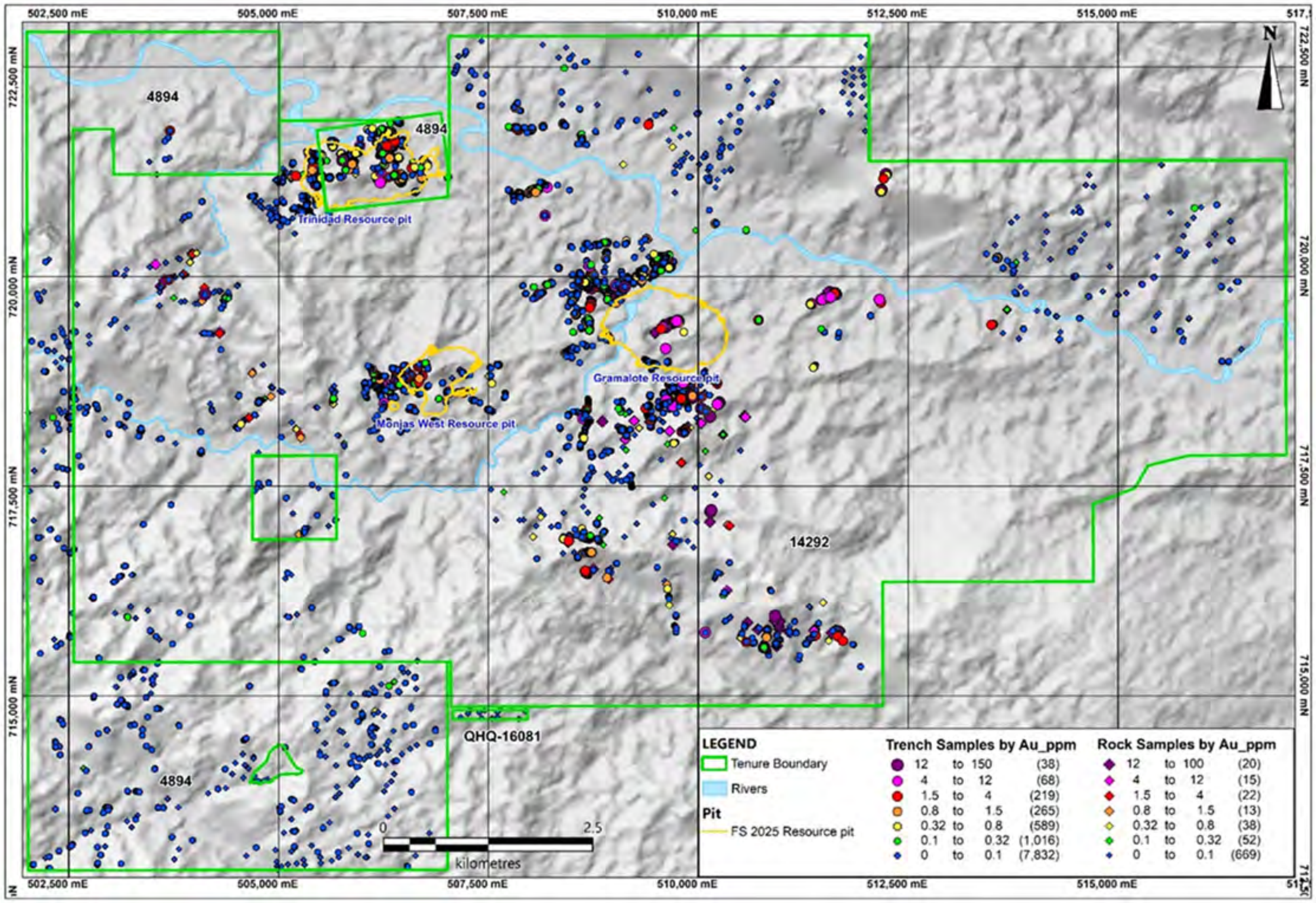
Host rocks to gold mineralization were classified as tonalites and granodiorites. The major minerals in veins and disseminations included pyrite, chalcopyrite, sphalerite, free gold, and rutile. Pyrite is the most common sulphide.

Figure 9-1: Location Plan, 2020–2024 Rock Chip and Trench Sampling



Note: Figure prepare by B2Gold, 2025.

Figure 9-2: Location Plan, 2020–2024 Rock Chip and Trench Sample Results



Note: Figure prepare by B2Gold, 2025.

Three types of pyrite and two types of chalcopyrite were differentiated, and correspond to the different mineralizing hydrothermal events.

The free gold occurs as fine elongated crystals (plates) or anhedral crystals, primarily as argentiferous gold coeval with several telluride and Bi-sulphosalt minerals. In massive sulphide veins, the argentiferous gold–telluride assemblage post-dates coarse-grained pyrite. Free gold occurs within fractured pyrite, and is closely associated with the second chalcopyrite type, and with the alteration minerals sericite and calcite. More rarely, free gold can be encapsulated within the second chalcopyrite type.

9.5 Exploration Potential

The Gramalote Ridge deposit remains open at depth. The mineralization is narrower to the east and west (Monjas East area) of the main deposit area.

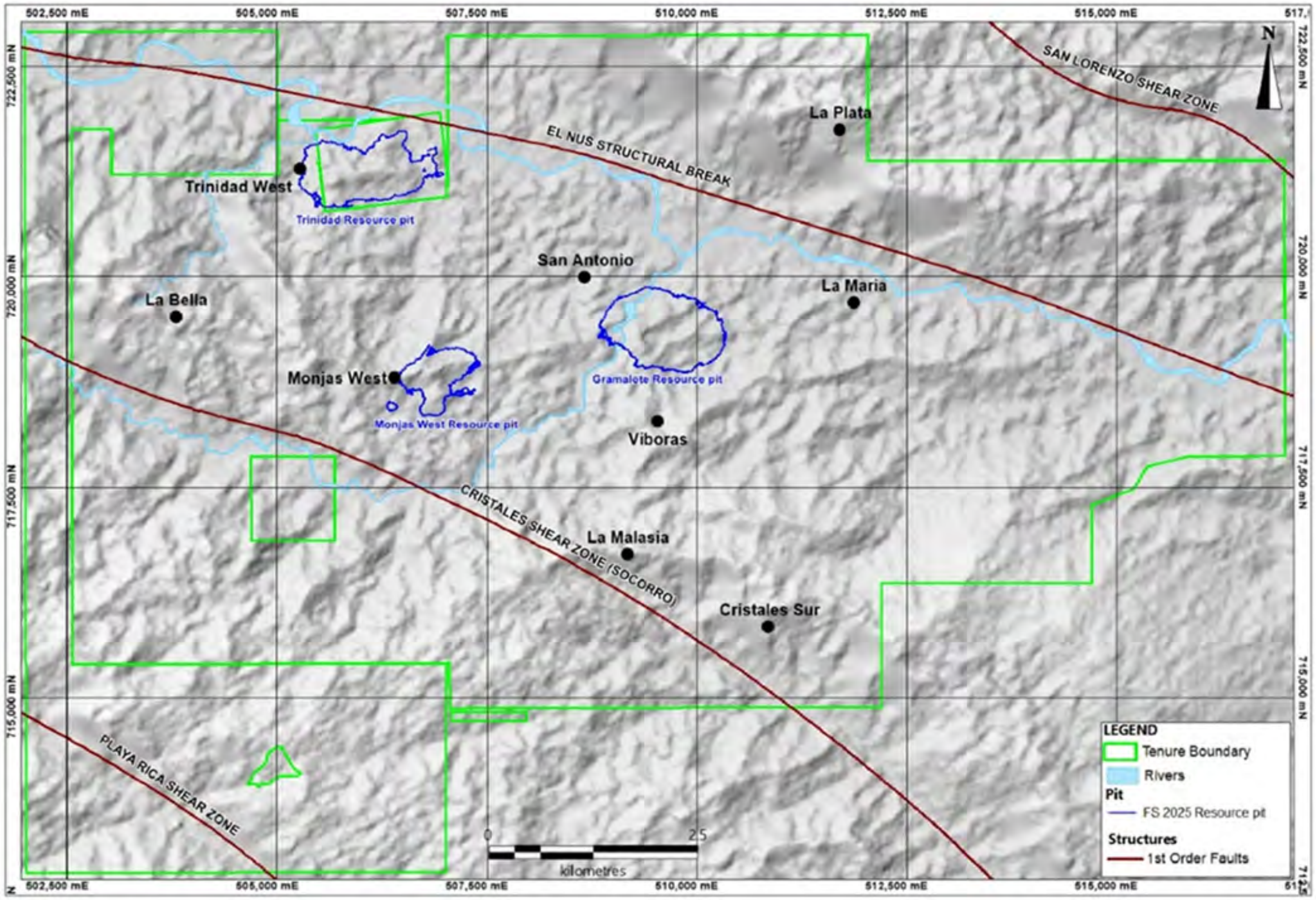
The Trinidad and Monjas West deposits remain open at depth, to the east, and particularly to the west.

A number of prospects were identified as summarized in Table 9-1. Prospect locations are shown in Figure 9-3.

Table 9-1: Exploration Potential

Target Name	Comment
Monjas West Extension	Situated 1,700 m west of Gramalote Ridge. Has artisanal workings. Soil and rock chip sampling identified a discontinuous gold anomaly with an area of 1,200 x 450 m. Tested by 53 core holes (19,977 m). Limited metallurgical testwork completed in two core holes. Western extension not drill tested.
Trinidad West Extension	Situated 4.8 km northwest of Gramalote Ridge. Northeast- and east–west-trending vein/veinlets with moderate quartz–sericite alteration. Rock and trenching samples define anomalous gold in possible western extension of Trinidad deposit. Western extension not drill tested.
San Antonio	Situated 500 m northwest of Gramalote Ridge. Northwest- and north–south-trending Type 1 vein mineralization. Selective vein sampling indicated anomalous gold values. Area was tested as part of condemnation drilling, with a total of 44 core holes (5,272 m) completed.
Viboras	Situated 1.8 km south of Gramalote Ridge. Has artisanal workings. Chalcopyrite-rich shear-veins with quartz–sericite alteration locally mined on a small scale. Soil sampling defined a 400 x 250 m gold anomaly. Not drill tested. Potential area to be used in the formalization program for small miners.
La Maria	Situated 2.5 km to the east of Gramalote Ridge. Has artisanal workings. Chalcopyrite-rich shear-veins with quartz-sericite alteration locally mined on a small scale. Soil sampling defined a 400 x 120 m gold anomaly. Tested by 26 core holes (7,562 m). First area used in the formalization program for small miners (mine currently in production).
La Plata	Situated 2.3 km northeast of Gramalote Ridge. Northwest- and north–south-oriented sheeted veinlets. Anomalous gold values returned from rock chip sampling within a 950 x 450 m area. Area proposed to be included in the formalization program for small miners. Not drill tested.
La Malasia	Situated 3 km south–southwest of Gramalote Ridge. Sub-parallel northeast, northwest and east–west-trending shear veins with strong quartz–sericite–pyrite ± chalcopyrite infillings along a 400 m section of La Malasia Creek. Gold-in-soil anomaly defined. Tested by 3 core holes (1,001 m). Area used in the formalization program for small miners.
La Bella	Situated 5.7 km west of Gramalote Ridge. Northeast- and north–south-trending Type 1 vein/veinlets mineralization. Rock sampling indicated anomalous gold values. Not drill tested.
Cristales South	Situated 3.9 km south–southeast of Gramalote Ridge. Northeast- and north–south-trending Type 1 vein/veinlets mineralization. Rock sampling indicated anomalous gold values. Not drill tested.

Figure 9-3: Prospect Areas



Note: Figure prepared by B2Gold, 2025.

10.0 DRILLING

10.1 Introduction

Table 10-1 summarizes the drilling to May 21, 2025 on a Project-wide basis. Figure 10-1 shows the collar locations. Drilling completed on the Project includes RC and core drilling, totalling 1,408 drill holes (269,049 m). There has been no drilling since May 21, 2025.

The resource drilling database cut-off date for Gramalote Ridge was March 9, 2022. There are 907 drill holes (approximately 193,126 m) and two adit samples (481 m) supporting the Mineral Resource estimate for Gramalote Ridge (Table 10-2). There are 147 holes (approximately 23,944 m) supporting the Trinidad Mineral Resource estimate (Table 10-3). The database cut-off date for the Trinidad estimate was July 14, 2025. The resource drilling database cut-off date for Monjas West was January 4, 2023. There are 55 holes (approximately 20,416 m) supporting the Monjas West Mineral Resource estimate (Table 10-4).

Drill hole collar locations used in the Mineral Resource estimates are shown in Figure 10-2 for Gramalote Ridge, Figure 10-3 for Trinidad drilling, and Figure 10-4 for Monjas West.

Drill holes completed for geotechnical and metallurgical purposes are included in the resource drilling totals. These drill holes were not systematically assayed, but the logging provided input on weathering surface and structural interpretations that are part of the resource models.

10.2 Drill Methods

AngloGold Ashanti contracted several Colombian drill companies, including Terramundo Drilling, Perfotec Ltda., Geominas Ltda., and Perforaciones Andina S.A., for the 2006–2007 drill program. The four contractors used a variety of core barrel sizes including HQ (63.5 mm core diameter), NQ (47.6 mm), and BQ (36.4 mm).

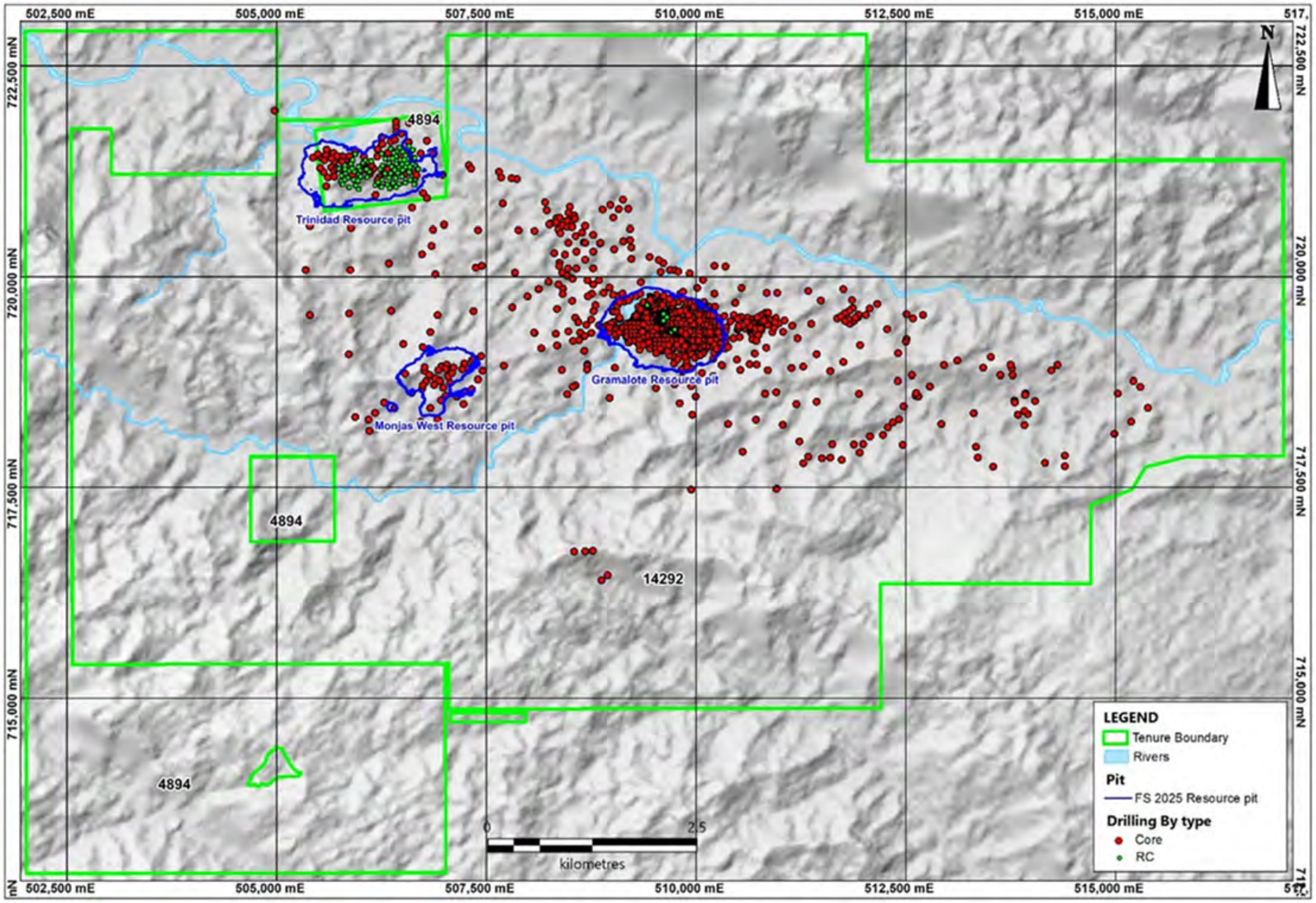
B2Gold used three drilling contractors during the 2008 drilling campaign: Kluane Colombia Empresa Unipersonal from Bogotá (drill rig KD600), AK Drilling International S.A. from Lima (drill rig UDR 200D LS), and Perfotec from Medellin (drill rig LF70PQ and Longyear 38). The three contractors used a variety of core barrel sizes including HQ, NQ, BTW (42.1 mm), and NTW (56 mm).

Gramalote Colombia has used its own drill fleet of five rigs and occasional contractors since 2008. A variety of core sizes were drilled, including NQ, NQ3 (45.1 mm), HQ, HQ3 (61.1 mm), PQ, and BTW. The RC hole diameter was 5½ inches.

Table 10-1: Project Drill Summary Table

Purpose	Year	Type	No. Drill Holes	Metres
2006–2007	Exploration	Core	46	13,062.11
2008	Condemnation	Core	1	200.9
2008	Exploration	Core	29	9,849.33
2008	Infill/definition	Core	60	20,138.37
2010–2017	Exploration	Core	74	24,928.89
2010–2017	Metallurgy	Core	18	4,035.26
2010–2017	Infill/definition	Core	350	58,902.38
2010–2017	Geotechnical	Core	32	6,629.13
2010–2017	Condemnation	Core	76	12,341.25
2010–2017	Infill/definition	RC	180	14,134.00
2019–2022	Infill/definition	Core	234	77,021.21
2019–2022	Exploration	RC	89	4,184.50
2019–2022	Metallurgy	Core	22	9,632.07
2019–2022	Exploration	Core	24	7,186.38
2019–2022	Infrastructure	Core	124	3668.02
2019–2022	Condemnation/exploration	Core	6	1699.64
2022	Hydrogeology	Core	11	527.62
2024	Infrastructure	Core	7	211.88
2025	Infrastructure	Core	25	695.54
Totals			***1,408***	***269,049***

Figure 10-1: Project Drill Collar Location Plan



Note: Figure prepared by B2Gold, 2025

Table 10-2: Gramalote Ridge Mineral Resource Drill Summary Table

Operator	Campaign	Purpose	Type	No. Holes	Metres Drilled
AngloGold Ashanti	2006–2007	Exploration	Adit	1	240.00
AngloGold Ashanti	2006–2007	Exploration	Core	43	12,312.34
B2Gold	2008	Exploration	Adit	1	241.10
B2Gold	2008	Exploration	Core	6	1,829.23
B2Gold	2008	Infill/definition	Core	60	20,138.37
B2Gold	2008	Condemnation	Core	1	200.90
AngloGold Ashanti	2010–2017	Metallurgy	Core	14	3,148.06
AngloGold Ashanti	2010–2017	Exploration	Core	23	8,183.11
AngloGold Ashanti	2010–2017	Infill/definition	Core	308	44,129.36
AngloGold Ashanti	2010–2017	Geotechnical	Core	17	6,048.91
AngloGold Ashanti	2010–2017	Condemnation	Core	16	2,488.63
AngloGold Ashanti	2010–2017	Infill/definition	RC	180	14,134.00
B2Gold	2019–2022	Metallurgy	Core	22	9,632.07
B2Gold	2019–2022	Infill/definition	Core	211	69,437.80
B2Gold	2019–2022	Infrastructure	Core	1	30.08
B2Gold	2019–2022	Exploration	Core	5	1,413.44
Subtotal adit				*2*	*481.1*
Subtotal drill holes				*907*	*193,126.3*
Total				***909***	***193,607.40***

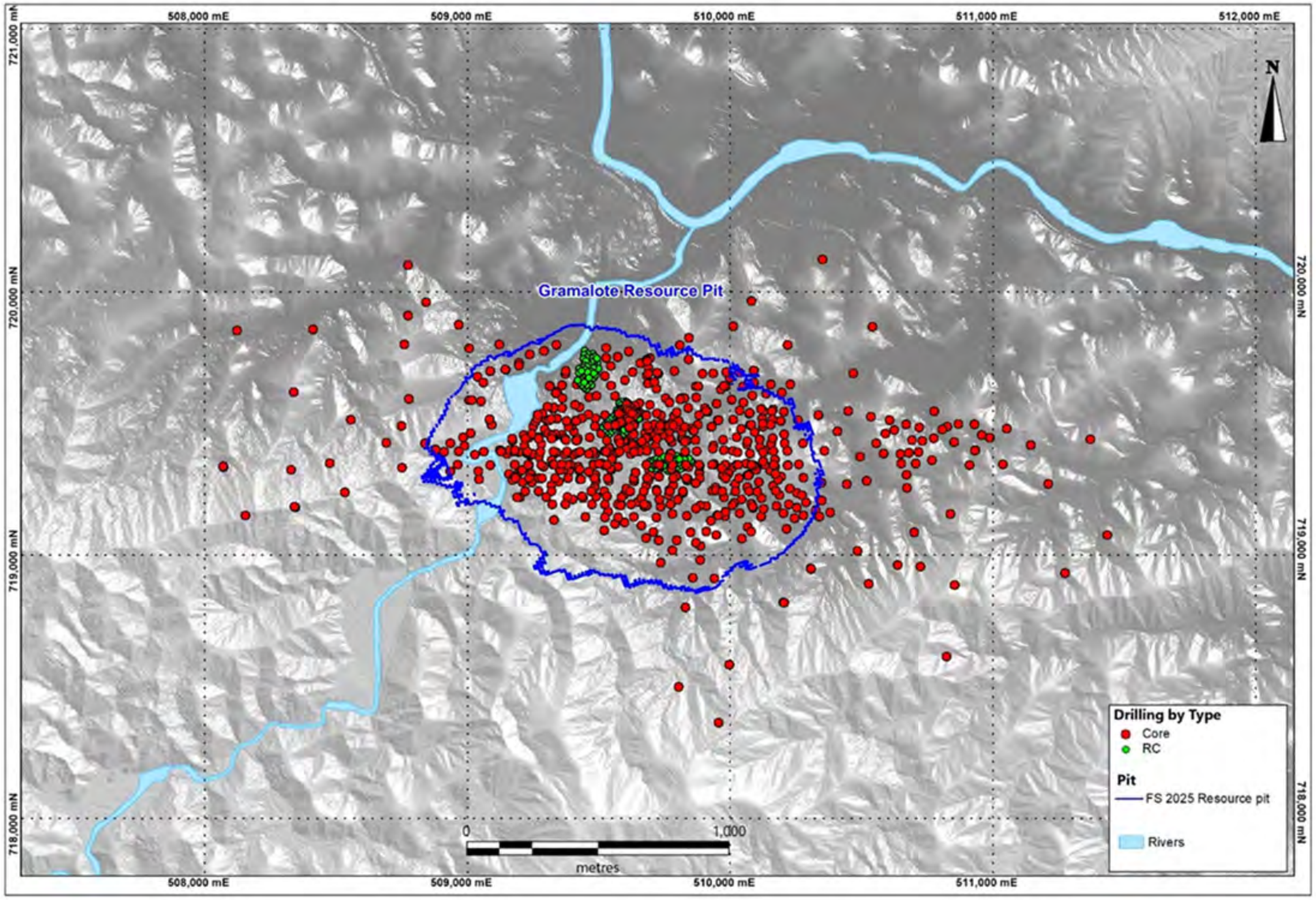
Table 10-3: Trinidad Mineral Resource Drill Summary Table

Operator	Campaign	Purpose	Type	No. Holes	Metres Drilled
B2Gold	2008	Exploration	Core	20	7,019.26
AngloGold Ashanti	2010–2017	Exploration	Core	12	4,230.10
AngloGold Ashanti	2010–2017	Metallurgy	Core	2	347.2
B2Gold	2019–2022	Exploration	RC	89	4,184.50
B2Gold	2019–2022	Infill/definition	Core	23	7,583.41
B2Gold	2019–2022	Exploration	Core	19	5,772.94
Total				***165***	***29,137.41***

Table 10-4: Monjas West Mineral Resource Drill Summary Table

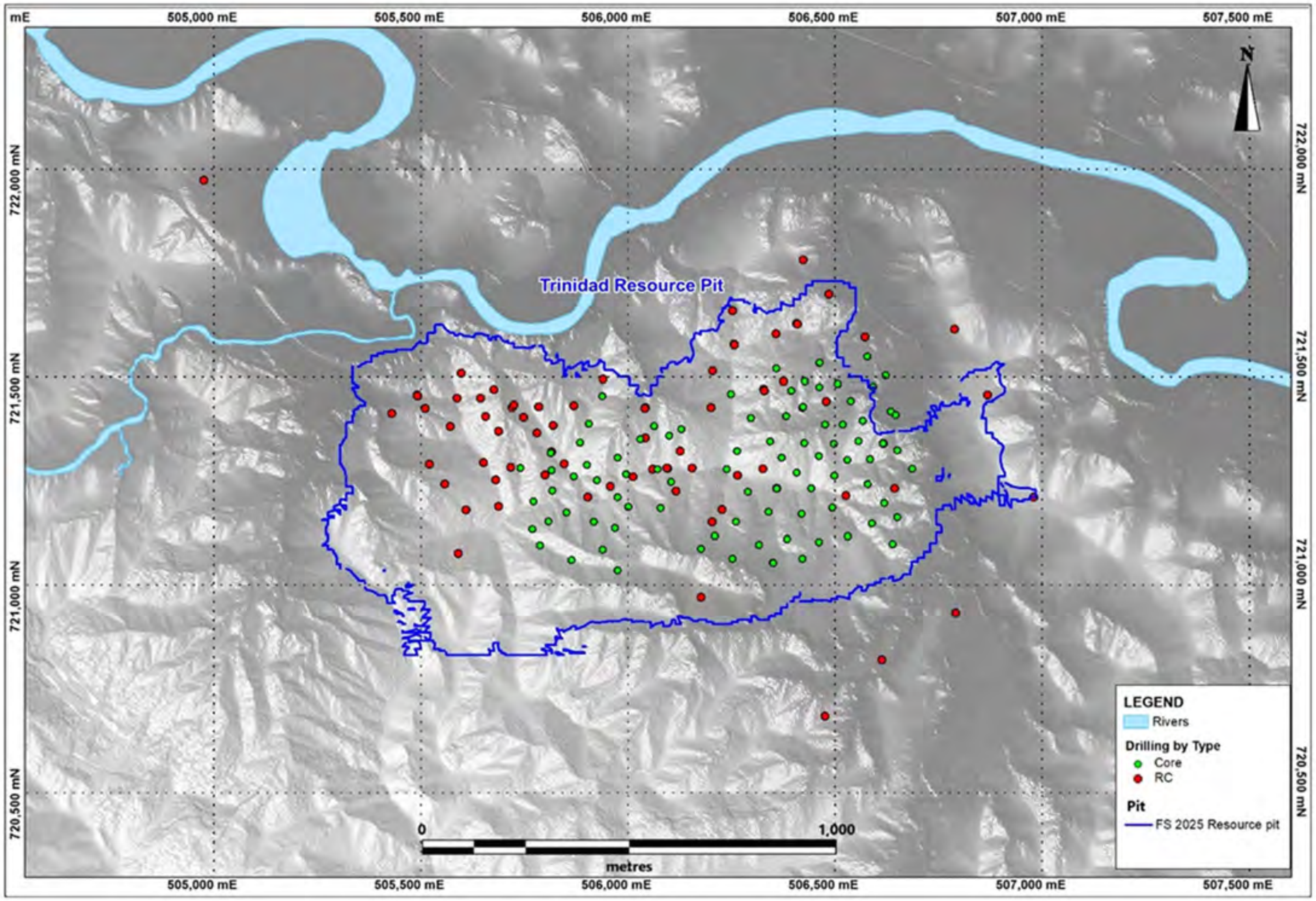
Operator	Campaign	Purpose	Type	No. Holes	Metres Drilled
AngloGold Ashanti	2010–2017	Condemnation	Core	1	150.00
AngloGold Ashanti	2010–2017	Exploration	Core	13	4,956.84
AngloGold Ashanti	2010–2017	Infill/definition	Core	39	14769.52
AngloGold Ashanti	2010–2017	Metallurgy	Core	2	540.00
Total				***55***	***20,416.36***

Figure 10-2: Gramalote Ridge Mineral Resource Drill Collar Location Plan



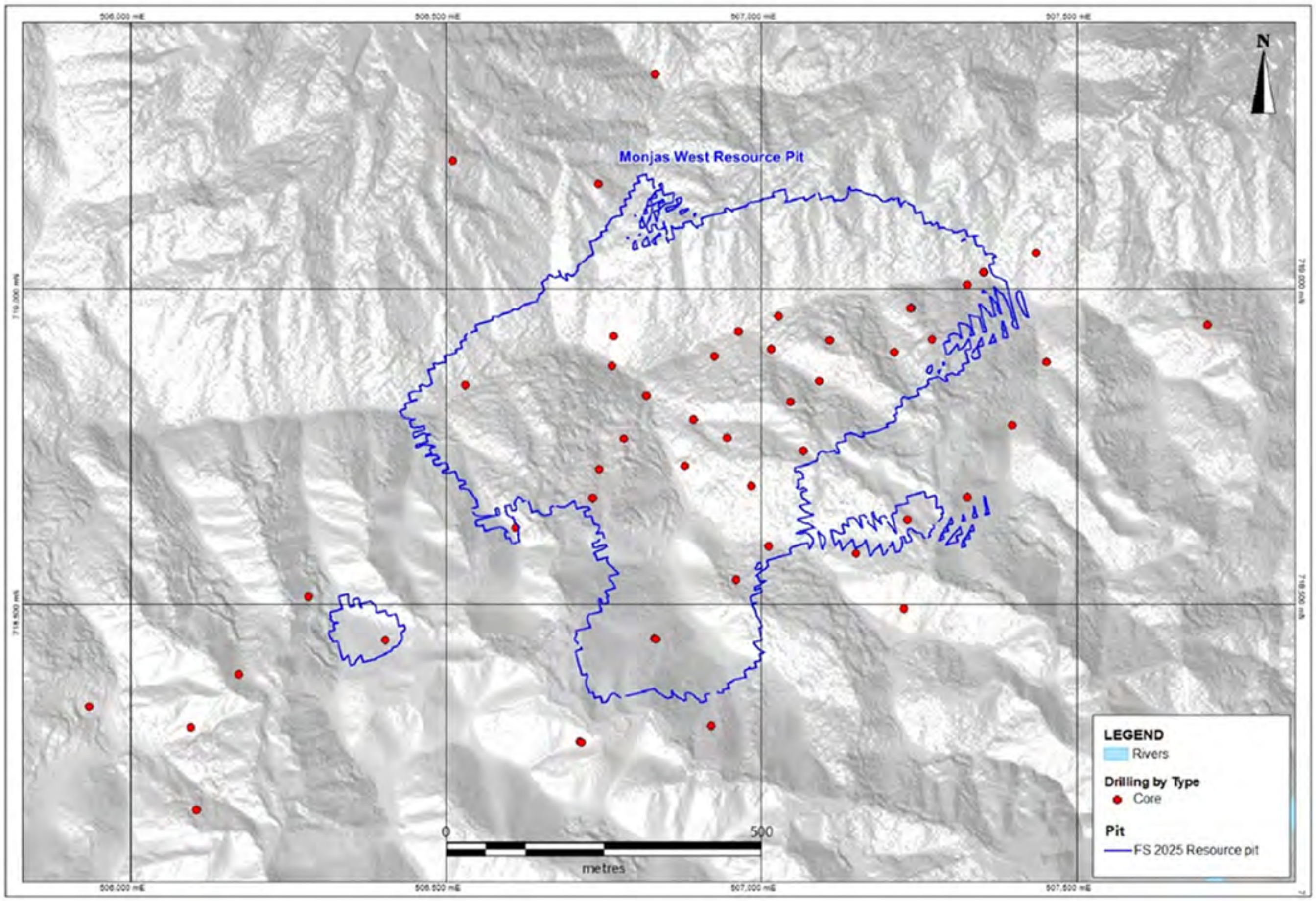
Note: Figure prepared by B2Gold, 2025

Figure 10-3: Trinidad Mineral Resource Drill Collar Location Plan



Note: Figure prepared by B2Gold, 2025

Figure 10-4: Monjas West Mineral Resource Drill Collar Location Plan



Note: Figure prepared by B2Gold, 2025

From 2019 to the Report Effective Date, B2Gold used drill contractors Kluane Drilling and Logan Drilling, and three Gramalote Colombia-owned drill rigs. The two contractors and Gramalote Colombia used HQ, NQ, HTW (81 mm), and NTW (56 mm) core sizes. The RC hole diameter was 114.3 mm.

Core and RC drilling is divided into four phases, by operator:

- AngloGold Ashanti: 2006–2007;
- B2Gold: 2008;
- AngloGold Ashanti: 2010–2018;
- B2Gold: 2019–to Report Effective Date.

Oriented core measurements were collected by B2Gold in the 2008 and 2019–2022 phases, using an Orishot Gen4 multishot core orientation tool.

RC drilling was completed in two phases:

- AngloGold Ashanti: 2013 and 2018; completed for test grade control purposes at Gramalote Ridge. Program comprised 180 drill holes spaced approximately 12.5 m apart in three separate areas; depths ranged from 27–100 m (average 79 m), with dip angles ranging from -54° to -72°;
- B2Gold: 2019-2020; testing oxide potential at Trinidad. Program comprised of 80 holes with depths ranging from 16–98 m (average 46 m), with dip angles ranging from -52° to -61°.

10.3 Logging Procedures

Drill core is transported from the rig in covered core boxes. Core is delivered to the logging area by the drill contractors or retrieved with company vehicles.

The geological logging protocols have undergone modifications over time.

During the 2008 B2Gold campaign, data was captured on paper, entered into Excel (dual data entry, comparison, correction), and then imported into and stored in an Access database. From 2010–2015, all geological data was entered onto A3-size log sheets with pre-set logging parameters that had to be noted, including lithology, alteration (dominant, subordinate and trace), types of veinlets, sulphide mineralization, and comments. These data were subsequently entered into DH Logger, then imported into a Century Systems Fusion database. Between 2015–2018 data entry was done directly into Excel capture templates that were subsequently imported into the Fusion database.

The most recent drill programs have the drill data entered directly into Excel and imported into a Project-specific Access database.

Core logging typically captures the lithology, alteration, and mineralization features. The Excel template includes tables for a quick and detailed geological logs, collar and downhole survey data, structural data, information on the core boxes, recovery, RQD and rock hardness, magnetic susceptibility and specific gravity, and information on any logging aspects that were subsequently modified.

Oriented core observations were captured using the detailed geological template during the oriented core campaigns. Geotechnical logging consisted of determination of RQD, core recovery and rock strength.

Magnetic susceptibility measurements were taken at 20 cm intervals, with a quality assurance and quality control (QA/QC) reading taken every 30 m.

Digital core photographs were taken of all drill core, prior to sampling, but after sample mark-up. Photographs were captured of two boxes simultaneously, with both a dry and a wet photograph taken.

RC logging included recording lithology, weathering profile, oxidation percentage, alteration, quartz percentage and sulphides percentage, if observed.

10.4 Recovery

Core recoveries are generally excellent with an overall project average of 96.8%.

Recoveries during the 2006–2007 AngloGold Ashanti campaigns averaged 96.8%.

Core recovery in 2008 averaged 96.47% for the duration of the drill program.

Core recovery for the Gramalote Colombia drilling completed in the period 2010–2017 averages 95.8%.

During the 2019–2022 B2Gold campaigns, core recovery averaged 98%.

10.5 Collar Surveys

The 2006–2007 program drill hole collars were surveyed using a high-precision differential GPS instrument.

During the 2008 program, drill hole collars were located on the drill pads by total station survey. Once the drill rig commenced drilling, a second high-precision total station survey was completed at the top and bottom of a drill rod in the drill chuck so that a starting azimuth and starting inclination could be calculated. The procedure was used in 2008–2009 by B2Gold for permanently marking drill hole locations.

During 2019–2022, the location of proposed drill holes was completed using GNSS Topcon GR-5 equipment, under the RTK mode. The final location of the actual drill hole and alignment to start drilling was completed using a total station instrument.

10.6 Downhole Surveys

Drill holes 1 through 7 of the 2006–2007 campaign were surveyed at the end of the hole using a Pajari instrument. Drill holes 8 through 43 were surveyed every 100 m down hole and at the end of the hole.

Down-hole surveying for the 2008 drill program was performed by a B2Gold geotechnician upon completion of the drill hole. Reflex Maxibor II, a continuous optical borehole survey system, recorded azimuth and dip every 3 m down the hole. In the case of spurious readings, the hole was re-surveyed.

Gramalote Colombia drill programs used a number of different instruments with varying survey depths:

- EZ-track: 12 m intervals;
- Maxibor 11: 6 m intervals;
- Icefield: 7 m intervals;
- Pajari: 30 m intervals.

From the 2019 drill campaign onward, down-hole surveying was performed using a Gyro Master tool. This system recorded azimuth and dip every metre down the hole. Downhole surveys were performed every 20 m during the first 100 m of every drill hole to ensure the hole reached the proposed target. After 100 m depth, the surveys were performed every 50 m until the end of the drill hole was reached.

10.7 Geotechnical and Hydrological Drilling

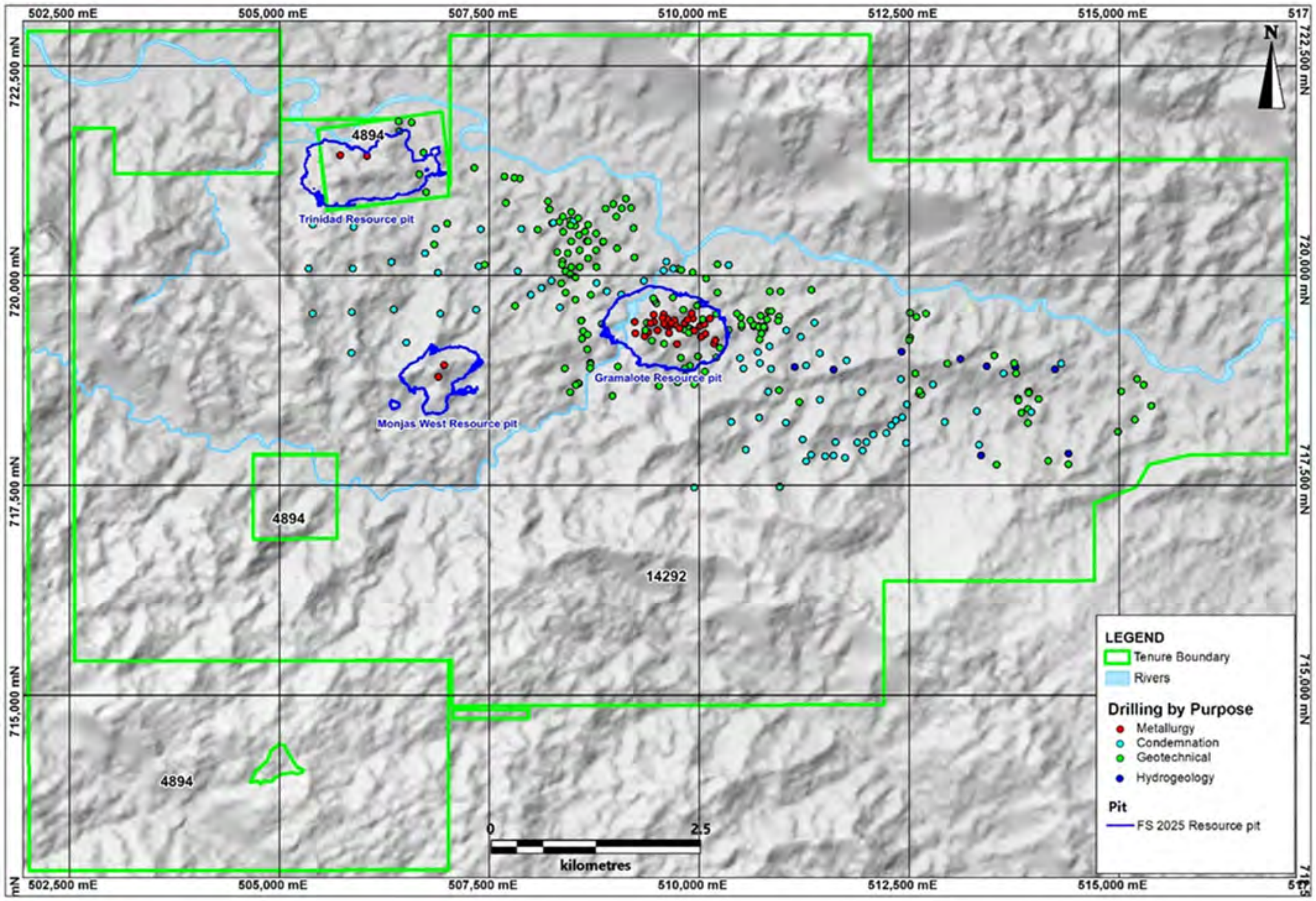
Geotechnical drilling provided geotechnical data in the areas of the proposed open pits and plant site infrastructure. For selected drill holes, geotechnical logging was complemented by structural logging, laboratory tests and acoustic Televiewer surveys.

These drill holes are included in the Project totals in Table 10-1. Drill collar locations are provided in Figure 10-5.

10.8 Metallurgical Drilling

Prior to 2019, drilling for metallurgical purposes included 14 drill holes (3,148.06 m) in the Gramalote Ridge area, two drill holes (347.20 m) in the Trinidad area, and two drill holes (540 m) in the Monjas West area. These drill holes are included in the Project totals in Table 10-1. Drill collar locations are included in Figure 10-5.

Figure 10-5: Collar Location Plan, Geotechnical, Metallurgical and Condemnation Drill Holes



Note: Figure prepared by B2Gold, 2025.

From 2019–2022, 22 drill holes (9,632.07 m) were completed in the Gramalote Ridge area for metallurgical tests. The 2019–2020 metallurgical drill program was executed in parallel to the infill drilling program. The intent was to use the same drill holes for infill purposes and to collect the remaining half core for metallurgical composites. The Master Comminution Composite and the Master Metallurgical Composite were collected from drill holes located in the core of the Gramalote Ridge deposit. One drill hole was completed for the Primary Domain Comminution test. Two drill holes were completed for the Master Metallurgical Composite, separated by 200 m along strike. These core holes were considered representative of the Gramalote Ridge mineralization for the recovery tests and process design. Additionally, nine comminution variability composites and 10 recovery variability composites were collected from the same number of drill holes. About 81% of Gramalote Ridge ore was sampled with the different metallurgical samples selected from the 2019–2020 program.

Early in 2022, one drill hole was completed for a high-pressure grinding (HPGR) recovery test. This drill hole is located in the central portion of Gramalote Ridge.

10.9 Condemnation

Prior to 2019, 17 condemnation drill holes (2,288.63 m) were completed in the immediate Gramalote Ridge area. An additional 60 condemnation drill holes (9,852.62 m) were completed in the San Antonio and Palestina areas.

During the 2022 drill campaign, six condemnation drill holes (1,699.64 m) were completed in the Concha area, near Gramalote Ridge.

These drill holes are included in the Project totals in Table 10-1. Drill collar locations are included in Figure 10-5.

10.10 Sample Length/True Thickness

During the 2006–2007 drill program, the holes were typically drilled either due east or due west with -60º dips and averaged 290 m in length. The selected orientation was based on early surface structural and geological mapping data. The drill spacing varied from 50–120 m and tested an area of about 900 x 400 m.

B2Gold used a predominantly northwest (315°) drill azimuth during 2008, based on an east–northeast multielement-based potassic enrichment and sodium depletion anomaly, and the results of surface structural and geological mapping available at the time. The northwest-oriented drill azimuth was considered appropriate to best intersect the different vein sets and alteration packages.

The Gramalote Colombia core holes were drilled predominantly with a northwest azimuth (315º), with 47–50º dips. Drill hole spacings range from 25 x 25 m to 100 x 100

m. An area of 200 x 100 m within the Gramalote Ridge high-grade zone was drilled on a 12.5 x 12.5 m grid for geostatistical purposes.

B2Gold used a northwest (315°) drill azimuth with 32–60º dips during the campaigns from 2019–2022. The northwest-oriented drill azimuth was still considered appropriate to best intersect the different vein sets and alteration packages. Drill hole spacings of 60 x 60 m were used within the infilled zones. Two areas of about 150 x 150 m area within the Gramalote Ridge medium- to lower-grade zones were drilled on a 25 x 25 m grid for geostatistical purposes. The first grid covered a distance of about 125 m along strike and 100 m of elevation range, with drill holes up to 175 m below surface. The first grid primarily targeted the Felipe Zone and the footwall wireframe areas. The second grid also covered about 125 m along strike and 100 m of elevation range; testing to 185 m below surface. This drilling targeted the Retiro and Main Gramalote wireframes.

Mineralized zones at Gramalote Ridge generally strike 075º and dip 75º to the south. Most drill holes were collared at an azimuth of 315º, but deviated to the right, so drill holes typically have azimuths of 317–318º and dip at about 45–50º. On average, the true width of the mineralization is about 75–80% of the downhole drilled length but varies depending on local orientation of the mineralized zones and the drill hole.

Trinidad mineralized zones generally strike 260–265° and dip 85° to the north. Most drill holes have a trend of 140° and dip of 45°. On average the true width of mineralization is approximately 70% of drilled length, but varies depending on the orientation of the mineralized zone and drill hole.

Mineralized zones at Monjas West generally strike 070–075º and dip 75º to the south. Most drill holes were collared at an azimuth of 290º, but deviated to the right, so drill holes typically have azimuths of 295–300º and dip at about 45–50º. On average, the true width of the mineralization is about 65–70% of the downhole drilled length, but varies depending on local orientation of the mineralized zones and the drill hole.

10.11 Drilling Since the Mineral Resource Database Close-off

No additional infill drilling was completed at Gramalote Ridge, Trinidad, and Monjas West after the resource estimate cut-off date. The drilling results to date confirm approximate mineralization widths as presented in the current Mineral Resource estimate.

Nine drill holes (1,795 m) post-date the resource estimate database cut-off for Gramalote Ridge. These holes were drilled for condemnation/infrastructure and are generally outside of the resource areas.

10.12 Comments on Drilling

In the opinion of the QP, the quantity and quality of the logged geological data, collar, and downhole survey data collected in the exploration drill programs are sufficient to support Mineral Resource and Mineral Reserve estimation and conceptual mining studies:

- Core and RC logging meets industry standards for gold and silver exploration;
- Collar surveys have been performed using industry standard instrumentation;
- Downhole surveys were performed using industry standard instrumentation;
- Recovery data from core and RC drill programs are acceptable;
- Drill orientations are generally appropriate for the mineralization style and the orientation of mineralization for the bulk of the deposit areas;
- Drilling was not specifically targeted to the high-grade portions of the deposits.

There are no drilling, sampling, or recovery factors known to the QP that could materially impact the accuracy and reliability of the drill results used in Mineral Resource and Mineral Reserve estimation.

11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Sampling Methods

11.1.1 RC

Samples are taken from 2 m runs. The sample is taken via a hose to the collector cyclone. Below the collector cyclone is located the splitter that has two chutes to divide the sample in two equal portions. The first one-half sample goes to the laboratory to be analyzed (sample), and the second one-half sample is returned to the drill hole (field reject). Only the coarse reject from the laboratory is preserved and stored. Between samples, the splitter is cleaned in three steps using air, then water, and a final air blast.

The sample is placed into tagged plastic bags which in turn go into rice bags. The rice bag markings include sample number, drill hole ID, and down hole depth. After packing the sample in double plastic bags, a portion is taken to place it in the chip tray for geology review. The plastic bags are then sealed and packed in the rice bags, normally one sample per rice bag as sample weights are 25–30 kg.

11.1.2 Core

Core sampling during the 2006–2007 AngloGold Ashanti programs was undertaken on nominal 2 m intervals. Samples were cut in half with a core saw. Variations in length from 2 m are due to mineralization or lithological changes.

B2Gold core holes from the 2008 program were sampled from top to bottom using variable core lengths observing breaks in alteration, mineralization intensities, and lithology differences. The majority of the samples are 1.6 m in length or less, with an average of 1.1 m.

Core sampling during the Gramalote Colombia programs from 2010–2018 was undertaken on nominal 2 m intervals. Samples were cut in half with a core saw. Variations in length from 2 m are due to mineralization or lithological changes.

During the B2Gold 2019–2022 campaign, samples were kept at 2 m in length, although sometimes shorter to avoid mixing very different alteration or mineralization features, especially at possible mineralization contacts. The minimum sample size was approximately 0.5 m. Samples were cut in half with a core saw.

11.2 Density Determinations

Since 2008, specific gravity (SG) or density determinations were collected on Project core samples using the Archimedes or wax immersion method. Additionally, pycnometer measures were collected for the first drilling campaigns. A summary of the available density data is included in Table 11-1.

Table 11-1: Density Data Summary

Drill Campaign Duration	Year SG Sample Taken	Number of Immersion SG Samples	Average of Immersion SG	Number of Pycnometer SG	Average of Pycnometer SG
2008	2008	1,142	2.67	261	2.62
2010–2017	2010	54	2.45	6	1.95
2010–2017	2011	1,065	2.57	68	2.40
2010–2017	2012	743	2.64	26	1.84
2010–2017	2013	1,054	2.64	27	1.82
2010–2017	2014	283	2.66	—	—
2010–2017	2015	126	2.68	—	—
2010–2017	2016	72	2.68	—	—
2010–2017	2017	164	2.49	14	1.89
2010–2017	2018	63	2.65	—	—
2019–2022	2019	150	2.66	—	—
2019–2022	2020	2,336	2.56	—	—
2019–2022	2021	2,389	2.56	—	—
2019-2022	2022	470	2.39	—	—
Total		***10,111***	***2.58***	***402***	***2.44***

Specific gravity information available before 2020 was not statistically representative for all types of material; measurements were predominantly performed on fresh tonalite and underrepresented weathered material. From 2020 onwards, and particularly during the 2021 drill campaign, specific gravity data from weathered material were measured, with samples collected every 3 m when possible.

A total of 29 test pits were excavated during September 2020 and used to calculate in situ saprolite densities to add confidence to the density values obtained from core samples.

11.3 Analytical and Test Laboratories

Sample preparation was completed by the following laboratories:

- ALS Bogota (prior to November 2012);
- ALS Bucaramanga (December 2012);
- ALS Medellin (January 2013–November 2018);
- Bureau Veritas Medellin (primary preparation laboratory; 2019–Report Effective Date);

- SGS Medellin (secondary preparation laboratory; November 2021–January 2022);
- ALS Medellin (secondary preparation laboratory; April, Sept–October 2020, February 2022 to the Report Effective Date).

ALS holds ISO 17025 accreditation for the preparation facilities in Colombia and the analytical facilities in Lima, Peru. Bureau Veritas holds ISO 9001:2015 accreditation for the preparation facilities in Colombia. SGS Colombia holds accreditation ISO/IEC 17021-1:2015 for the preparation facilities in Colombia.

ALS Lima was the primary analytical laboratory prior to 2019. The laboratory was accredited under INDECOPI, the Peruvian consumer agency, prior to 2010, and has held ISO 17025 accreditations for selected analytical techniques since 2010. ALS Lima has been used as the check laboratory for Bureau Veritas Lima and SGS Medellin originals since 2019.

Bureau Veritas Lima has served as the primary analytical laboratory since 2019. Bureau Veritas Lima holds accreditations for selected analytical techniques, including ISO 17025, ISO 9001:2015, ISO 14001:2015, and OSHAS 18001:2007. Bureau Veritas Lima was also used as the check laboratory on original ALS Lima assays.

SGS Medellin was used for primary analyses between November 2021 and January 2022. SGS Medellin was also used as an umpire laboratory (on ALS Lima original assays) prior to 2019. The laboratory currently holds the following certifications for selected analytical techniques: ISO 9001:2015, ISO 45001:2018, and ISO 14001:2015.

In 2020, as a results of Covid-related delays in Lima, both ALS Medellin and BV Medellin submitted some pulps to their respective Vancouver laboratories. ALS Vancouver holds ISO/IEC 17025:2017 accreditation since 2005. Bureau Veritas Vancouver holds ISO/IEC 17025:2017 since 2011. The Vancouver laboratories used the same gold–silver analytical techniques as their Peruvian equivalent but the coding on the silver determination was different (discussed below).

ACME Santiago was used as a check laboratory by B2Gold in early 2009 for 2008 ALS Lima originals. Accreditations used by the laboratory at the time are not known.

All of the laboratories listed in this sub-section were independent of the Project operator at the time the analytical work was performed.

11.4 Sample Preparation and Analysis

11.4.1 Sample Preparation

Sample preparation at the ALS laboratories consisted of drying, crushing to 70% passing 2 mm (pre-2011); crushing to 85% passing 2 mm (post-2011), and pulverizing to >85% passing 75 µm.

Sample preparation at Bureau Veritas Medellin consisted of drying, crushing to 85% passing 2 mm, and pulverizing to >85% passing 75 µm.

Sample preparation at SGS Medellin included drying, crushing to 90% passing 2 mm and pulverizing to >90% passing 106 µm.

11.4.2 Analyses

Analytical work by ALS Lima and ALS Vancouver comprised:

- Gold analysis by fire assay and atomic absorption spectrometry (AAS) ALS method Au-AA24, which has lower and upper detection limits of 0.005 g/t Au and 10 g/t Au respectively. For assays over 10 g/t Au, a gravimetric finish was applied (method Au-GRA22), with a lower detection limit of 0.05 g/t Au and an upper detection limit of 10,000 g/t Au;
- Silver analysis by four acid digestion and mass spectrometry (method MS62) which has lower and upper detection limits of 0.02 g/t Ag and 100 g/t Ag respectively. Prior to 2019 samples that assayed >100 g/t Ag were re-analyzed using method Ag-AA62, which had an upper detection limit of 1,500 g/t Ag. Where assay values were >1,500 g/t Ag, samples were analyzed using a gravimetric finish (method Ag-GRA21) that had an upper detection limit of 10,000 g/t Ag;
- Multi-element determination using a four-acid digest followed inductively coupled plasma (ICP) mass spectrometry (MS), using method ME-MS61 to provide a 48 element suite.

Analytical work by Bureau Veritas consisted of:

- Bureau Veritas Lima and Bureau Veritas Vancouver both completed gold analysis by fire assay with an atomic absorption (AA) finish (method FA450), with a lower and upper detection limit of 0.005 g/t Au and 10 g/t Au, respectively. For assays >10 g/t Au, re-assay was performed using a gravimetric (method FA550) with a lower and upper detection limit of 0.9 g/t Au and 1,000 g/t Au, respectively;
- Bureau Veritas Lima performed silver analysis using a multi-acid digest followed by ICP-MS (method 4A200-Ag), with a lower and upper detection limit of 0.1 g/t Ag and 200 g/t Ag, respectively;
- Bureau Veritas Vancouver conducted silver analyses using a multi-acid digest followed by ICP-MS or ICP emission spectroscopy (ES), method MA200-Ag, with a lower and upper detection limit of 0.1 g/t Ag and 200 g/t Ag, respectively. This was the same method as used by Bureau Veritas Lima; Bureau Veritas Vancouver uses a different method code;
- Bureau Veritas Lima completed multi-element analysis using a multi-acid digest followed by either ICP-MS or ICP-ES (method 4A250) to obtain a 59-element suite;

- Bureau Veritas Vancouver performed multi-element analysis using a multi-acid digest followed by either ICP-MS or ICP-ES (method MA250) to obtain a 59-element suite. This was the same method as used by Bureau Veritas Lima; Bureau Veritas Vancouver uses a different method code.

Analytical work by SGS Medellin comprised:

- Gold analysis consisted of fire assay with an AAS finish (method FAA515), with lower and upper detection limits of 0.05 g/t Au and 10 g/t Au respectively. For assays >10 g/t Au, samples were reassayed using a gravimetric finish (method FAG505), with lower and upper detection limits of 0.05 g/t Au and 10,000 g/t Au, respectively;
- Silver analysis was completed using a by four acid digest, and either ICP MS or ICP optical emission spectroscopy, and mass spectrometry quantification (ICM40B – ICP-OES/ICO-MS) with lower and upper detection limits of 0.02 g/t Ag and 50 g/t Ag, respectively. Overlimit samples (>50 g/t) were re-assayed using a multi-acid digest AAS (method AAS41B), with lower and upper detection limits of 10 g/t Ag and 4,000 g/t Ag respectively.

11.5 Quality Assurance and Quality Control

The analytical QA/QC program included submission of certified reference materials (CRMs or standards), blanks, and coarse reject and pulp duplicate samples.

CRMs were sourced from third-party suppliers, Geostats Pty Ltd, Rocklabs (now part of SCOTT), and CDN Resource Laboratories Ltd. Coarse blank material was purchased from SGS Colombia SA.

From 2008 to 2009, standards were inserted at the rate of 1:35 samples, blanks and duplicates were inserted at the rate of 1:40.

From 2010 to 2018, 14 controls samples were inserted in 100 sample groups based on the final digits of the sample number. Every 25 samples a standard or blank was inserted.

Since 2019, the analytical batch size has been set at 78 samples. Six QC samples are added to each batch, consisting of two standards, two blanks, and two duplicates. The duplicates comprise one preparation duplicate that is collected from the coarse reject after crushing and prior to pulverizing; and one field duplicate, which is either a quarter core sample or the lateral outcome of the rotary splitter in the case of RC samples. Sample crushing and pulverizing is monitored by sieve checks on randomly selected samples after both crushing and pulverizing processes. No significant preparation issues were identified.

The decision as to how many samples should be reanalyzed when a failure is identified depends on which quality control sample failed, a standard and/or a blank. If the two standards or all blanks inserted within a batch failed, a rerun will be requested for all of the samples in that batch. B2Gold personnel prepare monthly QA/QC reports. To date, no material issues have been noted as a result of the QA/QC evaluations.

Overall standard failure rates are less than 2%. The majority of the failed standards were rerun and passed on rerun. Blank failure rates are very low with only four failures in ~6,100 insertions. Overall standard analytical bias for assays associated with accepted results is neutral. The field duplicate precision curve is fairly flat at 92%. This suggests heterogeneity within the mineralization. Coarse reject duplicates have a precision of 17.6% at 0.5ppm and pulp duplicates have a precision of 27% at 0.5 ppm.

When B2Gold commenced using Bureau Veritas Medellin in 2019 as a sample preparation laboratory, a series of silver analyses and granulometry tests using 75 µm material was conducted from 2019 to end 2020 to verify the laboratory procedures. The results of the umpire granulometry tests were within the expected levels of performance.

11.6 Check Assays

During the period 2007–2013, AngloGold Ashanti sent samples for check assay to ACME and SGS Medellin.

No check assays were completed during the period 2014–2017.

B2Gold sent selected samples from the 2018 drill program for check assay at Bureau Veritas Lima. Unfortunately, pulps from the 2014–2017 period had deteriorated and could not be used. From January 2020 onward B2Gold routinely submitted pulp samples for check assay:

- Bureau Veritas Lima original samples were sent to ALS Lima;
- SGS Medellin original samples were sent to ALS Lima;
- ALS Lima original samples were sent to Bureau Veritas Lima.

Gold was analyzed by the umpire laboratory using the same method that was used for the original samples.

B2Gold submitted and evaluated the results of the check assay programs on a quarterly basis. Umpire assays of 2017–2022 originals average 97.7% of the originals, well within acceptable limits, and provide good support for the original assays.

11.7 Databases

The initial database, in 2006, was a Fusion database from Century Systems. In 2008, the data were migrated to an Access platform. When AngloGold Ashanti resumed Project control in 2010, the data were re-transferred to the Fusion platform.

B2Gold currently manages Project-specific data in Access. Data are entered into Excel capture templates that are imported and compiled into the Access database. The database is updated weekly with any new data generated, including new logging data, new assay results or logging data corrections. It is shared via Box.com with B2Gold's Vancouver head office, and the updated Excel export is stored on the Project server.

11.8 Sample Security

Sample security has not historically been monitored. Sample collection from drill point to laboratory relies upon the fact that samples are either always attended to, or stored in the locked on-site preparation facility, or stored in a secure area prior to laboratory shipment.

Drill core, pulps, and rejects are stored at the Project in purpose-built storage facilities and are under constant supervision and monitored by a closed camera circuit.

Chain-of-custody procedures consist of sample submittal forms to be sent to the laboratory with sample shipments to ensure that all samples are received by the laboratory.

11.9 Sample Storage

Drill core is stored at the Project in purpose-built storage facilities, and is under camera supervision.

Pulps and rejects are stored in a separate onsite facility within the Project area that is also monitored by the camera circuit.

11.10 Comments on Sample Preparation, Analyses and Security

In the opinion of the QP:

- Sample collection, preparation, analysis and security for RC and core drill programs are in line with industry-standard methods for gold–silver deposits;
- Drill programs included insertion of blank, duplicate, and standard reference material samples;
- QA/QC methods are practiced during density measurement programs, which are industry leading practices;
- QA/QC program results do not indicate any problems with the analytical programs (refer to discussion in Section 12);
- Data is subject to validation, which includes checks on surveys, collar co-ordinates, and assay data. The checks are appropriate, and consistent with industry standards (refer to discussion in Section 12);

- All core and RC chips have been catalogued and stored in designated areas.

The QP is of the opinion that the quality of the gold and silver analytical data is sufficiently reliable to support Mineral Resource estimation without limitations on Mineral Resource confidence categories. The data can support Mineral Reserve estimates.

12.0 DATA VERIFICATION

12.1 Internal Data Verification

12.1.1 Data Verification

Internal data verification includes the use of software tools that employ a set of scripts that identify and display any inconsistent data related to Project logging rules. Picklists, look-ups, and formulas within the Excel capture template help prevent missing or overlapping interval entries and entry of incorrect codes.

Validation query sets within the database evaluate the completeness/integrity of the data set for any given drill hole within and between data tables.

From 2020 onwards, senior geologists have periodically reviewed the database subset used for Mineral Resource estimation, for information consistency, consistency in use of designated codes and data completeness.

A detailed data validation was conducted in 2022 in support of the updated Gramalote Ridge model. Information reviewed as part of that process included detailed logging, structural data, collar, survey, recovery, RQD and magnetic susceptibility data, and verification consisted of comparing original source data against the data in the database. Where errors or omissions were noted, these were corrected as required.

12.1.2 Laboratory Inspections

Preparation facilities have been visited and detailed inspections performed by B2Gold, Gramalote Colombia and Anglo Gold Ashanti during the Project life.

The most recent visits were conducted by B2Gold personnel, who inspected the Bureau Veritas Medellin preparation facility in January 2021 and March 2022. No material issues were noted as a result of these reviews.

12.2 External Data Verification

12.2.1 Previous Technical Reports

Technical Reports were filed on the Project in 2008, 2009, 2012, and 2014, and a voluntarily filed report was completed in 2020. As part of the compilation of those documents, the QPs at the time reviewed the available QA/QC and supporting data. No material data issues were noted as a result of these reviews.

12.2.2 Laboratory Inspection

Third party consultants jsAnalytical Laboratory Consultants Ltd (jsAnalytical) audited the Bureau Veritas Medellin preparation facility in December 2019. No material issues were identified as a result of the review. jsAnalytical made some recommendations to improve selected laboratory practices, which Bureau Veritas Medellin implemented.

12.2.3 External Audits

AngloGold Ashanti requested two external audits of the then model and resource estimate, in 2013 (Parker, 2013), and 2019 (Wood, 2019). The audits provided comments on the estimation process and the Wood (2019) audit also addressed drill spacing. Wood (2019) observed that the drill hole spacing classifications for Indicated (50 x 50 m) and Inferred (100 x 100 m) were acceptable, but the implementation resulted in a significant number of blocks with wider drill spacing than stated in the reports and recommended. Additional infill drilling was warranted to reduce the spacing, particularly for support of Indicated Mineral Resource classifications.

The models reviewed during the external audits are superseded by the model and approach described in Section 14.

12.3 QP Data Verification

12.3.1 Mr. Stephen Jensen

Mr. Jensen performed site visits (see Section 2.4.1).

During those site visits he personally reviewed and inspected diamond core and RC drilling at various drills and sites and the core retrieval and handling procedures; core and RC logging and data collection procedures, protocols, and geological control; core photography procedures and quality; core and RC cutting and sampling procedures including sample QAQC protocol; drill collar and downhole survey practice; core storage and security; density measurement and density QA/QC procedures; and sample shipping and chain of custody procedures.

Mr. Jensen also reviewed data entry, verification and filing procedures including inspections of data organization and filing; database management procedures; and accuracy of geological and grade interpretations on sections and levels and in geological models.

Mr. Jensen reviewed the Mineral Resource estimation procedures and methods including data tabulations, modelling, and validation.

As a result of the data verification, Mr. Jensen concluded that the Project data and database are acceptable for use in Mineral Resource and Mineral Reserve estimation and can be used to support mine planning.

12.3.2 Mr. Peter Montano

Mr. Montano performed a site visit (see Section 2.4.2).

He performed a number of reviews in support of the open pit Mineral Reserves and cost assumptions that included: Pit design and optimization parameters; open pit geotechnical designs; equipment selection; production and development rates; sustaining and non-sustaining capital and operating costs; and sensitivity of costs to key input parameters. Mr. Montano reviewed the Mineral Reserves mine plan, production assumptions, and results.

He checked the financial model, in particular model values against the capital and operating cost estimates, and reviewed the model results.

As a result of the data verification, Mr. Montano concluded that the data are acceptable for use in Mineral Reserve estimation and can be used to support mine planning.

12.3.3 Mr. John Rajala

Mr. Rajala performed a site visit (see Section 2.4.3).

He has performed reviews of the available metallurgical testwork data supporting the metallurgical recoveries used in the LOM plan and amenability of the mineralization within the LOM plan to the proposed process facilities; assessed process plant consumable requirements for suitability for LOM plan purposes; and reviewed sustaining and operating cost forecasts for the process plant in the LOM plan. As a result of the data verification, Mr. Rajala considers that the metallurgical recovery forecasts used in the Mineral Resource, Mineral Reserve and economic analysis supporting the Mineral Reserves are appropriate. The process portion of the LOM plan can be used to support the Mineral Reserve estimates.

12.3.4 Mr. Ken Jones

Mr. Jones performed a site visit (see Section 2.4.4).

He undertook reviews of, and discussed aspects of, environmental approvals; environmental compliance and environmental issues; closure and reclamation planning and cost estimates for closure; and social engagement with local stakeholders and communities with appropriate B2Gold staff. He participated in reviews and discussions with staff responsible for obtaining and maintaining permits.

As a result of the data verification, Mr. Jones considers that the mine plan is achievable.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

13.1.1 Historical Testwork

Metallurgical testwork facilities involved in the initial testwork included SGS Lakefield in Lakefield, Ontario, Canada; the SGS analytical laboratory in Santiago, Chile; ALS analytical laboratories in Santiago and Perth, Australia; FLSmidth in Salt Lake City, Utah; Jenike and Johanson Limited (Jenike and Johanson) in Toronto, Canada; ADP Holdings (Lycopodium) in Perth, Australia, Metso Outotec Corporation (Metso Outotec) in Pennsylvania, USA; Julius Kruttschnitt Mineral Research Centre (JKMRC) in Brisbane, Australia; and the Cooperative Research Centre for Optimising Resource Extraction (CRC-ORE) in Brisbane, Australia. The laboratories and testwork facilities used were independent of B2Gold. Metallurgical laboratories are not typically accredited for testwork other than chemical analyses.

Testwork completed in the early stages was used to refine the approaches taken for the 2025 Feasibility Study.

13.1.2 Current Testwork

The majority of the testwork program supporting the 2025 Feasibility Study was conducted by SGS Lakefield during 2019–2020, with the remainder either completed at SGS Lakefield under supervision of a consultant/specialist or at another laboratory.

After the SGS Lakefield metallurgical testwork program was completed, additional metallurgical testwork to support high pressure grindings roll (HPGR) circuit selection was conducted by Metso Outotec during June 2022.

The programs completed are summarized in Figure 13-1.

13.2 Metallurgical Samples

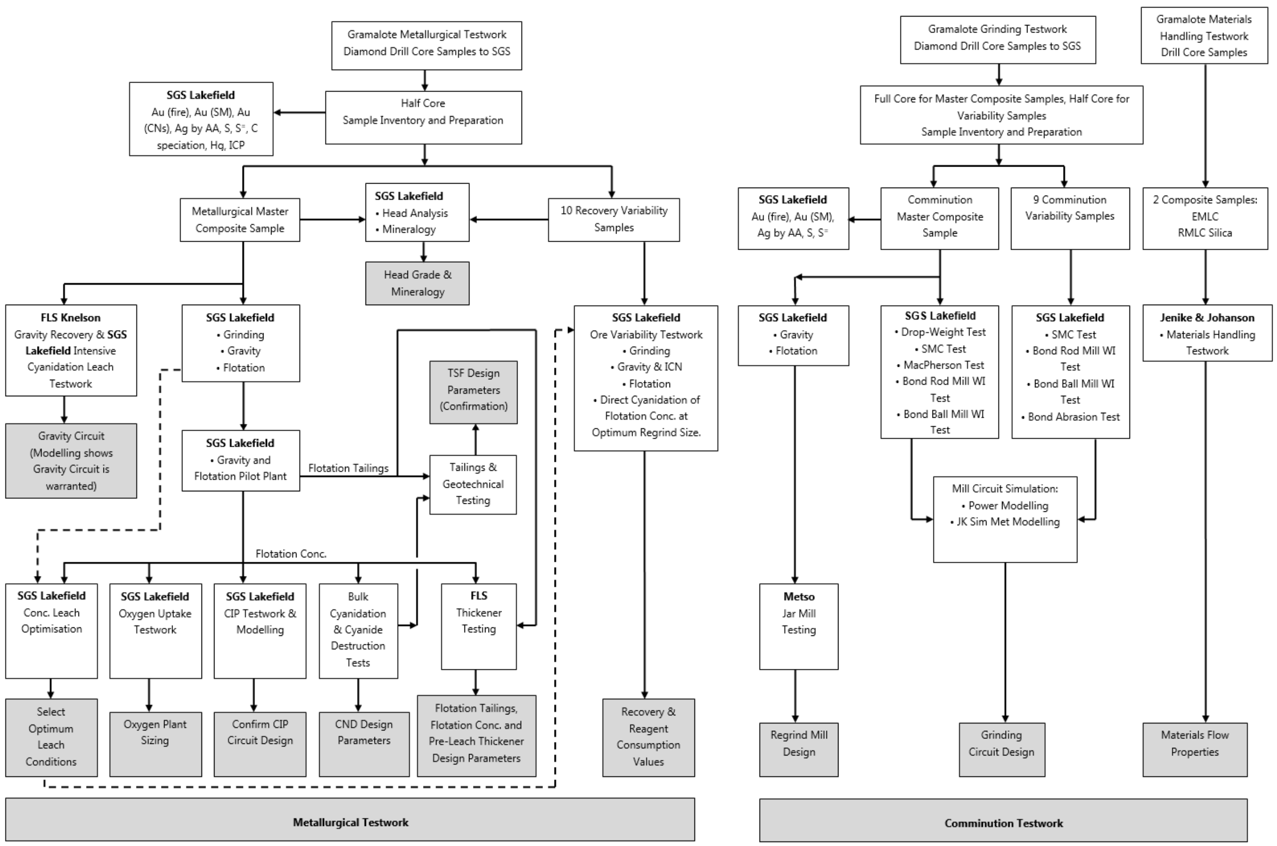
13.2.1 Metallurgical Domains

Due to the relatively consistent nature of the deposit in terms of host rock, the association of gold with veinlets, and the recommendation to domain on gold head grade, three gold head grade ranges were used to classify metallurgical samples rather than traditional metallurgical domains:

- Low-grade: 0.30–0.69 g/t Au;
- Medium-grade: 0.70–0.95 g/t Au ;
- High-grade: >1.1 g/t Au.

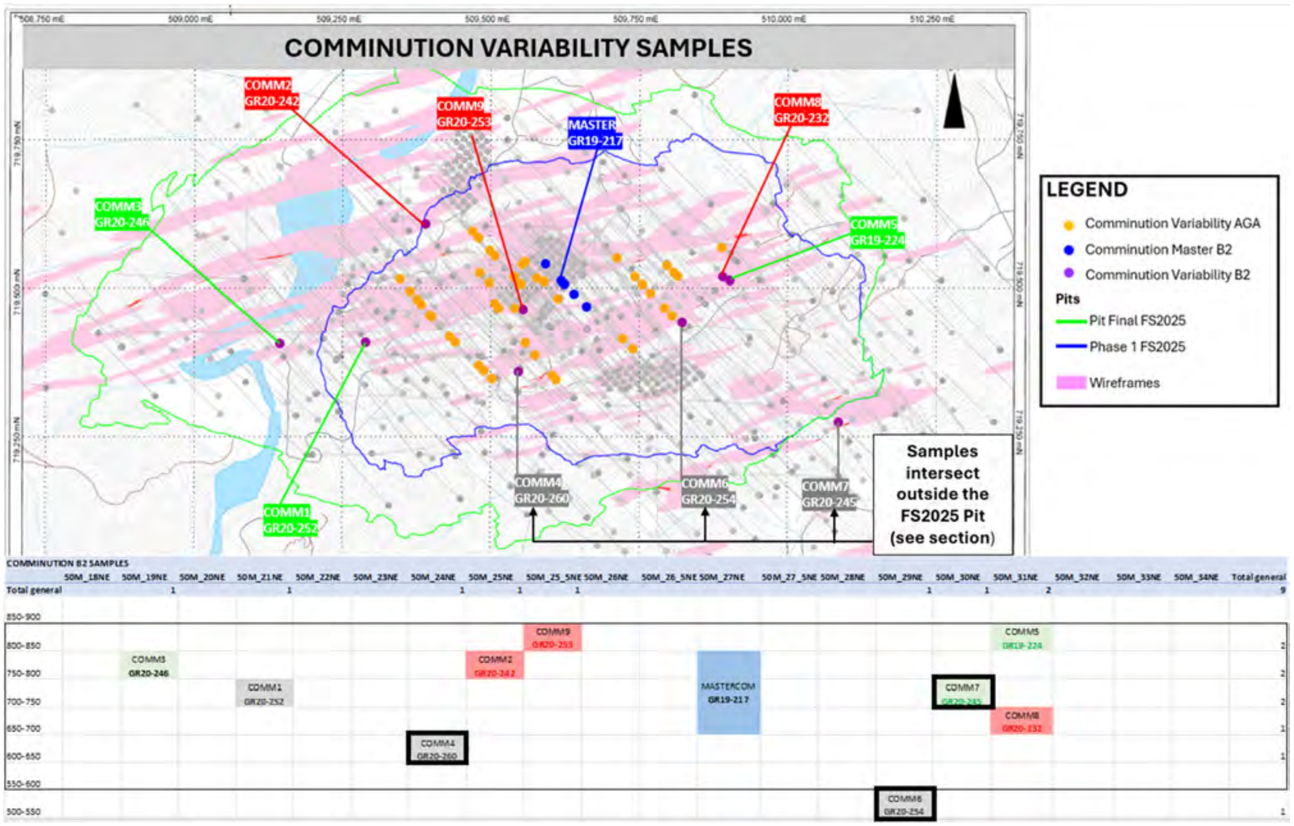
Sample locations for the various composites are shown on Figure 13-2 and Figure 13-3.

Figure 13-1: Metallurgical Testwork Program Flowchart



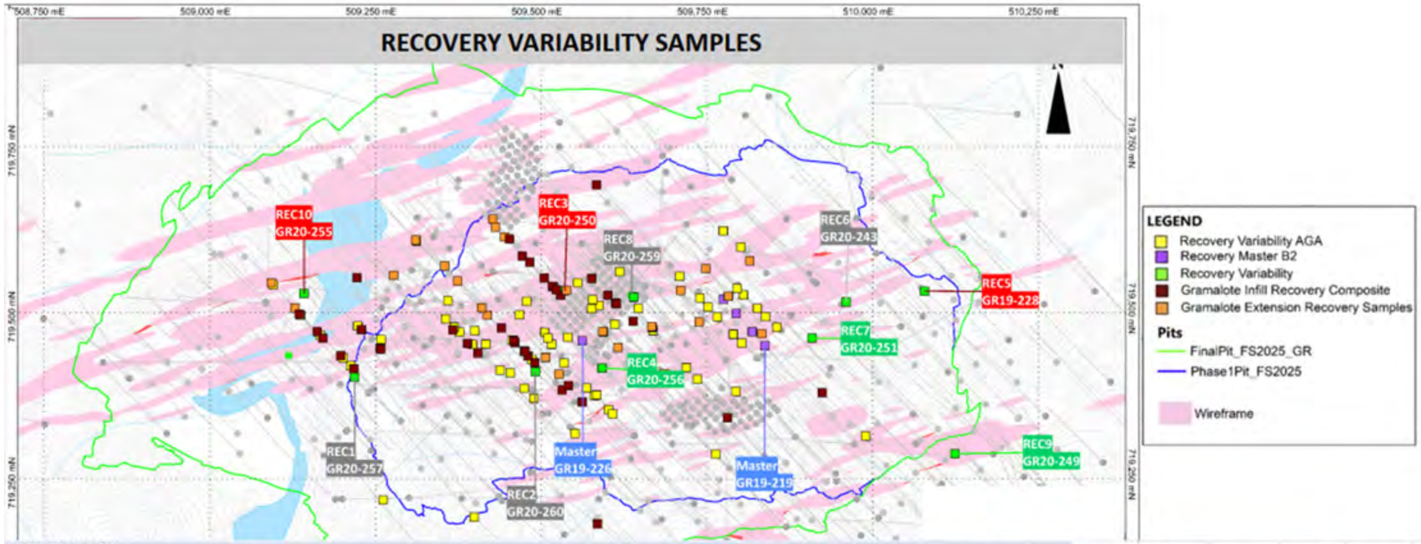
Note: Figure prepared by Lycopodium, 2025

Figure 13-2: Location of Master Comminution Composite and Comminution Variability Samples



Note: Figure prepared by Lycopodium, 2025

Figure 13-3: Location of Master Metallurgical Composite and Recovery Variability Samples



RECOVERY B2 SAMPLES

Etiquetas de fila	50M_18NE	50M_19NE	50M_20NE	50M_21NE	50M_22NE	50M_23NE	50M_24NE	50M_25NE	50M_25_5NE	50M_26NE	50M_26_5NE	50M_27NE	50M_27_5NE	50M_28NE	50M_29NE	50M_30NE	50M_31NE	50M_32NE	50M_33NE	50M_34NE	Total general
Total general			2				1	1		1			1			1	1	1		1	10
850-900								REC4 0.89g/t Au													1
800-850			REC10 1.65g/t Au																		1
750-800									MMCR				REC8 0.43g/t Au				REC9 0.79g/t Au				2
700-750			REC1 0.39g/t Au						MMCR						MMCR					REC5 1.50g/t Au	2
650-700										REC3 3.60g/t Au					MMCR			REC6 0.60g/t Au			2
600-650															MMCR						
550-600																REC7 0.94g/t Au					1
500-550							REC2 0.48g/t Au														1

Note: Figure prepared by Lycopodium, 2025

13.2.2 Master Composite Samples

The master comminution composite (MCC) and master metallurgical composite (MMC) samples were sourced from drill holes located in the central zone of the Gramalote Ridge deposit and are considered representative of Gramalote Ridge mineralisation for the recovery and comminution tests.

13.2.3 Variability Samples

Variability samples were selected at different elevation ranges and proposed pit phases.

The comminution and recovery variability samples were also distributed roughly proportionally to the pit phases.

Comminution Variability Samples

Nine comminution variability samples were collected, each from individual drill holes, with a particular focus on high-value RQD intervals. For these samples, 56% had 90–100% RQD, 33% had RQD between 80–85%, and one sample had a 57.9% RQD. Although the comminution variability samples were not collected for recovery purposes, the nine composites were also distributed across the grade ranges.

Recovery Variability Samples

Ten recovery variability samples were collected, with each sample sourced from a single drill hole. The selection basis was the grade ranges and the metal contribution of single samples within the composites. No external dilution was included; however, due to the common presence of very low-grade intervals within the Gramalote Ridge deposit, 80% of the composites included internal dilution with intervals below the 0.1 g/t Au cut-off grade. None of the composites included intervals >15 g/t Au.

13.3 Metallurgical Testing

13.3.1 Head Assays

A comprehensive suite of assays was conducted on the master (metallurgical) composite and the 10 metallurgical variability (recovery) composites, including the following:

- Gold: Triplicate fire assay; screened metallics; cyanide soluble;
- Silver: AA;
- Sulphur: S_T, $S^=$, S(0ɵ), $SO_4^=$;
- Carbon speciation: Total, graphitic, organic, carbonate;

- Multi-element: ICP scan.

Metallurgical variability composites were also assayed for mercury. The variability samples and master composite were submitted for specific gravity measurement by pycnometer.

The comminution master composite was assayed for the following:

- Gold: Triplicate fire assay; screened metallics;
- Silver: AA;
- Sulphur: S_T and $S^=$.

Major observations included:

- The master composite gold head grade was 1.59 g/t Au (screened metallics), with triplicate assays ranging from 1.66–2.22 g/t. The grade variability in the head assays indicated a nugget effect. As such, gravity testwork should be pursued;
- The silver assay for the master composite was 1.4 g/t Ag, which gave a gold to silver ratio of 1.1;
- Gold assays for the variability composites ranged from 0.40–6.07 g/t Au (screened metallics). A nugget effect was noted in the triplicate gold assays for a number of samples;
- The silver assays for the variability composites ranged from <0.5 g/t (below the detection limit) to 2.9 g/t, which gives gold to silver ratios ranging from 0.2–2.1 and averaging 1.2;
- The copper assays in all composites were low to moderate, indicating that a standard elution circuit without cold cyanide wash would be suitable;
- Sulphur speciation indicated that most sulphur is present as sulphide sulphur ($S^=$);
- Carbon speciation showed a low level of organic carbon, so preg-robbing was not considered to be an issue;
- Mercury is <0.3 g/t, which is below detection limit for all samples tested.

13.3.2 Mineralogy

A mineralogical evaluation was completed using QEMSCAN analysis and gold deportment on the metallurgical master composite.

Bulk Mineralogy

The metallurgical master composite contained major amounts of plagioclase (33%), moderate amounts of quartz (26.9%), micas (13.4%), and potassium feldspars (11.4%), minor amounts of clays (5.9%), chlorite (3%), calcite (2.8%), and trace (<2%) amounts of pyrite (0.9%), other silicates (0.9%), and other minerals. The main clay minerals included illite and montmorillonite.

Gold Deportment Study

The sample preparation procedure for the gold deportment study included pre-concentration by gravity separation, followed by heavy liquid separation (HLS) and superpanning. Optical microscopy and SEM-EDS were conducted on the polished sections prepared from all pre-concentrated products for gold mineral scanning, identification, grain size measurement, and association characteristics.

Approximately 95.8% of the gold was concentrated in the HLS Sink fraction, which accounted for 6.7% of the total mass. The gravity recovery was highly efficient for the sample. Chemical analyses were also used to determine the distribution of gold by grade and gold minerals by association category in the pre-concentration fractions, and weighted to the overall sample.

A total of 595 gold grains were found in the sample. These occurred as liberated (44.8%), exposed (29.4%), and locked particles (25.8%), with average sizes of 16.3 µm, 6.4 µm, and 3.9 µm, respectively. Gold minerals identified were native gold (82.3%), electrum (16.6%), and other minerals ~1%. The major association of gold was with pyrite (88.2%) with lesser amounts associated with chalcopyrite/pyrite composites (4.5%) and chalcopyrite (4.0%).

13.3.3 Comminution Testwork

A suite of comminution tests was conducted to determine the variability of comminution parameters throughout the deposit.

The comminution master composite was submitted for the full suite of comminution tests including the JK drop-weight test (DWT), the SMC test, the MacPherson autogenous grindability (AWi) test, the Bond rod mill (RWi) and ball mill (BWi) tests, as well as the Bond abrasion test (Ai).

An additional nine comminution variability samples were submitted for the SMC test as well as RWi, BWi and Ai determinations.

The testwork was completed by SGS Lakefield, with the results from the drop-weight and SMC tests being interpreted and ranked by JKTech.

For the comminution master composite, the following conclusions were drawn:

- The DWT value of 33.3 is similar to the SMC test value of 34.0 which gave confidence in the SMC test results for the variability samples;

- This sample is characterised as hard with respect to resistance to impact breakage (A x b) and abrasion (ta) breakage, falling in the medium range of hardness with respect to all the indices from the MacPherson autogenous grindability test;
- The RWi value was 14.8 kWh/t which is considered medium hardness;
- The BWi values using closing screen sizes of 425 µm and 355 µm were 16.7 kWh/t and 17.0 kWh/t respectively;
- The BWi value, for a closing screen of 300 µm (48 Tyler Mesh), was 15.8 kWh/t which is considered moderately hard. It is unusual to see lower BWi values when a finer closing screen is used. SGS Lakefield confirmed that a lower bulk density was used for this test, which may have influenced the results. Since the comminution master composite was not used for comminution modelling, this did not affect~~ed~~ equipment sizing;
- The Ai value was measured at 0.599 g which is considered abrasive.

For the comminution variability composites, the following conclusions can be drawn:

- The samples ranged from medium to hard in terms of A x b values, ranging from 51.8–30.1, averaging 37.7;
- The RWi values typically ranged from 12.4–15.6 kWh/t, placing them in the moderately soft to moderately hard range of the SGS database, with the exception of one sample, Comm 4, that was found to be significantly softer at 9.9 kwh/t. The overall average value was 13.6 kWh/t;
- The BWi values ranged from 12.2–17.0 kWh/t for a closing screen of 300 µm, placing them in the moderately soft to moderately hard range of the SGS database. The overall average was 14.9 kWh/t. Comm 4 was again shown to be softer than the other samples, with a BWi of 12.2 kWh/t;
- The Ai values for the variability samples ranged from 0.447–0.620 g covering the moderately abrasive to abrasive range;
- Overall, the variability in the comminution test results among the nine variability samples was similar to historical results; however, the A x b and BWi distributions were respectively 7% and 10% harder on the 80th percentile design basis for the nine new variability samples compared to the historical distributions.

13.3.4 Gravity Concentration Testwork

Extended Gravity Recoverable Gold Testwork

The metallurgical master composite was submitted for a complete extended gravity recoverable gold (E-GRG) testwork program. The three-stage gravity test was completed at the SGS Lakefield and results forwarded to FLSmidth for analysis and modelling.

The metallurgical master composite was considered coarse in terms of the AMIRA grain size classification. It is very amenable to gravity recovery due to the high amount of coarse gravity recoverable gold. Installation of a gravity circuit was recommended.

Gravity Separation Testwork

Three bulk gravity separation tests (G-1 to G-3) were completed using the metallurgical master composite to create samples for bench scale flotation testwork. One bulk test was completed using the comminution master composite (G-4) to create a sample for vendor testing. The target grind size for the first test (G-1) was P_{80} 300 µm. The tailing sample from test G-1 was primarily used for grind size optimisation tests that evaluated the flotation response of the samples at P_{80} 300, 250, 200 and 150 µm. The selected grind size of P_{80} 250 µm was used as the target for the remaining tests.

The ground ore was processed through a Knelson MD-3 laboratory concentrate using standard conditions. The Knelson concentrate was upgraded using a Mozley C-800 separator, with the exception of test G-2, where the Knelson concentrate was not further processed.

The average gold recovery to gravity concentrate for the metallurgical master composite was 30.4%. The mass recovery to concentrate was between 0.10–0.15% for these tests. The silver recovery to gravity concentrate ranged from approximately 7–13%.

The average calculated gold head grade was 1.92 g/t Au, slightly higher than the screened metallic head grade of 1.59 g/t Au. The average calculated silver head grade was 1.8 g/t Ag, which was slightly higher than the direct head grade of 1.4 g/t Au.

The results confirmed the E-GRG testwork conclusions that a gravity circuit should be included in the process flowsheet.

13.3.5 Bench Scale Flotation Testwork

Bench scale flotation testwork on the metallurgical master composite was divided into a number of sections:

- Tests F-1 to F-4, F-1R and F-2R, which evaluated effect of grind size on gold recovery;
- Tests R2, R3 and R4, which evaluated an alternative frother and site water at the selected grind size;
- Tests LCT-1 to LCT-4, which compared a rougher/scavenger flowsheet with recycled scavenger concentrate, and a rougher/cleaner circuit;

- Bulk tests F-5 and F-6 to produce sample for downstream concentrate leach testwork;
- Bulk test F-7 to produce sample for the concentrate regrind vendor testwork.

Effect of Grind Size on Recovery

Four tests were conducted using the tailings from the G-1 gravity testwork. Each test used a 2 kg charge of gravity tailings, with potassium amyl xanthate (PAX) as the collector and MIBC as the frother. A natural pH was used for all tests. Grind sizes tested were approximately P_{80}s 300 µm, 250 µm, 200 µm and 150 µm. Repeat tests were conducted for the two coarsest grind sizes to ensure that the test results were reproducible at such coarse sizes (F-1R and F-2R). The tailings assay for the 300 µm tests were 0.03 and <0.02 g/t Au respectively, while the 250 µm tests gave <0.02 g/t Au for both tests. These tests show good repeatability so there is confidence in the test results.

For these tests, the mass recovery after 20 minutes of rougher flotation ranged from 2.9–3.3% with overall gravity plus flotation gold recoveries of 98.4–98.9%. Sulphur recoveries ranged from 96.6–96.8%. All flotation tests achieved very high gold and sulphur recovery with low mass recovery.

Test F-1 at a grind size of P_{80} 301 µm showed distinctly slower kinetics than the repeat test, although, the mass and sulphur recoveries after 20 minutes were similar to other tests. Gold recoveries in the flotation stage were also slightly lower for this test at 97.6% compared with an average of 98.4% for other tests in this series.

Based on analysis of the test results, a primary grind size of P_{80} 250 µm was selected for all further flotation work.

Optimisation Testwork

These tests examined an alternative higher flash point frother and the use of site water.

The tests completed using W31 frother instead of MIBC gave slightly higher mass recoveries for similar overall gold and sulphur recoveries. F-2R2, which used SGS Lakefield river water, had a mass recovery of 3.9% for an overall recovery of 98.8% for gold and 96.7% for sulphur. F-2R4, which used site water warmed to ambient temperature, had a mass recovery of 4.6% for an overall recovery of 98.9% Au and 96.7% S. In comparison, the average mass recovery for the two tests using MIBC at a grind size of P_{80} 250 µm was 3.0%.

Test F-2R3 was conducted with site water that had been refrigerated. This test gave low sulphur recoveries and mass recoveries compared with the other W31 tests. However, the overall gold recovery was unaffected.

Locked Cycle Tests

A number of locked cycle tests were conducted to evaluate flowsheet options such as a rougher/scavenger flowsheet with recycled scavenger concentrate and a rougher/cleaner circuit. Tests LCT-1, LCT-3 and LCT-4 were rougher/scavenger tests, while LCT-2 was a rougher/cleaner test.

For all tests, the gravity tailings from the G-2 test was used. Six cycles were used for all of the locked cycle tests to achieve equilibrium.

For the rougher/scavenger testwork, the following conditions were used:

- 7 minutes of rougher flotation and 13 minutes of scavenger flotation;
- Scavenger concentrate recycled to the head of the rougher in the next cycle.

For the rougher/cleaner tests, the following conditions were used:

- 20 minutes of rougher flotation;
- 7 minutes of cleaner flotation;
- 3 minutes of cleaner scavenger flotation;
- Cleaner scavenger concentrate recycled to the head of the rougher in the next cycle.

The results of the locked cycle tests showed that for the rougher/scavenger tests, 98.9% overall gold recovery was achieved with a total sulphur recovery of 96.5 to 97.0%. Mass recoveries ranged from 2.1–2.9% depending on frother type. The recoveries from the rougher/cleaner test were 87.4% overall gold and 83.4% total sulphur. Although the mass recovery was much lower at 1.1%, and the concentrate grade higher for the rougher/cleaner test, as Project economics favoured the higher recovery case, the cleaner flowsheet was not pursued further.

W31 Tests

W31 is preferred from a plant design perspective as MIBC requires a hazardous area classification due to its low flash point, which is undesirable. The W31 test results have formed the basis for plant design.

Overall gold recoveries of 98.8 to 98.9% were achieved with 96.5 to 96.7% sulphur recovery.

The selected flowsheet for pilot plant work was the rougher-only flowsheet. Plant design allowed for both the rougher only and the rougher/scavenger flowsheet, in the event that higher concentrate grades or lower mass recoveries need to be targeted. However, it is anticipated that the rougher only flowsheet will be the predominant operating method.

Bulk Rougher Test Results

Two bulk rougher tests were conducted using the metallurgical master composite to produce material for downstream testing.

Concentrate from the F-6 test was used for the initial grind/recovery cyanide leach tests.

13.3.6 Flotation Pilot Plant Testwork

Following completion of the bench scale testwork on the metallurgical master composite, a 990 kg bulk sample was processed through a gravity/flotation pilot plant to create sufficient sample for downstream testwork.

The key targets for the pilot plant were as follows:

- Flotation feed size of K_{80} 250 µm;
- 50 kg/h processing rate;
- Rougher only circuit;
- 3% rougher concentrate mass pull;
- Rougher concentrate grades of approximately 40 g/t Au and 20% S;
- Rougher tailing grades of approximately 0.02 g/t Au and 0.02% S;
- Gold and sulphur flotation recoveries of 97 to 98%;
- PAX collector addition rate of 30 g/t;
- W31 frother addition rate of 30 g/t;
- Gravity (Knelson) concentrate mass pull of 0.1%.

The pilot plant ran for three days, with approximately 20 hours of operation; day 1 was start-up while days 2 and 3 were used to take composite samples for overall mass balance purposes. The average PAX and W31 additions were 33 g/t and 30 g/t respectively with an average pH of 8.1.

The pilot plant was able to maintain its target of K_{80} 250 µm during the three-day campaign.

Metallurgical Balances

Composite samples, collected over one hour during steady-state operation, were taken during day 2 and day 3. These composite samples were used for the metallurgical balances. For the flotation circuit, the assays from each of these runs was reconciled using a generalised least squares method (Bilmat 10.2).

The gold and sulphur recoveries for the balanced pilot plant were 95 to 98% gold and 92 to 94% sulphur. The tailings grades were 0.03 to 0.05 g/t Au and 0.05 %S which are slightly higher than the bench scale work. It is not uncommon to see higher tailings grades for pilot scale equipment, so these results compare well to the bench scale tests. With higher mass recoveries to concentrate, it is likely that tailings grades could be further reduced.

The average overall flotation plus gravity gold recovery for the pilot plant was 97.3%. This did not take into account the downstream intensive cyanide leach extraction. The average balanced rougher concentrate grade was 39.2 g/t Au and 17.7% S.

The pilot plant gravity silver recovery was 10.2% from a calculated head grade of 1.6 g/t Ag. The flotation recovery was 69.6% which gave a total overall gravity and flotation silver recovery of 72.7%.

Specific gravity measurements of concentrate and tailings were taken on samples from PP3 survey.

Bulk Gravity-Flotation Testwork

A 300 kg bulk gravity and flotation test (F-7) was conducted using the comminution master composite in order to produce sample for the regrind vendor test. The test was completed using the optimum primary grind size and flotation conditions established for the metallurgical master composite.

The sample was ground to a grind size of P_{80} 250 µm and then passed through a Knelson Concentrator in three 100 kg batches (G-4). The combined Knelson concentrate was submitted for an intensive cyanide leach test (CN-28). The Knelson tailings was combined, mixed and rotary split into 30 individual 10 kg samples. These charges were used for 30 rougher flotation tests.

The combined gravity and flotation recovery was 94.8% gold and 64.7% silver with a flotation mass recovery of 3.1%.

The gravity concentrate from test G-4 was submitted for an intensive cyanide leach

The gold and silver leach extractions after 48 hours were 98.7% and 83.3% respectively.

13.3.7 Cyanide Leach Testwork

Intensive Cyanide Leach of Gravity Concentrate

Four intensive cyanidation tests, CN-1, CN-2, CN-15 and CN-28, were completed using the metallurgical master composite gravity concentrate. In addition, a re-leach test was also performed, where the residue from test CN-15 was submitted for an additional leach test using the optimised flotation concentrate leach conditions. The sample was reground to P_{80} 21 µm prior to the test. This test simulated the intensive cyanidation tailings being directed the flotation concentrate leach circuit.

Re-leach of the intensive cyanidation tailings was not included in the overall plant recovery calculations and should be considered as a potential Project upside.

The conditions used for the intensive cyanidation tests were as follows:

- 20 to 40% solids (w/w), depending on sample size;
- pH 10.5 to 11 maintained with lime;
- 20 g/L NaCN, maintained;
- “Leach Aid” addition based on metallurgical master composite gold head grade and test mass;
- 48 hour leach duration, with 24 hour subsample;
- Ambient temperature;
- No aeration.

The 48 hour gold extractions from the intensive cyanidation tests ranged from 96.8–98.7%, with an average of 97.5%. The silver extractions ranged from 66.2–84.5%, with an average of 76.9%. The re-leach test gave a gold extraction of 81.9% and a silver extraction of 40.2% after 30 hours of leaching after grinding to a P_{80} of 21 µm.

All gravity concentrates tested were amenable to an intensive cyanide leach.

Cyanide Leach of Flotation Concentrate

Grind and leach recovery tests were performed on F-6 bulk flotation concentrate to determine the optimum target regrind size and leach residence time. Grind sizes from P_{80} 250 µm (as is) to P_{80} 16 µm were evaluated.

The cyanidation testwork was conducted in bottle rolls under the following conditions:

- 35% solids (w/w);
- pH 10.5 to 11 maintained with lime;
- 6 hours of pre-aeration ahead of leaching;
- 1,000 g/t lead nitrate added at pre-aeration;
- 2 g/L NaCN;
- Dissolved Oxygen target 20 to 25 mg/L;
- 72 hour retention time;
- 35°C temperature maintained.

Six tests were conducted initially, with a repeat series conducted to confirm observations. Residues were analysed for gold in triplicate, but only the average is presented for brevity.

Results indicate that in the first series residue grades decreased until a grind size of P_{80} 27 µm was reached, with an unexpected increase in residue grade at P_{80} 22 µm before a reduction again at P_{80} 16 µm. In the second series, residue grades decreased with grind size. Cyanide and lime consumptions generally increased with decreasing grind size. Improvements in gold recovery were relatively small between grind sizes of P_{80} 16–24 µm.

The overall gold extractions ranged from 84.6% for Test 9 at P_{80} 263 µm to 97.7% for Tests 8 and 14 at P_{80} 16 µm. Residue grades ranged from 11.2–0.17 g/t Au. Kinetics for the finer grind sizes were very fast, with most of the gold extracted within 24 hours.

Similar to gold, the silver showed a general trend of increasing recovery with decreasing grind size. The overall silver extractions ranged from 42.1% for Test 3 at P_{80} 263 µm to 70.2% for Test 14 at P_{80} 16 µm. Residue grades ranged from 7.0–26.7 g/t Ag.

A grind size of P_{80} 20 µm was selected as the target for all further work.

A further series of three hard stop tests was conducted to select the optimum leach residence time, a using pilot plant flotation concentrate. Conditions were otherwise unchanged from the grind/recovery leach tests. The measured grind size was P_{80} 17 µm in each test.

The gold extractions were identical for each of the three tests, with slightly higher silver extractions for the longer residence times. Gold leach extractions were 97.9%, which yielded an overall gold recovery of 95.7%. There was very little variation in repeat gold residue grades, which gave confidence to the test conclusions and indicated that the samples contained very little free gold.

Based on the test results, a leach residence time of 30 hours was selected for all further testing. The selected residence time included a design factor to allow for one leach tank to be bypassed and still achieve 24 hours residence time.

Bulk Cyanide Leach

A single bulk test, using approximately 6.7 kg of the pilot plant flotation concentrate, was completed to create sample for cyanide destruction testwork. After the 30-hour leach, the pulp was contacted with carbon for six hours to simulate a carbon-in-pulp (CIP) circuit. The loaded carbon was removed from the solution prior to the cyanide destruction testing. The barren leach solution was submitted for a comprehensive solution analysis to allow parameters for the cyanide destruction test programme to be specified.

The gold extractions for the bulk leach test were lower than the comparable hard stop test results at 93.1% compared with 95.7%. The residue grades for the bulk test were 2.03 g/t Au and 19.1 g/t Ag compared with an average of 0.81 g/t Au and 12.5 g/t Ag for the hard stop tests. It is not uncommon to see slightly higher residue grades in a small tank leach compared with bottle roll tests due to possible mixing and oxygen dispersion inefficiencies. The dissolved oxygen reading for 24–30 hours was only 13.6 mg/L, compared with the target of 20–25 mg/L. It is possible that the lower oxygen level impacted leach kinetics for the bulk test. Cyanide and lime consumptions were also lower than in the hard stop tests.

13.3.8 Variability Testwork

Testing of the standard flowsheet at optimum conditions was conducted using each of the 10 variability composites. Each test included the following steps:

- Gravity separation of a 20 kg sample using a Knelson concentrator, followed by a Mozley table cleaner stage to target 20 g (~0.1% mass) total gravity concentrate.
- Intensive cyanide leach of gravity concentrate at as received K_{80}.
- Rougher flotation using 10 kg of Knelson tailings targeting approximately 3% mass recovery.
- Regrind of rougher flotation concentrate to a grind size of approximately P_{80} 20 µm.
- Cyanide leach of reground rougher flotation concentrate for 30 hours.

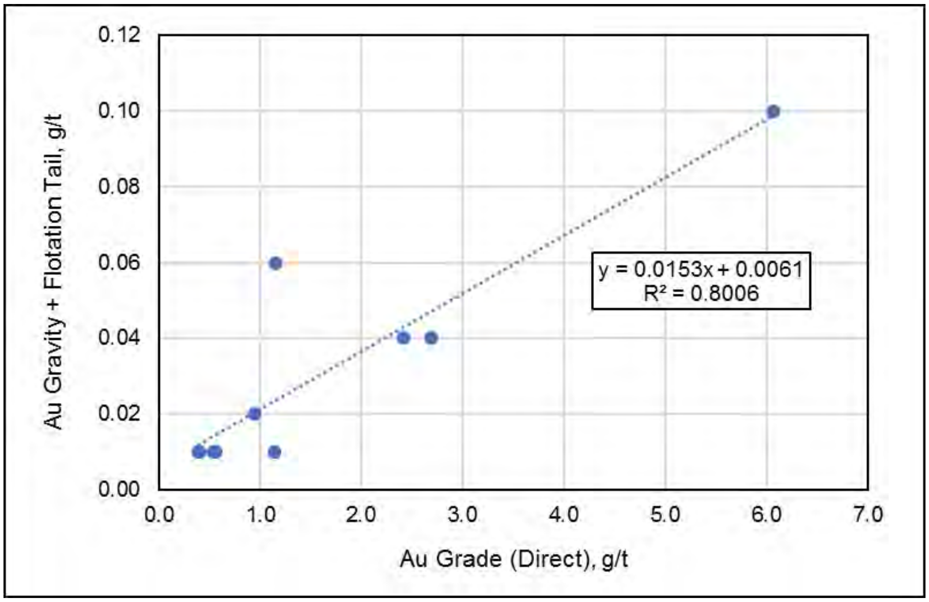
The overall gold and silver recoveries were calculated for each sample taking into account the gravity concentrate intensive cyanide leach extractions.

The gravity gold recovery values for the variability samples ranged from 16.6% to 46.7% and averaged 28.5%. The gravity silver recovery values ranged from 4.1% to 18.6% and averaged 8.5%.

The average flotation mass recovery for the 10 variability tests was 2.9%. The gold flotation unit recovery ranged from 93.0% to 97.8% and averaged 95.6%, which yielded an overall gravity plus flotation recovery of 96.8%. The average overall gravity plus flotation silver recovery was 60.7%. The average sulphur and sulphide recoveries were 93.6% and 88.0% respectively.

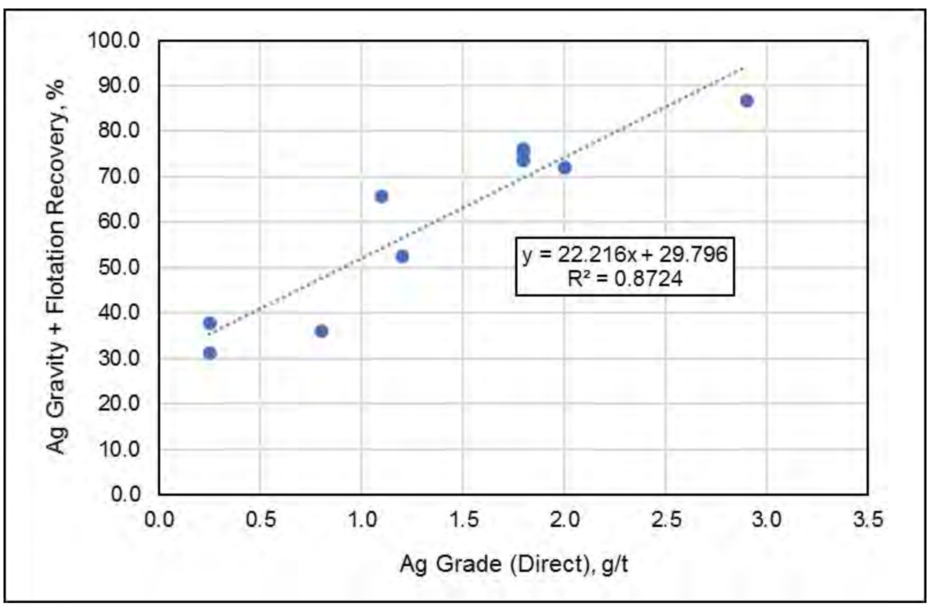
Figure 13-4 and Figure 13-5 show that there is a good relationship between head grade and rougher flotation tailings assay. This can be used to predict the overall gravity plus flotation recovery. There is also a relationship between sulphide head grade and flotation mass recovery that will be useful for operations predictive purposes.

Figure 13-4: Gold Gravity and Flotation Tail versus Head Grade



Note: Figure prepared by Lycopodium, 2022.

Figure 13-5: Silver Gravity and Flotation Recovery versus Head Grade



Note: Figure prepared by Lycopodium, 2022.

The intensive cyanide leach gold extraction ranged from 78.1% to 98.9% with an average of 93.2%. The silver extraction averaged 79.2%. The average overall gravity concentrate and intensive cyanide leach gold extraction was 27.0%.

The flotation concentrate gold leach extraction ranged from 97.1–99.2% with an average of 98.4%. The silver extraction ranged from 57.3–92.2% with an average of 75.1%. The calculated head grades compared well to the direct (flotation concentrate subsample) head grades. The average calculated gold head grade was 29.1 g/t Au compared with the direct head grade of 29.9 g/t Au. The average calculated silver head grade was 31.2 g/t Ag compared with the direct head grade of 32.4 g/t Ag. The reagent consumptions, on a kg/t cyanide leach feed basis, averaged 4.59 kg/t NaCN and 3.57 kg/t CaO.

There is a good relationship between cyanide consumption and the copper assay in the feed (to gravity and flotation). Copper was not measured in the concentrate leach feed, but it is likely that there would also be a relationship with this parameter as well, if it were available.

The average overall gold and silver recoveries, which take into account the intensive cyanide leach extractions, were 95.5% and 45.5% respectively. There is a logarithmic relationship between overall gold recovery and the calculated gold head grade, and a similar relationship for silver.

13.3.9 Overall Gold and Silver Recovery

The overall gravity, flotation plus cyanide leach gold and silver recoveries for the metallurgical master composite and the variability samples indicate that overall gold recoveries in the range of 92–98% and overall silver recoveries in the range of 27–60%, could be expected from the Gramalote Ridge ore when using the selected flowsheet. The average gold and silver recoveries for all the variability samples were 95.5% and 46.3%, respectively, while for the master composite recoveries were 95.7% and 54.1%, respectively.

13.3.10 Leach Kinetics and Carbon Modelling

Rougher concentrate from the F6 bulk rougher test was leached using the optimum leach conditions, for subsequent use in carbon adsorption and equilibrium testwork described below. Three leach tests were conducted:

- CN-23, ~200 g leach kinetic test conducted in a rolling bottle for a total of 72 hours. Solution sub-samples were taken at 1, 2, 4, 8, 12, 24, 30 and 48 hours.
- CN-24, ~1.2 kg bulk leach test conducted in a rolling bottle for 72 hours. Pulp from this test was used for equilibrium isotherm tests.

- CN-25, ~300 g leach kinetic test conducted in a rolling bottle for 72 hours. Solution sub-samples were taken at 1, 2, 4, 8, 12, 20, 24, 30 and 48 hours. This was for confirmation of leach kinetics seen in CN-23, and for the adsorption kinetic test.

Leach Kinetics

Gold leaching was very fast and began to plateau at approximately 12 hours. The leach kinetic constant, k_s, for the Gramalote Ridge sample was calculated at 0.325.

Carbon Adsorption Kinetics

The pulp from the bulk leach test was contacted with fresh, attritioned Calgon GRC-22 carbon in a batch test in a rolling bottle for 72 hours. Solution samples were removed for analysis at 1, 4, 6, 9, 24, 48 and 72 hours to establish the adsorption kinetic profile.

The kinetic (k) and equilibrium (K) adsorption constants generated from the sample tested were 0.023 h^{-1} and 10,388 g/t, respectively. The kinetic constant k is a first order rate constant that describes the rate of film diffusion of gold cyanide onto carbon in the initial stages of the adsorption process. The equilibrium constant K represents the predicted equilibrium loading of gold cyanide on the carbon (in g/t) from a solution containing 1 mg/L Au. The product of the two constants (kK) provides a useful guide as to whether or not CIP/carbon in leach (CIL) will perform efficiently when processing the particular pulp. In this regard, experience has shown that when the kK value is <50 h^{-1}, the carbon adsorption process will be slow and inefficient, requiring many adsorption stages and high carbon inventories. When the kK value is >100 h^{-1}, however, good CIP/CIL performance can be expected. Against this background, it can be seen that the Gramalote Ridge master composite gold adsorption properties are excellent, with a value of 239 h^{-1}.

Carbon Adsorption Equilibrium Isotherms

To generate the equilibrium data, predetermined amounts of pulp from the bulk leach test were contacted with predetermined amounts of activated carbon to achieve target solution/carbon ratios (or targeted carbon loadings). The slurries were agitated, in rolling bottles with carbon, for 72 hrs to equilibrate the activated carbon with the gold in solution, after which time the carbon was recovered by screening and samples of the slurry were filtered to generate carbon, solution and washed residue samples for gold and silver analysis.

Results

The leach kinetic constant and the carbon adsorption constants were calculated to be:

- Leach kinetic constant, k_s: 0.325;
- Kinetic adsorption constant, k, h^{-1}: 0.023;

- Equilibrium adsorption constant, K, g/t: 10,388;
- Product of equilibrium and kinetic constants, kK, h^{-1}: 239.

Overall, the product of the two constants is 239 h^{-1}, indicating excellent gold adsorption properties. Based on this constant and the very low organic carbon assays for the master and variability composites, CIP was selected for the proposed process plant. The SGS Lakefield modelling study indicated excellent CIP performance with low soluble gold losses.

13.3.11 Oxygen Uptake Testwork

An oxygen uptake test was performed using 200g of pilot plant flotation concentrate. The test was performed in a 1 L reaction kettle equipped with an overhead mixer, for a duration of 36 hours. The kettle was covered, and a mixture of air and oxygen was added to the to the pulp to achieve a dissolved oxygen concentration of between 20–25 mg/L. Lime was added throughout the test to achieve a pH of 10.5 to 11.0.

The dissolved oxygen content of the slurry was measured for a total time of 15 minutes, at 1-minute intervals. During these readings the oxygen was turned off. Dissolved oxygen readings were taken at 0, 1, 2, 3, 4, 5, and 6 hours during the pre-aeration period. Cyanide was then added to represent the leach conditions, with a concentration maintained at 0.5 g/L, and readings were continued at 1, 2, 3, 4, 5, 6, 7, 9, 24 and 30 hours.

The oxygen uptake rate was much higher during the pre-aeration period and dropped significantly after cyanide was added. This replicates what has been noted in plant operation at several sites, where the oxygen demand drops significantly in the presence of cyanide. Any future Gramalote plant should incorporate external oxygen contactors with cyanide dosed into the pump suction to take advantage of this effect.

13.3.12 Cyanide Destruction Testwork

After contact with carbon, the bulk leach slurry sample from test CN-26 (produced from the metallurgical master composite) was used for cyanide destruction testwork. The objective of this testwork was to yield a treated pulp containing <10 mg/L residual weakly acid dissociable cyanide (CN_{WAD}) using the SO_2/air destruction process, as well as to optimise reagent addition rates.

The main parameters adjusted in the testwork were the stoichiometric addition of sodium metabisulphite (SO_2 source) and the addition of copper sulphate. The tests were completed in batch and continuous mode. The sample responded well to cyanide destruction treatment using the SO_2/air process. All of the tests were conducted at 35°C, 35% solids (w/w), pH 8.5, and a retention time of approximately 195 minutes. pH regulation was carried out using lime slurry.

Upon completion of the cyanide destruction program, the following operating conditions achieved a discharge CN_{WAD} concentration of <10 mg/L:

- 35% solids (w/w);
- Addition ratio of 4.3 g of SO_2 per gram of CN_{WAD};
- pH of 8.5, using lime as required (~5 kg/t);
- 195 minutes residence time.

Copper sulphate was not required to achieve the target discharge concentration. However, it is recommended that a copper sulphate addition system be included in the process plant design for start-up and upset conditions.

The cyanide destruction testwork was directed by a metallurgical consultant and the following design conditions were recommended for the Gramalote cyanide destruction circuit:

- 35% solids (w/w);
- Addition ratio of 4.5 g of SO_2 per gram of CN_{WAD};
- Addition of 10 mg/L of Cu^{2+};
- pH of 8.5, using lime as required (~5 kg/t);
- Air addition of 3,500 scfm;
- 240 minutes residence time.

This testwork indicates the SO_2/air process can be successfully employed to treat the CIP tailings stream to reduce CN_{WAD} to the desired concentration.

13.3.13 Rheology Testwork

Rheology testwork was conducted on the following samples:

- Flotation tailings from the pilot plant;
- Cyanide destruction feed (at a pH of 11);
- Cyanide destruction discharge (at a pH of 8.5).

All samples were tested at the as received slurry density from the previous tests of 35% solids (w/w).

Viscosity Test Results

The cyanide destruction feed sample was submitted for a viscosity study that included measurements at a constant shear rate of 5 s^{-1} for 120 s at four different slurry densities. All tests were conducted with a pulp pH of 11 and at 37°C. A suitable starting point for the cyanide destruction circuit testwork was a slurry density of approximately 35% solids (w/w).

Rheology Test Results

The rheology test results were conducted using concentric cylinder rotation viscometry using a spindle and cup configuration.

The critical solids density for the flotation tailings sample was 71.5%. The critical solids density for the cyanide destruction sample feed and discharge samples was 57.5% and 59.5% respectively. The generally accepted limit for leaching and CIP is a yield stress of 10 Pa; therefore, approximately 48% solids (w/w) is the maximum value that could be used in leaching and CIP. The design leach density of 35% solids (w/w) is well below that value and therefore slurry rheology will not adversely impact leaching or carbon adsorption kinetics.

13.3.14 Thickener and Flocculation Testwork

Three samples from the pilot plant were provided to FLSmidth for thickener testwork; a flotation concentrate sample, a reground flotation concentrate (pre-leach thickener) sample, and a flotation tailings sample. Testwork comprised flocculant screening, static and dynamic settling tests and thickened mud rheology evaluation.

The thickening tests showed:

- An anionic polyacrylamide flocculant with a high molecular weight and medium charge density produced the best overflow clarity and settling velocity when compared to other flocculants. MF 1011 was used for the testing campaign, but any flocculant meeting the criteria above would likely be acceptable;
- Recommended flocculant dosages are approximately 20–25 g/t dry solids for the tailings and 50–60 g/t of dry solids for both concentrate samples;
- Flux testing showed that the optimum feedwell densities for flocculation were between 13–15% solids (w/w) for the flotation tailings, between 11–13% solids (w/w) for the flotation concentrate, and between 6–10% solids (w/w) for the reground flotation concentrate;
- The desired thickener underflow densities of 55–60% solids (w/w) for flotation tailings, 50% solids (w/w) for flotation concentrate and 45% solids (w/w) for reground flotation concentrate were readily achieved. It is possible to target higher densities if required, as underflow solids concentrations of 72%, 64% and 55% (w/w) solids respectively were achieved in the dynamic testing.

13.3.15 Regrind Testwork

A sample of flotation concentrate from bulk gravity and flotation test (F-7) was supplied to Metso Outotec for Jar Mill testing. The as-received material was blended and split using a Jones Riffle splitter to obtain a representative sample of the test feed, for determination of particle size distribution and bulk density.

A 20.3 cm by 25.4 cm Jar Mill rotating at 71.3 rpm (76% of theoretical critical speed) and charged with 15.9 kg of 19 mm diameter steel balls was used for testing. The required quantity of water was added to achieve the desired slurry density of 60% solids (w/w) inside the Jar Mill. Multiple subsamples were split to perform a series of Jar Mill runs with varying grinding retention times. The Jar Mill was loaded with grinding media, solids, and water in layers to generate a mixed grinding environment at the beginning of each run. The Jar Mill product was blended and split using a Jones Riffle and a subsample was obtained for particle size distribution measurement.

The specific energy of 24.19 kWh/t reported included a 10% safety factor. The conversion to Vertimill specific energy used an efficiency factor of 0.66 based on Metso Outotec experience and resulted in a value of 15.75 kWh/t.

13.3.16 Materials Handling Testwork

Materials handling testwork was conducted in 2011 by Jenike and Johanson on two samples labelled Early Mine Life Composite and Remaining Mine Life Composite Gramalote Silica. No additional testwork was performed in this area after 2011.

The test program consisted of the following tests:

- Compressibility, for determining the bulk density versus consolidating pressure relationship;
- Flow function, for calculating the critical outlet dimensions;
- Wall friction, for determining wall friction angles and mass-flow hopper angles;
- Bench scale angle of repose and drawdown, for determining angles of repose and drawdown.

General test conditions were as follows:

- Ambient laboratory conditions, ~22°C and 50% relative humidity;
- Size fraction tested, -6 mesh (3.35 mm);
- Moisture content, as received (0.37%, 0.32%);
- Time at rest, 0 and 24 hours.

The as-received moisture content is considered a very low point at which to conduct testwork and probably not representative of the potential Gramalote operation given the likely annual rainfall.

At the as-received moisture content, the expected bulk density range for the Early Mine Life Composite sample is between 1,455–1,557 kg/m^3.

At the as-received moisture content, the expected bulk density range for the Remaining Mine Life Composite Gramalote Silica sample is between 1,493–1,572 kg/m^3.

The test results for both samples show low cohesive strength and that no minimum outlet size is required to prevent cohesive arching. Therefore, given the high abrasive index of the ore, a rock box design is recommended for improved wear performance compared with the mass flow design.

The material is easy flowing with a slight tendency to rathole.

At the as-received moisture content, the angle of repose is expected to range from 31 to 37° (from horizontal) and the drawdown angle is expected to range from 54 to 59° (from horizontal).

The material tested behaves very well and is similar to other sulphide/fresh ores. A rock box design should be adopted to avoid excessive wear.

13.3.17 Site Water Analysis

Analysis of a site water sample, taken from La Palestina stream within the TMF footprint area, was conducted by SGS Lakefield. The sample was taken after rainfall, and therefore the turbidity would be expected to be higher than usual. The water is non-scaling based on pH, alkalinity and hardness, and would likely require little water treatment apart from filtration for general use in the processing plant. Access to this water source is likely to be stopped following the start of tailings deposition in the TMF.

An alternate source of fresh water is likely to be from the Guacas Creek. Four water samples were collected from the Guacas Creek in 2016 and analysed by MCS Consulting and Environmental Monitoring.

The water is non-scaling based on pH, alkalinity and hardness, and would likely require little water treatment apart from filtration for general use in the processing plant. As the variability of water quality throughout the year is not understood, anti-scalant reagent solution will be added at very low dosing rates to the water streams used throughout the processing plant. The plant design will include a water treatment plant with reverse osmosis technology as the elution circuit performs better with higher quality water.

Potable water will be produced to drinking water standards at the accommodation camp. This water will be further chlorinated and treated with UV light before being reticulated to buildings within the processing plant area.

13.3.18 High Pressure Grinding Roll Testwork

A series of HPGR tests were conducted by Metso Outotec in June 2022 on a 3.0 t drill core sample. The HPGR unit used for the testing was a new HRC 800e machine which is a full-scale version of the commercial unit with the exception of a reduced tire width.

The open circuit HPGR test program was completed to establish the effect of varying specific force on specific throughput, operating gap, power draw and particle breakage along with varying feed moisture and roll speed. Upon completion of the HPGR testing, BWi tests were conducted on the feed and all product samples.

Results indicated:

- The HPGR test results match overall expectations, including the specific energy increases with increasing force, while moisture and speed have little effect;
- The BWi values for a closing screen size of 300 µm was 16.1 kWh/t for the HPGR feed and ranged from 15.1–16.1 kWh/t for the products of the HPGR tests.

13.3.19 Comminution Circuit Selection

The Gramalote Ridge deposit is characterized as medium to hard competency and moderately soft to moderately hard in terms of BWi. It is considered moderately abrasive to abrasive.

The comminution circuit design is for a plant throughput of 6.0 Mtpa of sulphide ore with primary crusher product at a mill feed size of F_{80} ~150 mm to a final product grind size of P_{80} 250 µm. The design is based on the 85th percentile of the comminution testwork results. SGS Lakefield completed the comminution testwork, followed by modelling with process simulation software, JKSimMet.

Metcom Technologies conducted further comminution modelling using power-based mill sizing methodologies, including Bond, Barratt, Morrell and MacPherson. The average total of these four different modelling methodologies are in agreement with the current comminution modelling and mill size recommendations.

13.3.20 Gravity Circuit Modelling

A sample of the metallurgical master composite was tested for gravity recoverable gold content by SGS Lakefield. In 2012, two samples labelled GER and GIR were also tested for GRG content.

The size of the gravity recoverable gold can be expressed using the AMIRA scale. As a rule, the coarser the gravity recoverable gold, the higher the amenability to gravity recovery. The finer the gravity recoverable gold, the more amenable to recovery using technologies such as flash flotation. The metallurgical master composite is considered coarse on the AMIRA scale while the 2012 samples are considered moderately coarse. As such, all three are amenable to gravity gold recovery.

13.4 Metallurgical Variability

Samples selected for metallurgical testing were representative of the various types and styles of mineralization within the Gramalote Ridge deposit. Samples were selected from a range of locations within the deposit zones. Enough samples were taken so that tests were performed on sufficient sample mass.

13.5 Recovery Estimates

Test results indicate that overall gold recoveries in the range 92–98% and overall silver recoveries in the range 27–60%, could be expected from Gramalote Ridge ore when using the selected flowsheet. The average gold and silver recoveries for all the variability samples were 95.5% and 46.3%, respectively, while for the master composite recoveries were 95.7% and 54.1%, respectively.

The logarithmic relationship between overall gold recovery and the calculated gold head grade is shown by the following relationship:

- Overall Gold Recovery (%) = 1.7873 * ln (Gold Head Grade, g/t) + 95.526.

At a gold head grade of 2.5 g/t Au, the overall gold recovery is forecast to average 97.2%.

Typically, a plant recovery discount of up to 1.0% is applied to account for soluble gold losses, fine carbon losses and plant problems which may impact on the overall recovery. However, the metallurgical testwork has not considered the additional gold that will be recovered by further leaching of the intensive cyanidation residue in the concentrate leach circuit. It is planned to introduce this residue to the regrind circuit ahead of concentrate leach. As such, no recovery discount has been applied, as it is considered that the laboratory results are suitably conservative and further optimisation may be possible.

There is a similar relationship for silver, and shown by the following relationship:

- Overall Silver Recovery (%) = 19.951 * ln (Silver Head Grade, g/t) + 37.707.

At a silver head grade of 1.9 g/t Ag, the overall silver recovery is forecast to average 50.1%.

No discount will be applied for silver.

13.6 Deleterious Elements

No deleterious elements are known from the processing perspective.

13.7 Comments on Section 13

The QP notes:

- The Gramalote Ridge deposit is characterized as medium to hard competency and moderately soft to moderately hard in terms of BWi. The ore is suitable for a SABC circuit, which includes a pebble crusher. It is considered moderately abrasive to abrasive;
- A gravity separation circuit is warranted based on testwork conducted by SGS Lakefield and the modelling by FLSmidth;
- Gramalote Ridge ore is amenable to gold recovery via froth flotation at a coarse grind size of P_{80} 250 µm. A flotation time (laboratory) of 20 min and a simple PAX and frother reagent scheme will recover approximately 3 to 4% of the mass and 93 to 98% of the gold in the flotation feed. A rougher/scavenger only flowsheet is optimum, and no cleaning is required;
- A regrind size of approximately P_{80} 20 µm is required to adequately liberate the gold prior to concentrate leach;
- The concentrate leach conditions identified as achieving high gold extraction are: 6 h of pre-aeration followed by 30 h of cyanide leach at 35% solids (w/w), pH 10.5 to 11 maintained with lime, 1 kg/t lead nitrate, 2 g/L NaCN and dissolved oxygen levels of between 20 to 25 g/L;
- There is a logarithmic relationship between gold head grade and overall gold recovery (including gravity, gravity concentrate leach, flotation and flotation concentrate leach). There is also a logarithmic relationship between silver head grade and overall silver recovery;
- Overall gold and silver recoveries for the Gramalote Ridge ore are expected to range from 92–98% for gold and 27–60% for silver. The average overall gold and silver recoveries were 95.7% and 45% respectively for the variability samples;
- Based on the very low levels of preg-robbing elements and very good adsorption properties, a CIP circuit was selected for the proposed Gramalote process plant;
- The concentrate leach tailings responded well to cyanide destruction using the SO_2/air method. The required cyanide destruction residence time is 240 min;
- The flotation tailings have a specific thickener throughput rate of 0.035 m^2/t/day. The flotation tailings thickener will require 20 g/t of flocculant to achieve an underflow density of 55–60% (w/w).

14.0 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The Mineral Resource model for Gramalote Ridge was completed on May 11, 2022 using drilling to March 9, 2022. Nine drill holes (1,795 m) post-date the resource estimate database cut-off. These holes were drilled for condemnation/infrastructure purposes and are generally outside of the Mineral Resource areas.

The Mineral Resource model for Trinidad was completed on August 6, 2025 using drilling up to July 14, 2025. As of the Report Effective Date, there has been no additional drilling at Trinidad.

The Mineral Resource model for Monjas West was completed on February 17, 2023 using drilling up to January 4, 2023. As of the Report Effective Date, there has been no additional drilling at Monjas West.

The models were constrained using pit shell assumptions provided on August 6, 2025, and the Mineral Resource estimates have an effective date of August 6, 2025.

Drill hole logging and assays were used as a basis for 3D models of lithology, weathering, mineralization zones, and gold grade estimates.

The software used for the estimates included Leapfrog Geo (modelling), Supervisor (geostatistics and variography), Datamine Studio RM (estimation) and Whittle (pit optimization).

Block modeling used parent block size of 15 x 5 x 10 m at Gramalote Ridge with sub-blocks down to 5 x 0.2 x 0.5 m at domain boundaries. At Trinidad, 15 x 5 x 10 m parent blocks were used with sub-blocks down to 3 x 1 x 1 m. The parent block size at Monjas West was 10 x 10 x 10 m with sub-blocks down to 2 x 2 x 1 m.

For open pit mine planning and conversion to reserves, a fully diluted whole block model was created at Gramalote Ridge using a block size of 15 x 5 x 10 m.

Artisanal small miners (ASMs) have focused on material in weathered rock, but recently also mining narrow veins in fresh rock. The high-resolution LiDAR survey conducted in 2020 accounts for historic surface hydraulic mining; however, no allowance was made in the models to account for ASM tunnels due to the small amount of material involved along with inadequate tunnel surveys.

14.2 Exploratory Data Analysis

A review of drill core and a statistical analysis of the gold grades (relative to logged features) supports that the main mineralization controls at Gramalote Ridge are:

- Quartz–sericite, potassic (K-feldspar) and sericite alteration (of moderate intensity or greater);
- Veins designated as Type 1, Type 2, and Type 5 (refer to descriptions in Table 7-2);
- Higher grades at Gramalote Ridge are associated with elevated pyrite and chalcopyrite abundance.

At Trinidad and Monjas West, mineralization is associated with both stockwork veinlets and alteration along the veinlet margins. Alteration is predominantly quartz–sericite, although some potassic alteration has been observed. Veinlets are associated with the same types as observed at Gramalote Ridge (Type 1, Type 2 and Type 5).

Silver generally has the same associations as gold. The silver gold ratio varies between 1.1 and 1.6 at different deposits and grade ranges with an average of Au:Ag of 1.4.

14.3 Geological Models

Lithology, structural, weathering and mineralization domains were modeled.

14.3.1 Lithology Model

Gramalote Ridge, Trinidad and Monjas West are hosted within tonalitic/granodioritic rocks. There is a gradational northeast trending contact between tonalite that hosts the deposits and diorites to the southeast of Gramalote Ridge. Regional lithological contacts were modeled in 3D and applied to the models where applicable.

14.3.2 Structures and Veining

Models of brittle structure and structural corridors were created based on logged drillhole data and geophysical/topographic lineaments.

Models of larger veins were created based on logged drillhole data and mapping from artisanal miner tunnels.

The structure and veining models were used to guide trends in the mineralization zone interpretations and in grade estimation.

14.3.3 Mineralization Zones

Gramalote Ridge

LG and HG mineralization domains were interpreted for Gramalote Ridge from RC and core drill holes.

In the LG domain, drill hole intervals were tagged as low-grade if they met a nominal cut-off grade of 0.1 g/t Au. Samples at the threshold were omitted or included based on Type 1 + Type 2 + Type 5 vein content and surrounding drill holes. A general 6 m minimum thickness was used, but allowance was made for continuity with adjacent holes. LG wireframes were created by contouring (isoshell) a high-resolution indicator block model of the manual tags. Blocks with indicator values ≥0.5 were considered inside the domain. Local orientations were adjusted to mimic the structural and veining interpretations, and some manual points were added to control the model trends.

In the HG domain, intervals were tagged as HG if they met a nominal 0.5 g/t Au cut-off. As with the LG domain, samples at the threshold were omitted or included based on mineralizing vein abundance, pyrite and chalcopyrite content, and adjacent drill holes. A general minimum thickness of 4 m was used, sometimes less for the sake of continuity with adjacent holes. HG wireframes were created by contouring (isoshell) a high-resolution indicator block model of the manual tags. Blocks with indicator values ≥0.5 were considered inside the domain. Local orientations were adjusted to mimic the structural and veining interpretations, and some manual points were added to control the model.

In the very low-grade domain (halo), drill hole intervals were tagged if they were logged as being even weakly altered and contained >0.5% veining. A general 6 m minimum thickness was used. Halo material is generally sub-economic. Halo wireframes were created using the same indicator methodology as the HG and LG domains.

A 3D perspective view (Figure 14-1) shows the overall orientation and widths of the mineralized zones at Gramalote Ridge.

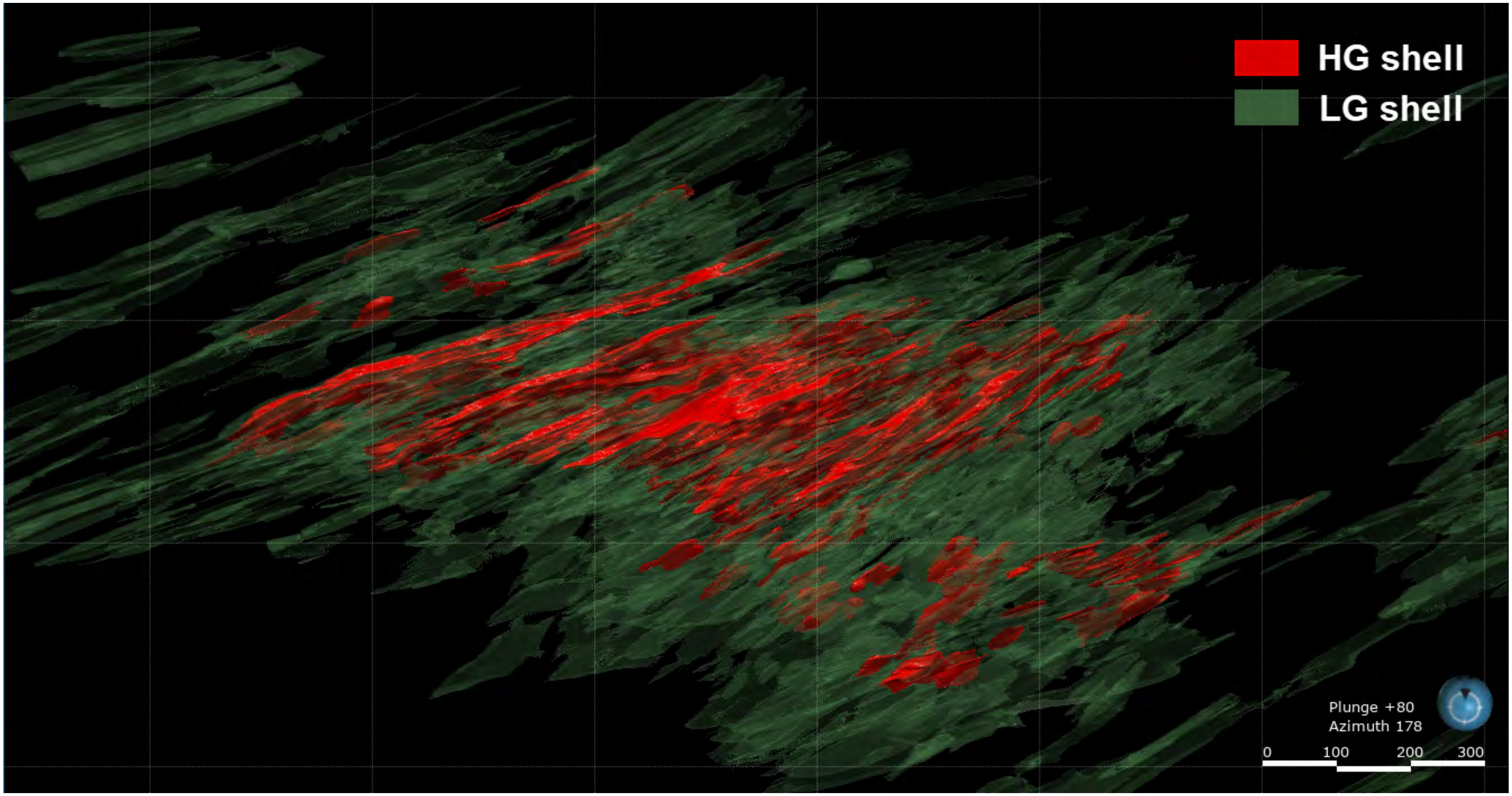
Trinidad

Low-grade mineralized domains at Trinidad were created using the same methodology as Gramalote Ridge. Drill hole intervals were tagged as low-grade if they met a nominal cut-off grade of 0.1 g/t Au. Samples at the threshold were omitted or included based on Type 1 + Type 2 + Type 5 vein content, and surrounding drill holes. A general 6 m minimum thickness was used, but allowance was made for continuity with adjacent holes.

LG wireframes were created by contouring (isoshell) a high-resolution indicator block model of the manual tags. Blocks with indicator values ≥0.5 were considered inside the domain. Local orientations were adjusted to mimic the structural and veining interpretations, and some manual points were added to control the model.

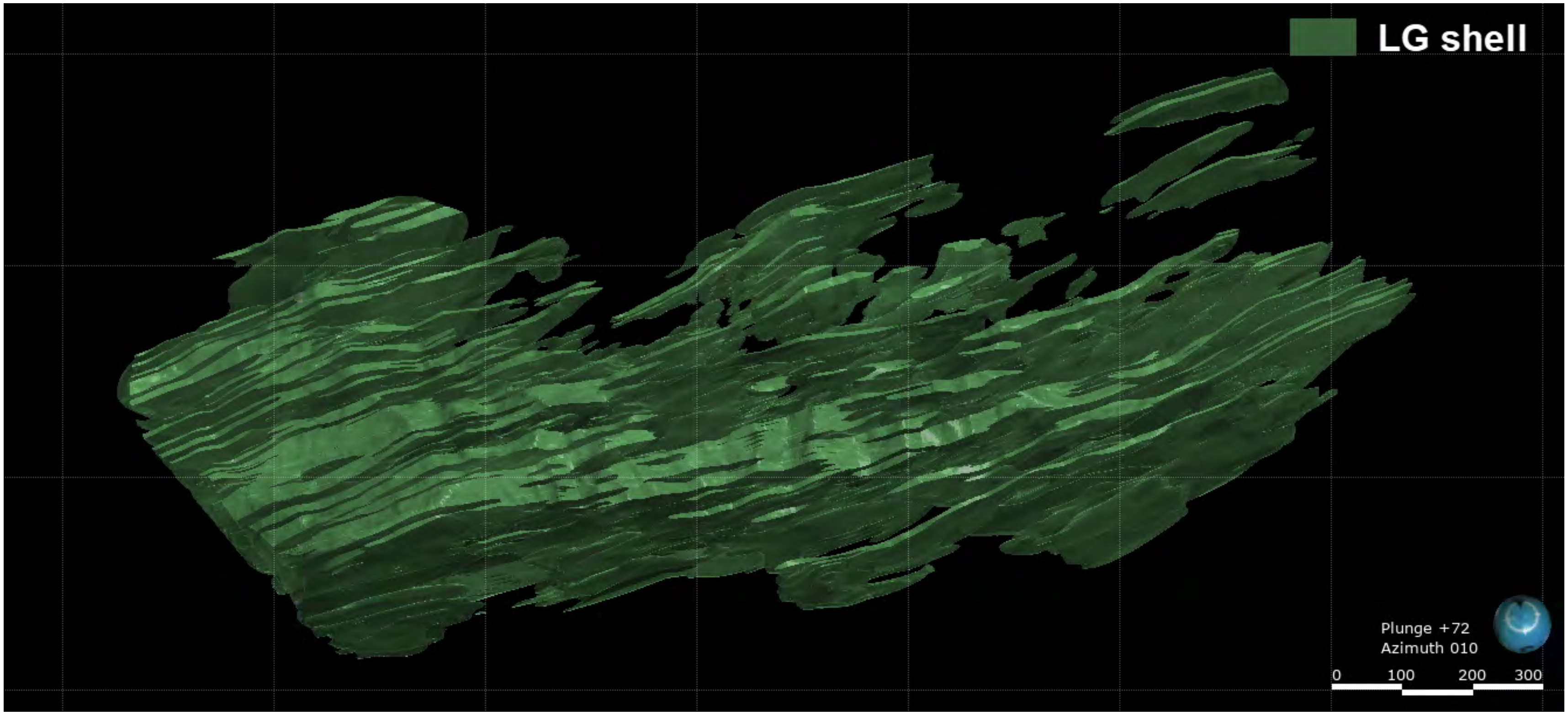
A 3D perspective view (Figure 14-2) shows the overall orientation and widths of the mineralized zones at Trinidad.

Figure 14-1: 3D Perspective View of Gramalote Ridge Mineralized Zones



Note: Figure prepared by B2Gold, 2025.

Figure 14-2: 3D Perspective View of Mineralized Zones at Trinidad



Note: Figure prepared by B2Gold, 2025.

Monjas West

At Monjas West low-grade mineralization domains were modeled based on a nominal cut-off of 0.1 g/t Au with consideration given to Type 1, Type 2 and Type 5 vein percentage and sulphide abundance.

3D solids of the low-grade mineralization domain were created in Leapfrog Geo, and are shown on Figure 14-3.

14.3.4 Weathering Model

Six weathering and overburden surfaces were created:

- Base of alluvial sediments;
- Base of soil/colluvium surface;
- Solid of backfill material (related to test RC areas);
- Solid of small miner discard piles;
- Base of laterite (Trinidad only);
- Base of saprolite;
- Base of saprock.

Soil/colluvium, alluvium, backfill and discard pile domains were used for density assignment; grades were not estimated for these material types. Saprolite and saprock were used to assign density and pit slope/work-index domains but not as boundaries to the grade estimates.

Mineralization within saprolite and saprock is weathered insitu and does not show significant displacement relative to underlying mineralization in unweathered rock.

Gramalote Ridge

The base of soil/colluvium, alluvium, backfill, discard, saprolite and saprock surfaces were based on logged weathering codes from drill holes as offsets of the topographic surface. Detailed surface mapping was also used to control the various surfaces.

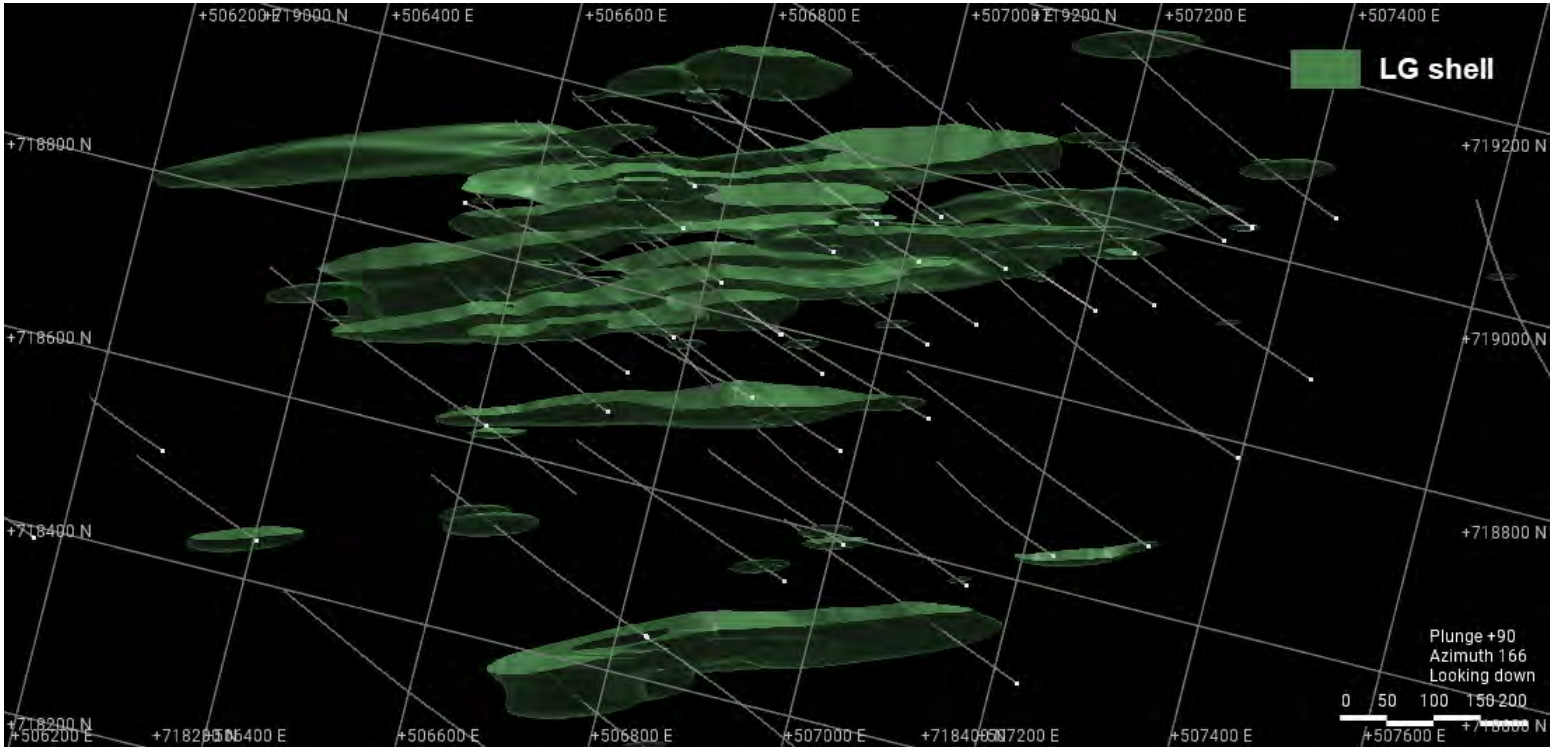
Trinidad

The base of soil/colluvium, laterite, saprolite and saprock surfaces were based on logged weathering codes from drill holes as offsets of the topographic surface.

Monjas West

The base of soil/colluvium, saprolite and saprock surfaces were based on logged weathering codes from drill holes as offsets of the topographic surface.

Figure 14-3: 3D Perspective View of Mineralized Zones at Monjas West



Note: Figure prepared by B2Gold, 2025.

14.4 Density Assignment

14.4.1 Gramalote Ridge

At Gramalote Ridge, bulk densities were estimated to the block model using ID2 interpolation. The mineralized zones and weathering domains were used as boundaries to this estimate. Obvious outliers were excluded from the dataset prior to estimation. Table 14-1 shows the mean density values of samples and the block model estimate.

14.4.2 Trinidad

At Trinidad, bulk densities were applied to the block model based on the average of SG samples by mineralized zone and weathering domains using the values in Table 14-2.

14.4.3 Monjas West

At Monjas West, bulk densities were applied to the block model based on the average of SG samples by mineralized zone and weathering domains using the values in Table 14-3.

14.5 Grade Capping/Outlier Restrictions

Capping levels were identified in each domain/sub-domain using distribution (probability) plots, deciles, and spatial observation of high-grade assays.

14.5.1 Gramalote Ridge

Assays within each mineralization domain (HG, LG, waste) were capped separately. In addition, assays were separated by structural sectors to represent different areas of the deposit. Capping levels are shown on Table 14-4. Overall, capping removes approximately 4% of the gold metal content.

14.5.2 Trinidad

Assays for each mineralization domain were capped separately. Additionally, assays were separated into structural sectors for the purposes of capping. Capping levels for Trinidad are shown in Table 14-5. Overall metal reduction is 5%. In the key domains, LG1 and LG5, capping removes 3% and 5% of the gold metal content respectively.

14.5.3 Monjas West

Assays for each mineralization domain were capped separately. Domains were further separated into capping sectors. Gold and silver capping levels are shown in Table 14-6. Capping removes 9% of gold metal content at Monjas West.

Table 14-1: Sample and Block Model Densities, Gramalote Ridge

Weathering	Mineralized					Waste				
	n	Mean SG	Std Dev	CV	Model SG	n	Mean SG	Std Dev	CV	Model SG
Discard (820)	n/a					0				1.60
Backfill (810)	n/a					0				1.60
Alluvium (710)	n/a					0				1.60
Soil/colluvium (700)	n/a					11	1.60	0.33	0.21	1.57
Saprolite (300)	162	1.66	0.31	0.19	1.66	115	1.64	0.28	0.17	1.65
Saprock (200)	63	2.23	0.38	0.17	2.15	59	2.18	0.40	0.18	2.19
Fresh (100)	5,477	2.65	0.05	0.02	2.64	1,305	2.65	0.05	0.02	2.65

Note: n = number, Std Dev = standard deviation, CV = co-efficient of variation, SG = specific gravity.

Table 14-2: Densities Assigned to the Block Model, Trinidad

Weathering	Mineralized				Waste			
	n	Mean SG	Std Dev	CV	n	Mean SG	Std Dev	CV
Soil/colluvium (700)	n/a				13	1.68	0.14	0.08
Laterite (400)	4	1.60	0.09	0.06	3	1.48	0.05	0.03
Saprolite (300)	53	1.59	0.14	0.09	118	1.61	0.25	0.16
Saprock (200)	14	2.29	0.31	0.14	28	2.30	0.31	0.13
Fresh (100)	350	2.65	0.04	0.02	699	2.63	0.06	0.02

Note: n = number, Std Dev = standard deviation, CV = co-efficient of variation, SG = specific gravity.

Table 14-3: Densities Assigned to the Block Model, Monjas West

Weathering	Mineralized				Waste			
	n	Mean SG	Std Dev	CV	n	Mean SG	Std Dev	CV
Soil/colluvium (700)	n/a				*	1.57	n/a	n/a
Saprolite (300)	*	1.65	n/a	n/a	*	1.65	n/a	n/a
Saprock (200)	*	2.40	n/a	n/a	4	2.47	0.11	0.04
Fresh (100)	96	2.63	0.07	0.03	646	2.62	0.11	0.04

Note: n = number, Std Dev = standard deviation, CV = co-efficient of variation, SG = specific gravity. *Values for domains with insufficient samples were extrapolated from other domains.

Table 14-4: Au and Ag Capping Levels, Gramalote Ridge

Structural Sector	Au Capping Levels			Ag Capping Levels		
	HG	LG	Waste	HG	LG	Waste
San Antonio	6.0	2.0	0.3	10.0	4.0	2.0
Felipe North	10.0	3.0	0.5	7.0	5.0	2.0
Felipe South	20.0	6.0	0.5	20.0	15.0	2.0
Gramalote East	5.0	3.0	1.0	2.0	5.0	0.5
Gramalote Main	45.0	6.0	1.0	50.0	40.0	1.0
Gramalote West	5.0	3.0	1.0	12.0	4.0	1.0
Retiro	20.0	5.0	1.0	60.0	20.0	3.0
Balsal	12.0	4.0	0.5	12.0	9.0	1.5
Reina	6.0	3.5	0.2	4.0	3.0	2.0
Limon	6.0	3.0	0.5	1.5	1.5	2.0

Table 14-5: Au and Ag Capping Levels, Trinidad

Structural Sector	Au Capping Levels	Ag Capping Levels
Waste	1	20
Alteration halo	2	8
LG-1	7	11
LG-2	4	4
LG-3	3.5	10
LG-5	6	35
LG-6	3	5

LG-7	2	2

Table 14-6: Au and Ag Capping Levels, Monjas West

Structural Sector	Au Capping Levels (g/t Au)	Ag Capping Levels (g/t Ag)
Waste	0.8	8
LG-1000	5	4
LG-2000	3.5	3

14.6 Composites

Down-hole composites of 4 m length were created for Gramalote Ridge, 2 m composites for Trinidad, and 4 m composites for Monjas West. The composite length was chosen as a balance between reducing the composite variance for variography and kriging while maintaining reasonable variance observed in the deposit.

Capping was done prior to compositing. New composites were started at the beginning of the mineralization zone boundaries and composite width was allowed to vary to avoid short composites at the end of intervals. Composite distribution statistics are presented in Table 14-7 for Gramalote Ridge, Table 14-8 for Trinidad and Table 14-9 for Monjas West.

14.7 Variography

14.7.1 Gramalote Ridge

Variograms were run on 4 m capped composites to evaluate spatial continuity and trends to gold mineralization and for use in gold grade estimation checks. For the ordinary kriged (OK) estimate, the nugget was adjusted down 40% and first ranges were increased by 50% so that it would better reconcile with change-of-support distributions. Sills for the other structures were increased proportionately. Table 14-10 summarizes the variogram models used at Gramalote Ridge.

14.7.2 Trinidad

Variograms were run on 2 m capped composites at Trinidad to evaluate spatial continuity and trends to gold mineralization and for use in gold grade estimation checks. For the ordinary kriged (OK) estimate, the nugget was adjusted down about 50% so that it would better reconcile with change-of-support distributions. Sills for the other structures were increased proportionately.

Table 14-11 summarizes the variogram models used at Trinidad.

Table 14-7: Gold Grade Statistics, 4m Composites, Gramalote Ridge

	HG domain				LG domain			
Statistic	**Uncapped Assays**	**Capped Assays**	**4 m Comps**	**Declustered 4 m Comps**	**Uncapped Assays**	**Capped Assays**	**4 m Comps**	**Declustered 4 m Comps**
N	11,840	11,840	4,937	4,937	29,623	29,623	13,207	13,207
Min	0.0025	0.0025	0.0025	0.0025	0.0025	0.0025	0.0025	0.0025
Mean	1.89	1.81	1.69	1.72	0.43	0.37	0.36	0.36
Median	0.87	0.87	1.08	1.08	0.17	0.17	0.22	1.08
Max	134.50	45.00	26.20	26.20	142.40	6.00	5.43	26.2
CV	2.21	1.85	1.18	1.20	3.78	1.89	1.29	1.26
Var	17.39	11.30	3.94	4.22	2.62	0.42	0.21	0.21

Note: LG domain is inclusive of the HG domain. N = number, min = minimum, max = maximum, CV = co-efficient of variation, Var = variance, Comps = composites.

Table 14-8: Gold Grade Statistics, 2m Composites Low-Grade Domain, Trinidad

Statistic	Uncapped Assays	Capped Assays	4 m Comps	Declustered 4 m Comps
N	6,249	6,249	4,716	4,716
Min	0.003	0.003	0.003	0.003
Mean	0.564	0.534	0.483	0.471
Median	0.267	0.267	0.289	0.286
Max	21.4	7	6.15	6.15
CV	1.89	1.47	1.232	1.223
Var	1.13	0.614	0.355	0.331

Note: N = number, min = minimum, max = maximum, CV = co-efficient of variation, Var = variance, Comps = composites.

Table 14-9: Gold Grade Statistics, 4m composites, Low-Grade Domain, Monjas West

Statistic	Uncapped Assays	Capped Assays	4 m Comps	Declustered 4 m Comps
N	1271	1271	676	676
Min	0.003	0.003	0.003	0.003
Mean	0.565	0.528	0.521	0.535
Median	0.193	0.195	0.297	0.312
Max	20.7	5	3.315	3.315
CV	2.053	1.561	1.099	1.085
Var	1.344	0.680	0.328	0.337

Note: N = number, min = minimum, max = maximum, CV = co-efficient of variation, Var = variance, Comps = composites.

Table 14-10: Gold Variogram Models, Gramalote Ridge

Domain	Nugget	Structure 1-Spherical				Structure 2-Spherical			
		Sill	Range-Strike	Range-Across	Range-Dip	Sill	Range-Strike	Range-Cross	Range-Dip
LG	0.50	0.40	20	25	40	0.10	110	40	200
HG	0.60	0.35	20	20	20	0.05	120	60	150

Table 14-11: Gold Variogram Models, Trinidad

Domain	Nugget	Structure 1-Spherical				Structure 2-Spherical			
		Sill	Range-Strike	Range-Across	Range-Dip	Sill	Range-Strike	Range-Cross	Range-Dip
LG	0.37	0.38	75	25	60	0.26	150	40	110

14.7.3 Monjas West

Variograms were run on 4 m capped composites at Monjas West. to evaluate spatial continuity and trends to gold mineralization and for use in gold grade estimation checks. For the ordinary kriged (OK) estimate, the nugget was adjusted down about 40% so that it would better reconcile with change-of-support distributions. Sills for the other structures were increased proportionately.

Table 14-12 summarizes the variogram models used at Monjas West.

Table 14-12: Gold Variogram Models, Monjas West

Domain	Nugget	Structure 1-Spherical				Structure 2-Spherical			
		Sill	Range-Strike	Range-Across	Range-Dip	Sill	Range-Strike	Range-Cross	Range-Dip
LG	0.28	0.42	75	25	60	0.3	150	40	110

14.8 Estimation/Interpolation Methods

14.8.1 Gramalote Ridge

Mineralization domain wireframes were coded to parent blocks of 15 x 5 x 10 m at Gramalote Ridge with sub-blocks down to 5 x 0.2 x 0.5 m at domain boundaries. The LG and HG mineralization domains served as a hard boundary relative to the waste domain for grade estimation.

Due to the variable nature of grades at the HG/LG contact, 2 m of material uphole and downhole of a contact were shared with the adjacent domain.

Using 4 m capped composites, gold and silver grades were estimated into 15 x 5 x 10 m blocks. Search orientations were controlled by Datamine's dynamic anisotropic search. With this method, the search ellipse is re-oriented (within user-set limits) to the local orientations of the mineralized zones.

Composites were shared across the oxide (saprolite)/fresh boundary for estimation. Where saprolite is mineralized it has a similar gold grade tenor relative to fresh rock.

All drill holes were used to estimate saprolite and saprock blocks; however, the specific Hydracore drilling focused on saprolite was excluded for estimates of fresh rock.

Gold and silver grades were estimated using ID3, OK and NN algorithms into parent-sized blocks. Mineral Resources are reported from the OK estimate.

The estimation plan is summarized in Table 14-13.

14.8.2 Trinidad

Mineralization domain wireframes were coded to 15 x 5 x 10 m parent blocks with sub-blocks down to 3 x 1 x 1 m at domain boundaries. The LG mineralization domain served as a hard boundary relative to the waste domain for grade estimation.

Using 2 m capped composites, gold and silver grades were estimated into 15 x 5 x 10 m blocks. Search orientations were controlled by Datamine's dynamic anisotropic search. With this method, the search ellipse is re-oriented (within user-set limits) to the local orientations of the mineralized zones.

Table 14-13:Gramalote Ridge Grade Estimation Plan

Pass	Search (m)	Min Comps	Max Comps	Max per Drill Hole
1	80 x 16 x 80	4	12	3
2	120 x 24 x 120	5	12	3
3	200 x 40 x 200	1	12	3

Composites were shared across the oxide (saprolite)/fresh boundary for estimation. Where saprolite is mineralized it has a similar gold grade tenor relative to fresh rock.

All drill holes were used to estimate saprolite, saprock and fresh blocks.

Gold and silver grades were estimated using ID3, OK and NN interpolations into parent-sized blocks. Mineral Resources are reported from the ID2 estimate.

Waste block grades were also estimated for mine planning purposes.

The estimation plan is summarized in Table 14-14.

14.8.3 Monjas West

Mineralization domain wireframes were coded to 10 x 10 x 10 m parent blocks with sub-blocks down to 2 x 2 x 1 m. The LG mineralization domain served as a hard boundary relative to the waste domain for grade estimation.

All drill holes were used to estimate saprolite, saprock and fresh blocks.

Gold and silver grades were estimated using ID3, OK and NN interpolations into parent-sized blocks (10 x 10 x 10 m). Mineral Resources are reported from the ID3 estimate.

Waste block grades were also estimated for mine planning purposes.

The estimation plan is summarized in Table 14-15.

14.9 Block Model Validation

Block grade estimates classified as Indicated and Inferred were validated using the following methods:

- Comparison of different declustering (NN and cell declustering) methods;
- Visual comparison of block grades to composites on cross sections and levels;
- Comparison of global block statistics for different estimation techniques;
- Swath plots to review potential local biases in the estimates;
- Global change of support comparisons.

Table 14-14: Trinidad Grade Estimation Plan

Pass	Search (m)	Min Comps	Max Comps	Max per Drill Hole
1	90 x 25 x 90	5	22	4
2	135 x 37.5 x 135	5	22	4
3	225 x 62.5 x 225	1	22	4

Table 14-15: Monjas West Grade Estimation Plan

Pass	Search (m)	Min Comps	Max Comps	Max per Drill Hole
1	80 x 25 x 80	5	16	4
2	120 x 37.5 x 120	5	16	4
3	200 x 62.5 x 200	1	16	4

14.9.1 Gramalote Ridge

Composite Declustering

Two methods of composite declustering were run, including a tonnage-weighted NN method and cell declustering (cell size of 60 x 20 x 60 m with eight offsets). The two declustering methods produced similar results.

Visual Comparison

Mineralization domains, composite grades and block model grades were reviewed in detail on 25 m-spaced vertical cross sections and 25 m-spaced levels. Generally, the model represents the drill hole grades well and there are no obvious over-projections of high or low grades observed in the final models.

Global Model Statistics

The global means at 0 g/t Au cut-off for ID2, ID3, OK and NN estimates compare within acceptable levels for Indicated blocks (approximately 1% difference) and Indicated plus Inferred blocks (2–3% difference). These comparisons are well within industry standard acceptable levels.

Swath Plots

Swath plots show the ID3 and OK models track the NN model (representing declustered composites) quite well with no obvious areas of under or over projection of grades.

Global Change of Support

A comparison of modeled tonnes and grade for NN, ID3, OK and theoretical change-of-support distributions using the discrete-Gaussian method was completed for the Indicated and Inferred blocks. Change of support assumed a 15 x 5 x 10 m selective mining unit, took into account the information-effect, and had no constraint from pit outlines.

At gold cut-off grades ranging from 0 g/t to 1.0 g/t, the OK model is within 5% of the theoretical tonnes and grade predicted by the change-of-support method.

Conclusions

The OK model was chosen over other estimates for resource reporting since it compared well to the change-of-support results and showed better representation from the visual review on cross sections.

14.9.2 Trinidad

Visual Comparison

Mineralization domains, composite grades, and block model grades were reviewed in detail on 50 m-spaced vertical cross sections and 25 m-spaced levels. The model is considered to reasonably represent the drill hole grades.

Swath Plots

Swath plots show the ID2 and OK models track the NN model (representing declustered composites) quite well, with no obvious areas of under or over projection of grades.

Global Model Statistics

The global means at a 0 g/t Au cut-off grade for the ID2, OK and NN estimates compare within acceptable levels for Inferred blocks (approximately 3% difference). These comparisons are well within industry-acceptable levels of difference, particularly for an Inferred Mineral Resource.

Global Change of Support

A comparison of modeled tonnes and grade for NN, ID2, OK and theoretical change-of-support using the discrete-Gaussian method was completed for Inferred blocks. Of support assumed a 15 x 5 x 10 m selective mining unit, and had no constraint from pit outlines.

At gold cut-off grades ranging from 0–1.0 g/t Au, the ID2 model is within 5% of the theoretical tonnes and grade predicted by the change-of-support method.

Conclusions

The checks completed on the Trinidad model support that the model can be used for Inferred Mineral Resources.

14.9.3 Monjas West

Visual Comparison

Mineralization domains, composite grades, and block model grades were reviewed in detail on 50 m-spaced vertical cross sections and 25 m-spaced levels. The model reasonably represents the drill hole grades.

Swath Plots

Swath plots show the ID3 and OK models track the NN model (representing declustered composites) quite well with no obvious areas of under or over projection of grades.

Global Model Statistics

The global means at a 0 g/t Au cut-off grade for the ID3, OK and NN estimates compare within acceptable levels for Inferred blocks (approximate 3% difference). These comparisons are well within industry-acceptable levels of difference, particularly for an Inferred Mineral Resource.

Conclusions

The checks completed on the Monjas West model support that the model can be used for Inferred Mineral Resources.

14.10 Classification of Mineral Resources

Drill hole spacing for resource classification at Gramalote Ridge is as follows:

- No Measured classified;
- Indicated: 50–60 m drill hole spacing (includes three 12.5 m-spaced RC test areas and two 25 m-spaced core hole test areas);
- Inferred: 100–120 m drill hole spacing.

Drill hole spacing for resource classification at Trinidad is as follows:

- No Measured or Indicated classified;
- Inferred: 100–120 m drill hole spacing or any block within 30-40 m of a drill hole.

Drill hole spacing for resource classification at Monjas West is as follows:

- No Measured or Indicated classified;
- Inferred: 100–120 m drill hole spacing or any block within 30 m of a drill hole.

14.11 Reasonable Prospects of Eventual Economic Extraction

Mineral Resources for Gramalote Ridge, Trinidad, and Monjas West considered potentially amenable to open pit mining methods were constrained within conceptual Lerchs–Grossmann optimized pit shells using the parameters in Table 14-16.

Based on these costs and assumptions, the calculated break-even cut-off grades are 0.13 g/t Au for oxide and range between 0.16–0.18 g/t Au for sulphide.

Mineral Resources potentially amenable to open pit mining from all deposits are stated above a cut-off of 0.13 g/t Au for oxide and >0.16 g/t Au for sulphide.

Constraints to the Gramalote Ridge pit optimization were applied around select planned infrastructure at Gramalote Ridge; specifically the plant, main haul road, and the Guacas River diversion. Figure 14-4 shows the location of constraining infrastructure.

14.12 Mineral Resource Statement

Indicated Mineral Resources are reported in Table 14-17, inclusive of those Indicated Mineral Resources converted to Probable Mineral Reserves. Mineral Resources that have not been converted to Mineral Reserves do not have demonstrated economic viability. Inferred Mineral Resources are provided in Table 14-18.

The QP for the estimate is Mr. Stephen Jensen, P.Geo., B2Gold's Exploration Manager for the Americas.

14.13 Factors That May Affect the Mineral Resource Estimate

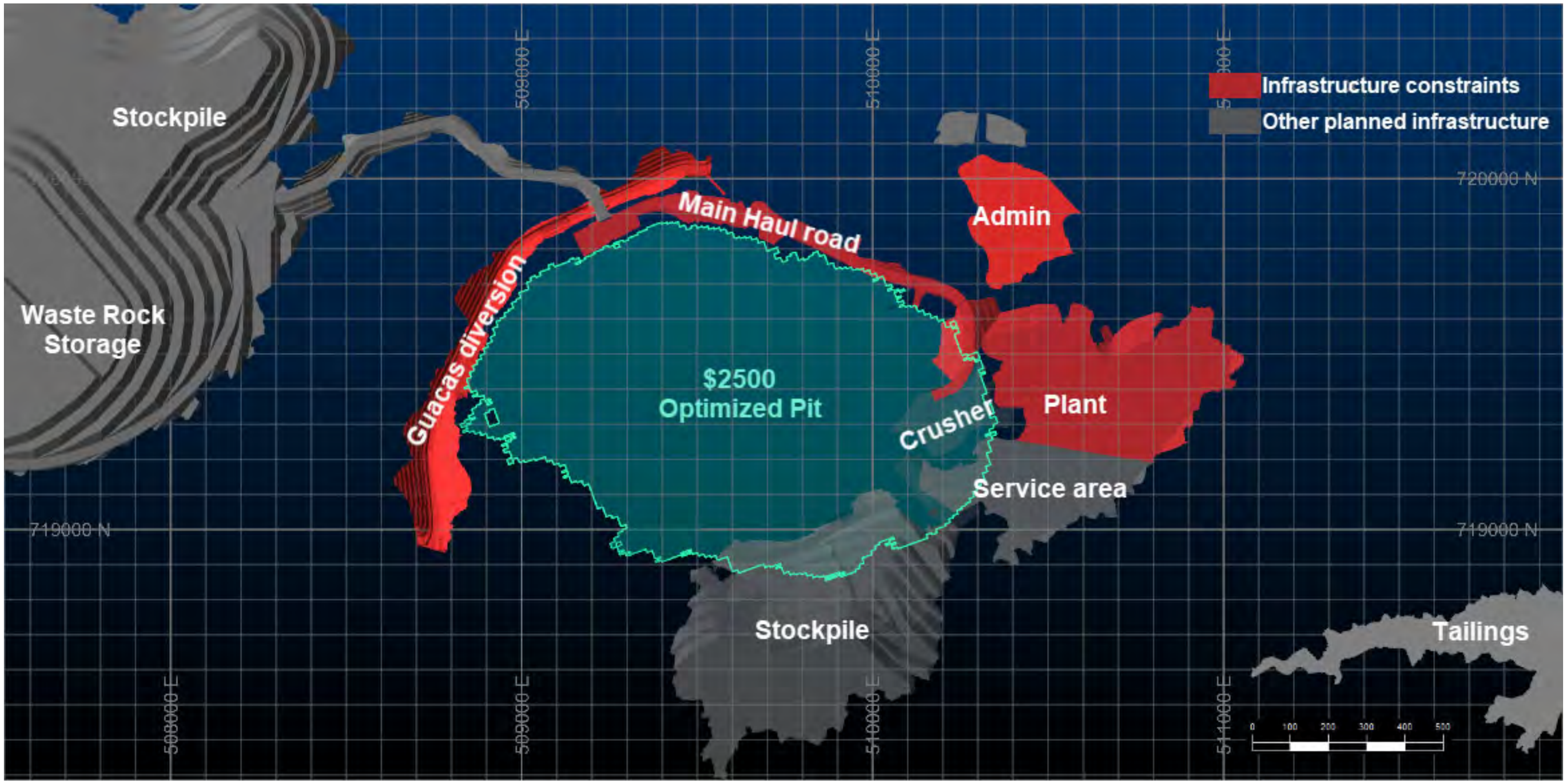
Factors that may affect the Mineral Resource estimates include:

- Metal price and exchange rate assumptions;
- Changes to the assumptions used to generate the gold grade cut-off grade;
- Changes in local interpretations of mineralization geometry and continuity of mineralized zones;
- Changes to geological and mineralization shape and geological and grade continuity assumptions;
- Density and domain assignments;
- Changes to geotechnical, mining and metallurgical recovery assumptions;

Table 14-16: Conceptual Pit Shell Parameters

Parameter	Unit	Gramalote Ridge	Trinidad	Monjas
Gold price	US$/oz	2,500	2,500	2,500
Silver price	US$/oz	28	28	28
Gold recovery (process) sulphide	%	92.7–97.6	90.9	87.6
Gold recovery (process) oxide	%	84	81.7	81.7
Silver recovery (process) sulphide	%	39.4	52.7	60.9
Silver recovery (process) Oxide	%	22	46.8	46.8
Average mining cost	US$/t mined	2.59	2.41	2.58
Process cost, sulphide	US$/t processed	9.74	9.74	9.89
Process cost, oxide	US$/t processed	6.13	6.13	6.28
Site general	US$/t processed	2.10	2.10	2.10
Selling cost	$/oz produced	82.84	82.84	82.84
Pit slopes	Degrees	31.6º in weathered 44.5–50.3º in fresh	28º in weathered 45–51º in fresh	30º in weathered 48º in fresh

Figure 14-4: Infrastructure Constraints To Pit Optimization



Note: Figure prepared by B2Gold, 2025. Infrastructure locations shown are proposed. North is to the top of the map.

Table 14-17:Indicated Mineral Resource Statement

Area	Tonnes (x 1,000)	Gold Grade (g/t Au)	Contained Gold Ounces (x 1,000)	Silver Grade (g/t Ag)	Contained Silver Ounces (x 1,000)
Gramalote Ridge Oxide	4,908	0.51	81	2.12	335
Gramalote Ridge Sulphide	151,501	0.71	3,443	0.88	4,307
Total Indicated Mineral Resources	***156,409***	***0.70***	***3,524***	***0.92***	***4,642***

Table 14-18:Inferred Mineral Resource Statement

Area	Tonnes (x 1,000)	Gold Grade (g/t Au)	Contained Gold Ounces (x 1,000)	Silver Grade (g/t Ag)	Contained Silver Ounces (x 1,000)
Gramalote Ridge Oxide	730	0.43	10	0.85	20
Trinidad Oxide	9,070	0.43	124	1.05	305
Monjas West Oxide	2,285	0.53	39	0.77	56
Subtotal Oxide Inferred	*12,085*	*0.45*	*173*	*0.98*	*381*
Gramalote Ridge Sulphide	9,666	0.53	164	0.81	251
Trinidad Sulphide	80,090	0.48	1,244	0.53	1,361
Monjas West Sulphide	21,118	0.63	430	0.40	274
Subtotal Sulphide Inferred	*110,873*	*0.52*	*1,839*	*0.53*	*1,886*
Total Inferred Mineral Resources	***122,958***	***0.51***	***2,012***	***0.57***	***2,267***

Notes to accompany Mineral Resource Tables:

1. Mineral Resources have been classified using the 2014 CIM Definition Standards.
2. Mineral Resources are reported inclusive of those Mineral Resources that were converted to Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
3. The QP for the resource estimate is Mr. Stephen Jensen, P.Geo., B2Gold's Exploration Manager for the Americas.
4. Mineral Resources are reported on a 100% project basis. The Gramalote Ridge, Trinidad and Monjas west estimates have an effective date of August 6, 2025.
5. Mineral Resources assume an open pit mining method and a gold price of US$2,500/oz.
6. Mineral Resources for Gramalote Ridge assume metallurgical recoveries of 84% for oxide and 92.7-97.6% for sulphide, and operating cost estimates of an average mining cost of US$2.59/t mined, processing cost of US$6.13/t processed for oxide and US$9.74/t processed for sulphide, general and administrative cost of US$2.10/t processed and selling cost of $82.84/oz of Au produced.
7. Mineral Resources for Trinidad assume metallurgical recoveries of 81.7% for oxide and 90.9% for sulphide, and operating cost estimates of an average mining cost of US$2.41/t mined, processing cost of US$6.13/t processed for oxide and US$9.74/t processed for sulphide, general and administrative cost of US$2.10/t processed and selling cost of $82.84/oz of Au produced.
8. Mineral Resources for Monjas West assume metallurgical recoveries of 81.7% for oxide and 87.6% for sulphide, and operating cost estimates of an average mining cost of US$2.58/t mined, processing cost of

US$6.28/t processed for oxide and US$9.89/t processed for sulphide, general and administrative cost of US$2.10/t processed and selling cost of $82.84/oz of Au produced.

9. Mineral Resources for Gramalote Ridge, Trinidad and Monjas West are reported at cut-offs of 0.13 g/t Au for oxide and 0.16 g/t Au for sulphide.
10. All tonnage, grade and contained metal content estimates have been rounded; rounding may result in apparent summation differences between tonnes, grade, and contained metal content.

- Changes to the input and design parameter assumptions that pertain to the conceptual pit constraining the estimates;
- Assumptions as to the continued ability to access the site, retain mineral and surface rights titles, maintain environment and other regulatory permits, and maintain the social license to explore and obtain the social license to operate.

14.14 Comments on Section 14

Mineral Resources are reported in accordance with the 2014 CIM Definition Standards.

There are no other environmental, legal, title, taxation, socioeconomic, marketing, political or other relevant factors known to the QP that would materially affect the estimation of Mineral Resources that are not discussed in this Report.

15.0 MINERAL RESERVE ESTIMATES

15.1 Introduction

Mineral Reserves at Gramalote Ridge have been converted from Indicated Mineral Resources. Inferred Mineral Resources were set to waste.

The mine plan assumes open pit mining using conventional mining methods and equipment.

15.2 Mineral Reserves Statement

The Mineral Reserve estimate for the Project reported within the Gramalote Ridge ultimate pit design is presented in Table 15-1.

The QP for the estimate is Mr. Peter Montano, P.E., B2Gold's Vice President, Projects. The estimate has an effective date of April 1, 2025.

15.3 Factors that May Affect the Mineral Reserves

Factors that may affect the Mineral Reserve estimates include:

- Changes to the gold price assumptions;
- Changes to pit slope and geotechnical assumptions;
- Unforeseen dilution or ore loss;
- Changes to hydrogeological and pit dewatering assumptions;
- Changes to inputs to capital and operating cost estimates;
- Changes to operating cost assumptions used in the constraining pit shell;
- Delays or changes to the resettlement and relocation plans;
- Changes to pit designs from those currently envisaged;
- Changes in mining or milling productivity assumptions;
- Changes to modifying factor assumptions, including environmental, permitting and social licence to operate.

Table 15-1: Mineral Reserves Statement

Area	Tonnes (x 1,000)	Gold Grade (g/t Au)	Contained Gold Ounces (x 1,000)
Gramalote Ridge Open Pit	76,700	0.96	2,360
Total Probable Reserves	***76,700***	***0.96***	***2,360***

Notes to accompany Mineral Reserves table:

1. Mineral Reserves have been classified using the CIM Definition Standards, are reported at the point of delivery to the process plant and have an effective date of April 1, 2025.
2. Mineral Reserves are reported on a 100% project basis.
3. The QP for the Mineral Reserve estimate is Mr. Peter Montano, P.E., Vice-president, Projects, B2Gold.
4. Mineral Reserves are based on a conventional open pit mining method, gold price of $1,750 per ounce, metallurgical recovery averaging 95.6%, selling costs of $60.00 per ounce including royalties, average mining cost of $2.70 per tonne mined, average processing cost of $8.50 per tonne processed, and average site general costs of $3.80 per tonne processed.
5. Reserve model dilution and ore loss was applied through whole block averaging such that at a 0.40 g/t Au cut-off there is a 1.2% increase in tonnes, a 4.6% reduction in grade, and 3.5% reduction in ounces when compared to the Mineral Resource model.
6. Mineral Reserves are reported above a cut-off grade of 0.40 g/t Au.
7. All tonnage, grade and contained metal content estimates have been rounded; rounding may result in apparent summation differences between tonnes, grade, and contained metal content.

15.4 Pit Optimization

Pit optimizations were completed using Geovia Whittle pit optimisation software. The pit shell sequences obtained from optimisations were analysed to define a practical mining sequence for the pit stage designs.

Utilizing the block model, cost, recovery, and slope data, the Whittle software determines a series of incremental pit shells, in which each shell is an optimum for a slightly higher price factor. In the analysis of the incremental pit shells, indicative net present values (NPV) were calculated by discounting the preliminary cash flows over time. The reported NPVs from the pit optimisation results provide indicative operating values for relative comparison purposes only. As well as the indicative NPVs, the incremental operating cost per ounce for the pit shells was also used to guide the pit shell selection and design process.

Additional optimisations were carried out to the base case optimisation to determine the sensitivities around the base case results.

The main optimization parameters are provided in Table 15-2.

15.5 Final Pit and Phase Selection

To define the internal and final pit phases, 86 optimizations were completed, starting at US$525/oz Au and running to US$3,500/oz Au. Different cash flows for each shell resulted from the optimization.

The final shell was selected to achieve the optimum project net present value (NPV), with one internal phase selected to improve the Project cash flow while considering other factors such as ore feed to the mill, mine fleet size, minimum mining width, practical sinking rates, and other planning considerations.

The US$1,750/oz Au pit shell was selected as the design basis for the ultimate pit. A smaller Phase 1 was defined using the US$768/oz Au shell, allowing for a lower strip ratio and early access to higher-grade material to support rapid payback.

Only two mining phases were selected in total (Figure 15-1). Phase 1 targets medium and high-grade zones that extend from the pit center toward the eastern pit wall. This phase is designed to deliver higher-grade ore during the initial years of operation, while allowing sufficient time for the Phase 2 ultimate pit to be developed. The sequencing strategy is detailed in Section 16.9.

Both the phase designs and the final pit layout are based on Whittle pit optimization results, and incorporate the selected mining equipment, geotechnical parameters, bench heights, slope angles for each phase, and berm widths.

15.6 Cut-Off Grade

Cut-off grades were shown in Table 15-2. For Mineral Reserve reporting, an applied cut-off grade of 0.40 g/t Au is used. Only sulphide material is processed in the mine plan. Oxide material is not included in the Mineral Reserves.

15.7 Dilution and Ore Loss

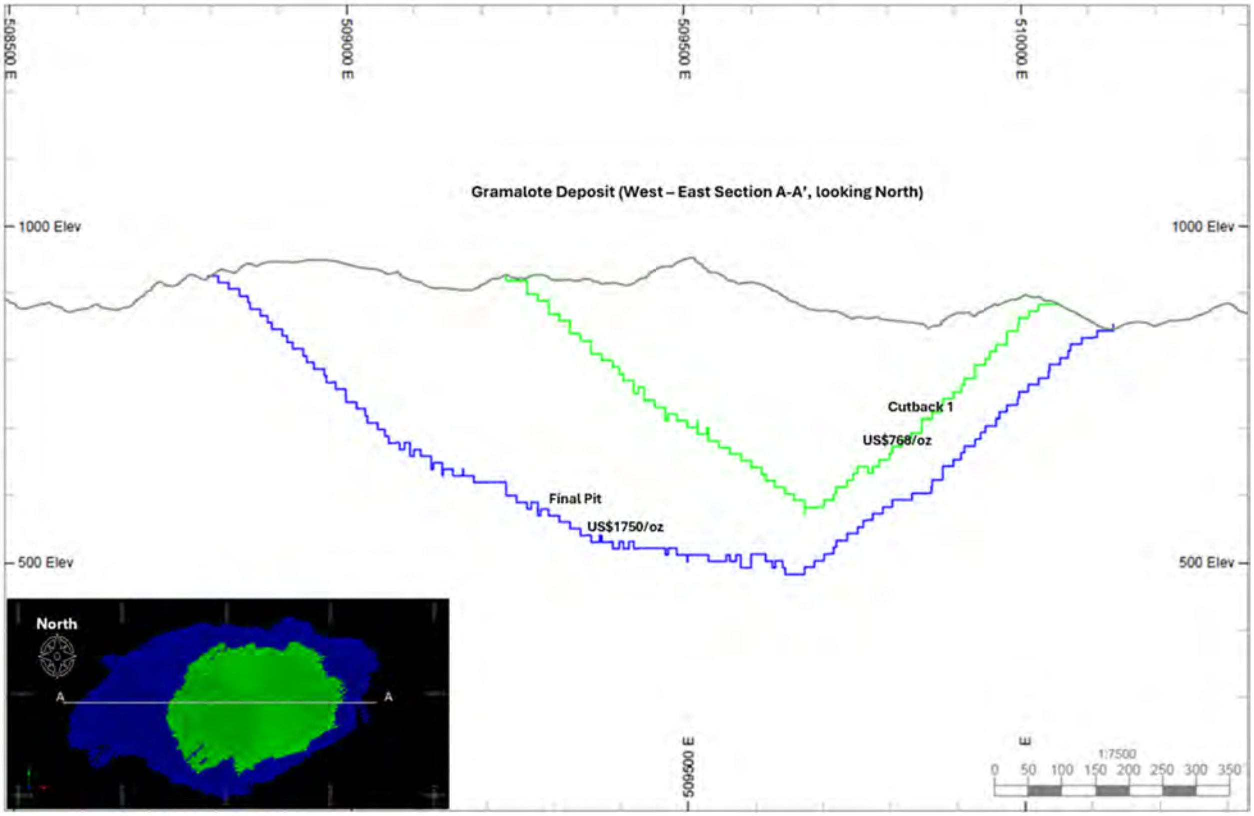
For Mineral Reserve reporting, the Resource model with 15 x 5 x 10 m parent blocks and 5 x 0.2 x 5 m sub-blocks was regularized to 15 x 5 x 10 m blocks. For Indicated blocks within the 2025 Feasibility Study Mineral Reserve pit, above a cut-off of 0.40 g/t Au, the regularized model compared to the resource model is +1.2% on tonnage, -4.6% on grade and -3.5% on contained gold. No additional dilution or ore loss was applied for Mineral Reserve reporting.

Table 15-2: Pit Optimization Parameters

Description	Unit	Value
Gold price	US$/oz Au	1,750
Silver price	US$/oz Au	24
Mining cost (820 m elevation)	US$/t mined	2.21
Mine sinking rate adjustment	US$/t 10m bench	-0.018 up 0.029 down
Mine rehabilitation cost	US$/t mined	0.05
Pit optimization mining cost	**US$/t mined**	**2.26**
Processing cost	US$/t processed	7.33
Tailings cost	US$/t processed	0.33
General and administrative cost	US$ M/year	22.8
Process plant throughput**	t/year	6,000,000
General and administrative cost	US$/t processed	3.80
Sustaining capital	US$/t processed	0.37
Closure capital	US$/t processed	0.46
Pit optimization processing cost	**US$/t processed**	**12.30**
Gold recovery	% of contained	0.927–0.976
Silver recovery	% of contained	0.394
Selling cost of gold	US$/oz produced	60.00
Selling cost of silver	US$/oz produced	2.04
Cut-off grade - calculated	g/t	0.24
Cut-off grade - applied	**g/t**	**0.40**
Pit slopes	degrees	31.6–50.3

Note: * Average mining cost after applying the sinking rate adjustment is $2.70/t mined ** Only sulphide ores are processed. Oxide material is not included in pit optimization ore reserves. *** Average gold recovery after ultimate pit shell selection is 95.6%.

Figure 15-1: Proposed Pit Phases



Note: Figure prepared by B2Gold, 2025. Green and blue lines show the revenue factor pit outlines.

15.8 Comments on Section 15

Mineral Reserves are reported in accordance with the 2014 CIM Definition Standards.

There are no other environmental, legal, title, taxation, socioeconomic, marketing, political or other relevant factors known to the QP that would materially affect the estimation of Mineral Reserves that are not discussed in this Report.

16.0 MINING METHODS

16.1 Overview

Mining operations will use conventional open pit mining methods and equipment.

The total active operational life is 11 years, including pre-stripping during construction. Processing will continue for an additional two years after active mining stops.

Mining of the pit will occur in two mining phases.

The basis for the mine plan is Mineral Reserves that have been converted from Indicated Mineral Resources. No Measured Resources were estimated in the block model. Inferred Mineral Resources were set to waste. Oxide material was also considered to be waste.

16.2 Geotechnical Considerations

Geotechnical designs are based on information collected from several years of surface mapping programs, geotechnical logging data, photo-logging observations, oriented core data, acoustic televiewer data, and results from laboratory testing to assess the intact rock strength.

Unconfined compressive strength tests indicated that unaltered tonalite has a “very strong” geotechnical rating, and the altered tonalite can be characterized as “strong”.

The geotechnical design parameters reflect mitigation methods against planar sliding hazards in specific design sectors, and the incorporation of a minimum empirical based berm width design.

All geotechnical data and recommendations were compiled in a geotechnical campaign completed in 2021 by the AngloGold Ashanti geotechnical team, a previous joint-venture partner in the Gramalote Project (Walker et al., 2021).

The geotechnical design parameters, based on 20 m-high benches, are presented in Table 16-1.

Since the 2025 Feasibility Study, pit design remains entirely within the pit boundaries defined in the 2021 AngloGold Ashanti study. The geotechnical parameters established at that time remain valid and are applicable to the current Mineral Reserves and pit designs.

The slope design recommendations, based on a 20 m bench height, are centred around defining an inter-ramp angle (IRA) over a 120 m stack height. This approach was primarily driven by study results indicating that potential for rockfall is the controlling factor for inter-ramp stability. As a result, the inter-ramp angle was limited to angles known to be achievable under similar mining conditions.

Table 16-1: Pit Slope Design Parameters

Domain	Wall Sector	Slope Orientation Range, Gramalote Ridge Pit (º)	Bench Height (m)	Bench Face Angle (º)	Bench Width (m)	Inter-Ramp Angle (º)
1 Weathered	Weathered	All	20	50	10	36.8
2 Fair	South	330–000–090	20	85	9.8	60
	West	090–150	20	75	9.8	52.8
	North	150–270	20	85	9.8	60
	East	270–330	20	75	9.8	52.8
3 Good	South	330–000–090	20	85	9.8	60
	West	090–150	20	75	9.8	52.8
	North	150–270	20	85	9.8	60
	East	270–330	20	85	9.8	60

The design also provides flexibility for future optimization, as actual mining practice may demonstrate that the proposed slope configuration can be safely steepened, provided that observed rockfall incidence remains sufficiently low.

The pit design was reviewed to ensure compliance to the defined geotechnical domains using Gem4D software. Pit designs were also checked for overbreak associated with geotechnical structures, rock fall potential, and formation of bullnoses. Geotechnical review concluded that the pit designs are acceptable for scheduling and budgeting purposes of the proposed mine plan.

To verify the methodology and ensure proper application of the geotechnical design parameters, several cross-sections were generated to check the bench face angles and berm configurations used in each pit sector.

16.3 Hydrogeological Considerations

A comprehensive hydrogeological investigation was carried out across the project site by Itasca to characterize the hydraulic behavior of various geological units. The campaign included the installation of 58 piezometers and execution of multiple test types including constant discharge, falling and rising head, and Lugeon (packer) tests. These were distributed across key areas such as the TMF, waste rock storage facility (WRSF), and the mining area, providing valuable data for groundwater modeling and slope stability assessments.

Results showed that alluvial sediments in valley bottoms have relatively high permeability, with hydraulic conductivity values ranging from 3.5×10^{-5} to 5.2×10^{-4} m/s and transmissivity between 20 and 180 m^2/day, consistent with an unconfined aquifer.

In contrast, saprolite materials exhibited low permeability, with hydraulic conductivities typically between 10^{-8} and 10^{-6} m/s. Falling and rising head tests were completed in saprolite due to insufficient water volumes for pump tests. The transition zone between saprolite and fresh rock showed variable permeability, depending on weathering and fracturing.

In the tonalite basement rocks, 61 Lugeon tests in the mining area and over 80 additional tests at the planned TMF and WRSF locations confirmed moderate to low permeability, with values from 10^{-9} to 10^{-5} m/s. Increased permeability correlated with fracture frequency, and no tests were conducted below 200 m due to equipment limitations. Overall, the testing program provided a solid baseline for understanding groundwater flow and supported the geotechnical and environmental design aspects of the project.

Groundwater infiltration is expected to occur at the contact between oxide and fresh rock. At this level in the pit, a 20m wide berm has been designed as a passive groundwater capture system. Much of the seepage water can be directed along the berm to exit points away from the pit.

Water inflows into the pit will be managed using mobile diesel pumps, which will be relocated as needed to various in-pit sumps, in accordance with the mine's development. The pumped water will be conveyed to the TMF via the access road that connects the process plant with the La Palestina Valley (see Section 18.3 for discussion on the TMF).

Estimated pumping rates range from 100–400 m^3/hour, depending on seasonal conditions. Under a 1/100-year rainfall event, rates are projected to increase to approximately 570 m^3/hour.

The first pit phase was designed to avoid the need for a diversion of Guacas Creek during the initial years. However, this diversion will be required prior to developing the final pit (see discussion in Section 18.5.4).

16.4 Pit Optimization

Pit optimization is discussed in Section 15.

16.5 Cut-off Grade

A cut-off grade was calculated for the sulphide material to determine the minimum gold grade that would generate a marginal economic benefit. The parameters used to support the cut-off grade determination for the Project are summarized in Table 15-2.

While the calculated cut-off grade for sulphide material was 0.24 g/t Au, an elevated cut-off grade of 0.40 g/t Au was applied for mine planning purposes. The material grading from 0.24–0.40 g/t, which is between the calculated cut-off and elevated cut-off for mine planning, will be stockpiled at the San Antonio WRSF. Oxide material is not processed, and therefore, does not have a defined cut-off grade. However, mineralized oxide material is also planned to be stockpiled at the San Antonio WRSF. This stockpiled material may represent a future source of mill feed material with additional studies.

16.6 Open Pit Design

The Gramalote Ridge deposit is planned to be mined in two phases:

- Pit Phase 1 is scheduled to begin in the second year of construction and continue through year 4 of operations, reaching a depth of 370 m. At the end of mining, pit Phase 1 will be approximately 900 m long, 500 m wide, and 370 m deep. This pit phase design contains 102 Mt, or 32% of the total LOM tonnage;
- Pit Phase 2 is planned to commence in year 3 of operations and progress to an ultimate depth of 510 m. At the end of mining, pit Phase 2 have extents of approximately 1,300 m long, 750 m wide, and 510 m deep. This pit phase design contains 217 Mt, or 68% of the total LOM tonnage.

Pit phasing for both phases are illustrated in Figure 16-1.

Pit Phase 1 was designed to avoid the necessity for a diversion of Guacas Creek, which is advantageous for production planning and capital costs. However, the Guacas Creek diversion will be required before commencing pit Phase 2.

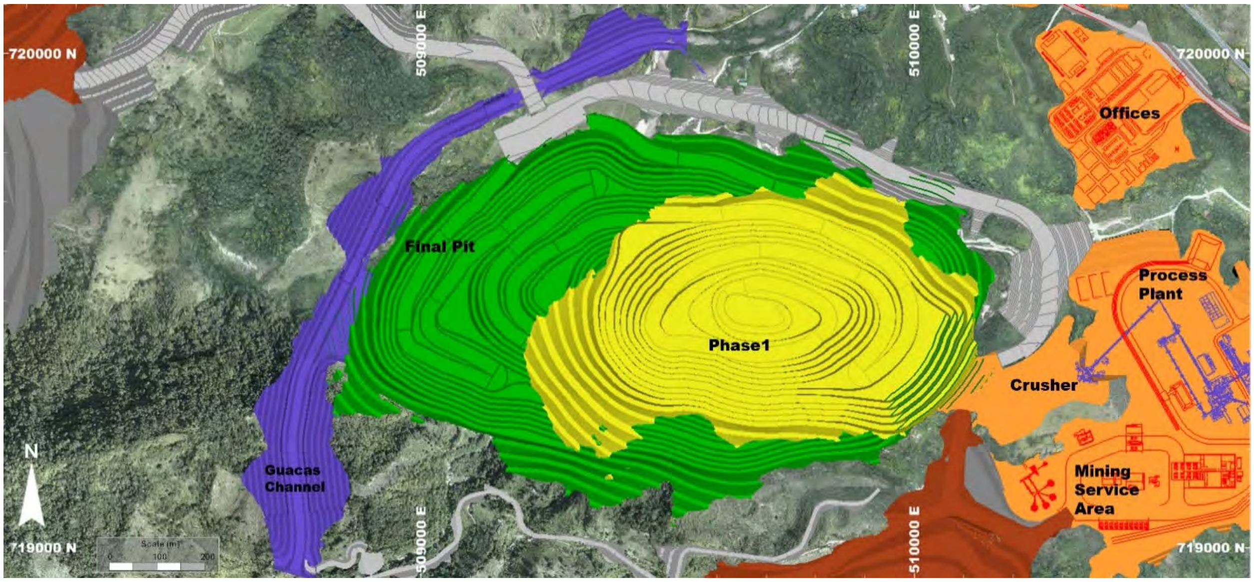
The main ramps were designed to 30 m width for two-way access, including allowances for drainage and berms and have a maximum 10% gradient. The dual ramp system will begin at elevation 660m and extend up to the pit exit to be located around 820 m elevation. A double-ramp system supports >75% of the total mined ounces. Below 660 m elevation, a single ramp system is planned, consisting of a 30-m-wide ramp with a 10% slope, extending to 500 m elevation.

Access to the pit was located on the northwest side to connect with a haul road. Both ramps will exit at elevation ~820. Additionally, three switchbacks are included on the west wall as shown in Figure 16-1.

During pre-stripping, a 30 m pioneering road will be established to create a wide platform to allow large/heavy mining equipment to operate.

Phase 1 includes a single access ramp, 30 m wide with a 10% gradient. The Phase 1 ramp exits on the northwestern side of the pit at elevation 834. Phase 1 features a switchback on the east wall.

Figure 16-1: Planned Pit Phasing



Note: Figure prepared by B2Gold, 2025.

16.7 Stockpiles

The mine planning strategy includes the use of stockpiles. The main stockpile will be located in the El Balzal area (Figure 16-2), and will have sufficient capacity to handle all tonnage requirements expected during the mine life.

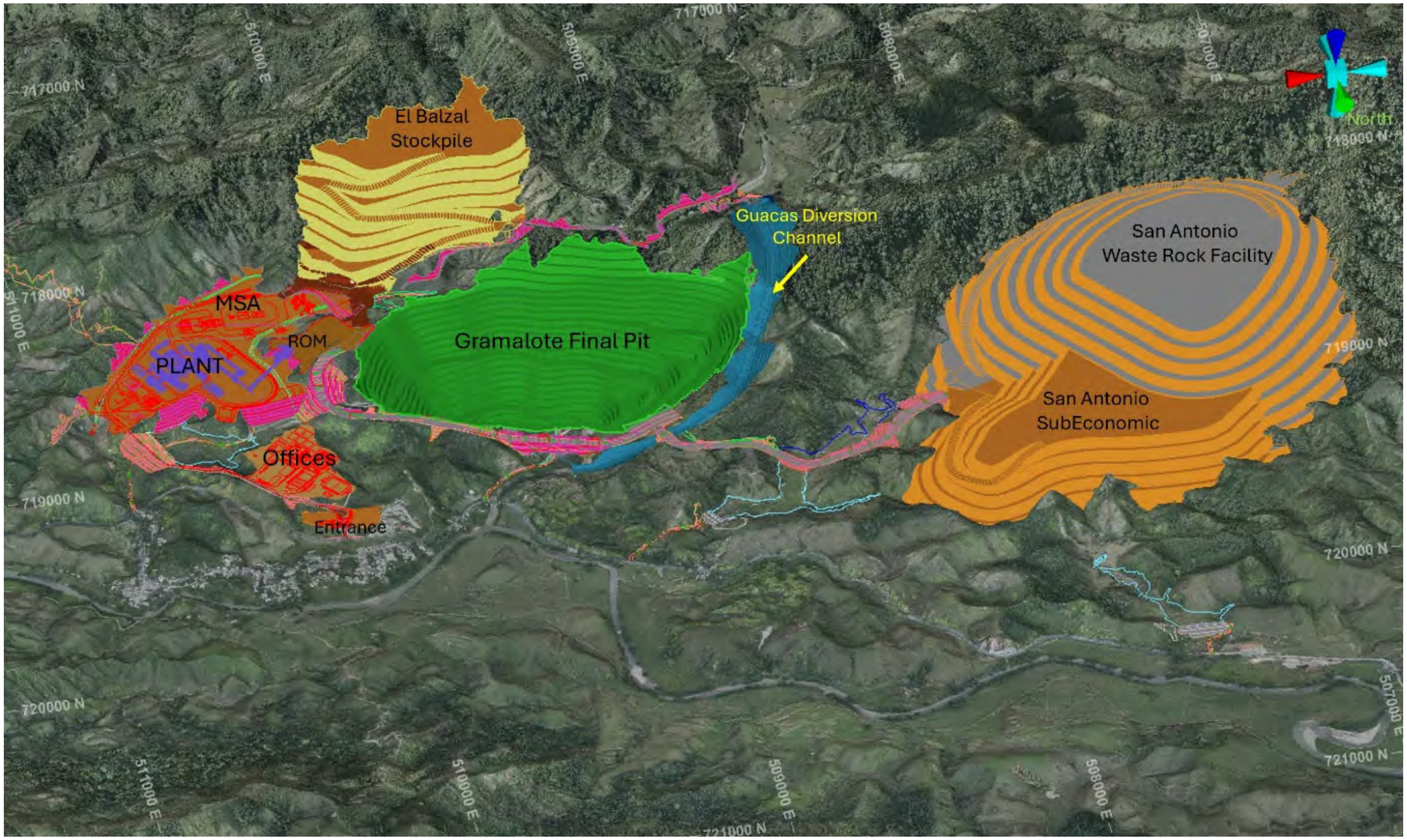
A cut-off strategy was developed dividing the ore in bins as follows:

- Fresh ore high-grade: ≥2.1 g/t Au;
- Fresh ore medium grade: 1.3–2.1 g/t Au;
- Fresh ore low-grade: 0.6–1.3 g/t Au;
- Fresh ore very low-grade: 0.4–0.6 g/t Au.

Mineralization grading 0.25–0.4 g/t Au, which is between the calculated cut-off and elevated cut-off for mine planning, will be placed in the San Antonio WRF area.

All stockpile areas will have access ramps that are 30 m wide, a maximum 10% grade, and a temporary face angle of 36º while in operations.

Figure 16-2: Proposed Open Pit, Stockpile, and WRSF Location Plan



Note: Figure prepared by B2Gold, 2025. MSA = mine services area; ROM = run-of-mine.

A comprehensive water management system was incorporated into the design, and includes contact and non-contact diversion channels (see discussion in Section 18.5). The contact water will connect at the mine services area with a channel that runs to the La Colorada sediment pond. The non-contact water will connect with the channel that goes on the eastern side of the process plant and discharge after the La Colorada sediment pond. These components are planned to be implemented during the construction phase.

16.8 Waste Rock Storage Facilities

The WRSF that will be located in the San Antonio area (refer to Figure 16-2) was designed to store approximately 240 Mt of waste, with additional capacity to accommodate future expansion if required. The facility design includes 20 m benches, 30 m berms, and 30 m haul roads, allowing for a final reclaimed slope of approximately 18° or a 3:1 ratio horizontal to vertical. The final bench reaches an elevation of 1,010 m, with the facility floor at 860 m.

Construction of the WRSF will begin by developing a large platform at the 870 m elevation, extending from the haul road to cover an area of approximately 70 ha. A bottom-up construction method will be applied, supported by a dual-ramp system to access both the waste rock disposal zone and the deposition area for the material grading 0.25–0.4 g/t Au.

A comprehensive water management system was incorporated into the design. This includes contact and non-contact diversion channels as well as two sediment ponds located in the San Antonio and El Banco basins. These components are planned to be implemented during the construction phase.

Some of the waste rock is classified as potentially acid generating (PAG). This material will be encapsulated with saprolite for a oxygen control barrier. Given the amount of saprolite material that will be available, studies and tests show the PAG material will be fully encapsulated preventing any discharge/affectation to the environment.

16.9 Production Schedule

The major constraints applied in the production schedule include:

- Providing 400 kt of ore prior to plant commissioning to enable sufficient buffer during start-up;
- Maintaining relatively consistent mining rates for better mining equipment utilisation throughout the mine life, ramping up from an initial pre-stripping pit and reaching steady state operations in year 3 of operations;
- Balancing haul distances at the end of the mine life to avoid purchasing additional trucks that will not have a long service life;

- Maintaining average vertical mining advance (sink) rates generally below 100 m per year, or approximately 10 benches per year. Sink rates tend to decrease near the base of each phase, where the working areas become narrow, water management requirements become challenging, the strip ratio is reduced, and the haul distances increase.

The mining sequence was developed using a combination of spreadsheets, mine design software, and other engineering tools. It considered production planning, haulage distances, and detailed tonnage scheduling by bench as well as by material type.

Pre-stripping is estimated at 13 Mt for the period prior to the first year of production. Initial ore will be stockpiled at the ROM area and later at the El Balzal area. A portion of the waste material will be used for the construction of mine platforms and haul roads.

A maximum extraction rate of 35 Mt/a is projected, with an 11-year LOM including the pre-stripping period. Processing will continue for an additional two years after active mining stops. The overall strip ratio is forecast at 3.1:1 (waste:ore). Mining operations will stockpile lower-grade material when applicable, which will be processed at the end of the mine life. The first pit phase will be mined early in the LOM to increase the plant feed grade during the initial years of production and to maintain a relatively steady gold production profile over the LOM.

Figure 16-3 is a graphic showing the projected total tonnes moved (ore and waste) and the strip ratio over the LOM. Figure 16-4 is a graphic showing the forecast ore tonnes and grade over the LOM. Figure 16-5 shows the material planned to be processed from stockpiles. Figure 16-6 summarizes the overall LOM gold production forecast.

Table 16-2 summarizes the LOM production forecast. Mining and process production forecasts on an annualized basis are provided in Table 16-3 and Table 16-4 respectively.

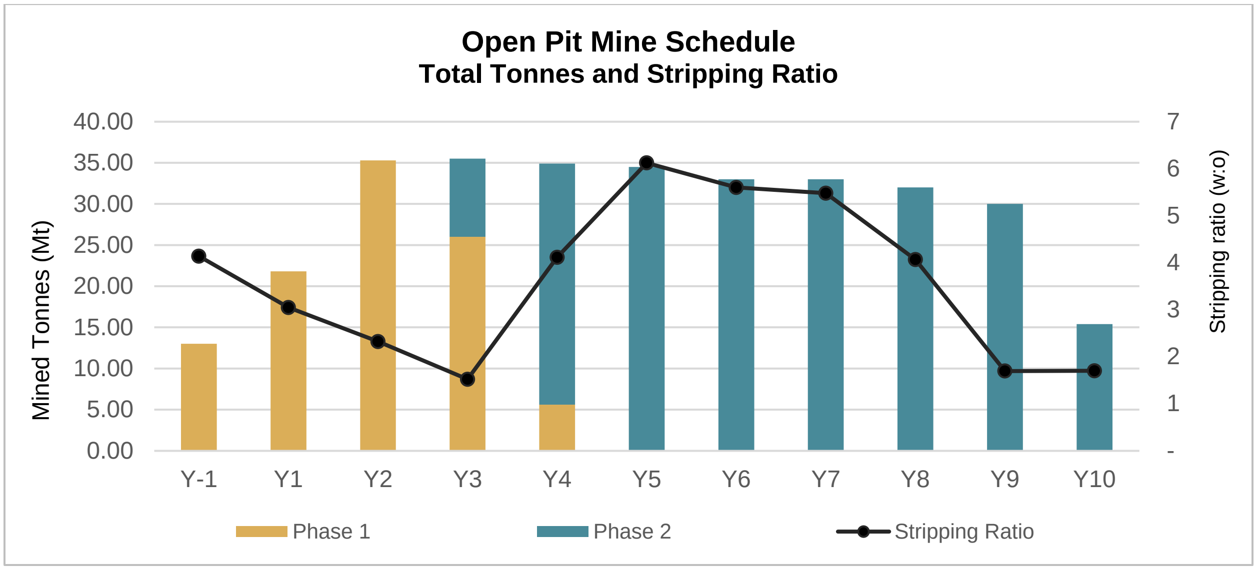
Mining will be carried out with an owner-operated fleet throughout the LOM.

16.10 Grade Control

Grade control drilling will be undertaken using 140 mm diameter RC drilling on a 15 x 10 m hole spacing for waste near ore, and 7.5 x 5 m for ore.

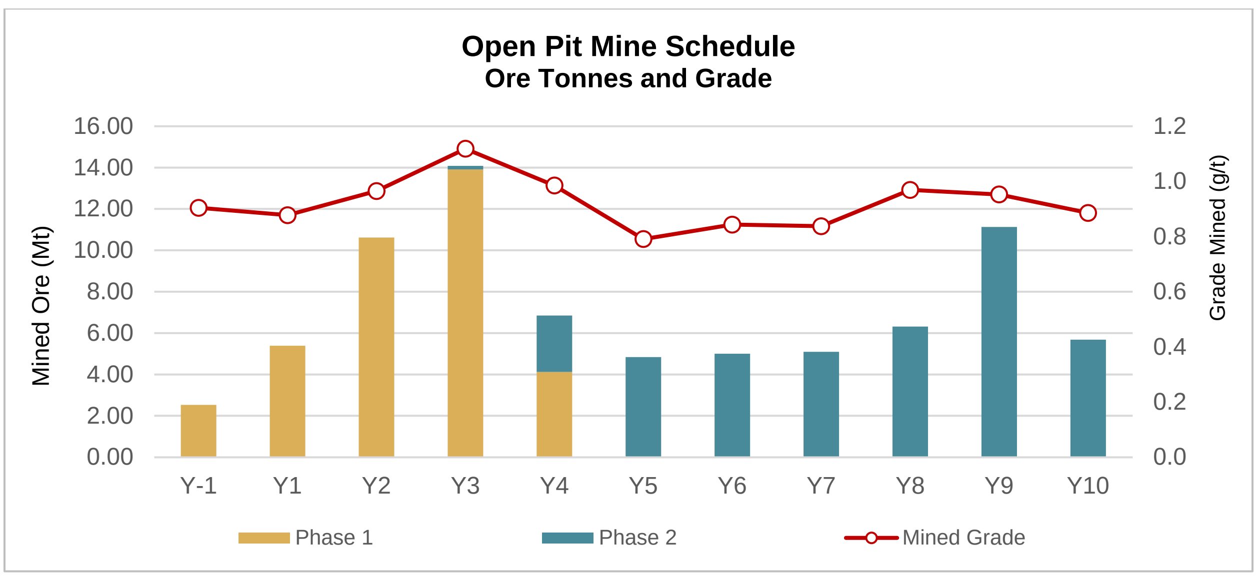
Samples will be sent to an onsite laboratory for gold and multi-element analysis, with pulp sample fractions remaining in storage.

Figure 16-3: Mine Production by Pit Phase, Material Movement and Strip Ratio



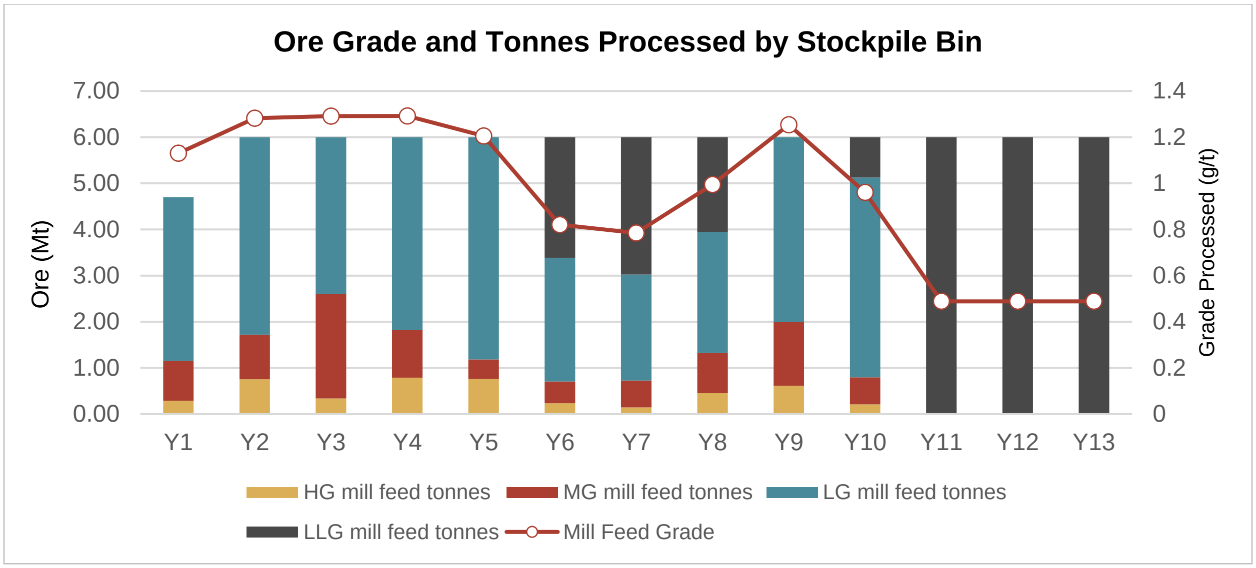
Note: Prepared by B2Gold, 2025.

Figure 16-4: Mine Production by Pit Phase, Ore Tonnage and Grade



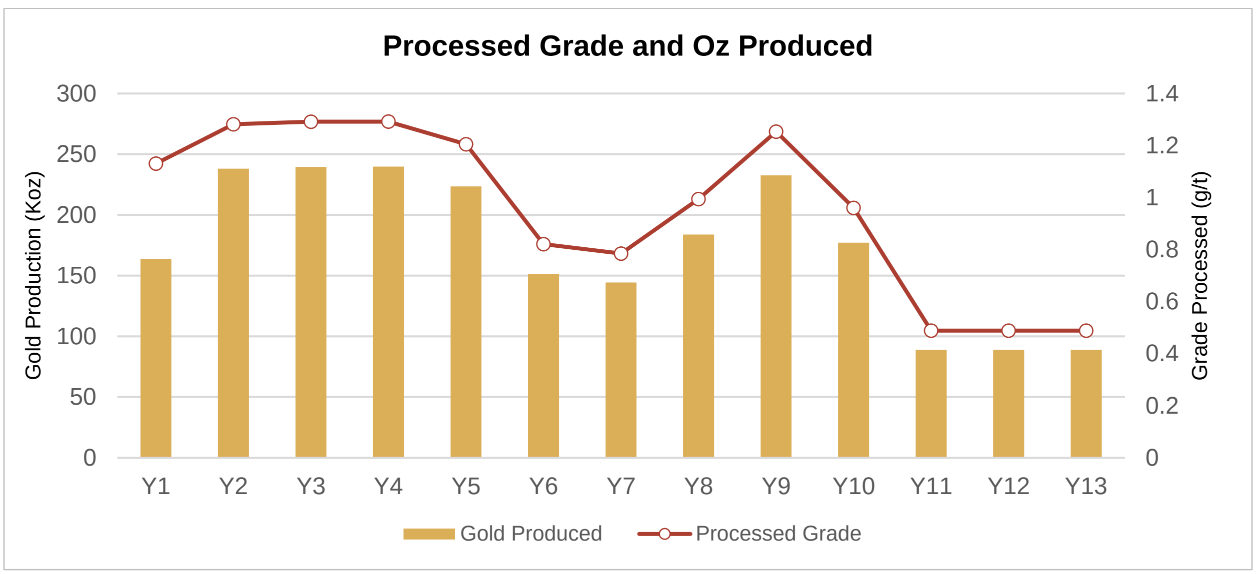
Note: Prepared by B2Gold, 2025.

Figure 16-5: Stockpile Grade and Tonnes Processed



Note: Figure prepared by B2Gold, 2025.

Figure 16-6: LOM Gold Production



Note: Figure prepared by B2Gold, 2025.

Table 16-2: LOM Production Schedule Summary

Item	Unit	Value
Open pit mine life	years	11
Open pit nominal production rate	Mt/a ore (max)	35.0
Process plant life	Years	13
Processing rate	Mt/a	6.0
Processing recovery	%	95.7
Average mined ore grade	g/t Au	0.96
Maximum long-term stockpile tonnage	Mt	19.2
Total life of mine gold production	koz	2,260
Average life of mine gold production	koz/a	177
Average gold production first five years	koz/a (first five years)	227

Note: Numbers have been rounded.

Table 16-3: LOM Mine Production Schedule

Combined LOM Mine Production Schedule	Units	Total LOM	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Open Pit Total													
Total waste	kt	**240,900**	10,500	16,400	24,700	21,400	28,200	29,700	28,000	27,900	25,700	18,900	9,700
Stripping ratio	w:o	**3.1**	4.1	3.0	2.3	1.5	4.1	6.1	5.6	5.5	4.1	1.7	1.7
Ore	kt	**77,500**	2,500	5,400	10,600	14,100	6,800	4,800	5,000	5,100	6,300	11,100	5,700
Grade	g/t Au	**0.78**	0.74	0.69	0.79	0.98	0.84	0.57	0.63	0.64	0.79	0.79	0.71
Contained gold	koz Au	**2,374**	74	152	330	506	217	123	136	137	197	341	162
Phase 1													
Total waste	kt	**65,200**	10,500	16,400	24,700	12,100	1,500	—	—	—	—	—	—
Stripping ratio	w:o	**1.8**	4.1	3.0	2.3	0.9	0.4	—	—	—	—	—	—
Ore	kt	**36,600**	2,500	5,400	10,600	13,900	4,100	—	—	—	—	—	—
Grade	g/t Au	**1.03**	0.90	0.88	0.97	1.12	1.17	—	—	—	—	—	—
Contained gold	koz Au	**1,214**	74	152	330	503	155	—	—	—	—	—	—
Phase 2													
Total waste	kt	**175,800**	—	—	—	9,300	26,600	29,700	28,000	27,900	25,700	18,900	9,700
Stripping ratio	w:o	**4.3**	—	—	—	56.2	9.8	6.1	5.6	5.5	4.1	1.7	1.7
Ore	kt	**41,000**	—	—	—	200	2,700	4,800	5,000	5,100	6,300	11,100	5,700
Grade	g/t Au	**0.88**	—	—	—	0.59	0.70	0.79	0.84	0.84	0.97	0.95	0.89
Contained gold	koz Au	**1,160**	—	—	—	3	62	123	136	137	197	341	162

Note: Numbers have been rounded.

Table 16-4: LOM Processing Summary

LOM Processing Summary	**Units**	**LOM Total**	**Y1**	**Y2**	**Y3**	**Y4**	**Y5**	**Y6**	**Y7**	**Y8**	**Y9**	**Y10**	**Y11**	**Y12**	**Y13**
Ore	kt	**76,700**	4,700	6,000	6,000	6,000	6,000	6,000	6,000	6,000	6,000	6,000	6,000	6,000	6,000
Grade	g/t Au	**0.96**	1.13	1.28	1.29	1.29	1.21	0.82	0.78	0.99	1.25	0.96	0.49	0.49	0.49
Contained gold	koz Au	**2,360**	171	247	249	249	233	158	151	192	242	185	94	94	94
Recovery	%	**95.7**	95.9	96.2	96.1	96.2	96.1	95.4	95.3	95.8	96.1	95.6	94.2	94.2	94.2
Produced gold	koz Au	**2,260**	164	238	239	240	223	151	144	184	233	177	89	89	89

Note: Numbers have been rounded.

16.11 Blasting

Blasting services will be a down-the-hole contract including the trucks, the explosives, and all the accessories at a contracted unit cost. Under Colombian law, all explosives and accessories can only be manufactured, supplied, and marketed by Indumil (Colombia's military industry).

The blast design considers ANFO, heavy ANFO, or emulsion as the main explosive for dry or wet holes, respectively.

Drilling and blasting operations will be carried out using widely accepted practices in the open pit mining industry.

The drilling and blasting pattern information was based on recommendations provided by Orica in 2021, which represents the most recent design review for the Project.

For production blasting of 10 m benches, 200 mm diameter blast holes will be used. Patterns will vary based on desired fragmentation. Ore blast patterns are planned at 5.2 x 5.0 m in production, while waste patters are planned at 7.5 x 8.7 m. Blast patterns, and hole sizes will be smaller in buffer blasts along pit walls.

The current average powder factor forecast varies from 0.46–1.18 kg/m^3. In near-surface oxide material, drilling and blasting requirements will be reduced.

16.12 Mining Equipment

Standard open pit mining equipment was selected with conventional drilling, blasting, loading, and hauling envisaged.

Initial access to the excavation areas will be executed using a contractor with a small fleet of articulated trucks, small excavators, and the required support equipment. This fleet will also be used for bulk earthworks during the plant platform construction and the initial road development.

Mine ancillary equipment will be used to support mass earthwork activities and expedite preparation of platforms to start pre-stripping.

The equipment was selected based on a standard open pit mining operation with conventional drill, blast, load and haul activities. The equipment selection considered bulk excavation of ore and waste using hydraulic excavators.

Mining will use hydraulic 200 t class excavators, with 100 t class haul trucks. Four main excavators will be required for the production period from Year 2 through Year 7, the number will then drop to the end of mine life as the material decreases.

An auxiliary fleet of equipment will support mine operations with development of access roads, haul road maintenance, building of safety berms, excavator support and cleanup, maintenance of the waste dumps, stockpile control, dust control, and drilling and blasting.

Table 16-5 details the peak heavy and auxiliary mining equipment planned to be used during the LOM.

Table 16-5: Owner Equipment Requirements

Equipment Type	Unit Numbers
Production drill at 200mm	3
Presplit/buffer drill at 140 mm	1
Grade control drill at 140 mm	2
100 t class haul truck	32
200–230 t class excavator	4
11–13 m^3 wheel loader	1
Mining bulldozers	5
Wheel dozers	2
Motor graders	2
100 t class water truck	4

17.0 RECOVERY METHODS

17.1 Process Flowsheet

The selected flowsheet is based on the metallurgical test work and trade-off studies described in Section 13 and uses unit operations that are well proven in industry. The flowsheet proposed is shown in Figure 17-1.

The key project and ore-specific criteria assumed for the plant design include:

- Throughput rate of 6 Mt/a;
- Pumps and conveyors will be sized with a 20% design margin, while all other plant equipment except for flotation concentrate leach stream equipment will have a 15% design margin;
- Flotation concentrate leach stream equipment will be sized for 5% mass pull without further design margin because typical flotation concentrate mass pull is expected to be 3%;
- Process plant availability of 92% supported by crushed ore storage, stand-by equipment in critical areas and stand-alone power supply;
- Sufficient automated plant control to minimize the need for continuous operator interface and allow manual override and control if and when required.

17.2 Plant Design

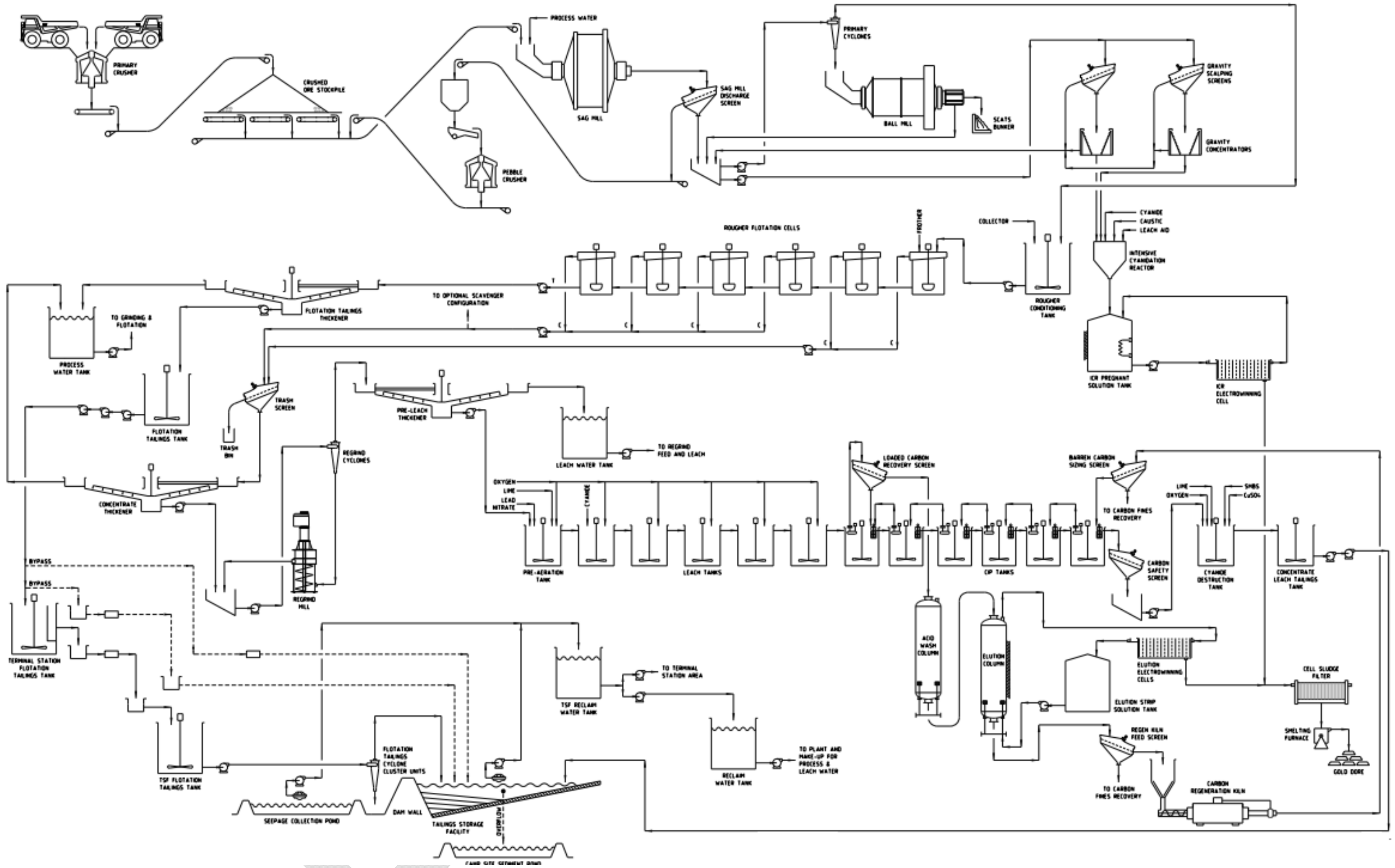
17.2.1 Crushing Circuit

Ore Receiving and Crushing

Run-of-mine (ROM) ore from the open pit, at maximum lump size of 900 mm, will be transported to the plant by 90 t capacity rear dump trucks. The trucks will tip directly into the ROM pocket; however, allowance will be made for the inclusion of a ROM stockpile. The ROM stockpile will be primarily used for emergency storage and ore blending. ROM ore will be reclaimed, from the stockpile to the ROM pocket via the ROM pad, by a front end loader.

A rock breaker will be installed to assist in breaking down oversize material retained on the ROM pocket. Ore will be withdrawn from the ROM pocket, with the material falling by gravity directly to the gyratory crusher. Crushed ore from the crusher discharges directly onto the sacrificial conveyor, prior to being conveyed, via the stockpile feed conveyor, to the crushed ore stockpile. The stockpile feed conveyor will be fitted with a weightometer to monitor the crushing area throughput and to control the flow of ore through the ROM pocket.

Figure 17-1: Proposed Process Flowsheet



Note: Figure prepared by Lycopodium, 2025.

The crushing circuit will be serviced by a single dust collection system. Dust collected from this system will be discharged onto the stockpile feed conveyor and then finely sprayed with water for dust suppression. A foam addition system for dust suppression of the crusher discharge product will also be installed for use, as required.

Crushed Ore Stockpile

The crushed ore stockpile will have a live capacity of approximately 9,000 t (equivalent to 12 h of mill feed at 6 Mt/a).

Crushed ore will be reclaimed from the stockpile by three variable speed apron feeders. The feeders will discharge onto the SAG mill feed conveyor which will convey the crushed ore to the SAG mill feed chute. The SAG mill feed conveyor will be fitted with a weightometer, used for controlling the speed of the reclaim feeders and for mass accounting of feed presented to the grinding circuit.

Grinding media for the SAG mill will be added onto the SAG mill feed conveyor from drums using a front end loader via the SAG mill ball hopper. This hopper, which is positioned after the weightometer, also enables clean up material to be returned to the circuit via the SAG mill feed conveyor.

A ventilation fan will force air into the concrete stockpile reclaim chamber to ensure fresh air ventilates the upper part of the chamber which would otherwise have limited natural ventilation.

17.2.2 Grinding and Classification

The grinding circuit will be a traditional SABC circuit, consisting of a single SAG mill and a single ball mill. The SAG mill will operate in closed circuit with a pebble crusher, whilst the ball mill will operate in closed circuit with hydro-cyclones. The product particle size exiting the grinding circuit (cyclone overflow stream) will contain 80% passing 250 µm material.

The SAG mill will be fitted with curved pulp lifters and discharge grates, which will allow slurry to pass through the mill and will also relieve the mill of pebble build-up. The SAG mill discharge stream will be screened by SAG mill discharge screen, with the oversize pebbles discharging onto the SAG mill oversize conveyor.

SAG mill discharge screen oversize will be conveyed to a pebble crushing circuit. Undersize from the screen will flow by gravity to the cyclone feed pump box, where it will combine with the discharge slurry from the ball mill and tailings from the gravity concentrators and scalping screens oversize. Part of the slurry will be pumped to the gravity circuit for recovery of gravity gold. The rest of the slurry stream will be pumped to the cyclone cluster. Process water will be added to the SAG mill feed chute for mill density control. Additionally, process water will be added to the cyclone feed pump box for cyclone feed density control.

The cyclone cluster overflow will flow by gravity through a process sampler for particle size analysis, followed by a metallurgical sampler for metallurgical accounting purposes, to the flotation circuit. Slurry from the cyclone underflow launder, will be returned to the ball mill feed chute for further grinding via a boil box. Ball mill discharge will pass through the ball mill trommel prior to discharging to the cyclone feed pump box. Reject oversize material, from the ball mill trommel screen, will be collected within the ball mill scats bunker.

Grinding media will be added to the ball mill via the ball mill feed chute from drums, utilizing a dedicated media hoist, kibble, and feed chute.

Spillage within the grinding circuit will be managed, utilizing a dedicated drive-in sump and sump pump. A second sump pump will be provided in the ball mill area. Any spillage generated in the grinding area will be returned to the cyclone feed pump box.

17.2.3 Pebble Crushing

Oversize from the SAG mill discharge screen will be conveyed to the pebble crusher feed bin via a series of belt conveyors. A self-cleaning belt magnet will be positioned at the head chute of the first conveyor to remove any scrap metal and steel media which could potentially damage the pebble crusher. A second self-cleaning belt magnet will be positioned across the belt of the second conveyor.

Downstream of the self-cleaning belt magnets, the pebbles pass under a metal detector prior to discharging into the pebble crusher feed bin. The feed bin will provide surge capacity ahead of the pebble crushers and allow a controlled feed to be presented to the crusher being operated. If the pebble crusher feed bin reaches a high level, then a diverter gate ahead of the pebble crusher feed bin will allow pebbles to bypass the pebble bin and crushers, and feed directly to the pebble crusher discharge conveyor below. If the metal detector detects tramp metal (not removed by the belt magnets), the diverter gate ahead of the pebble crusher feed bin will automatically enable pebbles to bypass the pebble bin and crushers, and discharge directly to the pebble crusher discharge conveyor below.

Pebbles will be withdrawn from the feed bin by a vibrating feeder and discharge into a duty only pebble crusher. The crushed pebbles will discharge directly onto the pebble crusher discharge conveyor, which in turn will return the crushed pebbles to the SAG mill feed conveyor.

17.2.4 Gravity Circuit

Recovery of gravity gold will be achieved via two duty gravity centrifugal concentrators operating in parallel. A gravity circuit feed pump will direct slurry to the centrifugal concentrators via scalping screens to prevent coarse heavy particles and mill ball steel chips from being recovered by the gravity concentrators.

Concentrate recovered by the gravity concentrators will flow by gravity to the gravity concentrate settling cone before being processed by the intensive cyanidation reactor. Oversize from the scalping screens and tailings from the gravity concentrators will flow by gravity to the cyclone feed pump box.

The intensive cyanidation reactor (ICR) will be programmed with an automatic sequence for gold dissolution using a concentrated cyanide solution. Upon completion of a batch, the dissolved gold in solution will be pumped to the ICR pregnant solution tanks for recovery by electrowinning. The ICR tailings stream will be pumped to the flotation concentrate regrind circuit for further grinding and gold extraction. Thus, any cyanide-bearing solution or solids will not be directed to the grinding or flotation circuit, as iron sulphide minerals such as pyrite can be easily depressed by free cyanide and negatively impact flotation circuit performance.

Spillage within the gravity circuit will be managed, utilizing a dedicated floor sump pump. Any spillage generated in the gravity area will be pumped to the regrind cyclone feed pump box to avoid any cyanide-bearing material from entering the grinding or flotation circuit.

17.2.5 Flotation Circuit

The cyclone overflow stream will flow by gravity to the rougher conditioning tank. Flotation will consist of rougher and scavenger flotation, with rougher concentrate reporting to the flotation concentrate thickener and scavenger concentrate either reporting to the flotation concentrate thickener or recycled to the rougher conditioning tank. It is anticipated that the rougher only flowsheet will be the predominant operating method. However, in the event that higher concentrate grades or lower mass recoveries need to be targeted, a rougher/scavenger flowsheet arrangement can be used instead.

Six flotation tank cells will be arranged in series. The reagent addition scheme with PAX collector and W31 frother will mimic the laboratory testwork for the selected size and number of flotation cells. Flotation circuit tailings will be transferred via dedicated pumps to the flotation tailings thickener.

A flotation tailings metallurgical sampler will be installed prior to the flotation tailings thickener feed box for metallurgical accounting purposes.

Spillage within the flotation circuit will be managed, utilizing a number of floor sump pumps. Any spillage generated in the flotation area will be returned to either the rougher conditioning tank or concentrate pump boxes.

17.2.6 Concentrate Thickening

The flotation concentrate will be pumped to a trash screen, where oversized material is collected in a trash bin. The undersize will then be directed to the concentrate thickener ahead of a regrind circuit. The thickener provides a buffer between the flotation circuit and the regrind mill. It also provides control of the water balance around the regrind circuit and helps to break down froth that may otherwise be an issue in the regrind circuit.

Overflow solution from the concentrate thickener will flow by gravity to the concentrate thickener overflow tank and then be pumped to the process water tank. Underflow from the concentrate thickener, at 50% solids, will be pumped to the regrind cyclone feed pump box.

The concentrate thickener area will be serviced by a dedicated floor sump pump. Spillage and wash down collected by the sump pump will be returned to the thickener feed box.

17.2.7 Concentrate Regrind

Flotation concentrate thickener underflow will be pumped directly to the regrind cyclone feed pump box. The regrind cyclone feed stream will be classified by hydrocyclones, and cyclone underflow will flow by gravity to the regrind mill for grinding. The cyclone overflow product size of P_{80} 20 µm is expected to be achieved with a low cyclone overflow slurry density of 15% solids (w/w) to promote better particle size separation efficiency. The cyclone overflow stream will flow by gravity to the pre-leach thickener.

Grinding media from drums will be added to the regrind mill via a chute, utilizing a dedicated media hoist and kibble.

The regrind mill area will be serviced by a dedicated floor sump pump. Spillage and wash down collected by the sump pump will be returned to the regrind cyclone feed pump box.

17.2.8 Pre-Leach Thickening

The regrind cyclone overflow slurry stream will flow by gravity directly to the pre-leach thickener feed box. Overflow solution from the pre-leach thickener will flow by gravity to the leach water tank. Underflow from the pre-leach thickener, at 45% solids, will be pumped to the leach feed distribution box.

The pre-leach thickener area will be serviced by a dedicated floor sump pump. Spillage and wash down collected by the sump pump will be returned to the thickener feed box.

17.2.9 Concentrate Leach Circuit

Leach thickener underflow will be pumped to the leach feed distribution box. The slurry from the leach feed distribution box will flow by gravity to one pre-aeration tank, followed by the five leach tanks.

The concentrate leach circuit will consist of six, mechanically agitated, slurry tanks operating in series. This equates to a residence time of 6 h for pre-aeration and 30 h for leaching at a design feed rate of 6 Mt/a with a flotation concentrate mass pull of 5%

and a slurry density of 35% solids (w/w). The selected leaching residence time of 30 h includes a design factor to allow for one slurry tank to be bypassed and still achieve 24 h residence time. Each slurry tank will have a live volume of 500 m^3.

The target concentrate leach conditions include a pH 10.5–11 maintained with slaked lime slurry, 1 kg/t of lead nitrate, 2 g/L of sodium cyanide and dissolved oxygen levels of 20 to 25 g/L.

Oxygen generated on-site will be added to each of the six slurry tanks via oxygen addition devices and dedicated oxygen contactor slurry pumps. Slaked lime slurry will be added via a ringmain to the leach feed distribution box and leach tanks as needed.

Cyanide analysers for on-line monitoring of the free cyanide concentration will allow for controlled addition of sodium cyanide solution to tanks as needed.

Should a leach tank be off-line for maintenance, it will be possible to bypass any of the leach tanks. The ability to bypass tanks will be made possible by the installation of pneumatic gates located within the leach inter-stage launders.

The leach circuit will be serviced by two floor sump pumps. Sump pumps will return slurry spillage to a nearby leach tank, and storm water to the pre-leach thickener feed box.

17.2.10 Carbon in Pulp Circuit

The CIP circuit will consist of six mechanically agitated CIP tanks, operating in series. Each CIP tank will have a live volume of 200 m^3. The total volume of the CIP tanks will exceed the required residence time of 6 h at design plant throughput and slurry density but will ensure sufficient residence time is achieved if the slurry density is diluted by sump pumps or loaded carbon recovery screen water sprays.

The leaching circuit will dissolve the gold present and the CIP circuit will recover this dissolved gold by carbon adsorption. Activated carbon will be retained in each of the CIP tanks by an inter-tank screen.

As the slurry flows through the CIP tanks, the carbon will be advanced counter-current to the slurry flow. This will be managed by balancing the carbon inventory in each CIP tank, by conducting regular measurements of the carbon concentration. Carbon advancement will be achieved by the CIP carbon recessed impeller transfer pumps.

Loaded carbon from the first CIP tank will be pumped to the loaded carbon recovery screen, where it will be washed to remove excess slurry. The excess slurry (screen underflow) will flow by gravity to the CIP tank of origin whilst the loaded carbon will flow by gravity to the acid wash column.

Regenerated carbon (or fresh carbon) will be added to the CIP circuit from the carbon regeneration circuit. The regenerated carbon (or fresh carbon) will be pumped to the

CIP circuit via the barren carbon sizing screen. The sizing screen will remove excess water and carbon fines. The dewatered carbon will discharge into the last online CIP tank with excess water and carbon fines directed to the carbon fines collection hopper. From there, it will be pumped through a carbon fines filter to the area sump pump, which then pumps the filtered water to the CIP tailings pump box for disposal.

Slurry discharging the last CIP tank will flow by gravity to the carbon safety screen. The carbon safety screen will capture and recover any carbon exiting the CIP circuit. The safety screen oversize will report to a fine carbon skip bin while the undersize will be pumped to the cyanide destruction feed box.

Should a CIP tank be offline for maintenance, it will be possible to bypass any of the CIP tanks. The ability to bypass tanks will be made possible by the installation of pneumatic gates located within the CIP tank inter-stage launders.

The CIP circuit will be serviced by floor sump pumps. Sump pumps will return spillage to either a nearby CIP tank or the CIP feed distribution box.

17.2.11 Acid Wash, Elution, Electrowinning and Gold Room

The carbon desorption circuit includes separate acid wash and elution columns of equivalent size. A cold acid wash will be utilized for removal of inorganic foulants from the carbon. Following acid wash, gold desorption from the carbon will be achieved utilizing a Pressure Zadra elution circuit. An elution column size of 4 t operated daily will satisfy the required carbon movements for the CIP circuit. The pressure Zadra circuit can be operated up to twice daily as needed. The dissolved gold will be recovered from the pregnant elution strip solution by electrowinning onto woven, stainless steel wire mesh cathodes.

The intensive cyanidation reactor will also produce pregnant solution for electrowinning. A dedicated electrowinning cell and ICR pregnant solution storage tanks will be used for gold recovery onto the electrowinning cell cathodes.

Gold sludge collected in the electrowinning cells will be recovered into a sludge hopper, from where it will be filtered via a sludge filter press.

The gold bearing filter cake will be thermally dried in an electric drying oven. Dried filter cake will be mixed with a prescribed flux mixture (silica, nitre, soda ash and borax), prior to being charged into the electric induction furnace. The molten metal will be poured into moulds to form doré ingots, which will be cleaned, assayed, stamped, and stored in a secure vault ready for dispatch. The slag produced will periodically be returned to the grinding circuit, via the SAG mill feed chute.

17.2.12 Carbon Regeneration

After elution, the barren carbon will be pumped from the elution column to the carbon regeneration circuit. The carbon and transfer water will be directed to the regen kiln feed screen, allowing excess water to be removed prior to the carbon discharging into the regen kiln feed hopper.

Carbon will be withdrawn from the regen kiln feed hopper via the kiln screw feeder and discharged directly into the carbon regeneration kiln. Within the electric horizontal rotary kiln, the carbon will be heated to 650°C to remove volatile organic foulants from the carbon surface, thereby restoring the carbon activity.

Re-activated carbon exiting the kiln will discharge to the carbon quench hopper and then be pumped to the barren carbon sizing screen located above the CIP circuit.

17.2.13 Cyanide Destruction

CIP tailings will be pumped to the cyanide destruction tank where cyanide destruction will be achieved using the SO_2/air process. In the SO_2/air process, sodium metabisulphite, air, copper sulphate (catalyst), and slaked lime slurry (as required) will be added to oxidize the residual free and weakly acid dissociable (WAD) cyanide to cyanate. The destruction circuit will reduce the residual cyanide contained within the tailings stream to below 10 mg/L WAD cyanide. The cyanide destruction circuit will consist of one mechanically agitated tank with a residence time of 240 min.

Slurry discharging the cyanide destruction circuit will flow by gravity to the concentrate leach tailings tank. A cyanide destruction discharge process sampler is installed on the discharge line to collect samples for monitoring and ensure that the cyanide destruction process has been completed effectively.

17.2.14 Concentrate Leach Tailings Disposal

Following cyanide destruction, the concentrate leach tailings will be pumped to the TMF. Discharge of concentrate leach tailings into the TMF is done sub-aqueously.

17.2.15 Flotation Tailings Thickening

Flotation circuit tailings will be pumped through a flotation tailings metallurgical sampler to the flotation tailings thickener. Overflow solution from the flotation tailings thickener will flow by gravity to the process water tank. Flotation tailings thickener underflow, at a slurry density of 55–60% solids, will be pumped to the flotation tailings tank for subsequent disposal to the TMF.

17.2.16 Flotation Tailings Pumping and Tailings Storage Facility

Flotation tailings slurry will be pumped via three slurry pumps in series to the terminal station flotation tailings tank. From there, it will flow by gravity through three dropboxes to a choke station, which regulates flow and pressure to prevent surges, then through a final dropbox before reaching the TMF flotation tailings tank. The slurry will be diluted and pumped to individual cyclones for desliming, where the cyclone overflow slurry stream or slimes will be discharged to the TMF, and the cyclone underflow slurry stream or sand with <15% passing 75 µm is used for dam wall embankment construction. The cyclones will be mounted on the dam wall, and typically four or five duty cyclones will be operated at a time to provide sand deposition where needed. As deposition progresses, the cyclone structures are periodically excavated and relocated to maintain required embankment profiles.

When flotation tailings slurry is not being deslimed, a bypass stream will flow by gravity through a similar dropbox and choke station arrangement before discharging into the TMF at multiple points. Additionally, an alternative bypass slurry stream without dropboxes will be used to discharge directly into the TMF via a separate choke station. A comprehensive tailings disposal strategy, including sand dam and sand stack development plans, has been developed by the tailings consultant.

Reclaim water from the TMF is transferred to the TMF reclaim water tank via decant pumps. One pump set will supply cyclone feed dilution water and gland water to the terminal station area, while another pump set returns water all the way to the main process plant reclaim water tank. Seepage from the TMF's impoundment and dam wall construction is collected in the seepage collection pond and pumped to the TMF reclaim water tank.

Excess water in the TMF is managed via a sidehill decant system, where it will flow by gravity through a tunnel and piping system to the campsite sediment pond.

Concentrate leach tailings slurry is discharged sub-aqueously into the TMF pond.

17.3 Equipment Sizing

The equipment list was derived based on the key process design criteria listed in Table 17-1.

Major equipment requirements are shown in Table 17-2.

Table 17-1: Key Process Design Criteria

Parameter	Units	Design
Plant throughput	t/a	6,000,000
Head grade, design	Au g/t	2.50
Gravity gold recovery	%	11.0
Overall gold recovery[1]	%	97.2
Crushing plant availability	%	65.0
Plant availability	%	92.0
Crushing work index (CWi)	kWh/t	17.0
Bond rod mill work index (RWi)	kWh/t	15.3
Bond ball mill work index (BWi)	kWh/t	15.7
SMC test parameter, A*b [1]		31.9
Bond abrasion index (Ai)		0.599
Grinding circuit product size, P_{80}	µm	250
Total flotation residence time (laboratory)	min	20
Plant flotation residence time	min	50
Mass recovery to flotation concentrate	%	5.0
Flotation concentrate thickener rise rate	m/h	2
Flotation concentrate thickener underflow density	% solids (w/w)	50
Regrind circuit product size, P_{80}	µm	20
Vertimill specific energy	kWh/t	15.75
Pre-leach thickener rise rate	m/h	2
Pre-leach thickener underflow density	% solids (w/w)	45
Concentrate leach density	% solids (w/w)	35
Pre-aeration residence time	h	6
Concentrate leach residence time	h	30
Cip residence time	h	6
Number of pre-aeration tanks		1
Number of leach tanks		5
Number of adsorption tanks (stages)		6
Conc. leach pH		10.5–11.0
Conc. leach lime addition [2]	kg/t conc.	4.13
Conc. leach sodium cyanide addition [3]	kg/t conc.	5.58
Conc. leach lead nitrate addition	kg/t conc.	1.0
Dissolved oxygen level in pre-aeration and conc. leach	ppm	20–25

Parameter	Units	Design
Elution circuit type		Pressure Zadra
Elution circuit size	t	4
Frequency of elution	strips / week	7
Cyanide destruction circuit type		SO_2 & air
Cyanide destruction residence time	h	4
SO_2 / CNwad weight ratio	g SO_2 : g CNwad	4.5
Flotation tailings thickener solids loading	m^2/t/day	0.035
Flotation tailings discharge slurry density	% solids (w/w)	55–60

Notes:

1. Design A x b value derived from the 15th percentile ranking of specific energies determined for each individual ore type.

2. Quicklime addition based on supply concentration of 88% CaO.

3 Sodium cyanide addition based on supply purity of 98.0% NaCN.

Table 17-2: Major Equipment List

Equipment Description	Details
Primary crusher	42” x 65” MKII Gyratory
SAG mill	32’ dia x 17.5’ EGL with 2 x 5MW drives
Ball mill	17’ dia x 27’ EGL with 1 x 5MW drive
Pebble crusher	1 x Raptor 450 cone crusher with provision for additional future crusher
Primary cyclones	10 x 650 mm cyclone cluster – 6 duty, 2 standby, 2 spare
Gravity concentrators	2 x KC-QS48
Intensive cyanidation reactor	CS4000
Flotation cells	6 x 300 m^3 tank cells
Concentrate thickener	12 m diameter high rate
Regrind mill	VTM-1000-WB
Regrind cyclones	7 x 150 mm cyclone cluster – 4 duty, 1 standby, 2 spare
Pre-leach thickener	12 m diameter high rate
Leach tanks	1 x pre-aeration + 5 x leach tanks, 508 m^3 each
CIP tanks	6 x 266 m^3 tanks, each with 2 m^2 intertank screens
Carbon safety screen	1.2 m x 2.4 m single deck vibrating screen
Cyanide destruction tank	1 x 407 m^3 tank
Flotation tailings thickener	30 m diameter high rate
Flotation tailings cyclones	10 x single 566 mm cyclones– 4 duty, 6 standby

Equipment Description	Details
Elution circuit	4t Pressure Zadra Circuit
Electrowinning cells	3 x 33 cathode cells, 1 x 13 cathode cell for gravity
Furnace	Induction furnace, 500 KC
Regeneration kiln	366 kg/hr electric kiln

17.4 Control Strategy

The plant control system will primarily consist of a distributed control system and be designed around a moderate level of automation and monitoring.

The starting and stopping of most electrical drives and actuated valves will be performed from the distributed control system operator interface terminals.

In general, the use of actuated isolation or control valves will be implemented around the plant for automatic control loops or sequencing as part of the plant control or the elution sequence.

The majority of equipment interlocks will be software configurable. All alarm and trip circuits from field or local panel mounted contacts will be based on fail-safe activation. The process plant will be provided with a one main control room.

17.5 Power and Consumables

17.5.1 Consumables

Reagents

The major reagents used within the process plant will include:

- Slaked lime slurry for pH control in concentrate leaching and cyanide destruction;
- Potassium amyl xanthate (PAX) collector for froth flotation;
- Frother to provide a stable froth for froth flotation;
- Sodium cyanide (NaCN) for gold dissolution in concentrate leaching and intensive cyanidation reactor and carbon desorption;
- Lead nitrate ($Pb(NO_3)_2$) for enhancing gold dissolution;
- Caustic soda or sodium hydroxide (NaOH) for pH control in intensive cyanidation reactor, carbon acid washing neutralization, carbon desorption and electrowinning;
- Hydrochloric acid (HCl) for carbon acid washing;
- Sodium metabisulphite (SMBS) for cyanide destruction;

- Copper sulphate pentahydrate ($CuSO_4.5H_2O$) for cyanide destruction;
- Flocculant for thickening;
- Fluxes for smelting;
- Anti-scalant to minimize scaling in the process water distribution, leach water distribution, reclaim water distribution, fresh water distribution, gland water distribution, and elution circuit.

Water Services

The process plant will use process water, leach water, reclaim water, fresh water, treated water, gland water and potable water. Make-up water requirements for process water and leach water will be pumped from the reclaim water tank, which receives water recovered from the tailings pond.

An event pond will hold any overflow from the process plant bunds (non-cyanide containing slurry), and storm water collected from the southern side of the process plant. The material in the event pond will be pumped to the concentrate thickener area sump pump if it contains any slurry, or the stormwater will be pumped to the reclaim water tank when necessary. A storm water pond, which will hold any stormwater collected from the northern side of the process plant, will be pumped to the reclaim water tank when necessary. Any spillage of slurry or water containing cyanide will be contained within the bunds in the process plant.

Air Services

High pressure air at 850 kPag will be provided by two high pressure air compressors.

Low pressure air will be supplied to the flotation and cyanide destruction areas by one of two, low pressure air blowers to be installed in each area.

Oxygen, for use within the concentrate leach circuit, will be supplied by one of two oxygen pressure swing adsorption plants.

17.5.2 Power

The power demand for the processing plant, along with the rest of the site and camp, will be provided by a self-generated natural gas power plant. Power is estimated at:

- Installed power: 38,210 kW;
- Average continuous draw: 20,388 kW;
- Annual process plant power consumption: 178,602,267 kW.

17.5.3 Water Consumption

A water balance for the process plant has been completed. Water from the thickeners overflow streams is recycled within the process plant to reduce the external water requirement. An approximate minimum average flowrate of 657 m^3/h of decant return water is needed to be recycled from the TMF to the process plant, while an additional 496 m^3/h is needed to be recycled from the TMF for use at the terminal station. Another approximately 29 m^3/h of fresh water is required to make-up the water consumption for the process plant.

18.0 PROJECT INFRASTRUCTURE

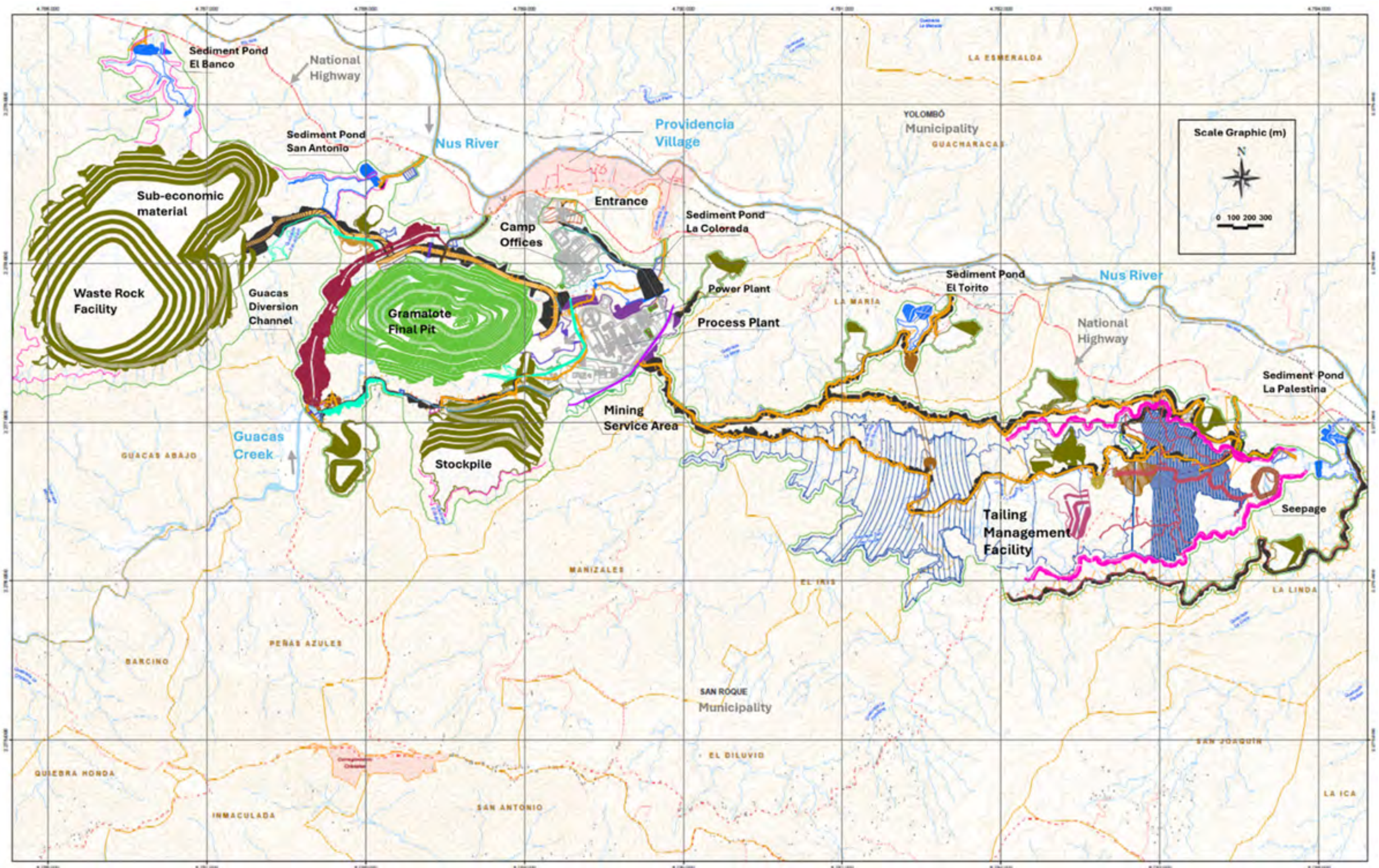
18.1 Introduction

Surface infrastructure to support operations as envisaged in the 2025 Feasibility Study includes:

- One open pit; to be mined in two phases;
- Waste rock storage facility, stockpiles, and water diversions, including Guacas Creek diversion;
- Process plant: grinding, flotation and leaching facilities, management and engineering offices, change house, workshop, warehouse, and electrical grid switchgear substation;
- Mine facilities: heavy equipment workshop, warehouse and laydown areas, fuel storage, fuel distribution bays, wash bay, tire bay, welding shop, assay laboratory, change house, cafeteria, explosive magazine, explosive transference plant, and concrete batch plant;
- Internal mine roads, including haul roads;
- A community bypass road to the Cristales village;
- Mine entrance and camp accommodation;
- Sediment control ponds;
- Quarry borrow pits;
- Water intake on Guacas Creek and associated water treatment plant;
- Tailings management facility;
- Utilities including potable and wastewater systems, fire protection, communications, and natural gas.

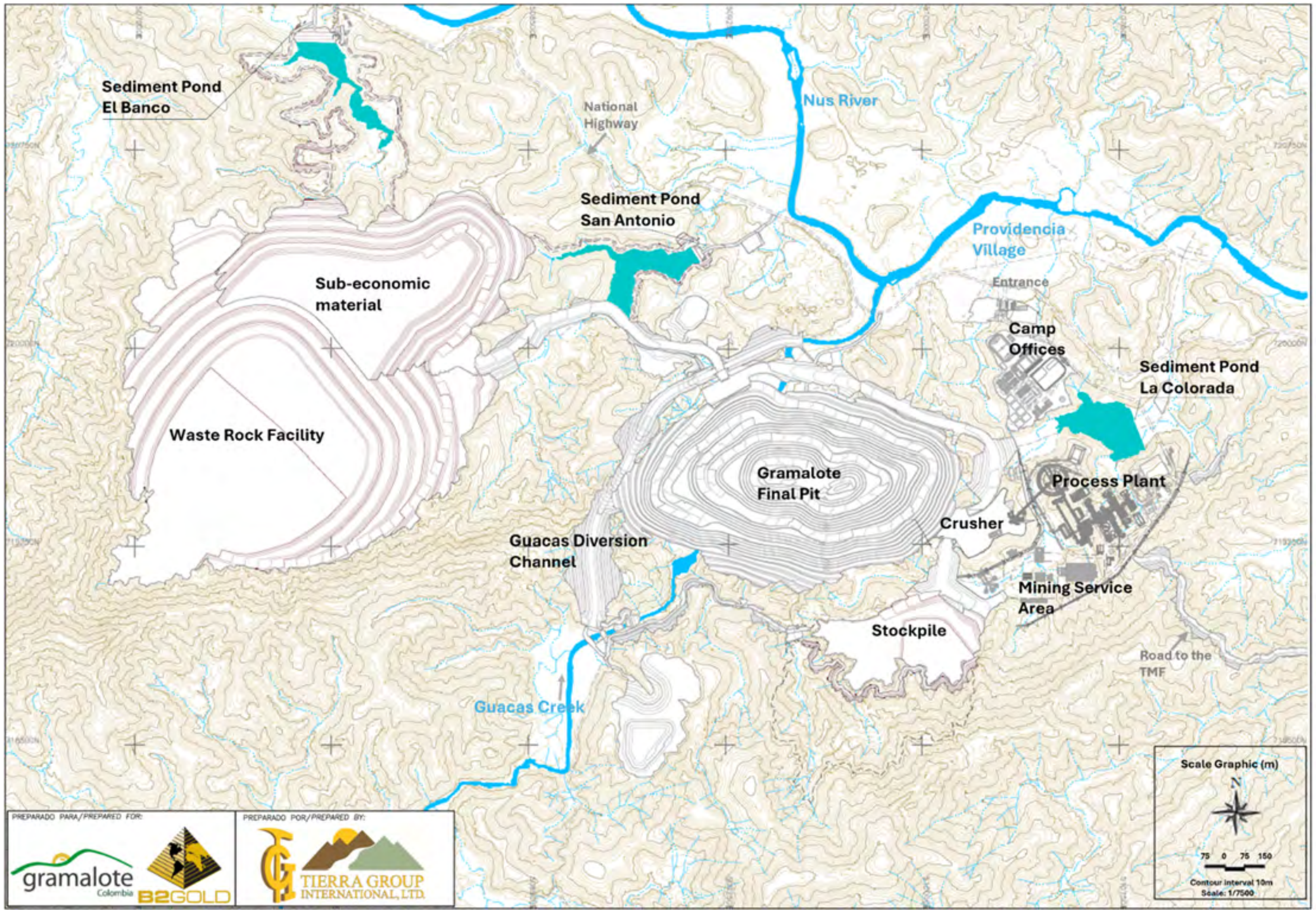
A layout plan showing the proposed infrastructure is provided as Figure 18-1. Figure 18-2 is an inset plan showing the general area of the planned open pit. Figure 18-3 is an inset figure illustrating the TMF location.

Figure 18-1: Proposed Infrastructure Layout Plan



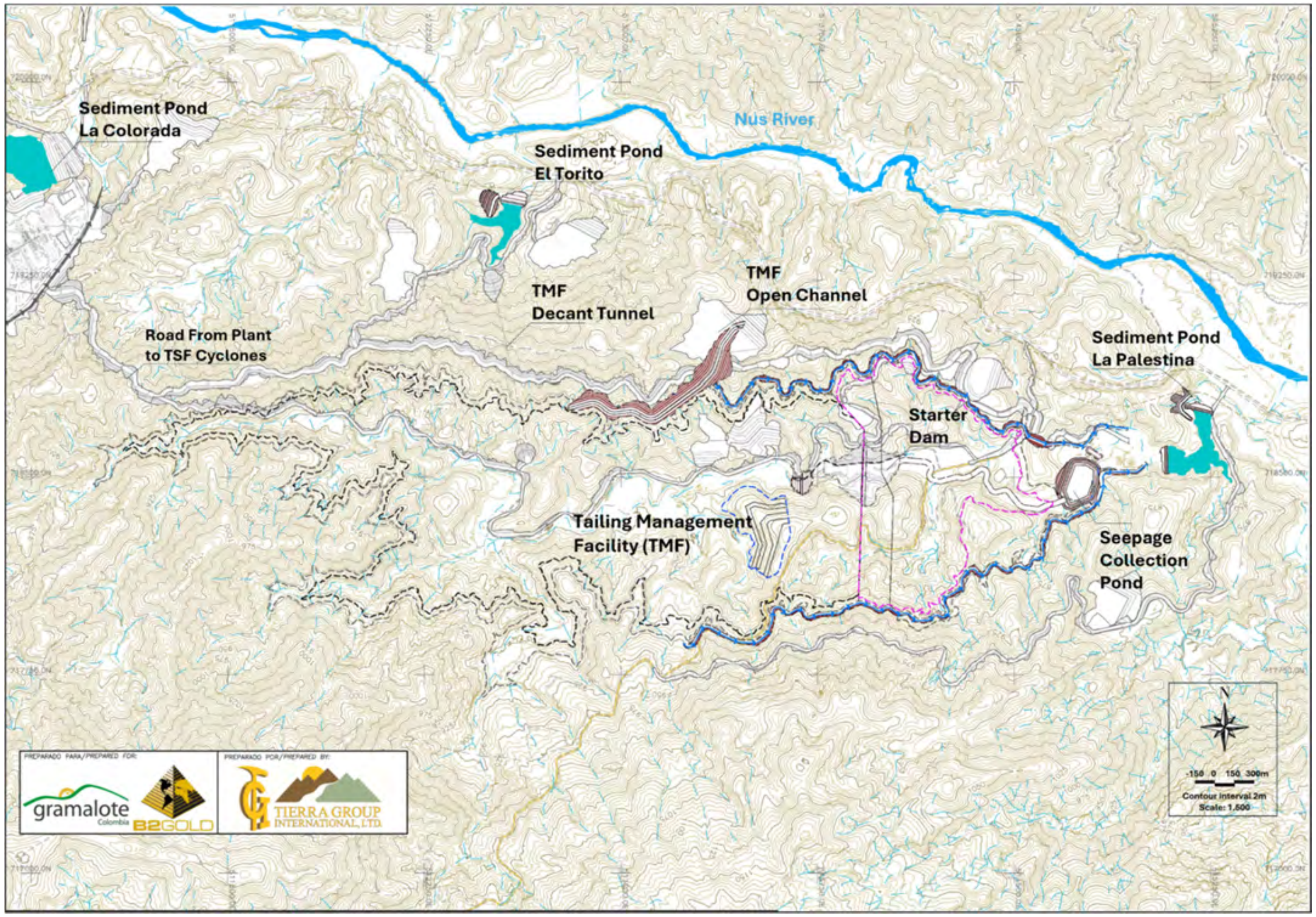
Note: Figure prepared by Integral Colombia, 2025, based on designs by Tierra Group International, B2Gold, and Integral Colombia.

Figure 18-2: Site Layout Plan Showing Proposed Pit, Stockpile, WRSF and Plant Locations



Note: Figure prepared by B2Gold, 2025.

Figure 18-3: Proposed TMF Location Plan



Note: Figure prepared by B2Gold, 2025. Designs by Tierra Group International, Integral Colombia, B2Gold.

18.2 Road and Logistics

The 100 km road-trip from Medellin to the Gramalote Project site was reduced to about 1.5–2.0 hours with the opening of the Tunel De La Quiebra–Vias Del Nus, completed in mid-2022.

Colombia has ports on both the Atlantic Ocean (Cartagena and Barranquilla) and Pacific Ocean (Buenaventura) that are equipped to receive large equipment. No significant issues affecting movement of normal or general cargo from the Atlantic or Pacific Ocean ports using road routes have been identified. Cargo from 3–5 m in width can be transported by road from the Pacific and Atlantic ports under a permit. Cargo over 5 m in width can be transported using a combination of barging on the Magdalena River with road transport from Puerto Berrio to the Project site, a distance of 73 km using National Route 62. Puerto Berrio is connected directly with Barranquilla and Cartagena by river. Barge service over the approximate distance of 700 km takes 7–13 days, depending on which shipping port is used.

The planned operations will be interconnected with 25 km of construction and haul roads. The roads were designed to minimize any interaction with public traffic on National Route 62.

A 4 km long community bypass is needed to route around the proposed TMF. It will be constructed early in the mine life. The bypass will allow the community of Cristales to maintain a road connection to National Route 62.

18.3 Tailings Management Facility

The TMF was designed to store 76.7 Mt of tailings over the 13-year process period. The Palestina Valley, where the TMF will be located, could accommodate a larger facility if required for any potential future mine expansion or mine life extension. A larger-capacity TMF, of 220 Mt capacity, was approved in 2016, and that approval is part of the current EIA.

The proposed TMF will be a cross-valley impoundment with a single embankment across the Palestina Valley, approximately 1 km upstream of the confluence of that valley with the Nus River. The facility will not be lined. Geotechnical investigations indicate that the Palestina Valley consists of a saprolite soil profile that averages about 30 m in thickness. Seepage analysis and hydrogeological models indicate minimal to negligible impact on groundwater, due to that soil type and the planned facility seepage collection system.

Flotation tailings and leach circuit cyanide destruction tailing streams will be separately managed:

- Flotation tailings will be pumped to a header tank and fed to hydrocyclones that will deposit the tailings as using a centreline raising method. Cyclone deposition will commence on a 30 m tall starter embankment to be constructed out of locally borrowed residual material;
- Leach circuit tailings will be pumped separately and deposited sub-aqueously within a supernatant pond to prevent oxidation and acid-generating potential.

The tailings material contains a high percentage of a sand-forming coarse fraction, which is planned to be placed directly downstream of the starter dam, creating a large sand dam.

The facility was designed to receive the 24-hour probable maximum precipitation without overtopping. A sidehill decant structure will be located in the centre of the north slope. The supernatant water flowing through the side-hill and tunnel will enter the El Torito sediment pond, where reclaim water can be abstracted and returned to the process plant. Water not required for processing will be discharged to the environment (see Section 18.5).

A spillway will be excavated through the northern Palestina valley-side slope as part of mine closure.

18.4 Waste Rock Storage Facilities and Stockpiles

Design considerations for the WRSF and mineralized stockpiles are discussed in Section 16.8.

A series of residual waste stockpiles, which will be located throughout the Project area during construction activities, will be used for overburden and topsoil storage.

18.5 Water Management

18.5.1 Overview

A water management system was developed to maximize the reuse of process water and minimize the use of make-up water from fresh water sources. The Project will have a positive water balance that will require discharge, primarily due to the large volumes of surface run-off that will be collected in the TMF. The TMF will be the primary reservoir and will have sufficient capacity to manage the probable maximum flood at all times.

Water control reservoirs, seepage and sedimentation ponds will be located in different catchment areas to manage sediment discharges, storm events, and contact water, including from the open pit, TMF, and WRSF.

The water balance model shows sediment ponds are adequately designed to manage all surface-water run-off, TMF and WRF seepage, and precipitation events across all climate scenarios.

Water quality models predicted concentrations of modeled constituents at all discharge ponds will meet all Colombian regulatory standards under all climate scenarios.

Contingency measures such as recycling and treatment are included in the mine plan to manage any potential exceedances. The WRSF is designed to be a phased development that will reduce initial environmental impacts and reduce downstream sediment loading.

The TMF water quality is predicted to meet discharge criteria requirements at all times. Therefore, under normal and wet conditions, TMF water will be continuously discharged via the decant structure to the El Torito sediment pond and from there to the Nus River.

Reclaim water from the TMF will be pumped directly to the main process plant reclaim water tank. Seepage from the TMF impoundment will be collected in a seepage collection pond and will be pumped to a reclaim water tank at the TMF.

18.5.2 Sediment/Seepage Ponds

Sediment/seepage ponds will be located downstream of each natural drainage basin that will host mine-related facilities. These ponds will collect the run-off, entrained sediment, and the seepage captured by drains. Ponds are envisaged to release water to the environment but could also be used for emergency water supply to the process plant. The sediment ponds were designed to avoid the use of flocculants to meet Colombia discharge limits (50 mg/L). This was achieved by raising the elevation of penstocks within the ponds to maintain a sufficient pond size to allow for sediment fallout.

There will be six sediment/seepage collection ponds, located downstream of all major drainages:

- El Banco sediment pond: WRSF west pond;
- San Antonio sediment pond: WRSF north-east pond;
- La Colorada sediment pond: plant/stockpile north pond;
- El Torito sediment pond: TMF discharge point;
- La Palestina sediment pond: TMF pond, will primarily be used during the TMF construction and will be replace by the seepage collection pond;
- TMF seepage collection box: TMF east pond.

In addition to the sediment ponds, each earthwork infrastructure disturbance will intercept sediment before it reaches the main collector ponds. Each sediment pond will discharge into the Nus River directly downstream of the embankment. An emergency spillway is designed at each sediment pond.

18.5.3 Water Storage and Water Discharges

The primary water storage locations, in addition to the sediment ponds, will include:

- Open pit;
- Intake facility at Guacas Creek (fresh water);
- TMF.

Water discharges, after ensuring compliance with the Colombian regulation, will include:

- Evaporation;
- TMF discharge through the decant tunnel to the El Torito sediment pond and then into the Nus River;
- Sediment ponds discharge into the Nus River.

Contact water will primarily be generated from open pit, WRSFs, stockpiles, and the TMF seepage pond. The anticipated quality of these contact waters is expected to be suitable for reclaim by the process plant.

The annual water balance reflects the collection and storage of all water within the TMF and open pit. The Project aim is to maximize use of reclaim water, and recycle as much water as practicable.

Fresh water will be drawn from the intake of Guacas Creek at a maximum rate of 35 m^3/hr. The remaining volume required for the process plant will be recirculated from the TMF seepage–cyclone station area at a rate of about 500 m^3/hr.

18.5.4 Guacas Creek Diversion

Guacas Creek must be diverted during the operational stages of the project as the final pit phase will impact the existing creek channel. A trade-off study evaluated open channel versus tunneling options, or a combination of both. An open channel design was adopted as that would reduce the risk of collapse/flooding.

The creek has a large catchment area of 62.5 km^2, and drains a high rainfall area. The diversion system is designed for a probable maximum precipitation event equivalent to 1,000,000-year return period storm, with a design flow of 950 m^3/s, with all flow to be contained within the bottom bench of the channel. Upstream, a 17 m high earth embankment diversion dam will be constructed to divert the creek into the open channel and protect the open pit from flooding.

A fresh water intake facility will be located at the entrance of the channel, and the water will be pumped through a 3.5 km pipeline to the water tank that will be located at the process plant.

18.6 Operations Area Infrastructure

Facilities will be constructed in three areas, connected by the 1.5 km long main access road:

- Administrative offices, to be constructed next to the main project entrance;
- Process plant offices, to be constructed at the process plant area;
- Mining service area. This will include a heavy equipment workshop consisting of several bays with overhead bridge cranes; a diesel storage and fuel area for both mining and light vehicles; warehouse; welding shop, tire storage, tire handling, and wash bay; and mine offices.

Light vehicle and mining equipment traffic was separated in the design process throughout the site to reduce interaction between the two types of equipment.

The explosive magazine and transfer will be located on the southeast edge of the Gramalote Ridge open pit, next to the stockpile area.

18.7 Camps and Accommodation

A new camp will be required to support both the construction and operational phases of the Project. A suitable site was identified adjacent to the main administrative offices for this purpose. A modular design was selected, and will include accommodation units, a cafeteria, kitchen, administrative area, laundry, recreational amenities, and a medical clinic.

The existing camp, 30 minutes from the Project area, will provide additional accommodations flexibility with a 70-bed capacity.

A significant portion of the construction workforce is expected to commute from nearby communities, thereby maximizing local opportunities for lodging and transportation services.

During operations, the proposed new camp will have an approximate 200-person capacity.

18.8 Power and Electrical

Power is planned to come via pipeline from gas-based self-generation, leveraging off the Project proximity of 2 km to the main gas pipeline that supplies the Medellín region. The 2025 Feasibility Study assumes that a take-or-pay agreement will be negotiated with the pipeline owner for the electricity supply. The agreement envisages that the pipeline owner will provide the pipeline to site, and construct the on-site power station.

The power plant will consist of four gas-powered engines that will have a total installed capacity of about 44 MW, sufficient to meet peak Project demands of about 30 MW. Three engines will be in operation, with one on stand-by. The plant will include load-shedding abilities.

18.9 Fuel

Fuel will be trucked in either from Medellin or Puerto Berrio for operational purposes.

During the construction phase, before the fuel station is built, the fuel will be sourced through the local retail market.

18.10 Water Supply

Guacas Creek will be the potable water source, as discussed in Section 4.6.

19.0 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

No market studies are currently relevant as the Gramalote Project will produce a readily-saleable commodity in the form of doré. Shipping and refining costs are estimated to be US$3.00/oz for gold and US$1.34/oz for silver.

19.2 Commodity Price Projections

Commodity prices used in Mineral Resource and Mineral Reserve estimates are set by B2Gold corporately. The gold price provided for Mineral Reserve estimation is US$1,750/oz, and US$2,500/oz was used for Mineral Resource estimation.

The financial model assumes a long-term gold price of US$2,500/oz.

19.3 Contracts

Quotes for major contracts were received to support the 2025 Feasibility Study.

Major contracts will include blasting explosives and accessories through Indumil (the only option for explosives under Colombian law), fuel provision, power provision, tire services, contacts related to infrastructure construction, and other mining and processing consumable contracts as needed.

Contracts will be negotiated and renewed as necessary with terms expected to align with industry standards and comparable to similar contracts previously executed by B2Gold.

19.4 Comments on Market Studies and Contracts

The QP notes the following.

The doré produced by the mine is readily marketable. Metal prices are set corporately for Mineral Resource and Mineral Reserve estimation, and the gold price used for Mineral Resources and Mineral Reserves in this Report was US$2,500/oz and US$1,750/oz, respectively.

The QP has reviewed commodity pricing assumptions, marketing assumptions and the current major contract areas, and considers the information acceptable for use in estimating Mineral Reserves and in the economic analysis that supports the Mineral Reserves.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

In Colombia, mining projects require an Environmental License authorised by ANLA to develop a project, as prescribed under Decree 1076 of 2015. The Environmental License authorizes the use of environmental resources for a mining project, including relevant necessary environmental permits for project development.

The Terms of Reference for the EIA for the Gramalote Project were issued by ANLA in 2012. The EIA was submitted for approval in February 2015. The Environmental License was granted under Resolution 1514 of 2015 and Resolution 0309 of 2016. In April 2018, B2Gold submitted a MEIA, which was approved under Resolution 00782 of 2019.

ANLA has approved further modifications to the Environmental License formalized in several resolutions. Resolution 2427, dated 2019, established phases for the resettlement of Project-affected persons. Resolution 1447, dated 2021, authorized the simultaneous execution of resettlement and construction activities. Resolutions 2292 and 2976, in 2024, updated the resettlement schedule and certain environmental management plans. B2Gold also holds various regional environmental permits and authorizations necessary for the advancement of complementary project activities.

The 2015 EIA and 2019 MEIA assessed the potential environmental and socio-economic impacts of Project development within the Project’s area of influence. These assessments included the proposed mitigation measures necessary to minimize potential impacts to acceptable levels. Stand-alone management programs were developed to address residual impacts, and to ensure that the Project complies with its regulatory and permitting requirements and B2Gold's Environmental and Social Standards.

To comply with Colombian environmental legislation, and based on the 2025 Feasibility Study results, B2Gold will submit an updated MEIA for approval to reflect Project changes. This update will be carried out in accordance with the Terms of Reference for Mining Projects (ANLA, 2016) and the General Methodology for the Preparation and Submission of Environmental Studies (Ministry of Environment and Sustainable Development (Colombia), 2018b).

For the updated MEIA submission, B2Gold must also assess the Project area of influence in accordance with The Guidelines for the Definition, Identification, and Delimitation of the Area of Influence (ANLA, 2018). This area is defined as the geographic zone where the Project's significant environmental impacts, across its various phases, are expected to occur, affecting abiotic, biotic, and socioeconomic components. Accordingly, the municipalities of San Roque, Maceo, Yolombó, and Cisneros remain identified as the most likely locations anticipated to have environmental and social impacts from the Project.

20.2 Baseline Studies

Collection of comprehensive baseline data commenced in 2011 with the support of environmental and social specialists, in alignment with both national and international standards to support Project design, impact assessment, and the acquisition of environmental permits. Updated baseline data are being collected to support the planned MEIA submission. Studies to date have included:

- Geology;
- Geomorphology;
- Geotechnical;
- Seismicity;
- Soils;
- Climate;
- Hydrology;
- Water quality;
- Sediment loads;
- Water users and uses;
- Landscape;
- Air quality;
- Noise;
- Flora;
- Terrestrial fauna;
- Aquatic ecosystems;
- Health characterization;
- Population studies;
- Archaeology.

20.3 Environmental Considerations/Monitoring Programs

The Environmental License set out specific management and monitoring programs that must be met for the biotic and abiotic environments surrounding the proposed mine site. These measures are included in a set of management programs that describe the activities aimed to prevent, mitigate, or correct significant environmental and social impacts that could be generated by Project development. These management programs will be in place for the LOM, beginning during planning and continuing until the construction, operation, and closure phases are completed:

The following programs will be required to prevent, mitigate, or correct significant Project impacts:

- Soil management program;
- Waste rock and debris management program;
- Water resources management and erosion control program;
- Air quality management program;
- Waste management program;
- Chemical substances and fuel management program;
- Blasting and explosives management program;
- Signage management program;
- Landscape management program;
- Cyanide management program;
- Fauna management and rescue program and protocols;
- Vegetation cover removal program;
- Fish relocation and rescue program;
- Restoration, rehabilitation, and land recovery program;
- Conservation program for endangered, endemic, and protected flora and fauna;
- Ecosystem conservation and ecosystem services management program.

In addition to the Management Programs, B2Gold must implement the following compensation programs:

- Biodiversity loss;
- Woody vegetation and threatened species;

- Changes in land use.

These compensation programs may include various conservation actions such as creating or extending public or private protected areas, generating voluntary conservation agreements, and/or undertaking ecological restoration actions. Approximately 3,500 ha will be subject to the proposed compensation programs.

B2Gold will also be required to have water management programs in place to monitor, conserve and preserve the watershed from which the Project abstracts water (referred to as the 1% Investment Program). The Project plans investment in the following three aspects:

- Preservation of vegetation cover in the Nus River basin, municipality of San Roque;
- Financing of a treatment plant for domestic residual waters in the village of Providencia;
- Installation of treatment systems for domestic residual water in rural areas of the municipality of San Roque.

20.4 Closure Plan

ANLA has approved a conceptual Closure Plan (approved under Resolution 1514 of 2015, Resolution 0309 of 2016, and Resolution 00782 of 2019) for the Project under the current Environmental License that includes a strategy to effectively and progressively rehabilitate areas that have been affected by Project activities. The Closure Plan includes measures such as the dismantling/demolition and removal of structures, stabilization of landforms, and rehabilitation of areas disturbed by Project activities. The conceptual Closure Plan will be modified and updated periodically throughout the Project life to reflect significant changes in the design, construction, operation, and closure phases. For updates of the mine Closure Plan, B2Gold will use The Guide For The Preparation Of The Closure And Abandonment Plan For Mining Projects Presented (ANLA, 2022).

B2Gold has estimated the cost of the Project's environmental reclamation and closure liabilities to be approximately US$51.2 M. The Mining Code (Law 685 of 2001) includes an obligation to provide financial guarantees to ensure the Closure Plan can be completed, cover the costs associated with environmental remediation, and comply with social obligations.

20.5 Permitting

As noted in Section 20.1, the Project has been granted an Environmental Licence through Resolutions 1514 (2015), 309 (2016), and 00782 (2019). This license sets out the conditions and requirements that must be met to support Project development, operation, and closure. B2Gold will submit an updated MEIA for approval to reflect

Project changes. This update will be carried out in accordance with the Terms of Reference for Mining Projects (ANLA, 2016) and the General Methodology for the Preparation and Submission of Environmental Studies (Ministry of Environment and Sustainable Development (Colombia), 2018b).

Key permits granted for the Project at this time are summarized in Table 20-1. Additional permits and authorizations will be required from other institutions and authorities, including:

- Authorization for acquisition and use of explosives;
- Certificates for controlled chemical substances and products;
- Sanitary authorization for potable water supply;
- Construction licenses;
- Realignment or intervention of national, regional, and rural access roads;
- Technical Regulations for Electrical Installations (RETIE);
- Relocation of existing powerlines.

B2Gold has a PTO in place, which was approved in 2018. The company will need to update the PTO to capture the changes to the mine plan that are proposed in this Report.

20.6 Considerations of Social and Community Impacts

As discussed in Section 20.2, the development of the 2015 EIA and 2019 MEIA included the collection of comprehensive baseline data for the Project area. The socio-economic baseline studies were used to assess potential impacts on surrounding communities due to Project development.

To assess the potential Project socio-economic impacts, the Project area was divided into direct and indirect areas of influence (Figure 20-1):

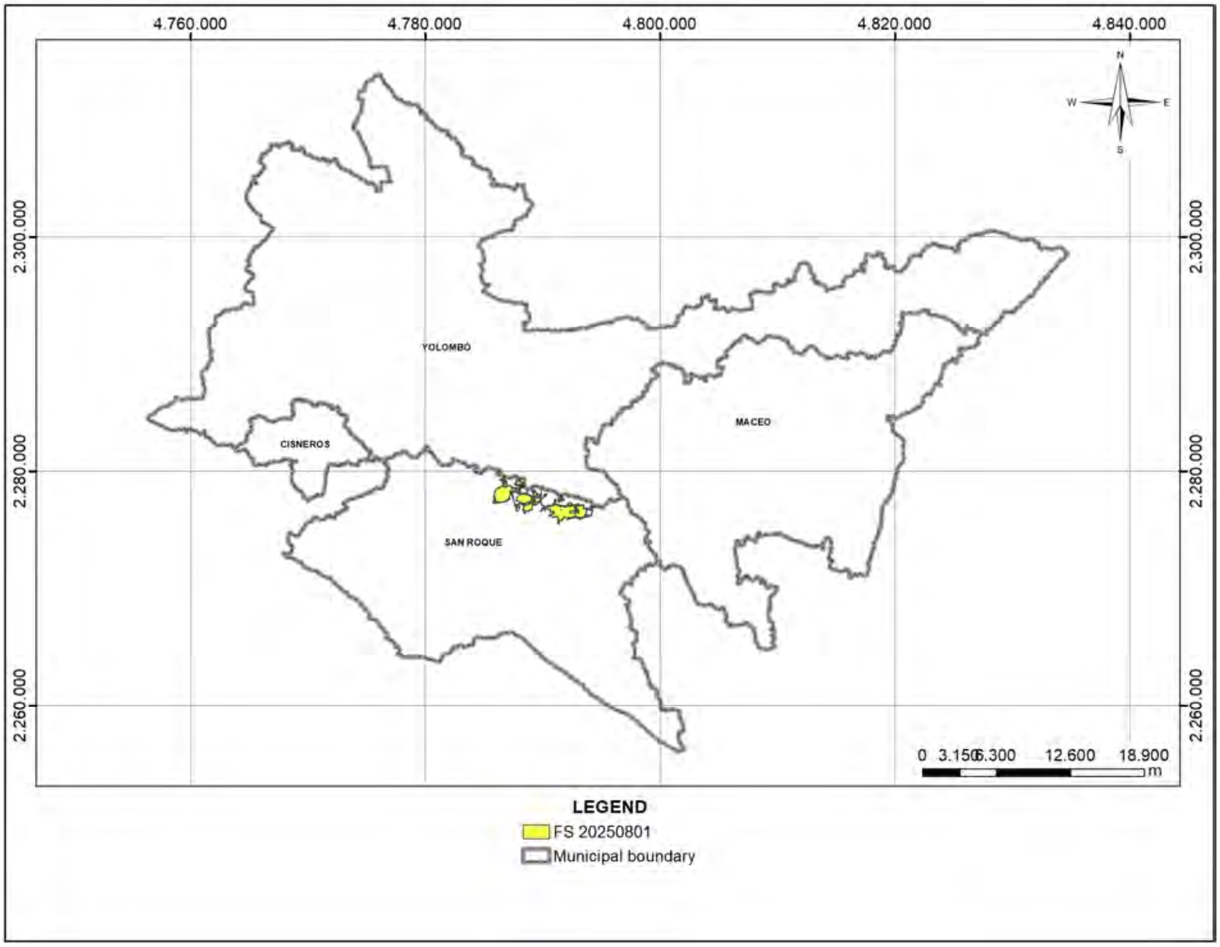
- Direct area: 18 localities (populated centres in villages and rural districts) in the local direct area of influence, within the municipalities of San Roque, Yolombo, and Maceo;
- Indirect area: Municipalities of Maceo, Cisneros, San Roque, and Yolombo.

During the preparation of the 2015 EIA and 2019 MEIA, community engagement processes were conducted, resulting in a total of 104 meetings with relevant stakeholders. For the proposed update to the MEIA, stakeholder engagement and communication activities will be conducted to identify and evaluate potential positive and negative impacts within the Project areas of influence.

Table 20-1: Key Granted Permits

Aspect	Permit/ Licence No	Description
Socio-environment	Resolution 1514 (25-Nov-2015) Resolution 0309 (29-Mar-2016)	Environmental license
Socio-environment	Resolution 0782 (08-May-2019)	Modification to the environmental license
Socio-environment	Resolution 02427 (11-Dec-2019) Resolution 01447 (18-Aug-2021) Resolution 2292 (21-Oct-2024)	Modification to the environmental license: • 2019: Resettlement phase extension • 2021: Concurrent resettlement • 2024: Resettlement schedule
Biodiversity	Resolution 943 (09-May-2023) Resolution 2649 (14-Nov-2023)	Management program for flora species under national and regional prohibition
Biodiversity	Auto 3607 (29-May-2019) Resolution 00725 (07-Apr-2022) Resolution 01380 (24-Jun-2022) Resolution 0356 (07-Mar-2024)	Compensation plan, biodiversity
Environment - water	Resolution 1514 (25-Nov-2015) Auto 3607 (29-May-2019) Resolution 0535 (14-Mar-2023)	1% investment plan – For water use
Archeology	Resolution 439 (23-Jun-2020) Resolution 1143 (31-Dec-2020)	Archaeological management

Figure 20-1: Project Area of Influence Studies



Note: Figure prepared by B2Gold, 2025.

The updated MEIA will outline specific management measures and actions designed to prevent, mitigate, and address potential significant impacts on communities located in the Project areas of influence;

The following programs were developed in close collaboration with the affected communities and in alignment with both Colombian regulations and international practices:

- Community assistance, information, and participation program;
- Procurement of goods and services program;
- Community training and education program;

- Environmental training and education program for workers;
- Resettlement program;
- Program to support public and private institutional management;
- Program for land access;
- Migratory pressure management program;
- Third-party involvement program;
- Promotion of cultural heritage program;
- Health promotion and prevention program;
- Preventive archaeology program.

Monitoring programs were developed for each social management program as described above. The monitoring programs seek to verify the efficiency and effectiveness of the environmental and social management measures proposed, and apply mitigative measures in a timely fashion, where necessary.

Two aspects that likely face potential significant negative impacts if not managed properly are the artisanal miners working in the Project area and the resettlement of individuals due to Project development. These aspects are discussed in the following sub-sections.

20.6.1 Resettlement

Project development will have a significant impact on nearby communities, primarily due to the need for land acquisition and the physical and economic resettlement of households located within the infrastructure footprint. B2Gold has been advancing resettlement planning in compliance with Colombian legislation, the obligations of the Environmental License, and in alignment with international standards.

Socioeconomic baseline data was collected during prior EIA studies and subsequently updated in 2018. Additional assessments conducted in 2020 confirmed the final number of households to be resettled and validated the impacts and mitigation measures with Project-affected persons. In total, 327 households have been identified as significantly impacted and included in the Resettlement Action Plan. Of these, 225 households will be physically relocated, while 102 households will experience economic displacement only.

The Environmental License requires that resettlement in the area of direct influence must be completed prior to the implementation of the approved mining plan (Mining Plan). Through Resolution 2427 (2019) and Resolution 1447 (2021), the National Environmental Licensing Authority (ANLA) authorized an extension of the resettlement phase and approved the commencement of project construction concurrently with the resettlement process, subject to compliance with technical requirements outlined in those resolutions. The resettlement schedule was further updated in 2024 via Resolution 2292, with completion currently planned for late 2026. It is possible that the schedule will require further adjustment to align with the current mine plan and construction timeline. If so, B2Gold will submit an updated schedule to ANLA for approval.

The resettlement process consists of the following phases:

- Establishment of baseline population and identification of Project-affected persons;
- Development and formalization of the Resettlement Action Plan;
- Signing of collective agreements with Project-affected persons;
- Signing of individual agreements with Project-affected persons;
- Monitoring of Resettlement Action Plan implementation;
- Completion audit of the Resettlement Action Plan.

B2Gold has acquired sufficient land to meet resettlement requirements. Planning and development activities are underway, with progress reported quarterly to ANLA. In 2022, the Resettlement Action Plan was formalized in agreement with the affected communities, and the phases of collective and individual negotiations were completed during 2023 and 2024. As of the Report Effective Date, 98% of affected households have signed individual agreements. Only 2% - equivalent to nine households - have not reached an agreement. Engagement and negotiations with these remaining households are ongoing and will continue until a resolution is achieved.

B2Gold has completed the design of housing, public facilities, and supporting infrastructure at the designated relocation site. Construction is progressing in accordance with approved designs and technical specifications. In addition, B2Gold is also implementing livelihood restoration programs, community integration initiatives, and measures to ensure the provision of necessary public services. Implementation efforts will continue to be closely monitored and reported to ANLA until all commitments are fulfilled and the RAP completion audit is successfully concluded.

20.6.2 Artisanal and Small-Scale Mining

ASM is a long-standing economic activity in the broader project area, but it also poses environmental and social risks, including land and water degradation, child labour, informal employment, and occupational health hazards.

B2Gold has monitored ASM activity since baseline studies began in 2011. The number of artisanal miners in the area has varied over time; in 2025, the average number of active miners within Mining Permit 14292 is approximately 500, with around 50% operating inside the Project footprint.

To reduce environmental impacts, support local communities, and foster social stability, B2Gold has advanced ASM formalization initiatives in coordination with local and national authorities. To date, 73 mining production units have been formalized under 22 mining associations, benefiting more than 180 miners and generating over 560 formal jobs with social security benefits.

These initiatives are framed within a strategy that includes:

- Assessing mining production units and supporting their formalization as small companies;
- Assisting with the formation of associative groups for formalized miners;
- Supporting the formalization process through mechanisms established in national legislation;
- Assisting formal mining production units in preparing and obtaining EIAs and complementary work plans as required by law;
- Providing technical training to improve operational, environmental, and safety performance.

B2Gold will continue advancing ASM formalization efforts, including assessing new areas for potential inclusion, in alignment with national policy, community priorities, and its commitment to responsible mining practices.

21.0 CAPITAL AND OPERATING COSTS

21.1 Introduction

Capital and operating cost estimates are based on forecast costs as of the Report Effective Date, April 1, 2025, to mine and process Mineral Reserves from open pit operations.

The LOM plan assumes owner-operated mine operations with contractor support in select areas. The construction period is assumed to be two years, scheduled as 2027 and 2028 in this Report, pending permitting.

Capital and operating costs are estimated in either Colombian Pesos or United States dollars depending on the category and are converted at exchange rate of 4,200 COP:US$ where applicable.

21.2 Capital Cost Estimates

21.2.1 Basis of Estimate

Capital costs consist largely of processing facilities, site infrastructure, mining equipment and rebuilds, and resettlement and land purchases. Capital costs are split into:

- Sustaining capital: Costs that support the existing LOM plan;
- Non-sustaining capital: Costs are for long-term structures or external project that do not necessarily depend on the mine plan. Non-sustaining capital includes camp, roads, initial mining equipment fleet, mine services area and workshops, process plant, tailings storage facility, resettlement, and other major infrastructure development required for mining and processing operations.

21.2.2 Labour Assumptions

Owner labour to support projects included in capital costs are included in operating costs after gold production begins. Where the labour is to be provided by some party other than the owner, labour costs are included in capital costs.

21.2.3 Construction Capital Costs

The construction phase occurs over two years, assumed to be 2027 and 2028 pending project permitting. This includes mining and related infrastructure, waste pre-strip and stockpile construction, processing related capital, site general infrastructure, resettlement and land purchases, and contingency. Post-construction capital includes the Guacas diversion and some remaining road construction not required before the start of commercial production.

A construction capital cost of US$740.1 M is estimated prior to commercial production. A post-construction capital cost of US$67.7 M is estimated after commercial production has begun.

21.2.4 Mine Capital Costs

Mine capital costs are estimated based on Owner operation of surface mining fleets. Fleet purchases, rebuilds, and equipment replacement costs are estimated based on forecast data from equipment manufacturers. Major mine equipment fleet replacements are carried out based on equipment utilization.

Total mining fleet and infrastructure capital is estimated to be US$140.5 M over the LOM.

After construction, US$149.5 M of capitalized waste is included in mining capital costs.

The total mining capital cost, after construction, is estimated to be US$289.9 M over the LOM.

21.2.5 Process Capital Costs

Process capital costs include sustaining capital for the process plant, tailings storage facility, and the sediment and process water ponds.

Process capital costs, after construction, are estimated at US$46.5 M over the LOM.

21.2.6 Site General Capital Costs

Site general capital cost estimates average US$0.5 M per year of operations.

Total site general capital costs are estimated to be US$6.0 M over the LOM.

21.2.7 Closure Costs

The total reclamation and closure capital cost is estimated at US$51.2 M, with costs occurring concurrently with operations where feasible, and most costs occurring at the end of the mining and processing operations.

21.2.8 Capital Cost Summary

Capital costs are summarized by category in Table 21-1.

Table 21-1: LOM Capital Cost Estimate

Area	Sub-Area	Units	Value
Non-sustaining capital	Mining and related infrastructure	$ M	87
	Waste pre-strip and ore stockpile	$ M	42
	Processing	$ M	313
	Site general infrastructure	$ M	186
	Resettlement and land purchase	$ M	32
	Contingency	$ M	81
	Subtotal construction (all site development prior to commercial production)	*$ M*	*740*
	Post-production construction (all site development after commercial production starts)	$ M	68
	Subtotal non-sustaining capital	*$ M*	*808*
Sustaining capital	Mining - capitalized waste	$ M	149
	Mining - excluding capitalized waste	$ M	140
	Processing	$ M	46
	Site general	$ M	6
	Contingency	$ M	34
	Subtotal sustaining capital	*$ M*	*376*
Closure capital	Closure costs	$ M	51
Total All Capital Costs		***$ M***	***1,236***

Note: Totals may not sum due to rounding.

21.3 Operating Cost Estimates

21.3.1 Basis of Estimate

Department costs are estimated independently. Some departments are treated as distributable costs, such as power generation and heavy equipment maintenance, and are allocated to other departments based on usage.

21.3.2 Mine Operating Costs

Mine operating costs are built up from first principles based on the mine plan, and are supported by life cycle costs or quotes from equipment and product suppliers, as well as benchmarked against comparable mining operations.

Mine operating costs are estimated at US$2.71/t rock mined including capitalized waste.

21.3.3 Process Operating Costs

Processing costs include all activities related to the process. Variable costs are costs which change with plant production, consisting largely of consumables/supplies and power costs, as well as maintenance and other allocations. Period costs are time-related costs which are incurred regardless of production, including labour, contractors, and a portion of maintenance and other distributed costs. Total process costs vary year over year depending on the operational plan.

Stockpile and ore rehandle costs are included with the processing costs.

The total process operating cost is estimated to be US$8.16/t milled over the life of processing operations.

21.3.4 Site General Operating Costs

Site general operating costs are modelled largely as period costs. These include period costs for administrative labour and supplies costs, camp costs, information technology services, personnel transport, health, and safety, environmental, security, supply chain, warehousing, site services, and site finance costs. Total site general operating costs vary year over year depending on the operational plan.

The total site operating cost is estimated at US$3.64/t processed over the life of operations.

21.3.5 Operating Cost Summary

The estimated LOM plan operating costs are presented in Table 21-2 and Table 21-3. Operating costs total US$1,583 M over the LOM. Mining costs will average US$11.27/t ore processed, process costs will average US$8.16/t ore processed and site general costs will average US$3.64/t ore processed, for an overall ore processed cost of US$23.08/t.

21.4 Comments on Capital and Operating Costs

The QPs note the following:

- The capital and operating costs for the proposed Gramalote Ridge Mine are based on forecast costs related to the Mineral Reserve LOM plan. The costs indicate operating and total costs below the Mineral Reserve cost basis (US$1,750/oz Au);
- LOM plan capital cost estimates total US$1,236 M, including capitalized waste;
- LOM plan operating cost estimates total US$783.16/oz Au produced, or US$23.08/t processed.

Table 21-2: LOM Operating Cost Totals

Area	Units	Value
Mining costs	$ M	823
Capitalized waste	$ M	-149
Processing	$ M	626
Site general	$ M	279
Power plant	$ M	0.4
Change in stockpiled ore	$ M	23
Silver sales/credit	$ M	-29
Dore transportation, security, insurance	$ M	3
Refinery charge	$ M	7
Total operating costs	***$ M***	**1,583**

Notes:

1. Distributable costs are included in the area totals. Totals may not sum due to rounding.
2. Operating cost totals are reported from production years only and exclude the estimated construction period.

Table 21-3: LOM Unit Operating Costs (Ore Processed)

Area	Ore Processed (US$/t)	Gold Produced (US$/oz Au)
Mining (all areas)	11.27	382.58
Processing	8.16	276.95
Site general	3.64	123.63
Total	***23.08***	***783.16***

Note: Mining costs are US$2.71/t mined for mining including capitalized waste. Processing costs include stockpile rehandle and ore haulage where applicable. Totals may not sum due to rounding.

22.0 ECONOMIC ANALYSIS

22.1 Forward Looking Information Statement

Identification of information that is forward-looking is included in the statement at the front of this Report.

22.2 Methodology Used

The financial model that supports the mineral reserve declaration in a standalone model that calculates annual cash flows based on scheduled ore production, assumed processing recoveries, metal sale prices, exchange rate of 4,200 COP/US$, projected operating and capital costs, and estimated taxes.

The financial analysis is based on an after-tax discount rate of 5%. All costs and prices are in unescalated real dollars.

All costs are based on the forecast costs for the proposed Gramalote Ridge Mine. Revenue is calculated from the recoverable metals and long-term metal prices described in Section 19.2, and exchange rate forecasts.

22.3 Financial Model Parameters

The economic analysis is based on the metallurgical recovery predictions in Section 13, the Mineral Reserve estimates in Section 15, the mine plan discussed in Section 16, the commodity price forecasts in Section 19, closure cost estimates in Section 20, and the capital and operating costs outlined in Section 21.

Royalites will average 3.2% over the LOM.

The economic analysis is reported on a 100% Project ownership basis. The economic analysis assumes constant prices with no inflationary adjustments, and a long-term gold price of US$2,500/oz as discussed in Section 19.2.

22.4 Taxes

Under Colombian mining law, producing mines are subject to a federal royalty of 4% of the gross value of gold and silver production. Gross metal value is determined by using 80% of the spot prices for gold and silver on the London Metals Exchange, so the effective royalty rate is 3.2% market value. The international gold price used as a benchmark is based on daily spot prices published by recognized financial sources such as the London Bullion Market Association (LBMA) or other widely accepted international commodities exchanges.

Additionally, gold mining companies are subject to a 35% corporate income tax, and while royalties were initially non-deductible under Colombia's 2022 tax reform, a 2024

Constitutional Court ruling restored their deductibility, allowing miners to subtract royalty payments from their taxable income.

22.5 Economic Analysis

The valuation date is January 1, 2027, which is assumed to be the date of a construction decision pending permitting activities. A discount rate of 5% is used. The after-tax project NPV is US$941 M. The internal rate of return is calculated to be 22.4%, with a payback period of 3.4 years after production start.

A summary of the financial results is provided in Table 22-1.

The annualized cashflow statement is provided in Table 22-2, Table 22-3, and Table 22-4.

22.6 Sensitivity Analysis

Changes in metal price, gold grade, capital costs and operating cost assumptions were tested using a range of 25% above and below the base case values. Results are shown in Figure 22-1. The sensitivity to gold grade is not shown as it mirrored the gold price sensitivity within this sensitivity range.

The Project is most sensitive to changes in the gold price and grade, and less sensitive to changes in operating costs and capital costs, which exhibit similar sensitivity.

Table 22-1: Key Financial Metrics

Item	Units	Value
Production Profile		
Contained gold ounces processed	Moz	2.36
Gold recovery	%	95.7
Average gold grade	g/t	0.96
Gold ounces produced	Moz	2.26
Average annual gold production	koz/yr	177
Mine life	Years	11.0
Mill life	Years	13.0
Ore tonnes processed	Mt	76.7
Waste material mined	Mt	241
Waste to ore strip ratio	Waste: ore	3.1
Project economics - $2,500 /oz project average gold price		
Non-sustaining capital, construction	$ M	740
Non-sustaining capital, post construction	$ M	68
Sustaining capital	$ M	376
External projects, reclamation, and closure capital	$ M	51
Gross gold revenue	$ M	5,650
Net cash flow	$ M	1,630
$NPV_{5.0\%}$	$ M	941
IRR	%	22.4
Payback period after production start	years	3.4
Unit Operating Costs		
LOM cash operating costs (mining, processing, and site general)	$/oz Au	700
LOM AISC	$/oz Au	985
Average LOM mining cost	$/t rock mined	2.71
Average LOM processing cost	$/t processed	8.16

Note: NPV = net present value; IRR = internal rate of return; AISC = all in sustaining costs. AISC per ounce is the sum of cash operating costs, royalties and production taxes, capital expenditures that are sustaining in nature, corporate general and administrative costs, community relations expenditures, all divided by the total gold ounces sold to arrive at a per ounce figure. AISC is both $/oz sold and $/oz produced as there is no timing delay because ounces are produced and sold in the same period.

Table 22-2: Cashflow Forecast on an Annualized Basis (Years 1–7)

All Figures US$ 000's	**Total**	**Y 1**	**Y 2**	**Y 3**	**Y 4**	**Y 5**	**Y 6**	**Y 7**
Total revenue	***5,650,111***	—	—	409,538	595,067	598,711	599,484	558,673
Cost of production	***1,615,441***	15,013	17,719	97,680	112,651	101,940	113,734	123,832
Royalties and production taxes	***182,203***	61	61	13,225	19,171	19,277	19,300	17,987
Corporate administration and CSR department	***82,910***	6,587	4,384	6,868	5,202	5,295	5,220	5,295
Corporate social responsibility projects	***20,919***	989	989	16,798	714	714	238	238
Non-sustaining capital costs	***733,166***	402,804	262,668	42,587	25,107	—	—	—
Sustaining capital costs, excluding capitalized waste	***226,996***	—	—	42,360	19,928	25,933	16,601	15,068
Mining capital costs, capitalized waste	***168,690***	—	19,217	19,128	14,824	23,664	38,968	21,082
External projects, reclamation, and closure capital costs	***51,246***	—	—	—	866	866	866	866
Total taxes & royalties paid	***924,056***	333	437	31,097	121,398	125,283	120,317	103,604
Working capital	***-24,002***	5,000	—	—	-24,002	—	—	—
Change in stockpiled ore and mill WIP	***0***	—	22,696	5,172	33,333	42,177	9,099	8,606
Net Cash Flow	***1,668,487***	***-430,787***	***-328,171***	***134,624***	***265,874***	***253,562***	***275,142***	***262,095***
AISC ($/oz)	***985***	—	—	1,205	725	737	810	822

Note: Table presented on a 100% basis and in US$ x 1,000. CSR = Corporate Social Responsibility. AISC = all-in sustaining costs. AISC per ounce is the sum of cash operating costs, royalties and production taxes, capital expenditures that are sustaining in nature, corporate general and administrative costs, community relations expenditures, all divided by the total gold ounces sold to arrive at a per ounce figure. AISC is both $/oz sold and $/oz produced as there is no timing delay because ounces are produced and sold in the same period. WIP = Work in Progress. Numbers have been rounded.

Table 22-3: Cashflow Forecast on an Annualized Basis (Years 8–14)

All Figures US$ 000's	Y 8	Y 9	Y 10	Y 11	Y 12	Y 13	Y 14
Total revenue	377,808	360,714	459,518	581,252	442,896	222,150	222,150
Cost of production	142,442	145,159	148,371	116,731	135,645	116,552	115,383
Royalties and production taxes	12,180	11,634	14,808	18,731	14,308	7,194	7,133
Corporate administration and CSR department	5,225	5,295	5,221	5,306	5,221	5,306	3,718
Corporate social responsibility projects	238	—	—	—	—	—	—
Non-sustaining capital costs	—	—	—	—	—	—	—
Sustaining capital costs, excluding capitalized waste	26,278	18,234	25,102	17,820	12,566	3,625	3,480
Mining capital costs, capitalized waste	16,336	15,470	—	—	—	—	—
External projects, reclamation, and closure capital costs	866	866	866	866	866	3,016	3,016
Total taxes & royalties paid	40,672	34,928	65,256	115,786	72,747	17,409	36,175
Working capital	—	—	—	—	—	—	-3,000
Change in stockpiled ore and mill WIP	-6,450	-6,933	4,278	41,387	-10,363	-45,998	-49,336
Net Cash Flow	***140,022***	***136,061***	***195,615***	***264,624***	***211,906***	***115,046***	***105,580***
AISC ($/oz)	1,343	1,357	1,053	680	937	1,511	1,441

Note: Table presented on a 100% basis and in US$ x 1,000. CSR = Corporate Social Responsibility. AISC = all-in sustaining costs. AISC per ounce is the sum of cash operating costs, royalties and production taxes, capital expenditures that are sustaining in nature, corporate general and administrative costs, community relations expenditures, all divided by the total gold ounces sold to arrive at a per ounce figure. AISC is both $/oz sold and $/oz produced as there is no timing delay because ounces are produced and sold in the same period. WIP = Work in Progress. Numbers have been rounded.

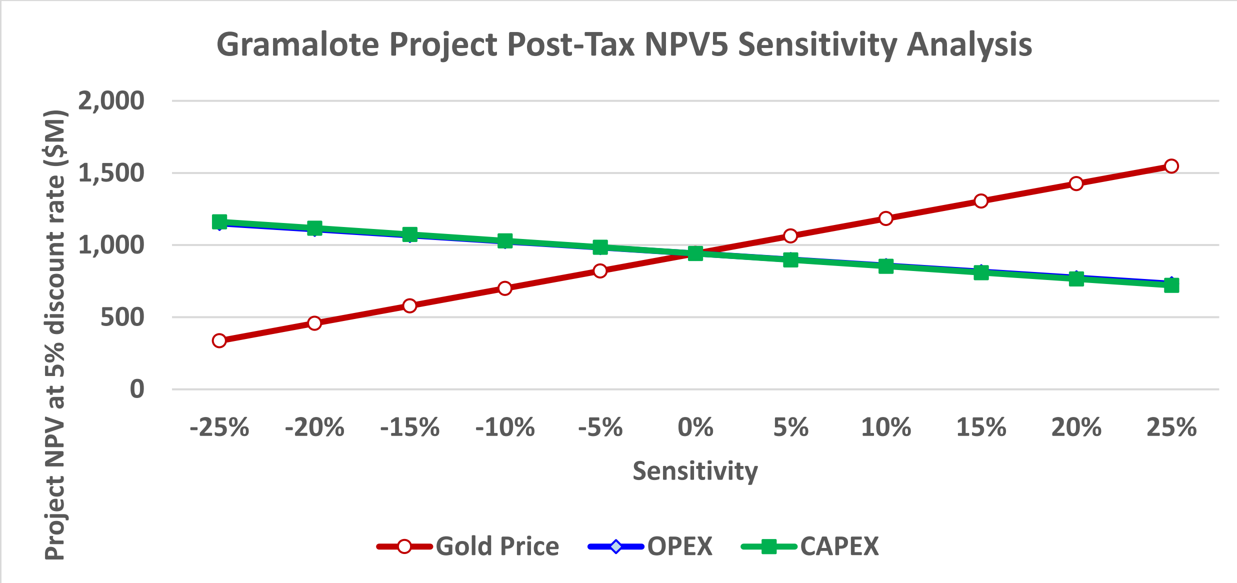
Table 22-4: Cashflow Forecast on an Annualized Basis (Years 15–20)

All Figures US$ 000's	Y 15	Y 16	Y 17	Y 18	Y 19	Y 20
Total revenue	222,150	—	—	—	—	—
Cost of production	110,991	1,597	—	—	—	—
Royalties and production taxes	7,133	-0	—	—	—	—
Corporate administration and CSR department	3,181	2,138	1,378	1,366	352	352
Corporate social responsibility projects	—	—	—	—	—	—
Non-sustaining capital costs	—	—	—	—	—	—
Sustaining capital costs, excluding capitalized waste	—	—	—	—	—	—
Mining capital costs, capitalized waste	—	—	—	—	—	—
External projects, reclamation, and closure capital costs	3,016	8,947	13,456	4,660	4,136	3,205
Total taxes & royalties paid	38,605	3	2	2	1	1
Working capital	-2,000	—	—	—	—	—
Change in stockpiled ore and mill WIP	-46,078	-1,589	—	—	—	—
Net Cash Flow	***107,301***	***-11,096***	***-14,835***	***-6,029***	***-4,488***	***-3,558***
AISC ($/oz)	1,383	—	—	—	—	—

Note:

Table presented on a 100% basis and in US$ x 1,000. CSR = Corporate Social Responsibility. AISC = all-in sustaining costs. AISC per ounce is the sum of cash operating costs, royalties and production taxes, capital expenditures that are sustaining in nature, corporate general and administrative costs, community relations expenditures, all divided by the total gold ounces sold to arrive at a per ounce figure. AISC is both $/oz sold and $/oz produced as there is no timing delay because ounces are produced and sold in the same period. WIP = Work in Progress. Numbers have been rounded.

Figure 22-1: Sensitivity Analysis



Note: Figure prepared by B2Gold, 2025. Opex = operating cost estimate, Capex = capital cost estimate

23.0 ADJACENT PROPERTIES

This section is not relevant to this Report.

24.0 OTHER RELEVANT DATA AND INFORMATION

This section is not relevant to this Report.

25.0 INTERPRETATION AND CONCLUSIONS

25.1 Introduction

The QPs note the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for this Report.

25.2 Mineral Tenure, Surface Rights, Water Rights, Royalties and Agreements

Information obtained from B2Gold experts supports that the mineral tenure held is valid.

Gramalote Colombia has commenced the acquisition of surface rights to support mining operations and purchases have been finalized on the majority of the area required. Based on the data and studies conducted, Gramalote Colombia has established the negotiation strategy for properties that are still to be acquired where they are necessary for the construction and operation stages.

Guacas Creek will provide the water for potable use. The water management system is designed to handle a positive water balance ensuring sufficient supply for all process water requirements.

Once in production, State royalties on the gold and silver are 4% of the gross metal value at the plant site (as per Article 16 of Law 141 in 1994). However, gross metal value is determined by using 80% of the spot prices for gold and silver on the London Metals Exchange, so the effective royalty rate is 3.2%.

Gramalote Colombia has lodged 17 land restitution proceedings. As of the Report Effective Date, a legal proceeding that was filed by a third party, Natalia Nohaba Vallejo, against the administrative act issued by the Mining Authority (Antioquia Secretary of Mines) is in progress. The Mining Authority (Antioquia Secretary of Mines) did not grant the mining rights that were requested by third parties over some “corridors” existing within the area of the integrated mining title 14292. The “corridors” arise from Colombian Mining Cadaster graphic defects.

Gramalote Colombia requested to become a party of the judicial proceedings, and was accepted as such on August 28, 2019. Gramalote Colombia will, therefore, be able to support the legal and technical thesis stated by the Mining Authority (Antioquia Secretary of Mines) throughout the proceedings.

In Gramalote Colombia's assessment, the risk of potential impact to the Project is low. If the legal precedent is maintained, the status quo of the mining area should be maintained as well. The “corridors” or graphic defects of the Colombian Mining Cadaster are not considered to be free areas. In the event that the interpretation does consider the “corridors” to be free areas, Gramalote Colombia would have a first right in time right, as it is the first applicant.

To the extent known to the QP, there are no other material factors or risks that may affect access, title, or the right or ability to conduct work on the Project that have not been disclosed in this Report.

25.3 Geology and Mineralization

The Gramalote Ridge and Trinidad deposits are considered to be examples of structurally-controlled, intrusive-related, gold deposits.

The geological understanding of the settings, lithologies, and structural and alteration controls on mineralization in the different zones is sufficient to support estimation of Mineral Resources. The geological knowledge of the area is also considered sufficiently acceptable to reliably inform conceptual mine planning.

The mineralization style and setting are well understood and can support declaration of Mineral Resources.

The Gramalote Ridge deposit remains open at depth. The mineralization is narrower to the east and west (Monjas East area) of the main deposit area. The Trinidad deposit remains open at depth, to the east, and particularly to the west.

A number of prospects have been identified that warrant additional exploration examination.

25.4 Exploration, Drilling and Analytical Data Collection in Support of Mineral Resource Estimation

The exploration programs completed to date are appropriate for the style of the deposits on the Project.

Sampling methods are acceptable for Mineral Resource estimation.

Sample preparation, analysis and security are generally performed in accordance with exploration best practices and industry standards.

The quantity and quality of the lithological, geotechnical, collar and down-hole survey data collected during the exploration and delineation drilling programs are sufficient to support Mineral Resource estimation. The collected sample data adequately reflect deposit dimensions, true widths of mineralization, and the style of the deposits. Sampling is representative of the gold and silver grades in the deposits, reflecting areas of higher and lower grades.

The QA/QC programs adequately address issues of precision, accuracy, and contamination. Drilling programs typically included blanks, duplicates, and CRM samples. QA/QC submission rates meet industry-accepted standards.

The data verification programs concluded that the data collected from the Project adequately support the geological interpretations and constitute a database of sufficient quality to support the use of the data in Mineral Resource estimation.

25.5 Metallurgical Testwork

Metallurgical testwork and associated analytical procedures are appropriate to the mineralization type, appropriate to establish the conceptual processing routes, and were performed using samples that are typical of the mineralization styles.

Samples selected for testing were representative of the various types and styles of mineralization. Samples were selected from a range of depths within the deposit. Sufficient samples were taken so that tests were performed on sufficient sample mass.

Recovery factors estimated are based on appropriate metallurgical testwork. At a gold head grade of 1.0 g/t Au, the overall gold recovery will be 95.5%. At a silver head grade of 1.0 g/t Ag, the overall silver recovery will be 38.0%. No plant recovery discounts are applied.

No deleterious elements are known from the processing perspective.

25.6 Mineral Resource Estimates

Mineral Resources are reported using the 2014 CIM Definition Standards, and assume open pit mining methods.

Factors that may affect the Mineral Resource estimate include: metal price and exchange rate assumptions; changes to the assumptions used to generate the gold grade cut-off grade; changes in local interpretations of mineralization geometry and continuity of mineralized zones; changes to geological and mineralization shape and geological and grade continuity assumptions; density and domain assignments; changes to geotechnical, mining and metallurgical recovery assumptions; changes to the input and design parameter assumptions that pertain to the conceptual pit constraining the estimates; and assumptions as to the continued ability to access the site, retain mineral and surface rights titles, maintain environment and other regulatory permits, and maintain the social license to operate.

25.7 Mineral Reserve Estimates

Mineral Reserves are reported using the 2014 CIM Definition Standards and are based on open pit mining methods.

Factors that may affect the Mineral Reserve estimates include: changes to the gold price assumptions; changes to pit slope and geotechnical assumptions; unforeseen dilution or ore loss; changes to hydrogeological and pit dewatering assumptions; changes to inputs to capital and operating cost estimates; changes to operating cost assumptions used in the constraining pit shell; delays or changes to the resettlement and relocation plans; changes to pit designs from those currently envisaged; changes in mining or milling productivity assumptions; and changes to modifying factor assumptions, including environmental, permitting and social licence to operate.

25.8 Mine Plan

The proposed mining operations will use conventional open pit mining methods and equipment.

Mining is based on a phased approach, which will see two phases in the main pit, and development of a satellite pit as the third phase.

The calculated cut-off grade for oxide material was 0.28 g/t Au, with an elevated cut-off grade of 0.40 g/t Au applied for mine and mill planning.

Four stockpiles are planned for operations. A fifth stockpile will store material grading 0.24–0.40 g/t Au, and will be located with the WRSF. Oxide material is not included in the processing plans or mineral reserves.

The WRSF will have a maximum total capacity of about 240 Mt, which is sufficient for the forecast LOM.

Pre-stripping is estimated at 13 Mt. Initial ore will be stockpiled at the ROM area and later at the El Balzal area. Some of the waste material will be used for the construction of mine platforms and haul roads.

A maximum extraction rate of 35 Mt/a is projected, with an 11-year LOM including the pre-stripping period. Processing will continue for an additional two years after active mining stops. The overall strip ratio is forecast at 3.1:1 (waste:ore).

The owner-operated equipment was selected based on a standard open pit mining operation. Auxiliary equipment will support mining, processing, and site general operations, but will not directly be involved in mine production

25.9 Recovery Plan

The proposed plant will process ROM mill feed material at a rate of 6 Mt/a to produce doré bars, using conventional equipment and processes. The process plant design for the Project is based on a robust metallurgical flowsheet designed for optimum recovery with minimum operating costs. The flowsheet is based upon unit operations that are well proven in industry.

The key criteria for equipment selection were suitability for duty, reliability, and ease of maintenance. The plant layout will provide ease of access to all equipment for operating and maintenance requirements whilst maintaining a layout that will facilitate concurrent construction progress in multiple areas.

The process plant will consist of single-stage primary crushing, closed circuit secondary (dry) screening and secondary cone crushing, a single-stage HPGR in closed circuit with HPGR (wet) screens and a ball mill in closed circuit with cyclones to achieve the target product size. It will include a gravity circuit with centrifugal gravity concentrators and an intensive cyanidation leach reactor. The flotation area will include a flotation circuit, flotation concentrate thickener, a trash screen and regrind circuit, pre-leach thickener, and a concentrate leach circuit. The gold circuit will consist of CIP and Pressure Zadra circuits.

25.10 Infrastructure

The major infrastructure envisaged in the 2025 Feasibility Study includes: one open pit; to be mined in two phases; waste rock storage facility, stockpiles, and water diversions including Guacas Creek diversion; process plant: grinding, flotation and leaching facilities, management and engineering offices, change house, workshop, warehouse and laydown facilities, and electrical grid switchgear substation; mine facilities: heavy equipment workshop, warehouse, fuel storage, fuel distribution bays, wash bay, tire bay, welding shop, assay laboratory, change house, cafeteria, explosive magazine, explosive transference plant, and concrete batch plant; internal mine roads, including haul roads; a community bypass road to the Cristales village; mine entrance and camp accommodation; water intake on the Guacas Creek and water treatment plant, sediment control ponds; quarry borrow pits; a TMF; and utilities.

25.11 Environmental, Permitting and Social Considerations

25.11.1 Environmental Considerations

The Terms of Reference for the Project EIA were issued by ANLA in 2012. The Environmental License was granted in late 2015. A MEIA was completed in 2019. Modifications to the Environmental License were completed in 2019 and 2024. An updated MEIA that reflects the changes in the 2025 Feasibility Study will be submitted for approval.

Collection of baseline data commenced in 2011. Updated baseline data is being collected to support the planned MEIA submission. The collection and updating of data and models is progressing with the support of environmental and social specialists.

The Environmental License sets out specific management and monitoring programs that describe the activities aimed to prevent, mitigate, and correct significant impacts that could be generated on the environmental and social components derived from Project development. These management programs will in place for the LOM, beginning during planning and continuing until the construction, operation, and closure phases are completed.

B2Gold must also implement compensation programs for biodiversity loss, woody vegetation and threatened species, and changes in land use. Water management programs must also be in place.

25.11.2 Closure Considerations

ANLA has approved a conceptual Closure Plan for the Project under the current Environmental License. The conceptual Closure Plan will be modified and updated periodically throughout the Project life to reflect significant changes in the design, construction, operation, and closure phases.

B2Gold has estimated the cost of the Project's environmental reclamation and closure liabilities to be approximately US$51.2 M. The Mining Code (Law 685 of 2001) includes an obligation to provide financial guarantees to ensure the Closure Plan can be completed, cover the costs associated with environmental remediation, and comply with social obligations.

25.11.3 Permitting Considerations

As of the Report Effective Date, B2Gold held a number of the key permits required to support Project development. The Environmental Licence will need to be updated to include the changes resulting from the 2025 Feasibility Study. Several additional permits and authorizations will be required for the Project as envisaged in this Report.

25.11.4 Social Considerations

Community engagement processes were conducted from 2015–2019, resulting in a total of 104 meetings with relevant stakeholders. Additional stakeholder consultation will be completed as part of the planned MEIA update.

Project development will have a significant impact on nearby communities, primarily due to the need for land acquisition and the physical and economic resettlement of households located within the infrastructure footprint. B2Gold has been advancing resettlement planning in compliance with Colombian legislation, the obligations of the Environmental License, and in alignment with international standards.

Socioeconomic baseline data were collected during prior EIA studies and subsequently updated in 2018. Additional assessments conducted in 2020 confirmed the final number of households to be resettled and validated the impacts and mitigation measures with Project-affected persons. In total, 327 households have been identified as significantly impacted and included in the Resettlement Action Plan. Of these, 225 households will be physically relocated while 102 households will experience economic displacement only.

The Environmental License requires that resettlement in the area of direct influence must be completed prior to the implementation of the approved Mining Plan. Through

Resolution 2427 (2019) and Resolution 1447 (2021), the National Environmental Licensing Authority (ANLA) authorized an extension of the resettlement phase and approved the commencement of project construction concurrently with the resettlement process, subject to compliance with technical requirements outlined in those resolutions. The resettlement schedule was further updated in 2024 via Resolution 2292, with completion currently planned for late 2026. It is possible that the schedule will require further adjustment to align with the current mine plan and construction timeline. If so, B2Gold will submit an updated schedule to ANLA for approval.

B2Gold has acquired sufficient land to meet resettlement requirements. Planning and development activities are underway, with progress reported quarterly to ANLA. In 2022, the Resettlement Action Plan was formalized in agreement with the affected communities, and the phases of collective and individual negotiations were completed during 2023 and 2024. As of the Report Effective Date, 98% of the affected households have signed individual agreements. Only 2% - equivalent to nine households - have not reached an agreement. Engagement and negotiations with these remaining households are ongoing and will continue until a resolution is achieved.

B2Gold has completed the design of housing, public facilities, and supporting infrastructure at the designated relocation site. Construction is progressing in accordance with approved designs and technical specifications. In addition, B2Gold is implementing livelihood restoration programs, community integration initiatives, and measures to ensure the provision of necessary public services. Implementation efforts will continue to be closely monitored and reported to ANLA until all commitments are fulfilled and the RAP completion audit is successfully concluded.

To reduce environmental impacts, support local communities, and foster social stability, B2Gold has advanced ASM formalization initiatives in coordination with local and national authorities. To date, 73 mining production units have been formalized under 22 mining associations, benefiting more than 180 miners and generating over 560 formal jobs with social security benefits. B2Gold will continue to advance ASM formalization efforts, including the assessment of new areas for potential inclusion, in alignment with national policy, community priorities, and its commitment to responsible mining practices.

25.12 Markets and Contracts

No market studies have been completed to date. The doré that will be produced by the mine is readily marketable.

Metal prices used for Mineral Resource and Mineral Reserve estimation are established the corporate level.

Quotes for major contracts have been received to support the 2025 Feasibility Study. These contracts will include blasting explosives and accessories through Indumil (the sole authorized provider for explosives under Colombian law), fuel provision, power provision, tire services, infrastructure construction, and other mining and processing consumable contracts as required.

Contracts will be negotiated and renewed as necessary, with terms expected to align with industry standards and comparable to similar contracts previously executed by B2Gold. The establishment of these key contracts is not considered a risk to the Project.

25.13 Capital Cost Estimates

The LOM plan assumes owner-operated mining operations, with contractor support in select areas A two-year pre-production period is anticipated, subject to permitting.

Capital costs consist largely of processing facilities, site infrastructure, mining equipment and rebuilds, and resettlement and land purchases.

A construction capital cost of US$740.1 M is estimated prior to commercial production. A post-construction capital cost of US$67.7 M is estimated after commercial production has begun.

Total mining fleet and infrastructure capital is estimated to be US$140.5 M over the LOM. After construction, US$149.5 M of capitalized waste is included in mining capital costs. The total mining capital cost, after construction, is estimated to be US$289.9 M over the LOM.

Process capital costs, after construction, are estimated at US$46.5 M over the LOM.

Total site general capital costs are estimated to be US$6.0 M over the LOM.

The total reclamation and closure capital cost is estimated at US$51.2 M.

The overall capital cost estimate is estimated at US$1,236 M.

25.14 Operating Cost Estimates

Departmental costs are estimated independently. Certain departments, such as power generation and heavy equipment maintenance, are treated as distributable costs and their costs are allocated to other departments based on usage.

Mine operating costs are estimated at US$2.71/t rock mined including capitalized waste.

Stockpile and ore rehandle costs are included with the processing costs. The total process operating cost is estimated to be US$8.16/t milled over the life of processing operations.

The total site general operating cost is estimated at US$3.64/t processed over the life of operations.

Operating costs total US$1,583 M over the LOM. Mining costs will average US$11.27/t ore processed, process costs will average US$8.16/t ore processed and site general costs will average US$3.64/t ore processed, for an overall ore processed cost of US$23.08/t.

LOM plan operating cost estimates total US$783.16/oz Au produced, or US$23.08/t processed.

25.15 Economic Analysis

The valuation date is January 1, 2027, which is modelled to be the date of a construction decision, pending permitting activities. A discount rate of 5% is used. The after-tax project NPV is US$941 M. The internal rate of return is calculated to be 22.4%, with a payback period of 3.4 years after production start.

The Project is most sensitive to changes in the gold price and grade, and less sensitive to changes in operating costs and capital costs, which exhibit similar sensitivity.

25.16 Risks and Opportunities

25.16.1 Risks

Permit timing assumes that all key permits will be obtained by 2027 to support Project construction. Delays in permit grants may affect the assumed dates in this Report for construction and operations.

Resettlement is assumed to be completed as the Project progresses. Delays in the resettlement process may affect the assumed dates in this Report for construction and operations.

The Project design basis incorporates the regulatory requirements that were current at the Report Effective Date. Updates or changes to the regulations after the Report Effective Date may require additional controls to be implemented.

The continued economic uncertainty with the current inflation and tariff imposts could affect the assumptions related to capital and operating cost estimates and therefore affect the economic analysis in this Report.

The formalization of the ASM activities, identification of designated ASM locations, and management of potential ASM-related impacts in the Project area are important due to their potential effects on site access and development, local security, and receiving environment.

25.16.2 Opportunities

The Mineral Resources at the Trinidad and Monjas deposits represent an upside opportunity if the Indicated Mineral Resources can be converted to Mineral Reserves and included in an updated mine plan with additional studies.

Further upside potential exists for the estimates if mineralization that is currently classified as Inferred can be upgraded to higher-confidence Mineral Resource categories.

Additionally, the numerous undrilled prospects discussed in Section 9 represent upside exploration potential, which will require drilling to evaluate their potential.

25.17 Conclusions

Under the assumptions outlined in this Report, the Project demonstrates a positive cash flow over the LOM and supports the Mineral Reserve estimates. The mine plan is achievable within the parameters and assumptions applied.

26.0 RECOMMENDATIONS

As the 2025 Feasibility Study shows a project with positive economics, and a decision to proceed with construction is at the discretion and purview of B2Gold's Board of Directors, the QPs have no meaningful recommendations to make.

27.0 REFERENCES

AngloGold, 2017: Gramalote Enhanced Pre-feasibility Study: internal report prepared for B2Gold and AngloGold, May 2017.

Aspden, J.A. and Mccourt, W.J., 1986: Mesozoic Oceanic Terrane in the Central Andes of Colombia: Geology. 14, pp. 415–418.

Cabral, Alexandre, 2007: Ore Microscopy of Samples from Gramalote Colombia: internal report prepared for Gramalote Colombia, December 2007.

Canadian Institute of Mining, Metallurgy and Petroleum (CIM), 2014: CIM Definition Standards – for Mineral Resources and Mineral Reserves, prepared by the CIM Standing Committee on Reserve Definitions: adopted by the CIM Council, May, 2014.

Canadian Securities Administrators (CSA), 2011: National Instrument 43-101, Standards of Disclosure for Mineral Projects, Canadian Securities Administrators.

Cediel, F., and C. Cáceres, 2000: Geological Map of Colombia, 1: 2,000,000: Geotec Ltd., Bogotá, Third Edition, digital format with legend and tectono-stratigraphic chart.

Cediel, F., Shaw, R.P. and Cáceres, C., 2003: Tectonic Assembly of the Northern Andean Block: *in* Bartolini, C., R.T. Buffler, and J. Blickwede, eds., The Circum Gulf of Mexico and Caribbean: Hydrocarbon Habitats, Basin Formation, and Plate Tectonics: AAPG Memoir 79, pp. 815–848.

Colombia Mining Agency, 2019: Brief Guide to Invest in Mining in Colombia: notes prepared by the Colombia Mining Agency, http://mineriaencolombia.anm.gov.co/index.php/en/regulations

Gorham, J.P., 2008: Updated Report on the Gramalote Property, Department of Antioquia, Colombia, 43-101 Technical Report prepared for B2Gold Corp., June 2008.

Hart, C.J.R., 2005: Classifying, Distinguishing and Exploring for Intrusion-Related Gold Systems: *in* The Gangue, Issue 87, October, 2005, published by Geological Association of Canada, Mineral Deposits Division, 18 p.

Hart, C., 2007: Reduced Intrusion-Related Gold Systems: *in* Goodfellow, W.D., ed., Mineral Deposits of Canada: A Synthesis of Major Deposit Types, District Metallogeny, the Evolution of Geological Provinces, and Exploration Methods: Geological Association of Canada, Mineral Deposits Division, Special Publications 5, pp. 95–112.

Hulse, D.E., 2012: NI 43-101 Technical Report on Resources, Gramalote Project, Providencia, Colombia: Report prepared by Gustavson Associates for B2Gold, effective date 1 June, 2012;

Hulse, D.E., Sobering, G., Newton, M.C. III, Malhotra, D., and Daviess, F., 2014: NI 43-101 Preliminary Economic Assessment, Gramalote Project, Northwest Colombia: report prepared by Gustavson Associates for B2Gold, effective date 1 February, 2014.

Lefebure, D.V., and Hart, C., 2005: Plutonic-related Au Quartz Veins and Veinlets, Model L02: *in* Fonseca, A., and Bradshaw, G., eds., Yukon Mineral Deposit Profiles: Yukon Geological Survey Open File Report 2005-5, pp.121–128.

McCourt, W.J., Feininger, T. and Brook, M., 1984: New Geological and Geochronological Data from the Colombian Andes: Continental Growth by Multiple Accretion: Journal of the Geological Society, London, vol 141, pp. 831–845.

Meister, S.N., 2009: Gramalote Ridge Project, Department of Antioquia, Colombia, NI 43-101 Technical Report: report prepared for B2Gold, effective date 27 February 2009;

Ministry of Environment and Sustainable Development (Colombia), 2015: Decree 1076 of 2015, By Which the Single Regulatory Decree of the Environment and Sustainable Development Sector Is Issued: 653 p., 26 May 2015.

Ministry of Environment and Sustainable Development (Colombia), 2018a: Manual of Biotic Component Compensations (Resolution 256 of 2018): 61 p., 22 February 2018.

Ministry of Environment and Sustainable Development (Colombia), 2018b: General Methodology for the Preparation and Submission Of Environmental Studies (Resolution 1402 of 2018): 228 p. 25 July, 2018.

National Environmental Licensing Authority, 2016: Terms of Reference for the Preparation of the Environmental Impact Assessment (EIA) – Mining Exploitation Projects (Resolution 2206 of 2016). 159 p., 27 December, 2016.

National Environmental Licensing Authority, 2018: Guidelines for the Definition, Identification, And Delimitation of the Area of Influence: 45 p. July, 2018.

National Environmental Licensing Authority, 2022: Guideline for the Preparation of the Closure and Abandonment Plan for Mining Projects: 55 p., August 2022.

Rodriguez, O.F., 2009: Ore Paragenesis of the Gramalote Gold Deposit: MSc thesis, Queens University, Canada, August 2009.

Salazar, D., 2019: Mining in Colombia: Overview: report prepared by Posse Herrera Ruiz, https://uk.practicallaw.thomsonreuters.com/w-019-7418?transitionType=Default&contextData=(sc.Default)&firstPage=true&bhcp=1

Sillitoe, R.H., 1991: Intrusion-Related Gold Deposits: *in* Foster, R.P. (ed.), Gold Metallogeny and Exploration, Blackie, Glasgow, pp. 165–209.

Sillitoe, R.H., 2005: Comments on the Quinchia, Quebradona and Gramalote Prospects, Colombia: internal report prepared for Sociedad Kedahda S.A., June, 2006, 11 p.

Sillitoe, R.H., 2006: Comments on the Initial Drilling Results from the Quinchia, La Mina and Gramalote Gold Prospects, Colombia: internal report prepared for Sociedad Kedahda S.A., July, 2006, 13 p.

Valencia, M., 2006, Reporte Tecnico: Cartografía Geológica – Estructural, Proyecto Gramalote: internal report prepared for Sociedad Kedahda S.A. September, 2006, 25 p.

Valencia, M., 2007. Estilos Estructurales y Ocurrencias Minerales Gramalote Regional - Fase II: internal report, AngloGold Ashanti Colombia.

Walker, D., Venter, J., Hamman, E. 2021: Gramalote Gold Project Geotechnical Feasibility Report: report prepared by AngloGold Ashanti, May, 2021.