NI 43-101 Preliminary Economic Assessment (PEA) Independent Technical Report for the Luanga PGM + Au + Ni Project Pará, Brazil

Prepared by:

GE21 Consultoria Mineral Ltda.

3130 Afonso Pena Ave., 9th Floor, Belo Horizonte, MG, 30130-910, Brazil
portal@grupoge21.com

On behalf of:

Bravo Mining Corp.

GE21 Project No.: 250510
Effective date: July 7th, 2025

Qualified Persons:

Porfirio Cabaleiro Rodriguez, BSc Mining Engineering, FAIG

Bernardo Viana, BSc Geology, FAIG

Paulo Roberto Bergmann Moreira, B.Sc Mine Eng, FAusIMM

Juliano Lima, B.Sc Geology Eng, MAIG

Eduardo Dequech de Carvalho, B.Sc Mine Eng, MAusIMM

NI 43-101 Preliminary Economic Assessment (PEA) Independent Technical Report for the Luanga PGM + Au + Ni Project, Pará State, Brazil

GE21 Project No:	250510
Effective date:	July 7th, 2025
Issue date:	August 20th, 2025
Version:	Rev02
Work directory:	S:\Projetos\Bravo\250510-Projeto-Lavra-Luanga
Copies:	Bravo Mining Corp. GE21 Consultoria Mineral Ltda.

Review	Description	Author(s)	Date

DATE AND SIGNATURE

This report, titled "NI 43-101 Preliminary Economic Assessment (PEA) Independent Technical Report for the Luanga PGM + Au + Ni Project, Pará State, Brazil", having an effective date of July 7th, 2025, was prepared by GE21 Consultoria Mineral Ltda. on behalf of Bravo Mining Corp., and signed.

Dated at Belo Horizonte, Brazil, on August 20th, 2025.

<Signed and sealed in the original>

Porfirio Cabaleiro Rodriguez, BSc Mining Engineering, FAIG
Author

<Signed and sealed in the original>

Bernardo Viana, BSc Geology, FAIG
Author

<Signed and sealed in the original>

Paulo Roberto Bergmann Moreira, B.Sc Mine Eng, FAusIMM
Author

<Signed and sealed in the original>

Juliano Felix de Lima, B.Sc Geology Eng, MAIG
Author

<Signed and sealed in the original>

Eduardo Dequech de Carvalho, B.Sc Mine Eng, MAusIMM
Author

IMPORTANT NOTICE

This report was prepared following the requirements of the Canadian Institute of Mining, Metallurgy and Petroleum NI 43-101 Standards of Disclosure for Mineral Projects by GE21 Consultoria Mineral Ltda. (GE21) on behalf of Bravo Mining Corp. (Bravo). The quality of the information, conclusions, and estimates contained herein is consistent with the level of effort involved in the report authors' services, based on i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by Bravo, subject to terms and conditions of its individual contracts with the report authors and relevant securities legislation. The contract permits Bravo to file this report as a Technical Report with Canadian securities regulatory authorities pursuant to the Canadian Securities Administrators' National Instrument 43-101, Standards of Disclosure for Mineral Projects, Companion Policy 43-101CP and form 43-101F1 (collectively, NI 43-101). Except for the purposes legislated under provincial securities law, any other uses of this report by any third party are at that party's sole risk. The responsibility for this disclosure remains with Bravo. GE21 is under no obligation to update this technical report, except as may be agreed to between Bravo and GE21 by contract from time to time. The user of this document should ensure that this is the most recent Technical Report for the property as it is not valid if a new Technical Report has been issued.

Currency is expressed in U.S. dollars, and all other units comply with the International System of Units, unless otherwise stated.

FORWARD LOOKING STATEMENTS

This news release contains forward-looking information which is not comprised of historical facts. Forward-looking information is characterized by words such as “assume”, “substantial”, “solid”, ‘’improve”, ‘’increase”, ‘’significant ”, ‘’success”, ‘’potential ”, ‘’opportunity”, “well positioned”, variants of these words and other similar words, phrases, or statements that certain events or conditions “could”, “may”, “should”, “will” or “would” occur. This news release contains forward-looking information pertaining to the Company’s 2025 PEA; the potential for future MRE growth from deeper drilling; the potential to incorporate mineralized material within the pit but stockpiled as below cut-off into future economic studies; the potential to process the oxide mineralized material and the economics thereof; whether or not current or future discoveries of copper-gold mineralization at Luanga will have sufficient economic merit to consider development; potential repeatability and improvements to the economic assumptions and/or to metallurgical recoveries used in the PEA and MRE in future studies; the potential to convert some or all of the MRE to mineral reserves through economic studies and the timing and results of any such studies; the assumption that onsite vertical integration/downstream processing will be technically and economically feasible and that an experienced partner will be identified and terms of a relationship be acceptable to Bravo; the carbon intensity of any future operation; whether of funding from Brazilian National Bank for Economic and Social Development (“BNDES”) and FINEP will be made available to Bravo and, if so, on what terms; the ultimate cost of power for any future mine developed; the duration and prices in future commodity cycles; whether more value can be unlocked within the Luanga in the future; changes in the exchange rate between the US$ and Brazilian Real (R$); the results of subsequent stages of permitting, including but not limited to the timing, granting and conditions of the LI and LO referred to herein; the outcomes of future economic studies and the Company’s plans in respect thereof. Forward-looking information involves risks, uncertainties and other factors that could cause actual events, results, and opportunities to differ materially from those expressed or implied by such forward-looking information. Factors that could cause actual results to differ materially from such forward-looking information include, but are not limited to, unexpected results from exploration programs, changes in the state of equity and debt markets, fluctuations in commodity prices, delays in obtaining required regulatory or governmental approvals, environmental risks, limitations on insurance coverage; and other risks and uncertainties involved in the mineral exploration and development industry. Forward-looking information in this news release is based on the opinions and assumptions of management considered reasonable as of the date hereof, including, but not limited to, the assumption that the assay results confirm that the interpreted mineralization contains significant values of nickel, PGMs and Au; that the mineralization remains open at depth, that PGM and/or Ni grades and mineralized thicknesses are improving at depth; that activities will not be adversely disrupted or impeded by regulatory, political, community, economic, environmental and/or healthy and safety risks; that the Luanga Project will not be materially affected by potential supply chain disruptions; and general business and economic conditions will not change in a materially adverse manner. Although the Company believes that the assumptions and factors used in preparing the forward-looking information in this news release are reasonable, undue reliance should not be placed on such information. The Company disclaims any intention or obligation to update or revise any forward-looking information, other than as required by applicable securities laws.

Cautionary Note for U.S. Investors Concerning Mineral Resources and PEA (Scoping Study in SEC S-K 1300)

This news release has been prepared in accordance with the requirements of the securities laws in effect in Canada, which differ from the requirements of United States securities laws. The terms “mineral resource”, “measured mineral resource”, “indicated mineral resource” and “inferred mineral resource” are defined in and required to be disclosed by NI 43-101; however, these terms are not defined terms under the U.S. Securities and Exchange Commission (“SEC”) modernization rules, known as “S-K 1300”, and are normally not permitted to be used in reports and registration statements filed with the SEC. Investors are cautioned not to assume that all or any part of an “measured mineral resource”, “indicated mineral resource” or “inferred mineral resource” will ever be upgraded to a higher category or converted into mineral reserves in accordance with S-K 1300. “Inferred mineral resources” have a great amount of uncertainty as to their existence, and great uncertainty as to their economic and legal feasibility. Under Canadian rules, estimates of inferred mineral resources may not form the basis of feasibility or pre-feasibility studies, except in rare cases. Investors are cautioned not to assume that all or any part of an inferred mineral resource exists or is economically or legally mineable. Disclosure of “contained ounces” in a mineral resource is permitted disclosure under Canadian regulations; however, the SEC normally only permits issuers to report mineralization that does not constitute “reserves” by SEC S-K 1300 standards as in place tonnage and grade without reference to unit measures. Accordingly, information contained in this News Release contain descriptions of the Company’s mineral deposits that may not be comparable to similar information made public by U.S. companies subject to the reporting and disclosure requirements under the United States federal securities laws and the rules and regulations thereunder.

Non IFRS Financial Performance Measures

“All-in Sustaining Cost”, “Total Site Costs”, “Total Operating Costs”, “Net Revenue”, and “Free Cashflow” are not performance measures reported in accordance with International Financial Reporting Standards (“IFRS”). These performance measures are included because these statistics are key performance measures that management uses to monitor performance. Management uses these statistics to assess how the Luanga Project compares against its peer projects and to assess the overall effectiveness and efficiency of the contemplated mining operations. These performance measures do not have a meaning within IFRS and, therefore, amounts presented may not be comparable to similar data presented by other mining companies. These performance measures should not be considered in isolation as a substitute for measures of performance in accordance with IFRS.

UNITS, SYMBOLS, AND ABBREVIATIONS

Abbreviations	
3E	Palladium, Platinum and Rhodium
4E	Palladium, Platinum, Rhodium and Gold
AACE	Association for the Advancement of Cost Engineering
AIG	Australian Institute of Geoscientists
ANM	Brazilian Mining National Agency
AusIMM	Australasian Institute of Mining and Metallurgy
BNDES	Brazilian Development Bank
CAPEX	Capital Expenditures
CFEM	Financial Compensation for the Exploitation of Mineral Resources
CIL	Carbon-in-Leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CMM	Meridional Mining Company
CMP	Carajás Mineral Province
DSF	Dry Stacking Facility
EIA	Environmental Impact Study
ESG	Environmental, Social, and Governance
G&A	General and Administrative
GE21	GE21 Consultoria Mineral Ltda
GPS	Global Positioning System
IOCG	Iron Oxide Copper Gold
IRR	Internal Rate of Return
LI	Installation License
LO	Operating License
LP	Preliminary License
MRE	Mineral Resource Estimates
NPV	Net Present Value
NSR	Net Smelter Return
OK	Ordinary Kriging
OPEX	Operating Expenditures
PAE	Economic Exploitation Plan
PEA	Preliminary Economic Assessment
PGM	Platinum Group Metals
QA/QC	Quality Assurance / Quality Control
QP	Qualified Person
RIMA	Environmental Impact Report
ROM	Run-of-Mine
RPEEE	Reasonable Prospects for Eventual Economic Extraction
SAG	Semi-Autogenous Grinding
SEMAS	State Secretariat for Environment and Sustainability
SIRGAS	Geocentric Reference System for the Americas
SMU	Selective Mining Unit
SUDAM	Federal Agency of the Amazon Development Superintendence
UTM	Universal Transverse Mercator
WRSF	Waste Rock Storage Facility

	Units and Symbols			
%	Percentage		kVA	Kilovolt-Ampere
“	Inches		L	Litre
°C	Celsius		m	Meters
R$	Brazilian Reals		M	Millions
Au	Gold		Mt	Megaton
cm	Centimetre(s)		Mtpa	Million Tonnes per Annum
cm^3	Cubic centimetre(s)		MW	Megawatt
g/t	Grams per Tonne		Ni	Nickel

	Units and Symbols			
Ga	Giga-annum		Oz	Troy Ounce
h	hours		Pd	Palladium
k	Thousands		ppb	Parts per Billion
Kg	Kilogram		ppm	Parts per Million
km	Kilometre(s)		Pt	Platinum
km^2	Square Kilometre(s)		Rh	Rhodium
kOz	Kilo-ounce		t	Tonne
kPa	Kilopascal		US$	United States Dollars
kV	Kilovolt		V	Volts

TABLE OF CONTENTS

LIST OF TABLES

LIST OF FIGURES

1 EXECUTIVE SUMMARY

1.1 Introduction

GE21 Consultoria Mineral Ltda. (GE21) is an independent mineral consulting company composed of a team of professionals qualified to report Mineral Resources and Reserves in accordance with the guidelines of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) code, as required by "National Instrument 43-101 – Standards of Disclosure for Mineral Projects" (NI 43-101).

Bravo Mining Corp. (TSX.V: BRVO, OTCQX: BRVMF – or "Bravo") is a Canadian and Brazil-based mineral exploration and development company focused on advancing its Luanga PGM+Au+Ni (Platinum Group Metals, Gold and Nickel) Project in the Carajás Mineral Province of Brazil.

Bravo has commissioned GE21 to prepare a Preliminary Economic Report (PEA) Technical Report for the Luanga Project in Pará, Brazil, in accordance with the requirements of NI 43-101.

The Effective Date of this report is July 7th, 2025. The Authors have relied on information provided by Bravo, which was provided in a database with full access given to the Qualified Persons (QPs).

1.2 Qualifications, Experience and Independence

GE21 is a specialized, independent mineral consulting company. The geological reconnaissance, due diligence evaluation, technical parameters and economic assessment have been conducted by GE21 staff members, who are members of the Australian Institute of Geoscientists (AIG) and/or the Australasian Institute of Mining and Metallurgy (AusIMM) and meet the requirements of independent QPs as defined in NI 43-101.

The QPs responsible for this independent Technical Report are Mr. Porfirio Cabaleiro Rodriguez, Mr. Bernardo Viana, Mr. Paulo Roberto Bergmann Moreira, Mr. Juliano Felix de Lima and Mr. Eduardo Dequech de Carvalho.

1.3 Reliance on Other Experts

The authors of this report are QPs under NI 43-101, with expertise in mineral exploration, data validation, resources and reserves estimation, and projects economic assessments.

Information regarding tenure, status, and permitted work within Bravo's property is based on publicly available data from Brazil's National Mining Agency (ANM).

Bravo Mining Corp. engaged Linneu de Albuquerque Mello, a qualified lawyer in Brazil, to provide a title opinion on the Luanga Mineral Rights, confirming their validity and good standing as of January 31, 2025.

1.4 Property Description and Location

The Luanga Project is an advanced-stage mineral project located in Pará, Brazil, which contains platinum group metals (PGM), gold (Au), and nickel (Ni). It is held under Exploration License Nº.1961 (ANM.851.966/1992) and covers 7,810.02 hectares (Table 1-1).

The project is located on private farmland used for cattle ranching, with no indigenous claims or protected forests. Land access agreements with six key landowners cover 97% of the known mineralized area. The land access agreements are renewable every two years.

The initial exploration permit expired in 1998; however, due to bureaucratic delays at ANM, its renewal was granted only in 2005 for an additional three years. Vale submitted a Final Exploration Report in 2008, and a Mining License application was filed in 2013. The ANM has not yet approved the Final Exploration Report, and Bravo expects this delay to continue until a new feasibility study is submitted.

Table 1-1: Mineral tenement summary

ANM Process	Municipality	Stage	Mineral	Title Owner	Size (hectares)	License No.
851.966/1992	Curionópolis	Application for Mining License	Au, Pd, Pt, Ni	Bravo Mineração Ltda	7,810.02	1961

Notes:
Comments: Mining License pending
ANM = Mining National Agency
Source: ANM, February 2025.

The Luanga mineral property is centred approximately at coordinates -05º57’24.34” S/-49º32’51.00” W (Figure 1-1).

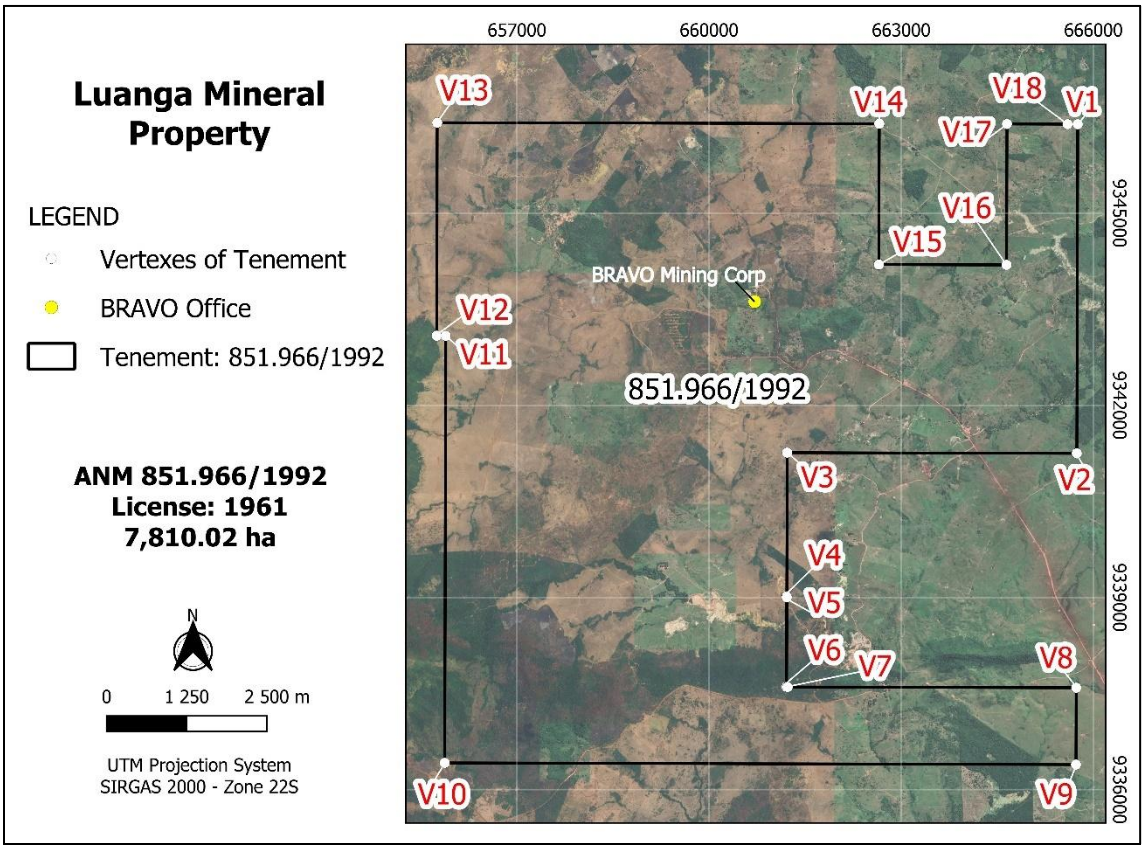


Figure 1-1: Luanga Project tenement map

Source: GE21, 2025.

1.5 Accessibility, Climate, Local Resources, Infrastructure, and Physiography

The Luanga Project is in Curionópolis, Pará, Brazil, about 500 km south of Belém. It is accessible via paved roads from Parauapebas and Marabá, both of which have commercial airports with multiple daily flights. The project is reached via a municipal paved road off Highway PA-275 (Figure 1-2).

The nearest towns are Curionópolis (17,846 people, 17 km south-southwest) and Serra Pelada (12 km west). Parauapebas, located 40 km away, is a key mining hub that provides labour, services, and infrastructure for the region.

The project is situated in the Carajás Mineral Province, within the South Pará Plateau (elevation 500-700m), near the Serra Seringue range. The Sereno River and its tributaries drain the area.

The climate is equatorial, with warm, dry winters and wet, humid summers. 75% of annual rainfall occurs from December to April, ensuring ample water availability for potential mining activities.

GE21 Consultoria Mineral

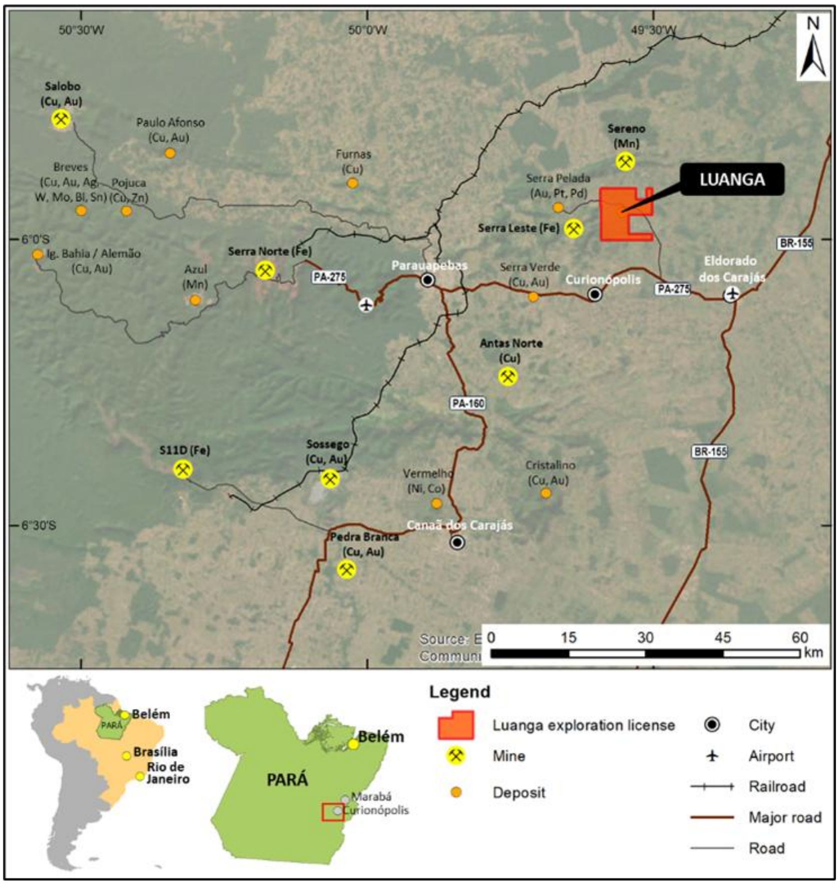


Figure 1-2: Access map for Luanga Project

Source: Bravo, 2025.

1.6 History

Due to access challenges, geographical studies in the Carajás region were limited until the 1960s. In 1966, CODIM, a subsidiary of Union Carbide, discovered the Serra do Sereno manganese deposit, sparking interest from US Steel. In 1967, a Brazilian team identified high-grade iron mineralization (66% Fe), though the government restricted foreign control.

In 1970, the Brazilian government established Amazônias Mineração SA (AMSA), with Vale (formerly Companhia Vale do Rio Doce, or CVRD) holding a 51% stake and CMM owning 49%. By 1974, AMSA secured exclusive rights to Carajás iron mineralization, and by 1977, Vale acquired the remaining 49% of CMM for US$55 million, gaining full control over the Carajás Mineral Province.

Vale began the construction of the Carajás Railroad in 1978, linking the mine to Ponta da Madeira, Maranhão. The Carajás Iron Mineralization Project (total cost: US$3 billion) included significant investments in infrastructure:

The Luanga Project exploration dates back to 1983 when DOCEGEO identified mafic-ultramafic rocks in the Luanga Complex, discovering chromitite mineralization and conducting initial geochemical surveys. By 1997, a joint venture between DOCEGEO and Barrick Gold had identified gold anomalies in stream sediments.

In 2000, Vale expanded its exploration with a soil geochemistry survey, revealing a 4 km Pt-Pd anomalous trend in the Luanga Complex. This led to further integration of geological, geophysical, and remote sensing data by Vale in 2001.

Between 1992 and 2003, Vale carried out comprehensive diamond drilling campaigns, totaling 256 drill holes (50,786.74m):

This extensive exploration established the Luanga Project as a key PGM-Au-Ni target within the Carajás Mineral Province, laying the foundation for continued resource evaluation and development.

In 2023, GE21 conducted a Mineral Resource Estimate (MRE) for Bravo, accompanied by a pit optimization study, to classify the Mineral Resources. The pit-constrained MRE had an effective date of October 22, 2023, and it was comprised of 73 Mt grading 1.75 g/t PdEq for a total of 4.1 Moz of PdEq in the Indicated category and 118 Mt grading 1.50 g/t PdEq for 5.7 Moz PdEq in the Inferred category. This 2023 MRE has been superseded by the current MRE.

1.7 Geological Setting and Mineralization

The Carajás Mineral Province (CMP), located on the southeastern margin of the Amazon Craton, is one of South America's most important mineral provinces. It hosts world-class Fe, Cu-Au, and Ni deposits, including the Serra Pelada, Salobo, and Igarapé Bahia Cu-Au deposits. The province is bounded by the Araguaia Neoproterozoic Belt to the east and south and overlain by Paleoproterozoic sequences to the west. The geological boundaries of the CMP are not precisely defined to the north, where Paleoproterozoic gneiss-migmatite-granulite terrains dominate.

Within the Carajás Domain, several mafic-ultramafic complexes intrude into both the Xingu Complex and the Archean volcano-sedimentary sequences. These intrusions are host to major Ni laterite deposits (e.g., Onça-Puma, Vermelho, Jacaré) and PGM deposits (e.g., Luanga, Lago Grande). The magmatic structure and evolution of these layered intrusions suggest that they belong to different Neoarchean magmatic suites, contrary to previous regional studies that ascribed them all to the Cateté Suite.

The Luanga Layered Mafic-Ultramafic Complex, commonly referred to as the Luanga Complex, is a significant geological feature situated within the Carajás Mineral Province. The complex spans approximately 6 km in length and up to 3.5 km in width, covering an area of around 18 km². It belongs to the Neoarchean Large Igneous Province (LIP) of the Carajás Mineral Province. The intrusion is characterized by abundant unweathered outcrops, massive blocks, and boulders, which are more prominent compared to the surrounding areas of the province.

The geomorphology of the complex features a smooth hill that is elongated in shape and arc-like, underlain primarily by ultramafic rocks. This hill is up to 60 meters higher than the surrounding flat areas, where gabbroic rocks prevail. The layering within the complex forms an arc-shaped structure that mirrors the overall morphology of the region

GE21 Consultoria Mineral

1.7.1 *Stratigraphy and Geological Features*

The central part of the Luanga Complex contains the thickest sequence of layered rocks. Moving toward the north and northeast, the layered sequence is truncated by granitic intrusions, while to the south, the sequence becomes progressively thinner. The complex and its host rocks are intersected by NNW-SSE dolerite dykes that are up to several meters wide.

Geological sections from drilling data reveal that the igneous layers in the central and southwestern portions of the complex exhibit steep dips to the southeast. These sections show that the Ultramafic Zone (UZ) overlies the Transition Zone (TZ), which in turn overlies the Mafic Zone (MZ), indicating that the layered sequence is tectonically overturned. Previous studies (Ferreira Filho et al., 2007; Teixeira et al., 2015) also described an overturned layered sequence in the Luanga and Lago Grande Complexes, suggesting that regional structural features led to the formation of large, overturned blocks in the Serra Leste region.

1.7.2 *Mineralization at the Luanga Complex*

The Luanga PGM + Au + Ni mineralized envelope follows the arc-shaped structure of the Mafic-Ultramafic Complex for approximately 8.1 km. The deposit is divided into three main mineralized sectors: North, Central, and Southwest. The mineralization is primarily found within the Transition Zone (TZ) of the Luanga Complex, which contains several PGM mineralized units.

The Main Sulphide Zone (MSZ) is the most significant mineralized zone, hosting the bulk of the PGM resources within the Luanga Complex. Additional mineralized layers are identified both within the Transition Zone (TZ) and the Ultramafic Zone (UZ).

In addition to the primary PGM mineralization, the Luanga Complex also contains thin chromitite layers or lenses. These are found either in the Transition Zone's upper or lower stratigraphic portions. The chromitite layers hosted in the lower portions of the TZ are associated with ultramafic cumulates, while the upper chromitite layers are located in immediate contact with the overlying Mafic Zone (MZ). These upper layers are hosted by plagioclase-bearing norite cumulates, indicating a complex stratigraphic relationship within the layered sequence of the complex.

1.8 Deposit Types

The majority of PGM resources come from mafic-ultramafic layered intrusions, particularly from stratiform mineralized layers near the transition from mafic to ultramafic cumulate rocks. The Bushveld Complex (South Africa) exemplifies this type of mineralization, known as reef-type deposits. These intrusions also host deposits of base metal sulphides (Ni-Cu-Co), chromite, and magnetite-ilmenite due to magmatic processes during the cooling and emplacement of mafic-ultramafic magmas.

Ni-Cu-PGM sulphides accumulate through the separation of sulphide liquid from silicate magma. In the Luanga Complex, textural features in the MSZ and other zones confirm the magmatic origin of PGM, Au, and Ni mineralization. Sulphide blebs are found interstitial to olivine and pyroxene, with rounded faces supporting a magmatic origin.

The Luanga Complex exhibits several mineralized horizons with varying metal ratios, indicating multiple mineralizing events during its magmatic evolution, similar to the Bushveld and Stillwater (U.S.A.) deposits.

While widespread alteration has affected the Luanga Complex, the primary magmatic features of cumulate rocks and PGM zones are preserved, despite partial alteration of sulphides to magnetite and Fe-hydroxides. The alteration is heterogeneous, maintaining key textures throughout the intrusion.

1.9 Exploration

1.9.1 Topography

In 2023, RR Topografia & Engenharia of Brazil completed a new Orthophotography and Digital Elevation Model (DEM).

1.9.2 Geophysics

The first geophysical work for Bravo was completed in 2021 by Southern Geoscience Consultants of Australia (SGC) and Southernrock Geophysics of Chile (Southernrock). Southernrock reprocessed the historic Induced Polarization (IP) data, while SGC reprocessed the historic magnetic data.

Ground geophysical activities conducted during 2022, and January 2023 included borehole electromagnetics and surface electromagnetic surveys. Both surveys were conducted by Geomag S/A (Geomag).

Borehole electromagnetic surveys (BHEM) were carried out along five drill holes totalling 1,109 linear metres. The best BHEM response was associated with drill hole DDH22LU047, which intersected 11 metres of massive sulphides.

A Fixed-Loop Transient Electromagnetics (FLTEM) survey was concentrated on the Central and North Sectors along 34 survey transversal lines (total of 30.27km). Loop dimensions were 600 x 400 metres, and survey lines were spaced 100 metres apart.

In 2024, Bravo conducted a ground geophysics survey consisting of magnetometry and micro-gravity, using a 100 m line-spacing grid performed over an area of approximately 18.7 km^2, covering the ultramafic and transition zones of the Luanga Complex.

Also in 2024, a helicopter-based electromagnetic survey (HeliTEM) was carried out over an area of 99.72 km^2, covering the whole Luanga mineral property. The survey was conducted by Xcalibur Multiphysics (Xcalibur) and consisted of a total of 771.2 km of lines.

1.9.3 Trenching

The trenching program started in Q4/2022 and aims to provide detailed information about the mineralized zones at the surface level. Up to the Effective Date of this report, 45 trenches were opened, totalling 9,065.73 linear meters. All opened trenches were mapped and sampled, and their channel samples were precisely surveyed with an RTK. After the work was completed, all trenches were closed. A total of 9,521 channel samples, including Quality Assurance and Quality Control (QA/QC) samples, were collected and analyzed for 3PGM and Au at independent laboratories.

1.9.4 Petrography

To enhance the geological understanding of the deposit, a petrographic study was conducted between 2022 and 2024 using 117 selected from the drill core representing the lithological diversity and multiple mineralized styles, and polished thin sections were prepared for all of them.

1.9.5 Mapping

Bravo hired PRCZ Consultores Associados (PRCZ) to carry out detailed geological and structural mapping work at the Luanga Project. This work was performed from November 2022 to July 2023.

Geological units identified in the Project area included rocks from the basement of the Carajás Domain, consisting mainly of gneiss, migmatite, and granulite terrains of the Xingu Complex, as well as metavolcanic and metaplutonic rocks of the Grão Pará Group.

1.10 Drilling

Approximately 123,610 meters in 601 drill holes have been drilled on the Property since 1992. Of these, 256 (50,787 m) are diamond drill holes (DDH) executed by DOCEGEO (Vale) and include Luanga's North, Central, and Southwest geological targets. Bravo's drilling totals 72,823.45 m and 345 holes, representing 59% of the project's drilled length (Table 1-2).

Drilling by Bravo commenced in March 2022 and has continued since then. The program was designed primarily for infill drilling and resource definition at Luanga. Bravo also performed 8 drill holes for metallurgical sample collection purposes. In 2023/2024, Bravo´s drilling program also included some exploration drill holes over several geophysical targets located outside the Luanga PGM+Au+Ni deposit. The whole diamond drilling developed by Bravo was performed using a mixture of HQ and NQ2 diameters.

Table 1-2: Drilling summary for Luanga

Year	Drill Type	Drill Holes	Total Metres	Company	Contractor
1992	DD	4	643.69	DOCEGEO	DOCEGEO
2001	DD	86	14,584.35		Geosol
2002	DD	71	15,423.25		Geosol
2003	DD	95	20,135.45		Geosol / Rede
2022	DD	135	23,258.20	Bravo	Servdrill
2023	DD	116	30,296.60		Servdrill
2024	DD	94	19,268.65		Servdrill
TOTAL		**601**	**123,610.19**		

Source: Bravo, 2025.

1.11 Sample Preparation, Analyses and Security

Bravo's QA/QC Policy is designed to ensure the reliability of exploration data and laboratory analytical results, maintaining an accurate and secure database for the project. The procedures followed include detailed processes for the transportation, verification, and analysis of samples to guarantee data integrity.

Bravo's diamond drilling campaign includes the use of Field Duplicates, Certified Reference Materials (CRMs), Blank samples, and Umpire Assay samples. Control samples (blank, CRM, and duplicate) are inserted into the analytical batch at a 1:20 ratio of regular samples. Blank and CRM samples are sourced from reputable suppliers like OREAS, AMIS, and Brasil Minas.

Bravo's QA/QC program accounts for 14,159 control samples, including Certified Reference Materials, Blank Samples, Field Duplicates and Umpire Check Assays, representing 10.7 % of the total samples.

Bravo conducted a resampling and assay campaign to validate Vale's historical data and to establish a correlation between total Ni grades and potentially recoverable sulphide-hosted Ni. Results from this campaign were entered into the drilling database, replacing the original data.

Bravo's team produces regular QA/QC internal reports to monitor the quality of assay results, ensuring consistent data quality. These reports specify which batches should be reanalyzed based on any discrepancies detected.

Although Vale's historical database did not include CRM insertion, the correlation procedures applied by Bravo have validated the database for estimation work. Despite attempts, Bravo was unable to obtain internal QA/QC results from Vale's laboratory (SGS Geosol).

Overall, Bravo's QA/QC procedures meet the industry's best practices, and the Luanga Project database is considered suitable for Mineral Resource Estimation.

1.12 Data Verification

Since 2022, GE21 team members have conducted multiple field visits to the Luanga Project to assess Bravo's exploration procedures, infrastructure, and data collection processes, ensuring the adequacy and reliability of data for the Mineral Resource estimate. Key visits took place on July 4-7, 2023; October 3-6, 2023; and January 27-31, 2025. During the last two visits, the GE21 Qualified Persons team included Geologist Bernardo Viana and Engineer Porfirio Rodriguez, an independent consultant assisting Bravo in developing the resource estimate.

Drill hole logging adheres to industry-standard practices, which Bravo has successfully established. GE21 reviewed procedures on a randomly selected set of drill cores and confirmed log completeness, with only minor omissions that were not considered significant.

Based on these observations, GE21 concludes that the exploration data at the Luanga Project is suitable for the Mineral Resource estimate, with data collection, quality control, and storage procedures meeting industry standards and no significant issues identified in Bravo's exploration activities.

1.13 Mineral Processing and Metallurgical Testing

Bravo developed metallurgical tests aimed at achieving results comparable to those of Vale's preliminary test work on PGM mineralization from the Luanga Project and recorded recommendations on metallurgical input parameters considered for the Mineral Resource Estimate (MRE). The scope of test work completed to date includes:

- Extensive comminution and flotation test work was conducted from 2022 to 2025, including a review and validation of historical work conducted by the previous project owner between 2001 and 2004.
- Several test programs were conducted on oxide material, including exploratory leaching and physical separation tests, as well as parameter optimization tests, between 2022 and 2025.
- Preliminary pyrometallurgical tests to evaluate the treatability of Luanga concentrates.

The metallurgical tests have been conducted since 2002, including by Vale, using Mintek. Laboratories in South Africa, followed by tests in CDM Vale laboratory using the circuit developed by Mintek.

Bravo resumed the metallurgical test work, from 2022 to 2023, submitting approximately three tonnes of sulphide metallurgical samples and 150kg of oxide samples to CETEM and TESTWORK laboratories in Brazil.

1.13.1 Bravo 2024/2025 Program

Ultimately, Bravo prepared a global bulk composite from the north zone of the Luanga deposit, totalling approximately 3.6 tonnes, which was constituted from diamond drill core samples.

Global composites were also prepared for the Central and SW zones of the deposit. The Central and SW composite was used in conjunction with the North composite for detailed comminution test work, while only the North composite was used in further minerals processing tests thus far.

Similar to the 2022/2023 scope, the 2024/2025 test work program was developed in phases. Laboratories used in those phases are listed below:

- Comminution – Metso Brazil
- Flotation – CETEM
- Flotation test work, North Sector – Base Metal Laboratories
- Mini Plant Program – CIT SENAI Laboratory
- Concentrate Production Run – CIT SENAI Laboratory
- Parameter Circuit Run – CETEM

1.13.2 2025 MRE Metallurgical Assumption Recommendations

The current metallurgical model shows recoveries of ca. 75 – 84% across a feed grade of 0.9 – 7.0 g/t PGM+Au, generating concentrate grades above 80 g/t PGM+Au, which is than that assumed in the 2023 MRE of 76 – 85%.

Recent locked cycle test work performed by Bravo at Base Metal Labs demonstrated 62% Ni recovery in both sets of North Sector tests at feed grades of 0.21% Ni. Locked cycle testing on the Central Sector sample demonstrated a 44% recovery from a 0.3% feed grade while producing the highest Ni grade in concentrate yet seen at Luanga at 16% Ni in concentrate (202 g/t PGM+Au in concentrate) (Table 1-3).

Table 1-3: MRE recommendation fresh sulphide rock global recovery

MRE Recommendation Fresh Rock Material	4E PGM	Pt	Pd	Rh	Au	Ni
Global Recovery (2 g/t)	78%	81%	77%	51%	48%	50%
MRE Recommendation High-Talc Domain	**-**	**Pt**	**Pd**	**Rh**	**Au**	**Ni**
Global Recovery (2 g/t)	-	51%	56%	27%	27%	0%

Source: Bravo, 2025.

It is important to note in the table below that the quantity of oxide material has decreased by almost 50% to around 5% – compared to the 10% oxide which made up the 2023 MRE. This reducing oxide tread is likely to continue as Luanga is expanded by further drilling. This reduces the significance of the oxide zone and the priority for further metallurgical study relating to the oxides.

Input assumptions for the 2025 MRE have relied on data from extensive follow-up parameter investigation test work, which has resulted in improved assumptions for Pd, similar assumptions for Au and more conservative assumptions for Pt and Rh as compared to those assumed in the 2023 MRE (Table 1-4).

GE21 Consultoria Mineral

Table 1-4: MRE recommendation oxide global recovery

MRE Recommendation Oxide Material	Au	Pd	Pt	Rh
Global Recovery (1-3 g/t)	90%	81%	23%	54%

Source: Bravo, 2025.

1.14 Mineral Resource

With the intention to simplify the Mineral Resource statement, an additional variable, based on the valuation of a calculation of palladium equivalent grade (PdEq), was created based on the following assumed metal prices and recoveries:

Metal price assumptions are based on 10-year trailing averages: Pd price of US$1,380/oz, Pt price of US$1,100/oz, Rh price of US$6,200/oz, Au price of US$1,500/oz, Ni price of US$7.10 US$/lb.

Palladium Equivalent (PdEq) Calculation

The PdEq equation is: PdEq = Pd g/t + F1 + F2 + F3 + F4

Where: *e:* $\boldsymbol{F1} = \frac{(Pt_p * Pt_R)}{(Pd_p * Pd_R)} \boldsymbol{Pt_t}$ $\boldsymbol{F2} = \frac{(Rh_p * Rh_R)}{(Pd_p * Pd_R)} \boldsymbol{Rh_t}$ $\boldsymbol{F3} = \frac{(Au_p * Au_R)}{(Pd_p * Pd_R)} \boldsymbol{Au_t}$ $\boldsymbol{F4} = \frac{(Ni_p * Ni_R)}{(Pd_p * Pd_R)} \boldsymbol{Ni_t}$

$_P$ *= Metal Price* $_R$ *= Recovery*

Several of these considerations (metallurgical recovery, metal price projections, for example) should be regarded as preliminary in nature, and therefore, PdEq calculations should be regarded as preliminary in nature.

Luanga Project's pit-constrained Mineral Resource Estimate (MRE) has an effective date of February 18, 2025, and is tabulated below.

GE21 Consultoria Mineral

Table 1-5: MRE statement at a cut-off of 0.5g/t Pd Eq*

Resource	Classification	Domain		*Average Value*						*Material Content*					
			Mass	**Pd eq**	**Pd**	**Pt**	**Au**	**Rh**	**Ni**	**Pd eq**	**Pd**	**Pt**	**Au**	**Rh**	**Ni**
			Mt	**ppm**	**g/t**	**g/t**	**g/t**	**g/t**	**%**	**koz**	**koz**	**koz**	**koz**	**koz**	**klb**
Open Pit	Measured	Ox	4	1.51	0.90	0.88	0.05	0.12	0.00	197	117	115	7	15	—
		High Talc	—	—	—	—	—	—	—	—	—	—	—	—	—
		Fresh	32	2.06	0.97	0.67	0.04	0.08	0.11	2,144	1,009	694	46	88	77,621
		Total	36	2.00	0.96	0.69	0.04	0.09	0.10	2,340	1,126	809	53	104	77,621
	Indicated	Ox	6	1.51	0.97	0.73	0.04	0.11	0.00	314	200	151	9	23	0
		High Talc	2	1.83	1.12	0.54	0.11	0.08	0.13	146	89	43	9	6	6,952
		Fresh	113	2.09	0.99	0.59	0.05	0.09	0.14	7,599	3,583	2,133	193	318	344,092
		Total	122	2.06	0.99	0.59	0.05	0.09	0.13	8,058	3,872	2,326	210	348	351,044
	Measured + Indicated	Ox	10	1.51	0.94	0.79	0.04	0.11	0.00	510	317	266	15	38	—
		High Talc	2	1.83	1.12	0.54	0.11	0.08	0.13	146	89	43	9	6	6,952
		Fresh	145	2.08	0.98	0.60	0.05	0.09	0.13	9,743	4,592	2,827	239	407	421,713
		Total	158	2.04	0.98	0.62	0.05	0.09	0.12	10,399	4,998	3,135	262	451	428,665
	Inferred	Ox	3	1.57	0.88	1.04	0.05	0.13	—	130	73	86	4	11	—
		High Talc	0	1.76	1.08	0.53	0.10	0.07	0.14	5	3	2	0	0	292
		Fresh	75	2.02	0.97	0.58	0.05	0.08	0.13	4,878	2,344	1,389	123	191	214,690
		Total	78	2.01	0.97	0.59	0.05	0.08	0.13	5,013	2,421	1,476	128	202	214,981

Notes:

1. The MRE has been prepared by Porfirio Cabaleiro Rodriguez, Mining Engineer, BSc (Mine Eng), MAIG, director of GE21 Consultoria Mineral Ltda., an independent Qualified Persons (QP) under NI43-101. The effective date of the MRE is February 18, 2025.
2. Mineral resources are reported using the 2014 CIM Definition Standards and were estimated in accordance with the CIM 2019 Best Practices Guidelines, as required by National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).
3. Mineral resources that are not Mineral Reserves do not have demonstrated economic viability. There is no certainty that all Mineral Resources will be converted into Mineral Reserves.
4. Chemical elements are estimated using different estimation methodologies according to the Weathering Model. Ordinary Kriging was applied to the Oxidized domain, while the Turning Bands Simulation was applied to fresh rock.
5. This MRE includes Inferred Mineral Resources, which have had insufficient work to classify them as Indicated Mineral Resources. It is uncertain but reasonably expected that inferred Mineral Resources could be upgraded to indicated Mineral Resources with continued exploration.
    - The Mineral Resource Estimate is reported/confined within an economic pit shell generated by Dassault Geovia Whittle software, using the following assumptions (Generated from work completed for Bravo and historical test work):
        - Metallurgical recovery in sulphide material of 77% Pd, 81% Pt, 51% Rh, 48% Au, 50% Ni to a saleable Ni-PGM concentrate.
        - Metallurgical recovery in oxide material of 81% Pd, 23% Pt, 54% Rh, 90% Au to a saleable PGM ash residue (Ni not applicable).
        - Metallurgical recovery in high-talc sulphide material of 51% Pd, 55% Pt, 27% Rh, 27% Au, 0% Ni to a saleable Ni-PGM concentrate Independent Geotechnical Testwork – Overall pit slopes of 40 degrees in oxide and 50 degrees in Fresh Rock.
        - Densities are based on 27,170 drill hole cores and 112 in situ sample density measurements. The Mineral Resources are reported on a dry density basis.
        - External downstream payability has not been included, as the base case MRE assumption considers internal downstream processing.

GE21 Consultoria Mineral

a. Payable royalties of 2%, (only considering CFEM, for reserves, a complete set of royalties must be considered)

6. Metal Pricing
   a. Metal price assumptions are based on 10-year trailing averages (2014-2023): Pd price of US$1,380/oz, Pt price of US$1,100/oz, Rh price of US$6,200/oz, Au price of US$1,500/oz, Ni price of US$7,10/lb.
   b. Palladium Equivalent (PdEq) Calculation
   c. The PdEq equation is: PdEq = Pd g/t + F1 + F2 + F3 + F4

   Where: $\boldsymbol{F1} = \frac{(Pt_p*Pt_R)}{(Pd_p*Pd_R)}\boldsymbol{Pt_t}$ $\boldsymbol{F2} = \frac{(Rh_p*Rh_R)}{(Pd_p*Pd_R)}\boldsymbol{Rh_t}$ $\boldsymbol{F3} = \frac{(Au_p*Au_R)}{(Pd_p*Pd_R)}\boldsymbol{Au_t}$ $\boldsymbol{F4} = \frac{(Ni_p*Ni_R)}{(Pd_p*Pd_R)}\boldsymbol{Ni_t}$

   $_P$ = Metal Price

   $_R$ = Metallurgical Recovery

7. Costs are taken from comparable projects in GE21's extensive database of mining operations in Brazil, which includes not only operating mines, but recent actual costs from what could potentially be similarly sized operating mines in the Carajás. Costs considered a throughput rate of ca. 10Mtpa:
   a. Mining costs: US$2.00/t oxide, US$3.00/t Fresh Rock. Processing costs: US$9.00/t fresh rock, US$7.50/t oxide. US$1.50/t processed, for General & Administration. US$1.00/t processed for grade control. US$0.50/t processed for rehabilitation.
   b. Several of these considerations (metallurgical recovery, metal price projections, for example) should be regarded as preliminary in nature, and therefore, PdEq calculations should be regarded as preliminary in nature.

8. The current MRE supersedes and replaces the Previous Estimate (2023), which should no longer be relied upon.
9. The QP is not aware of political, environmental, or other risks that could materially affect the potential development of the Mineral Resources other than those typical for mining projects at this stage of development and as identified in this report.
10. Totals may not sum due to rounding.

Source: GE21, 2025.

1.15 Mineral Reserve Estimates

No Mineral Reserves have been declared.

1.16 Mining Methods

The Project contemplates conventional open-pit mining using drill-and-blast, loading, and hauling operations. The mine plan is based on open pit optimization, final pit design, and phased scheduling, considering Measured, Indicated, and Inferred Mineral Resources.

Mining operations will be conducted in 10 m benches, with 1-year ramp-up and phase sequencing designed to prioritize higher-grade zones and lower strip ratios in the early years, improving early cash flow and project payback.

The final pit design supports a total mine life of 17 years. The mining plan was based on a 10 Mt per year ROM production, and the total mining rate is expected to reach up to 103.6 Mt per year (including ROM, waste, and re-handling), supported by a fleet of up to six hydraulic excavators, 44 off-highway trucks (up to 242 t capacity), and nine drill rigs. Although mining will be contractor-operated, fleet sizing was conducted to ensure production targets are met safely and efficiently.

Waste material will be hauled to a Waste Rock Storage Facility (WRSF), while tailings will be filtered and disposed of on a Dry Stacking Facility (DSF) within the project site. Oxidized mineralized material is currently excluded from the mine plan and will be stockpiled for potential future evaluation.

Supporting infrastructure was considered in the layout to control environmental impacts and enable safe long-term operations.

1.17 Recovery Methods

The processing is based on metallurgical testwork and is conceptually estimated for fresh mineralized material. Initial plant capacity is planned at 5 Mtpa, ramping up to 10 Mtpa in Year 2.

Run-of-mine (ROM) material is delivered to a primary crushing area, equipped with a grizzly and MMD crusher. Crushed material is transported via a long-distance belt conveyor to a coarse stockpile, then reclaimed and processed through a three-stage comminution circuit comprising a SAG mill, an open-circuit secondary ball mill, and two tertiary ball mills in closed circuit with hydrocyclones.

Classification targets a final grind of 28 μm. Cyclone overflow is thickened to increase solids content before flotation. Flotation includes rougher, scavenger, cleaner, and recleaner stages, with tailings sent to filtration and dry stacking. Final concentrate is thickened, stored, and filtered via pressure plates, targeting an annual production of 184 kt of dry concentrate.

Tailings will be filtered and deposited in a Dry Stacking Facility (DSF). The DSF will have a storage capacity of 65.7 Mm^3 over an area of 122 ha. Stacking will follow defined drainage slopes and compaction protocols, including roller passes, moisture control, and field testing.

A 15-m-high water dam within the project area will form a reservoir with 2.7 Mm^3 capacity, sufficient for average operational demand. A contingency system will allow diversion of up to 136 m^3/h from the Sereno River via a 4.9 km pipeline.

Process circuits will rely on recirculated water recovered from tailings and concentrate thickeners, stored in a process water tank and distributed throughout the plant to minimize raw water use.

1.18 Project Infrastructure

The Project covers an area of approximately 1,600 hectares and will include key infrastructure required for mining and processing operations. Main facilities comprise an open pit, haul and access roads, a WRSF, a DSF, settling ponds, process water treatment, a tailings dam for the first two years of operations, a water reservoir, maintenance shops, administrative buildings, explosives storage, and a main electrical substation connected via a 230 kV transmission line.

The Project is accessible by paved roads from either Marabá (134 km) or Curionópolis (32 km). Marabá and Parauapebas offer commercial flight connections. On-site roads will support haulage and internal circulation.

Electrical power will be supplied via a 230 kV transmission line for 35 km connecting the Project to the Carajás Substation. The system is designed for a peak load of 100 MW. The main substation, located near the processing plant, will reduce voltage from 230 kV to 13.8 kV and distribute it across the facility. The substation includes high-voltage pads, control rooms, emergency power, protection systems, and fire safety infrastructure.

Mine water will be managed using perimeter drainage ditches, sedimentation sumps, and pit dewatering pumps. Excess water, once treated, will be discharged in accordance with environmental regulations. Process water will operate in a closed circuit, minimizing consumption. Raw and potable water will be sourced from wells and a planned 2.7 Mm^3 water dam.

Tailings will be deposited in a conventional tailings dam during the initial two years and later managed by dry stacking. The WRSF and DSF are designed for Life-of-Mine capacity with a 5% volume contingency. Drainage systems and settling ponds will mitigate sediment runoff.

The site will include administrative offices, workshops, medical and security posts, cafeteria, locker rooms, control room, and laboratory. A communications system using a hybrid fiber and copper backbone will support operations, along with radio networks across key locations. Support areas will include maintenance facilities, storage warehouses, and staff amenities designed to ensure operational efficiency, safety, and regulatory compliance.

Figure 1-3 presents the Project's Master Plan.

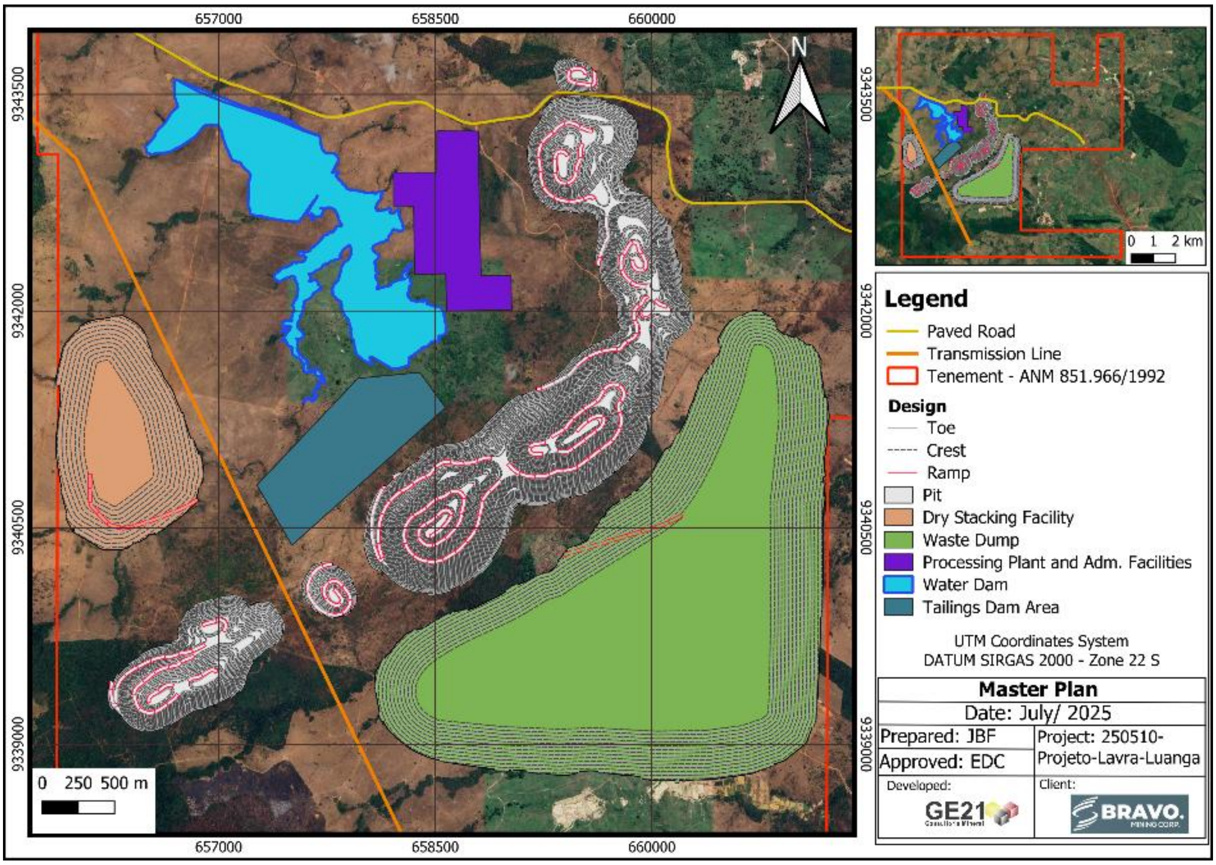


Figure 1-3: Master Plan

Source: GE21, 2025.

1.19 Market Studies and Contracts

The Project is expected to enter the market during a period of structural deficits in several key metals—particularly platinum, palladium, rhodium, gold, and nickel. Demand for PGMs remains robust, driven by the automotive and hydrogen sectors, while global supply is constrained by geopolitical instability and declining production in South Africa. The Project will produce a concentrate containing PGMs (Pt, Pd, Rh), Au, and Ni, and is strategically positioned near Atlantic ports with access to Southern African, Chinese, European, and North American markets. These factors enhance its strategic value amid tightening global supply.

PGM concentrates are readily accepted by global smelting and refining facilities, with established infrastructure in South Africa, Europe, and growing capacity in Asia and North America. Contracts typically address payable metal percentages, TC/RCs, deleterious elements, logistics, and dispute resolution clauses. The Project's concentrate is expected to meet smelter specifications, with low Cr_2O_3 and acceptable MgO levels for conventional electric arc furnace processing.

GE21 Consultoria Mineral

Platinum demand is supported by both industrial uses (particularly fuel cell electric vehicles) and jewelry, especially in Asia. Supply is concentrated in South Africa and Russia, with declining output highlighting the need for new producers.

Palladium is key in automotive emission control, Pd demand remains high despite substitution efforts. Supply risks are elevated due to geopolitical factors and aging Russian and South African mines.

Rhodium is a niche but high-value metal essential for NOx reduction in vehicles. Even minor supply changes significantly impact price. The Project's rhodium content is economically significant despite low tonnage.

Gold is a globally liquid commodity with pricing influenced by macroeconomic conditions. It represents less than 3% of the Project's revenue.

Nickel demand is rising due to EV battery applications, in addition to stainless steel. The Project's nickel output is expected to achieve 75–90% payability depending on form and destination.

Preliminary offtake discussions have been initiated with international refiners. A flexible strategy is planned, including tolling, long-term sales contracts, and potential local downstream partnerships. Commercial terms will be optimized to maximize revenue from all payable metals, with consideration for advance payments and hedging strategies.

The economic model uses long-term, real pricing assumptions aligned with consensus forecasts, as presented in Table 1-6.

Table 1-6 –Metal price deck

Commodity	Luanga 2025 PEA Price Deck	Source	Spot price at time of determination	Unit
Palladium	1,271	Investec LT Real June 2025	1,172	US$/oz
Platinum	1,500	Investec LT Real June 2025	1,444	US$/oz
Rhodium	6,000	GE21	5,540	US$/oz
Gold	3,251	Consensus Economics LT Real June 2025	3,336	US$/oz
Nickel	17,637	Investec LT Real June 2025	15,100	US$/t

Source: various, as presented in column #3.

1.20 Environmental Studies, Permitting, and Social or Community Impacts

The Project is in the Preliminary License (LP) application phase, with the Environmental Impact Assessment (EIA/RIMA) submitted to SEMAS-PA in June 2024. Environmental studies conducted between 2022 and 2024 followed the approved Terms of Reference. These included surveys on flora, fauna, water resources, hydrogeology, and local socioeconomic conditions.

Environmental licensing in Brazil follows a three-stage process (Preliminary License - LP, Instalation License - LI, Operation License - LO) and is governed by federal and state laws. In addition to the LP, the Project will require permits for water use and vegetation suppression. The area is mostly anthropized, with pastures dominating land use and some remaining forest fragments hosting protected species.

Socioeconomically, the Project is expected to stimulate the local economy through job creation and tax revenue. Negative impacts such as vegetation loss and hydrological alterations will be mitigated by a comprehensive set of environmental and social programs. A closure plan has been developed in line with Brazilian and international standards.

1.21 Capital and Operating Costs

The capital cost estimate was based on GE21 database, usual indexes from mineral industry (Cost Mine Magazine) and public reports from similar operations in the region. Due to the methodology used to develop the capital estimate and the conceptual level of engineering definition, the estimate has an accuracy of -30% +50%.

The total CAPEX (Capital Expenditures) for the base-case is estimated in US$ 592.9, comprising US$ 495.8 refers initial investment and US$ 97.1 in sustaining capital. The capital cost summary is presented in Table 1-7.

Table 1-7: Capital Costs Estimates

CAPEX Summary (MUSD)			
AREA	TOTAL	INITIAL	SUSTAINING
Mine	59.4	36.7	22.7
Plant & Dry Stack	401.9	319.2	74.4
Infrastructure	19.4	19.4	-
Transmission Line and Substation	17.3	17.3	-
Indirect	94.8	94.8	-
Total CAPEX (Base Case)	**592.9**	**495.8**	**97.1**
Mine Closure	17.9		

Source: GE21, 2025.

Operating costs (OPEX) include the ongoing cost of operations related to mining, processing, tailings disposal and general administrative (G&A) costs. It covers all direct costs necessary for the operation of the enterprise.

Table 1-8 summarizes the operational costs estimated for the Project.

Table 1-8: Operational costs summary.

Description	Unit	OPEX
Mine	US$/t processed	22.80
Process	US$/t processed	12.12
Freight	US$/t processed	0.94
G&A	US$/t processed	5.00
Total	**US$/t processed**	**40.86**

Source: GE21, 2025.

1.22 Economic Analysis

The economic evaluation was based on a discounted cash flow model, incorporating post-tax cash flows and assuming an 8% discount rate. The analysis considered a fixed exchange rate of BRL 5.80 / 1.00 US$, projected mine plan and recoveries, and constant-dollar assumptions (i.e., no inflation). All production is assumed to be sold in the year of production.

Revenues are derived from a multi-metal basket including PGM (Pd, Pt, Rh), gold, and nickel. Tax assumptions include CFEM royalties (1.5% for gold and 2% for other metals), a simplified 25% corporate income tax, and a 9% social contribution. The model also includes the TFRM fee applied in Pará State and contractual royalties to third parties, including BNDES (2.0%), Vale (1.0%), and surface rights holders (up to 1.0%).

Working capital and depreciation were estimated using simplified assumptions based on typical parameters for similar projects.

The cash flow results are pretented in Table 1-9, and provide a preliminary assessment of the project’s financial viability.

Table 1-9 – Cash Flow Results

Discount Rate	8%
NPV (US$ X 1000)	$ 1,249
IRR (%)	49%
Discounted Payback After-Tax (years)	2.42

Source: GE21, 2025.

1.23 Adjacent Properties

Within a 10 km radius of the Project, two main mineral deposits are of note: the Serra Pelada Au + PGM deposit and the Serra Leste iron ore deposit. These deposits are geologically unrelated to Luanga, as they are not associated with mafic-ultramafic intrusions. Serra Pelada, located 8 km west, was historically Brazil’s largest gold mine during the 1980s, with over 1 Moz Au extracted through artisanal mining. Serra Leste, approximately 8.5 km southwest, is an active iron ore operation owned by Vale.

The region surrounding is fully allocated, with no open ground available for new exploration claims. Vale is the dominant tenement holder in the area.

1.24 Other Relevant Data and Information

For effect of this PEA, a second scenario was developed, considering the on-site smelting of the flotation concentrate.

The proposed smelter is estimated to process up to 200,000 tonnes/year (wet basis) of PGM-rich nickel sulphide concentrate, yielding up to 800,000 ounces of PGM, 16,000 tonnes of nickel, and 120,000 tonnes of sulphuric acid. The selected metallurgical route includes a fluidized bed roaster and a DC electric arc furnace (EAF), producing a PGM-rich ferronickel alloy for subsequent refining. Sulphur is recovered and converted to sulphuric acid.

An internal technology review assessed various smelting and hydrometallurgical options, considering technical feasibility, recovery efficiency, environmental impact, and economic viability.

DC reductive EAF smelting was identified as the most suitable option and confirmed by preliminary bench-scale tests in collaboration with Arxo Metals (South Africa).

Table 1-10 and Table 1-11 present the estimated smelter CAPEX and OPEX.

Table 1-10: Smelter CAPEX estimates summary

TOTAL SMELTER CAPEX ESTIMATE	
Item	**Total (US$)**
Major Equipment	83.5
Auxiliary Equipment	25.0
Shipping & Logistics	5.5
Engineering, Construction & Installation	44.0
Contingency (15%)	23.9
Total CAPEX	**181.9**

Source: Bravo, 2025.

Table 1-11: Smelter OPEX estimates summary

SMELTER OPERATING COST			
Category	**Annual Cost ($M)**	**Unit Cost (US$/t)***	**% of Total**
Feed (Variable)	1,023.6		91.6%
Variable Costs	37.2	186.00	3.3%
Fixed Costs	17.7	88.50	1.6%
Byproduct Credits	(2.5)		-0.2%
Total OPEX	1,076.0	274.50	100%

*** Cost per tonne of concentrate.**

Source: Bravo, 2025.

The smelter scenario cash flow results are presented in Table 1-12

Table 1-12 - Results

Discount Rate	8%
NPV (US$ X 1000)	$ 1,861
IRR (%)	49%
Discounted Payback After-Tax (years)	2.43

Source: GE21, 2025.

1.25 Interpretation and Conclusions

1.25.1 Mineral Exploration and Geology

In general terms, the geological descriptions, sampling procedures and density tests that were evaluated were found to be of acceptable quality and in accordance with industry best practices.

It was noted that the data collection process was executed with the aim of maintaining data security. Data was stored in a standardized database, which was found to be secure and auditable.

GE21 reviewed the process through which density was determined and concluded that it was in conformity with industry best practices.

1.25.2 QA/QC

GE21 performed the evaluation of the QA/QC data, which includes Blanks, CRMs, Field Duplicates, Check Assays, and Umpire Check Assays.

QA/QC procedures, sampling methodology, and analytical methods applied by Bravo are within the industry's best practices standard. The QP responsible for this report, considering the data presented in Section 11, is of the opinion that the Luanga Project's Database is suited for Mineral Resource Estimation work.

1.25.3 Geological Model

The procedure that was adopted to produce the 3D geological models (wireframes), consisting of generating triangulations between interpreted geological cross sections, was executed properly and in accordance with the opinions of GE21 staff.

1.25.4 Grade Estimation

The heterogeneity of the geological model leads GE21 to select the Turning Bands Simulation method to estimate the grades for the Luanga Project.

The variograms that were used in the estimation method are satisfactory and consistent with respect to the grade estimation that was calculated via Simulation (E-Type), making use of search anisotropy determined in the variography study. A valid conditional simulation in geostatistics ensured that simulated values honour both spatial continuity and data distribution.

To classify Mineral Resources, a study of spatial continuity for Pd Equivalent was conducted using variography followed by ordinary kriging interpolation. This study established a continuity zone suitable for considering:

- The Measured Mineral Resource was classified according to a reference grid of approximately 45mx45m, with a minimum number of 3 holes in the section along the strike and dip directions, surrounded by the pit shell.
- The Indicated Mineral Resource classification had as a reference a drilling grid of approximately 75m x 75m, extending both along the strike and dip directions and requiring a minimum of two drill holes.
- Manual post-processing was undertaken to construct wireframes representing the volumes categorized as Measured and Indicated while considering the blocks within the resource pit shell.
- The Inferred Mineral Resource classification is all remaining estimated blocks within the resource pit shell.

GE21 considers the Mineral Resource classification model and the analysis of criteria for the classification of those Mineral Resources to be satisfactory, although some recommendations have been made for future improvements.

GE21 Consultoria Mineral

1.25.5 Mineral Resource Estimate

The Luanga Project's pit-constrained MRE has an effective date of February 18, 2025. In summary, It comprises 36 Mt at 2.00 g/t Pd Eq for a total of 2.3 Moz Pd Eq in the Measured category, 122 Mt at 2.06 g/t Pd Eq for 8.0 Moz Pd Eq in the Indicated category, 158 Mt at 2.04 g/t Pd Eq for a total of 10.4 Moz Pd Eq in the Measured + Indicated categories, and 78 Mt at 2.01 g/t Pd Eq for a total of 5.0 Moz Pd Eq in the Inferred category.

1.25.6 Mine Plan

The mine plan was developed based on the Mineral Resource estimate and an open pit optimization process. The final pit design resulted in 113 Mt at 2.68 g/t Pd_Eq of Measured + Indicated Resources, and 52 Mt at 2.59 g/t Pd_Eq of Inferred Resources, using a 0.87 g/t Pd_Eq cut-off, 95% mining recovery and 5% mining dilution factors. Waste within the pit design sum 1,124 Mt, and Low Grade material of 30 Mt, resulting in 7.0 strip ratio.

The mining schedule was developed with a focus on operational features of the mining operations and optimizing concentrate production. The production rate was defined as 10.0 Mt/year of run-of-mine delivered to the processing plant, including a ramp-up period of 1 year at 50% the full production rate. The mine life is estimated at approximately 17 years.

Drilling and blasting operations are expected for production purposes. A staggered blast pattern forming equilateral triangles was selected for the blast design, as it provides optimal distribution of explosive energy and promotes efficient rock fragmentation.

The ROM and waste will be loaded into rigid frame off-road trucks and hauled to the processing plant. The waste will be disposed in a Waste Rock Storage Facility, and filtered tailings will be disposed within a Dry Stacking Facility.

1.25.7 Mineral Processing

The recovery method proposed for the Project is based on the results of initial metallurgical testing carried out since and benchmarking against similar operations.

For the purposes of this Report, the selected metallurgical route is based on the most recent locked cycle tests (LCT) campaign. Although previous processing routes and historical tests results have been referenced, the selected route was developed specifically for the concentration of hard rock, given the limited amount of oxidized material. Data from the previous testworks should be taken into account during the development of the feasibility study.

The processing plant is planned to operate initially at 5 Mtpy, ramping up to 10 Mtpy by the second year of operation. The proposed plant includes a three-stage grinding circuit followed by a flotation process composed of rougher, scavenger, cleaner, and recleaner stages. Concentrate is expected to be produced at a rate of 184,000 dry tonnes per year.

While the proposed recovery method is robust and consistent with the characteristics of the mineralization, further metallurgical testwork is planned to support detailed engineering and equipment selection, particularly in grinding, thickening, and filtration stages.

1.25.8 Infrastructure

The main infrastructure for the Project includes the processing plant, Waste Rock Storage Facility, and Dry Stacking Facility, tailings dam (for the first 2 years of operation), and water dam, covering an area of approximately 1,600 hectares.

Supporting infrastructure comprises site access roads, settling ponds, maintenance facilities (truck shop, plant workshop and warehouse), process water treatment facility, administrative buildings (admin & finance, management, engineering and geology offices, support services, parking lot, restrooms, security, medical post, HSEC (Health, Safety, Environment & Communities), access gate, laboratory, explosives magazine, main electrical substation, 230 kV transmission line, water catchment system, communication system, and control room.

1.25.9 Environmental, Permitting, and Social Considerations

The Project is advancing through the environmental licensing process in accordance with Brazilian legislation. Following the issuance of the Terms of Reference by SEMAS-PA, the EIA/RIMA studies were completed between 2022 and 2024 and formally submitted in June 2024 to support the application for the Preliminary License (LP) which was granted on to Bravo by the Pará State Environmental Agency in February 2025. These studies covered both climatic seasons and addressed key aspects such as fauna, flora, hydrology, hydrogeology, and socioeconomic conditions. Additional permitting requirements, such as the Water Use Grant and Vegetation Suppression Authorization, will be addressed in subsequent phases.

The area of influence is predominantly anthropized, with pasture and rural uses, and does not overlap with protected areas or traditional communities. Socioeconomic conditions in Curionópolis reflect a dependence on mining and cattle ranching, with limited health infrastructure and partial basic sanitation coverage. The Project is expected to generate positive socioeconomic impacts, including job creation, local economic growth, and tax revenue, particularly benefiting Curionópolis and neighboring municipalities.

The Project also presents environmental challenges, notably the suppression of native vegetation and changes to the local hydrological regime due to water reservoir construction. Mitigation strategies include a comprehensive set of environmental and social programs aligned with ESG principles, as well as a mine closure plan following national and international standards. The plan provides for environmental rehabilitation, structural stability, and long-term monitoring.

1.25.10 Capital and Operating Costs

The total Base Case Project CAPEX over the 17-year operational life is estimated at US$ 592.9 million, comprising US$ 495.8 million in Initial CAPEX and US$ 97.1 million in Sustaining CAPEX. These estimates include the following components:

- Site Preparation
- Processing Plant
- Infrastructure:
  - Transmission Line and Substation
  - Maintenance Facilities
  - Administrative Buildings
  - Internal Roads
  - External Roads
  - WRSF and DSF
- Mine Closure
- Indirect costs:
  - Engineering
  - Management
  - Spare Parts
  - Commissioning
  - Supervision
  - Initial Plant Charge

The total Base Case Project OPEX over the 17-year operational life is estimated at US$ 6,755 million. The average OPEX was estimated at US$ 40.86/t processed. These estimates include the following components:

- Mining
- Processing
- General and administrative
- Logistics

1.25.11 Economic Analysis

The preliminary economic analysis demonstrates that the Project has sufficient economic potential under the Base Case scenario to warrant advancing the Project towards a PFS or FS. At average prices of US$1,271/oz Pd, US$1,500/oz Pt, US$6,000/oz Rh, US$3,251/oz Au, and US$8.00/lb Ni, the after-tax NPV (8%) is estimated at US$ 1,249 million, an IRR of 49% and a payback period of 2.4 years. These results indicate a financially attractive opportunity, supporting continued advancement of the Project.

GE21 Consultoria Mineral

1.25.12 Alternate Case (Vertical Integration)

A second scenario was developed, considering the on-site smelting of the flotation concentrate, with capacity to treat up to 200,000 tonnes wet basis of PGM-rich, nickel sulphide concentrate, up to 800,000 ounces of PGM, 16,000 of nickel, and 120,000 tonnes of sulphuric acid annually. The metallurgical process comprises a single line fluidized bed roaster and direct-current electric arc furnace producing a PGM-enriched, ferronickel alloy for further precious and base metal refining. Extracted sulphur will be processed to sulphuric acid in a sulphuric acid plant.

The total CAPEX for the on-site facilies is estimated at US$ 181.9 million, over and above the CAPEX estimate for the Base Case, including the following components:

- Auxiliary Equipment
- Shipping & Logistics
- Engineering, Construction & Installation

The annual incremental OPEX for the smelter facilies is estimated at US$ 54.9 million, over and above the OPEX for the Base Case, resulting in an average OPEX of US$ 274.50/t concentrate. These estimates do not include feed costs and byproduct credits, and include the following components:

- Energy and water
- Reagents and consumables
- Labor
- Maintenance and overheads

The Alternate Case preliminary economic analysis results in after-tax NPV (8%) estimated at US$ 1,861 million, an IRR of 49% and a payback period of 2.4 years.

1.26 Recommended Work Program

The recommended work program comprises:

PHASE 5A – Metallurgical testwork at Luanga

Completion of any outstanding metallurgical testwork and optimization:	
Estimate	US$0.50M
Continuation of carbon sequestration study:	
Estimate	US$0.05M
Continued Metallurgical testwork and optimization	
Estimate	US$0.20M
Sub-total – Phase 5A	***US$0.75M***

PHASE 5B – Prefeasibility Study ("PFS") following favorable results from a PEA

Completion of a PFS:	
Estimate	US$1.0M
Sub-total – Phase 5B	***US$1.0M***

PHASE 5C – Deep Drilling below the Luanga PGM+Au+Ni deposit

Deep drilling at the Luanga PGM+Au+Ni deposit.

8 holes @ ~500m = 4,000m @ US$450/m

Estimate	US$1.8M
Sub-total – Phase 5C	***US$1.8M***

Phase 5D – Regional Exploration

Exploration of new (IOCG and/or massive sulphide Ni/Cu/ PGM targets):

Geological, geophysical and drilling programs to evaluate the potential for the discovery of additional zones of mineralization:

Geophysics	US$0.1M
Drilling: 70 holes x 200m for 14,000m @ US$400m (all inclusive)	US$5.6M
Sub-total – Phase 5D	***US$5.7M***
TOTAL – PHASE 5	***US$9.25M***

PHASE 6 – Feasibility Study (“FS”) following favorable results from a PFS

Completion of a FS, including any required infill drilling, geotechnical drilling:

Estimate	US$5.0M
TOTAL – PHASE 6	***US$5.0M***

Source: GE21, 2025.

2 INTRODUCTION

GE21 Consultoria Mineral Ltda. (GE21), headquartered in Belo Horizonte, Minas Gerais, Brazil, is an independent mineral consulting company composed of a team of professionals qualified to report Mineral Resources and Reserves in accordance with the guidelines of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) code, as required by "National Instrument 43-101 – Standards of Disclosure for Mineral Projects" (NI 43-101).

Bravo Mining Corp. (TSX.V: BRVO, OTCQX: BRVMF) is a Canadian and Brazil-based mineral exploration and development company focused on advancing its Luanga PGM+Au+Ni (Platinum Group Metals, Gold and Nickel) Project in the Carajás Mineral Province of Brazil. The Company's head office is located at Av. Jornalista Ricardo Marinho, nº. 360, room 247, Barra da Tijuca, Rio de Janeiro, RJ, Brazil, Zip code 22631-350 and its registered office is located at Bentall 5, 550 Burrard Street, Suite 2501, Vancouver, British Columbia, V6C 2B5.

Bravo has commissioned GE21 to prepare a Preliminary Economic Report (PEA) Technical Report for the Luanga Project in Pará, Brazil, in accordance with the requirements of NI 43-101.

The Effective Date of July 7th, 2025, is based on the receipt date for the Project database. Bravo indirectly owns 100% of the Luanga Project. The organizational structure of Bravo and ownership of the Luanga Project is shown in Figure 2-1.

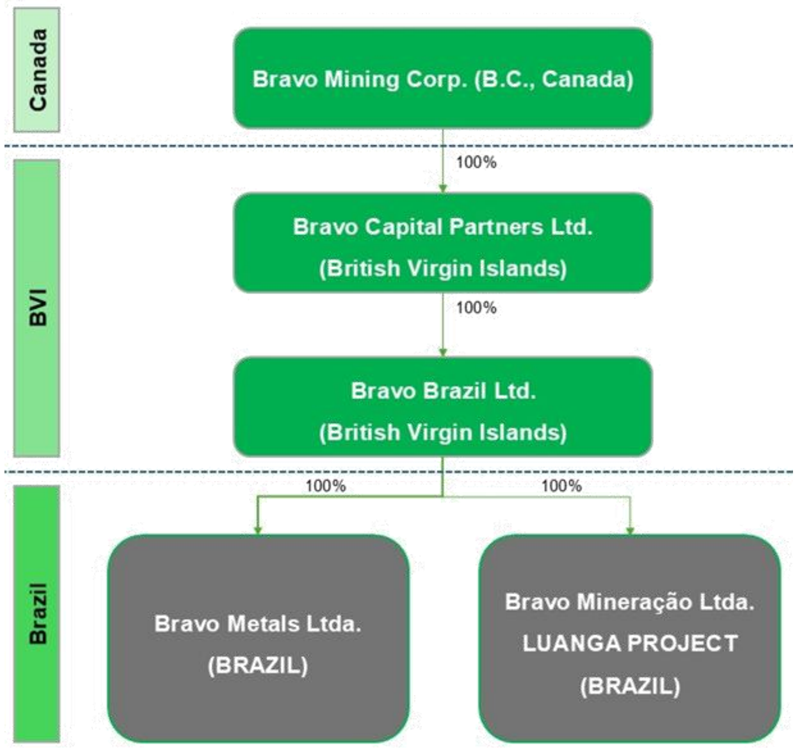


Figure 2-1: Bravo Organization Chart

Source: Bravo, 2025.

2.1 Terms of Reference

This report was prepared as a NI 43-101 Technical Report for Bravo, on the Luanga Project.

The quality of information, conclusions, and estimates contained herein are based on information available at the time of preparation provided by the Bravo's staff and the assumptions, conditions, and qualifications set forth in this report.

2.2 Qualifications, Experience and Independence

GE21 is a specialized, independent mineral consulting company. The geological reconnaissance, due diligence evaluation, technical parameters and economic assessment have been conducted by GE21 staff members, who are members of the Australian Institute of Geoscientists (AIG) and/or the Australasian Institute of Mining and Metallurgy (AusIMM) and meet the requirements of independent QPs (Qualified Persons) as defined in NI 43-101.

Neither GE21 nor the Authors of this Technical Report have had any material interest invested in Bravo or any of its related entities. GE21's and the Author's relationship with Bravo is strictly professional, consistent with that held between a client and an independent consultant. This report was prepared in exchange for payment based on fees that were stipulated in a commercial agreement. Payment of these fees is not dependent on the results of this report.

2.3 Qualified Persons

The QPs responsible for this independent Technical Report are Mr. Porfirio Cabaleiro Rodriguez, Mr. Bernardo Viana, Mr. Paulo Roberto Bergmann Moreira, Mr. Juliano Felix de Lima and Mr. Eduardo Dequech de Carvalho.

Mr. Porfirio Cabaleiro Rodriguez is one of the QPs regarding the objectives of this report. Mr. Porfirio was responsible for Sections 2, 3, 4, 5, 6, 13, and 14, with co-responsibility for Sections 1, 11, 12, 25, 26 and 27. Mr. Rodriguez is an engineer and a FAIG, and has sufficient experience relevant to the style of mineralization and type of deposit under consideration. Mr. Rodriguez has more than 40 years of experience working with exploration and mining projects. Mr. Rodriguez is considered as an independent QP, as defined in NI 43-101.

Mr. Bernardo Viana is also QP responsible for this Report. Mr. Viana was responsible for Sections 7, 8, 9, 10, 20 and 23, with co-with responsibility for Sections 1, 11, 12, 25, 26 and 27. Mr. Viana is a geologist and a FAIG and has sufficient experience relevant to the style of mineralization and type of deposit under consideration for being considered an independent QP, as defined in NI 43-101. Mr. Viana has more than 20 years of experience working with exploration and mining projects.

Mr. Paulo Roberto Bergmann Moreira is also QP responsible for this Report. Mr. Moreira was responsible for Sections 17 and 24, with co-with responsibility for Sections 1, 21, 25, 26 and

GE21 Consultoria Mineral

27. Mr. Moreira is an engineer and a FAusIMM, with over 42 years' experience in mineral processing and mineral reserve estimate, and has sufficient experience relevant to the style of mineralization and type of deposit under consideration.

Mr. Juliano Felix de Lima is also QP responsible for this Report. Mr. Lima was responsible for Section 18, with co-with responsibility for Sections 1, 21, 25, 26 and 27. Mr. Lima is an geology engineer and a MAIG, with over 25 years' relevant experience in exploration, resource and reserves estimation and open pit and underground mining, including numerous mineral properties in Brazil, and has sufficient experience relevant to the style of mineralization and type of deposit under consideration.

Mr. Eduardo Dequech de Carvalho is also QP responsible for this Report. Mr. Carvalho was responsible for Sections 16, 19, and 22, with co-with responsibility for Sections 1, 21, 25, 26 and 27. Mr. Carvalho is an engineer and a MAusIMM, with 7 years of experience in mineral reserve estimation and mine planning, and has sufficient experience relevant to the style of mineralization and type of deposit under consideration.

Table 2-1 presents each QP with their report items' responsibilities, and Appendix A presents the Certificates of QP.

Table 2-1: Report Items and assigned QP responsibilities

Company	QP	Section Responsibility	Site Visit	Responsibility
GE21	Porfirio Cabaleiro Rodriguez, FAIG	2, 3, 4, 5, 6, 13, and 14, with co-responsibility for Sections 1, 11, 12, 25, 26 and 27	July 4 to 7, 2023; October 3 to 6, 2023; and January 27 to 31, 2025	Author
GE21	Bernardo Viana, FAIG	7, 8, 9, 10, 20 and 23, with co-with responsibility for Sections 1, 11, 12, 25, 26 and 27	October 3 to 6, 2023; and January 27 to 31, 2025	Author
GE21	Paulo Roberto Bergmann Moreira, FAusIMM	17 and 24, with co-with responsibility for Sections 1, 21, 25, 26 and 27	-	Author
GE21	Juliano Felix de Lima, MAIG	18, with co-with responsibility for Sections 1, 21, 25, 26 and 27	-	Author
GE21	Eduardo Dequech de Carvalho, MAusIMM	16, 19, and 22, with co-with responsibility for Sections 1, 21, 25, 26 and 27	-	Author

Source: GE21, 2025.

2.4 Site Visits and Details of Inspection

Mr. Rodriguez visited the property from July 4 to 7, 2023, October 3 to 6, 2023, and January 27 to January 31, 2025. On the last two visits, GE21's qualified persons team was composed of Mr. Rodriguez and Mr. Viana. During the site visit, some diamond drill collars were located, their recorded coordinates were validated with a handheld GPS (Global Positioning System), and the core was inspected in the onsite core storage facility.

Also relevant aspects of the project, such as topographic, vegetation, water and power supply conditions, accessibility, nearby communities, and general location were assessed by the QPs.

2.5 Effective Dates

The Effective Date of this report is July 7th, 2025. The Authors have relied on information provided by Bravo, which was provided in a database with full access given to the QPs.

2.6 Previous Technical Reports

This PEA builds upon the results presented in the independent Mineral Resource Estimate (MRE) report titled 'Independent Technical Report on the Luanga PGM + Au + Ni Project, Pará State, Brazil', with an effective date of February 18, 2025. The MRE was prepared following the National Instrument 43-101 - Standards of Disclosure for Mineral Projects of the Canadian Securities Administrators (NI 43-101) requirements. The current PEA uses the resource estimate as a foundation for evaluating the potential economic viability of the project. No additional drilling or geological modeling was performed after the effective date of the MRE.

GE21 conducted a previous Mineral Resource Estimate (MRE) for Bravo, accompanied by a pit optimization study, to classify the Mineral Resources. This technical report is titled "Independent Technical Report on Resources estimate for Luanga PGM + Au + Ni Project Pará State, Brazil" dated December 1, 2023 with an effective date of October 22, 2023, and has been superseded by the current MRE mentioned on the above paragraph.

2.7 Currency, Units, and Definitions

Unless otherwise stated, the units of measurement in this report are compliant with the International System of Units (SI). All currency is in United States dollars (US$) unless otherwise indicated. An exchange rate of R$ 5.80/US$ 1.00 was used based on parameters adopted in international projects.

The UTM (Universal Transverse Mercator) projection, Zone 22 South, SIRGAS2000 (Geocentric Reference System for the Americas) datum was adopted as a spatial reference.

3 RELIANCE ON OTHER EXPERTS

3.1 Introduction

The authors of this Report are QPs, as defined under NI 43-101, with relevant experience in mineral exploration, data validation, and Mineral Resource estimation.

The information presented regarding the tenure, status and work permitted by permit type within the Bravo property in Section 4 – Property Description and Location, is based on information published by the National Mining Agency of Brazil (Agência Nacional de Mineração, ANM) and is available to the public.

Bravo retained Linneu de Albuquerque Mello, whose lawyers are qualified to practice law in the Federative Republic of Brazil. According to a title opinion by Linneu de Albuquerque Mello dated 31th January, 2025, the Luanga Mineral Rights were valid and in good standing at that time.

The environmental licensing status information and work plans related to community and social outreach included in Section 20 – Environmental Studies, Permitting and Social or Community Impact, were compiled by GE21, based on information provided by Bravo.

GE21 determined that the economic factors used in the determination of specific technical parameters of this Report, including gold, PGM, nickel and the US$: BRL assumptions used, were in line with industry norms and broader market consensus and are acceptable for use in the current report.

The authors of this Report have not identified any significant risks in the underlying assumptions as, in addition to the above, the underlying assumptions are in line with spot market conditions as of the date of this Report.

GE21 Consultoria Mineral

4 PROPERTY DESCRIPTION AND LOCATION

Luanga is an advanced-stage mineral project comprising the Luanga deposit in Pará State, Brazil, which contains PGM+Au+Ni. It is held under the Exploration Licence Nº.1961 and designated ANM 851.966/1992, comprising an area of 7,810.02 hectares in extent. The Project is located in the municipality of Curionópolis in the central-eastern region of Pará State, approximately 500 km south of Belém (a sizeable coastal port city and capital of Pará, Figure 4-1).

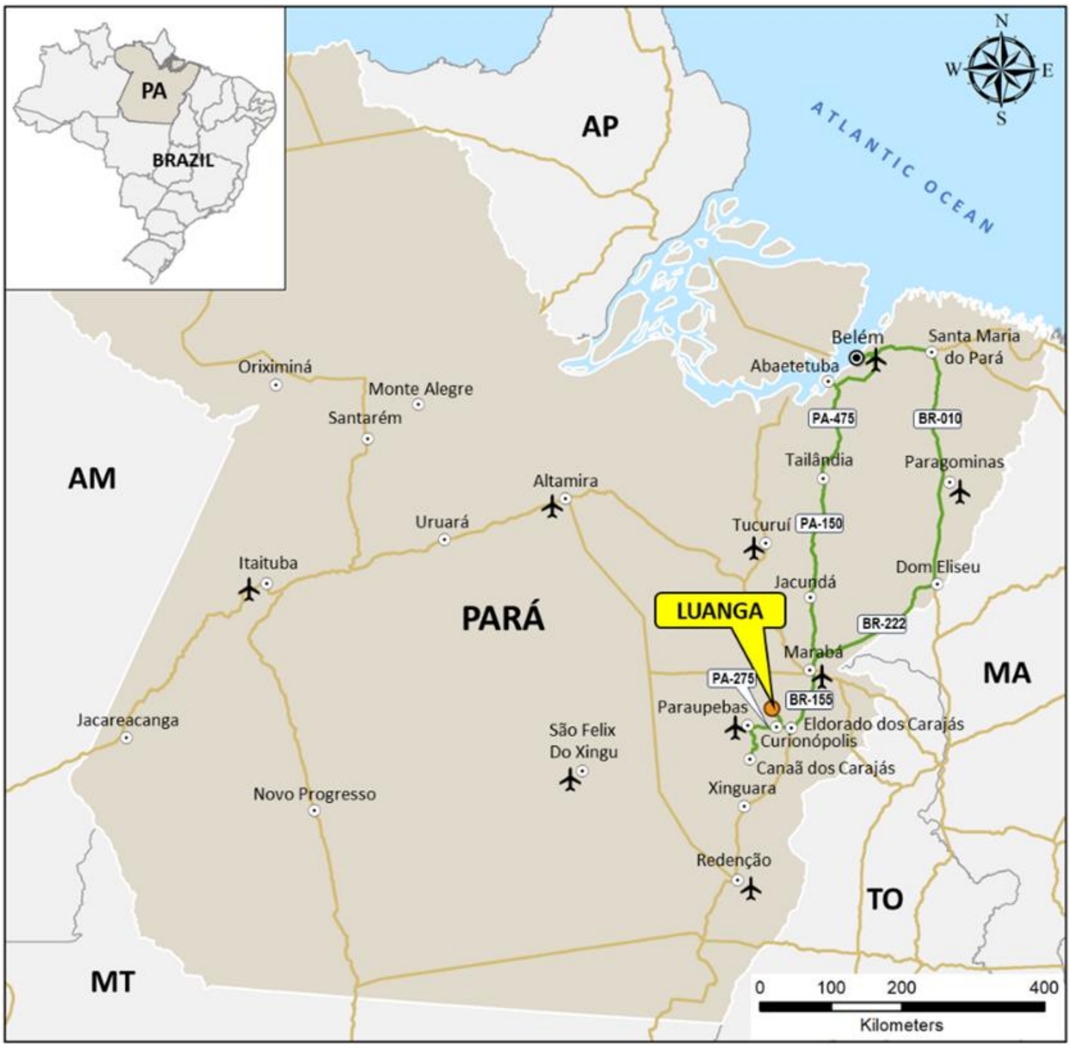


Figure 4-1: Regional location of Luanga Project in Pará State, Brazil

Source: GE21, 2023.

The area is located on private farmland, used for cattle farming. There are no indigenous claims or protected forests in the area. To carry out exploration/feasibility works, such as drilling, an access agreement is required with the surface rights owner (landowner). Land access agreements (Figure 4-2) are currently in place with six key landowners, covering 97% of the known mineralized envelope of the Luanga deposit. These agreements have recently been renewed and are valid for two years each time.

GE21 Consultoria Mineral

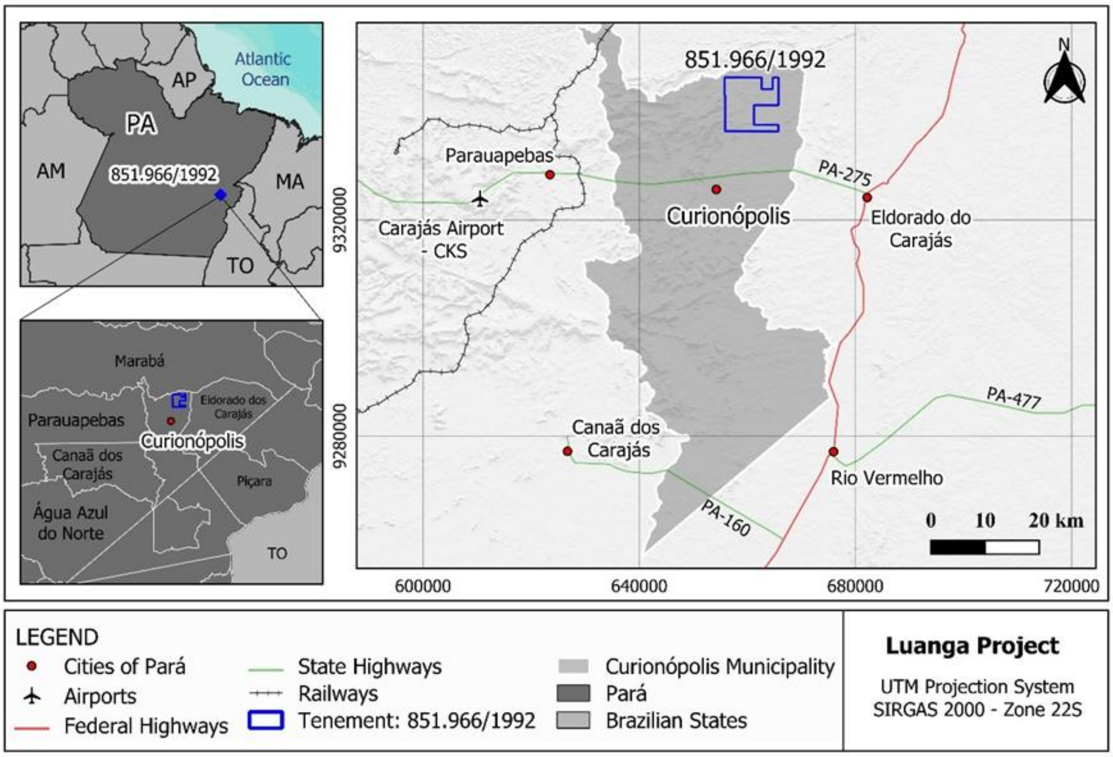


Figure 4-2: Luanga Project location map

Source: GE21, 2025.

4.1 Mineral Tenure

On 5th September, 1995, the Ministério de Minas e Energia (Ministry of Minerals and Energy – MME) issued Vale Exploration, Licence No. 1961 under the process designated ANM 851.966/1992. The ANM administers Exploration Licences. This Exploration License is located 40 km northeast of Parauapebas in Pará State, Brazil.

The license, which covers the Project, comprises an area of 7,810.02 hectares, currently in Bravo Mineração Ltda.'s name, as summarized in Table 4-1 and illustrated in Figure 4-3. Exploration License 851.966/1992 remains valid while the Mining License application is pending.

Table 4-1: Mineral tenement summary

ANM Process	Municipality	Stage	Mineral	Title Owner	Size (hectares)	License No.	Expiry Date
851.966/1992	Curionópolis	Application for Mining License	Au, Pd, Pt, Rh, Ni	Bravo Mineração Ltda	7,810.02	1961	
				Total	**7,810.02**		

Notes:
Comments: Mining License pending.
ANM = National Mining Agency.

Source: ANM, February 2025.

GE21 Consultoria Mineral

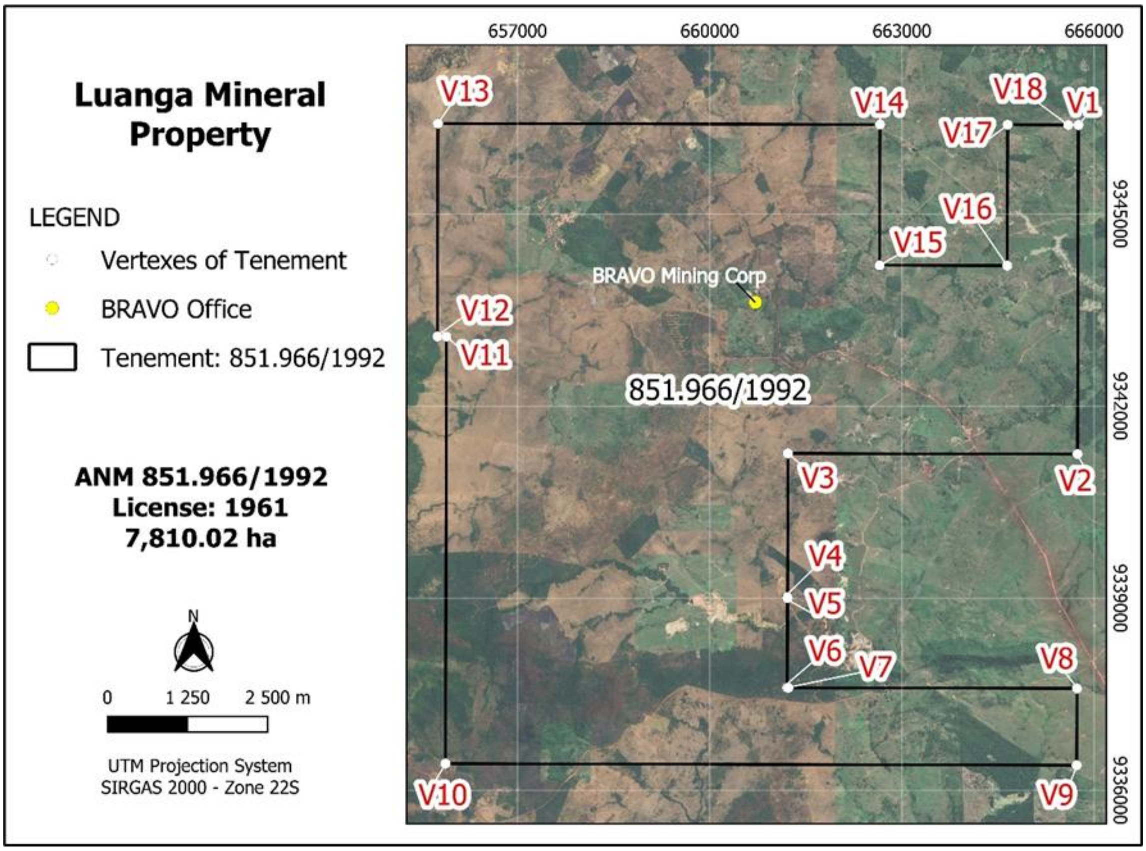


Figure 4-3: Luanga Project tenement map

Source: GE21, 2025.

The Luanga mineral property is centred approximately at coordinates -05º57'24.34" S/-49º32'51.00" W. Bounding coordinates of Exploration License No.1961 from ANM title documents are presented as vertices of yhe Luanga mineral property (Table 4-2).

Table 4-2: Vertices of Luanga mineral property

Vertex	Latitude	Longitude	Vertex	Latitude	Longitude
v1	-05°54'40"284	-49°30'09"580	v10	-06°00'05"795	-49°35'30"045
v2	-05°57'27"643	-49°30'09"580	v11	-05°56'28"677	-49°35'30"072
v3	-05°57'27"638	-49°32'36"608	v12	-05°56'28"677	-49°35'34"710
v4	-05°58'41"177	-49°32'36"614	v13	-05°54'40"336	-49°35'34"693
v5	-05°58'41"177	-49°32'36"617	v14	-05°54'40"300	-49°31'50"304
v6	-05°59'26"752	-49°32'36"617	v15	-05°55'51"911	-49°31'50"304
v7	-05°59'26"758	-49°32'36"617	v16	-05°55'51"911	-49°30'45"503
v8	-05°59'26"758	-49°30'09"580	v17	-05°54'40"289	-49°30'45"503
v9	-06°00'05"822	-49°30'09"580	v18	-05°54'40"284	-49°30'14"770

Exploration License Nº 1961, ANM.851.966/1992 – Datum SIRGAS 2000

Source: ANM, October 2023.

Although the original exploration permit expired in 1998, the ANM renewed it only in 2005 for an additional three years. A Final Exploration Report was submitted by Vale in 2008, followed by a Mining License application in 2013. However, the ANM has not issued a decision to date. Bravo expects this pending status to continue until a new study demonstrating the project's technical and economic feasibility is submitted.

4.1.1 *Acquisition or Transaction Terms*

On 3rd June, 2020, Vale, FFA Holding e Mineração Ltda (FFAH) and Brazil Americas Investments and Participation Mineração Ltda (BAIP), where Bravo is the beneficiary party, appointed FFAH and BAIP to acquire the Project. Payment terms were as follows, with royalties shown in Section 4.3.

- USD300k paid on 7th December 2021.
- USD500k paid on 9th November 2022.
- USD500k paid on 19th November 2023.
- Total: USD1.3 million

On 24th January 2022, Bravo´s wholly owned subsidiary acquired 100% of the shares of AIPL (Americas Investments & Participation Ltd.), giving it a 100%, undivided interest in the Project.

4.2 Environmental and Social Liabilities

No environmental liabilities have been identified within the Luanga Exploration License. The current land use at the Project is solely agricultural cattle grazing. There are no significant rivers within the property. There are also no existing forests on the property, thus no deforestation is required. However, as of the date of this Report, Bravo has planted more than 36,000 trees, or approximately 105 trees per hole drilled (Figure 4-4).

Figure 4-4: Seedling nursery

Source: Bravo, 2025.

The most significant activity to be completed by the company in the next few years is relatively low-impact drilling. Bravo will concurrently rehabilitate drill sites.

Social or community impact is expected to be negligible since the nearest community is the village of Serra Pelada, which is approximately 12 km away. There are no indigenous communities within 25 km of Luanga.

The ashpalt road to Serra Pelada crosses Luanga in the northern half of the property. A low-voltage power line parallels this road. Bravo does not expect to encounter major difficulties in moving the road and associated power line if the Project advances to a construction decision. The location of the road and power line has not, and is not expected to, impact planned exploration activities.

4.3 Royalties and Other Encumbrances

The following royalties are applicable:

- 1% NSR royalty to Vale.
- 2% NSR royalty to BNDES (Brazilian Development Bank).
- CFEM (Financial Compensation for the Exploitation of Mineral Resources) Government Royalties:
  - 1.5% NSR royalty on Au.
  - 2% NSR royalty on precious metals (Pd, Pt, Rh).
  - 2% NSR royalty on base metals (Ni, Cu).
- The Private Landowner Royalty is equal to 50% of CFEM royalties.

Bravo is subject to a corporate income tax rate of 25%, which is applied to the pre-tax profit. The Company can apply for a tax incentive under Superintendence for the Development of the Amazon (Superintendência do Desenvolvimento da Amazônia, SUDAM) based on Federal Law Nº 13,799, 3rd January, 2019. If granted, this reduces the 25% income tax by 75% for a 10-year period, starting from the year in which the Appraisal Certificate from SUDAM is issued. The total tax burden in this case is 15.25% (75% of 25% + 9% social contribution tax). Bravo has not applied for SUDAM status as of the date of this Rreport.

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

5.1 Accessibility & Physiography

Luanga is accessible via paved roads from two regional centres, Parauapebas and Marabá. Both cities have commercial airports with multiple flights a day to Belo Horizonte and Belém from Parauapebas, and to Brasília, São Paulo, Rio de Janeiro, Salvador, and others from Marabá. Access to the Project is via a municipal paved road that from Highway PA-275.The closest population centres to the Project are the small town of Curionópolis, with a population of approximately 17,846, approximately 17 km south-southwest of Luanga and the mining community of Serra Pelada, approximately 12 km west of Luanga. There are no communities within the Project boundary.

Parauapebas, located approximately 40 km to the west-southwest of Luanga, is the region's critical service provider and labour source. Parauapebas is the largest mining town in the state, with a significant labour force of residents supporting multiple world-class iron ore and copper-gold mines in the Carajás. Parauapebas is also home to all of the region's mining-related services and mining infrastructure. Parauapebas was recorded as having a population of 213,576 in 2020. Any future operation is expected to be able to source all labour from the local region.

The nearest rail services are those privately owned by Vale in Parauapebas, which connect to Marabá. Bravo has access to this rail line as part of the Luanga purchase agreement with Vale. The nearest commercial-scale port facility is Vila da Conde, located adjacent to the state capital, Belém, approximately 660 km to the north of Project. The port facilities can also be accessed via barge on the Tocantins River, which can be accessed from Marabá.

The Luanga Project is located in the Carajás Mineral Province, which lies within the South Pará Plateau, where the altitudes vary from 500 m to 700 m above sea level. A series of NNE-SSW trending ranges project above the plateau, remnants of an older surface that was eroded to a peneplain and uplifted during the Paleozoic. Luanga lies on the southeast flank of the Serra Sereno range, with peaks up to 600 m above sea level. The stream banks are terraced and capped with iron-aluminous laterite, which is currently being actively eroded (Figure 5-1).

The main drainage of the area flows into the Sereno River, part of the Rio Vermelho system. A system of tributaries flows from Serra Leste to the northeast, crossing the Luanga mineral property, until it discharges into the Sereno River.

Inside the Luanga Project area, vegetation has been cleared for pasture and subsistence cultivation, which is indicated in Figure 5-2 by the pink areas, versus dark green, which are forested areas. The Luanga Project covers 7,810 ha, much of which has been deforested in decades past to create grazing land for cattle. This area is more than sufficient for any contemplated future mining-related activities, including waste rock and tailings disposal, process plant and related infrastructure.

GE21 Consultoria Mineral

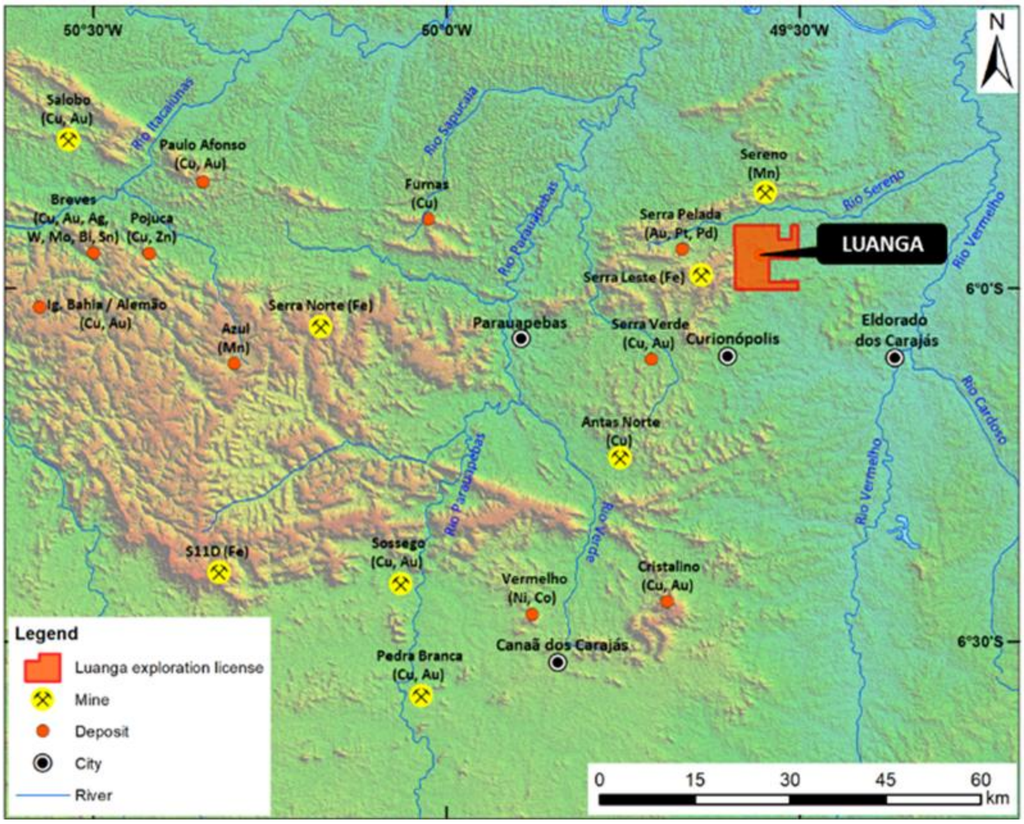


Figure 5-1: Physiography of Carajás region

Source: GE21, 2023.

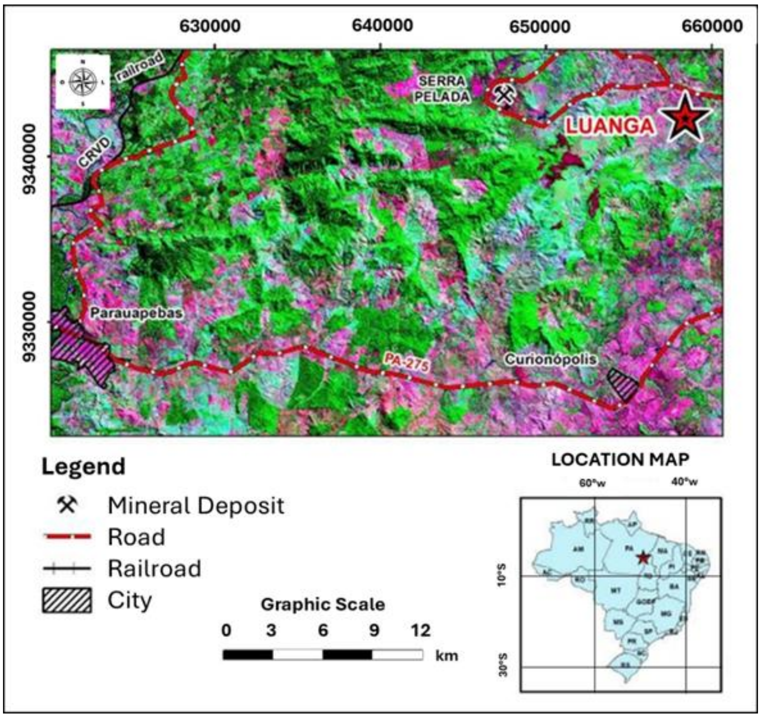


Figure 5-2: Sat image (RGB composition 342) with relief and vegetation of Carajás region

Source: GE21, 2025.

GE21 Consultoria Mineral

5.2 Climate and Length of Operating Season

Situated approximately 6º south of the equator, the climate is typically equatorial, with slight variation in mean monthly temperatures throughout the year. The average maximum temperature is 32ºC while the average minimum is 22ºC. There are two distinct seasons: the winter is warm and dry, while the summer is wet and humid. Three-quarters of the annual precipitation falls from December through April. In August, the average rainfall is 10 mm, while in January, February and March, the monthly rainfall exceeds 150 mm. Rainfall intensity can be high. For these reasons, water availability for any contemplated future mining-related activity in the region is plentiful and readily accessible. Figure 5-3 shows the climate data for Curionópolis. The annual rainfall average is 2,082 mm.

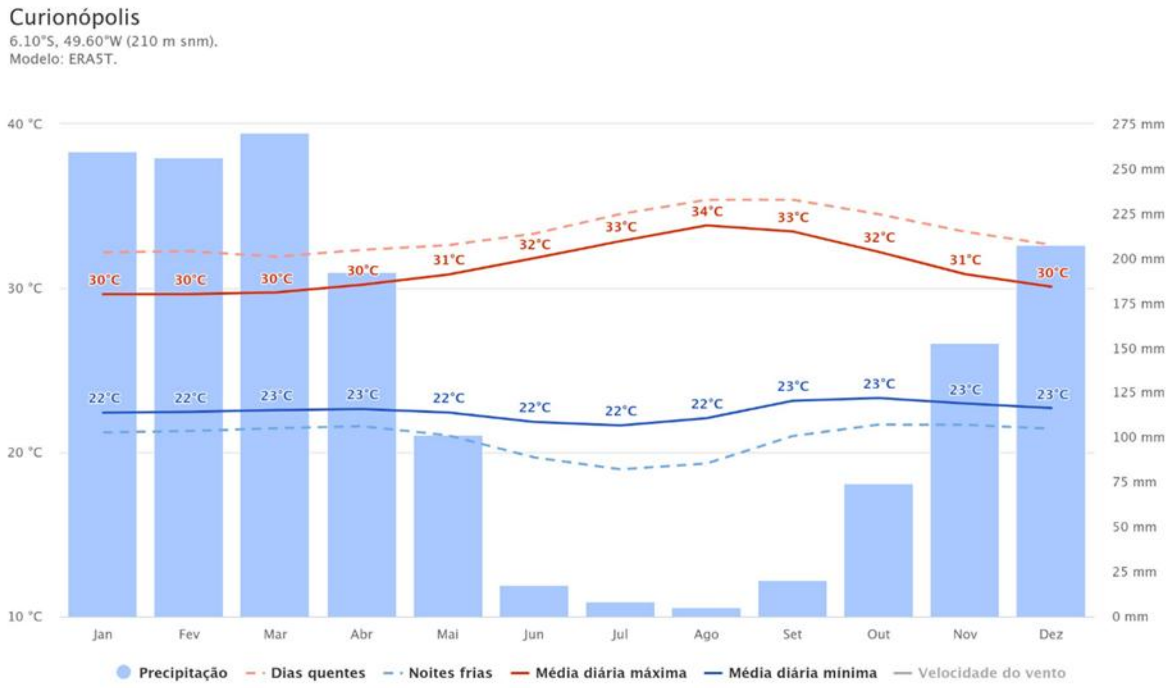


Figure 5-3: Average monthly temperature and rainfall at Curionópolis

Source: Meteoblue.com, 2025.

5.3 Local Resources and Infrastructure

The area is in a moderately fertile red/yellow podsols region. Agricultural production includes rice, corn, beans, palm oil, banana, tomato, watermelon, coffee, avocado, guava and cashew nuts. Throughout the region, there is extensive cattle ranching, producing both milk and meat, using natural pastures that are annually burnt to stimulate the growth of young grasses. Total stock numbers include up to 400,000 head of cattle and 50,000 pigs in the region.

The burgeoning mining industry in the Carajás Mineral Province required a massive investment in infrastructure to create transport routes for industrial and agricultural exports. One of Brazil's most significant mining projects is based on the iron mineralization deposits in the Serra dos Carajás near Parauapebas. With a reported 18 billion tonnes of mineralization, this is one of the world's most significant iron deposits. The Project lies within the Grande Carajás Project Mining and Industrial Zone (Projeto Grande Carajás, PGC) that is reportedly gazetted over an area of 400,000 km^2 and involves a total investment of US$62 billion. The town of Carajás has been completely rebuilt and is closed to all but Vale's workers. Vale constructed a heavy-duty rail line over 892 km from the iron mines to the Atlantic port of São Luís. The nearest railhead is at Parauapebas, close to the Serra Leste mine, located 40 km by road from the Project. Another option is a railhead at Carajás, 55 km from Luanga.

Besides iron ore, other minerals such as gold, copper, nickel, manganese, and bauxite have been discovered in significant quantities in the Carajás Mineral Province, with additional discoveries a regular occurrence. Much of the metal mined in the region is exported in its raw form, but there has been some attempt at metal refining. These include the aluminum smelter in Belém (Latin America's most significant industrial plant) and a steel mill in São Luís. Mining developments have led to increased energy demands, spurring the construction of dams for hydroelectric power generation.

The region's economy is heavily dependent on mining, principally from the iron ore mines of the Carajás. Vale is reportedly developing five projects in southern Pará located within a radius of 90 km from Carajás, three of them to the southeast and two to the northeast.

The city of Parauapebas is equipped with all the local amenities, such as banks, hospitals, hotels, schools, university and supermarkets. In addition, the long history of mining in the town has provided the area with a skilled workforce experienced in disciplines that support mining, such as machinery mechanics and general maintenance.

Marabá is the market centre for the region and a hub for road, rail, and river transport. Along with the mining industry, the city economy relies on agriculture, cattle raising, handcraft production and commerce. Many experienced miners are in the vicinity, and the university in Marabá is focused on training professionals for the mineral industry.

The Tocantins River and its tributaries are of vital economic importance to the region, both as a source of fresh water for the population and industry and as a source of hydroelectric power. Downstream from Marabá, the Tucuruí hydroelectric dam expanded its capacity in 2005 to lift output to 8,370 MW. Three other hydroelectric plants on the Tocantins River have a combined capacity of 2,630 MW, and an additional plant is near completion. Seven more hydroelectric plants are planned on the Tocantins River (Figure 5-4).

A branch of the main 230 kV hydroelectric power transmission line from Tucuruí to Carajás has the available capacity to supply the necessary power to the Project area for any contemplated future mining-related activity. This power transmission line is approximately 25 km to the NW of the Luanga property.

Bravo has office facilities close to the Project, on land that is leased by Bravo, within a local farm, and close to Curionópolis and Parauapebas (Figure 5-5).

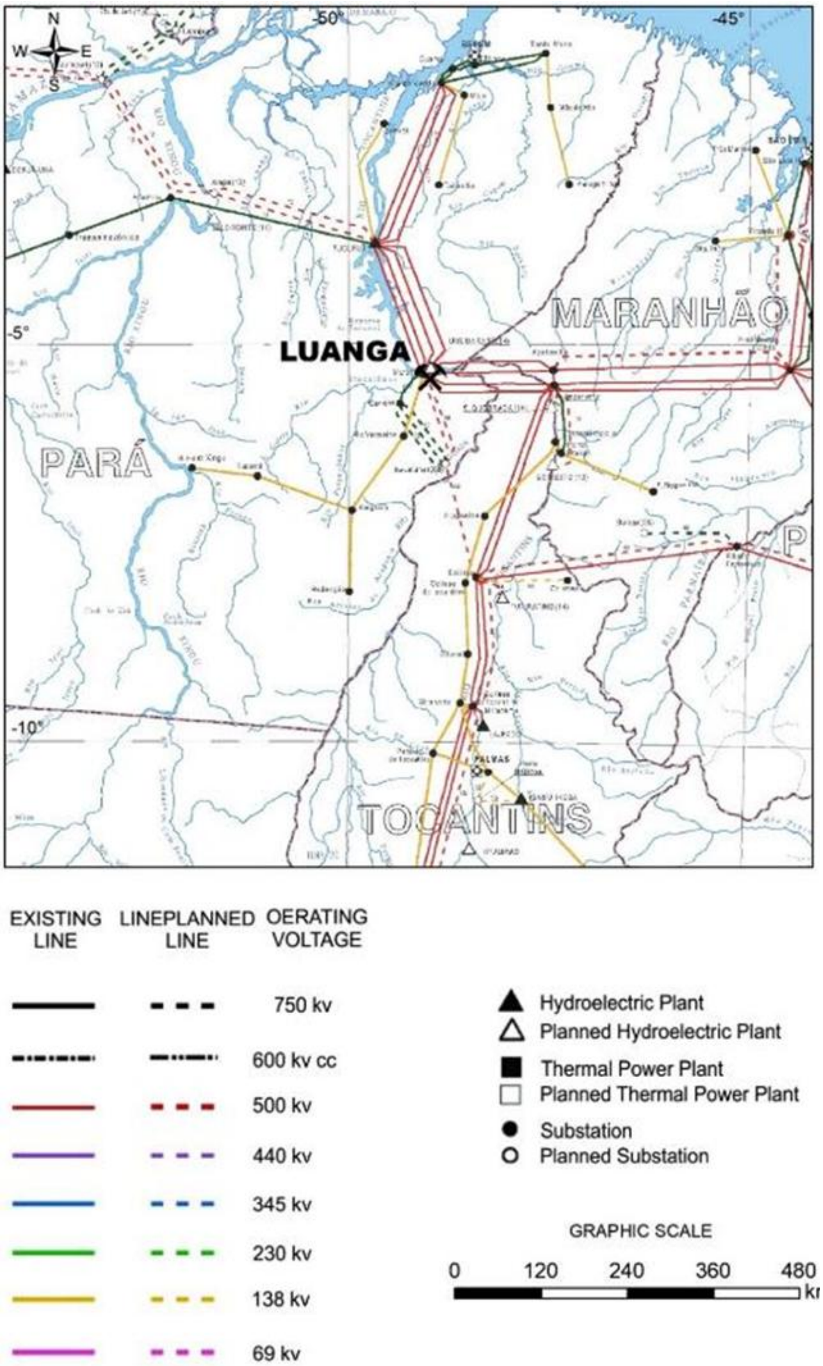


Figure 5-4: Power transmission lines in the region of Luanga Project

Source: Bravo, 2023.

Figure 5-5: Bravo facilities at Luanga Project

Source: Bravo, 2025.

5.4 Social and Community

Bravo has implemented several initiatives related to Environmental, Social, and Governance (ESG) performance during the implementation and development of Luanga. One of the ESG initiatives is related to environmental management. Bravo has implemented measures to minimize the impact of its operations on the environment. For example, it has developed a water management plan that includes monitoring water quality and quantity and implementing measures to reduce water consumption. Additionally, Bravo has implemented a waste management plan that provides recycling and proper disposal of waste material. In order to reduce impact on the natural environment, Bravo created an internal procedure to plant at least ten new trees for each drill hole carried out on the Project area and has to date planted more than 36,000 trees (or approximately 105 trees per hole drilled). Currently, the company has over 30,000 trees in its local nursery, awaiting planting (Figure 5-6).

Figure 5-6: Bravo’s nursery at Luanga camp

Source: Bravo, 2023.

The company also provides training programs for residents to develop skills that can be used in the mining industry or in other areas of the local economy, establishing partnerships with local communities to promote sustainable development in the region. Currently, 80% of Bravo’s workforce is from the local communities in the Carajás region, exceeding Bravo’s goals for local hiring.

Bravo has also set a goal of local spending, with significant contracts such as drilling and assaying being let to companies within the Carajás region.

As part of Bravo’s ESG strategy, the Company has developed, approved, disseminated, and trained its personel and contractors on the following policies :

- Code of Conduct & Ethics Policy;
- Anti-Bribery & Anti-Corruption Policy;
- Disclosure & Confidentiality Policy;
- Diversity & Inclusion Policy;
- Whistleblower Policy;
- ESG Policy (including Health & Safety, and Environment).

6 HISTORY

6.1 The Carajás Mineral Province

The history of the Carajás region is essential to contextualize the discovery and subsequent evaluation of the Luanga deposit. Until the 1960's, geological work carried out in the Carajás region was restricted by a lack of access except in the vicinity of the major rivers. In 1966, DNPM/PROSPEC (Brazilian National Department of Mineral Production / Mineral Prospection sector) published the results of Project Araguaia. This involved the acquisition of aerial photo coverage and photo interpretation of the Carajás region. No mineral discoveries were reported as the fieldwork was restricted to the major drainages.

The first mineral exploration in the Carajás region was carried out by Companhia de Desenvolvimento de Indústrias Minerais (CODIM), which, in 1966, discovered the manganese deposit of Serra do Sereno. This discovery motivated US Steel to commence broad-scale exploration in the region through its subsidiary Companhia Meridional de Minerações (CMM). In July 1967, a Brazilian team discovered high-grade iron ore with an average grade of 66% Fe. US Steel wanted to develop the Carajás iron deposit. However, the Brazilian Government was unwilling to give a foreign company control over such an important national asset. Instead, the Brazilian Government created in April 1970 a joint venture company, Amazônias Mineração SA (AMSA), owned 51 percent by Rio Doce (CVRD, which now is Vale) and 49 percent by CMM. By presidential decree on 6th September 1974, AMSA was granted the rights to all iron ore in the Carajás Mineral Province.

Exploration continued until 1977, when CMM, concerned over the high capital cost and poor outlook of the international market for iron ore at the time, withdrew from the project. Vale purchased CMM's 49% for US$55 million. AMSA, now wholly owned by Vale, was granted the rights for mineral exploration and development.

In June 1978, at the commencement of laying the Carajás railroad, linking Ponta da Madeira on the Maranhão coastline to the Carajás reserves effectively launched the implementation of the Carajás Iron Ore Project.

With the establishment of the Carajás Iron Ore Project and its associated infrastructure, the Carajás Mineral Province was established and recognized. Decades on, it is one of the largest mineral provinces in the world and the most significant mining region in Brazil. As a result of the recognition of the global importance of the Carajás Mineral Province, meaningful exploration was undertaken over the following decades by Vale and other domestic and foreign mining companies. This work resulted in the discovery of several deposits in the province and the development of several mines.

6.2 The Luanga Project

Mafic-ultramafic rocks of the Luanga Complex were identified in 1983 during regional exploration developed by DOCEGEO (the exploration division of Vale) in the Serra Leste region. After discovering up to 2 m thick chromitites, DOCEGEO conducted geological mapping, soil geochemistry survey (400 m x 40 m grid) and ground magnetic survey in the Luanga Complex. Four diamond boreholes were drilled to test the thickness and lateral continuity of outcropping chromitites. The drilling results were not favourable for chromium mineralization but intersected anomalous concentrations of Pt and Pd, including 9 metres at 2.57 ppm of Pt+Pd (drill hole PPT-LUAN-FD0004).

In 1997, a joint venture between DOCEGEO-Barrick Gold, carried out a stream sediment campaign over the Luanga Complex area that identified Au anomalism.

In 2000, Vale carried out a new soil geochemistry survey to test the Au anomalies outlined by Barrick Gold. The sampling grid, covering the southern portion of Luanga Complex, defined a 1 km long trend of Pt and Pd anomalies. Due to this anomalous trend, Vale carried out an additional soil geochemistry survey in the northern portion of the Luanga Complex (next to chromitite layers), identifying another 1 km long Pd and Pt anomalous trend. The geochemical survey was extended to the central portion of the layered complex, adding a further 2 km extension, now joining up to form a continuous Pt-Pd anomalous trend along the entire layered intrusion.

In 2001, Vale started an exploration program for PGM in the Serra Leste region. Systematic geological and structural mapping using RADARSAT and Landsat-TM5 integrated data, along with airborne geophysical survey, led to the discovery of several other layered mafic intrusions.

6.3 Vale / DOCEGEO Drilling

Vale completed considerable diamond drilling in the 1992, 2001, 2002 and 2003 campaigns. Drill logs and assay summaries and certificates for all historical drill holes are available and have been compiled into a database along with more recent drill data. This historical work has been thoroughly documented.

Historical Vale drilling consisted of 256 diamond drill holes (50,786.74 linear metres) at Luanga between 1992 and 2003 (Table 6-1). Most diamond drilling occurred between 2001 and 2003, and it involved two main targets, Luanga and Luanga South. At Luanga, 232 diamond drill holes (45,599.59 linear metres) were completed, representing approximately 90% of the drilling program. At Luanga South, 24 drill holes (5,187.15 linear metres) were completed.

Most of the diamond drilling was carried out by two Brazilian companies, Geologia e Sondagem S.A. (Geosol) and Rede Engenharia e Sondagem Ltda. (Rede). DOCEGEO was responsible for the first four drill holes at the Project.

Table 6-1: DOCEGEO Drilling Summary

Year	Drill Type	Drill Holes	Total Metres	Contractor	Target
1992	DD	4	643.69	DOCEGEO	Luanga
2001	DD	86	14,584.35	Geosol	Luanga
2002	DD	71	15,423.25	Geosol	Luanga
2003	DD	67	14,535.15	Geosol - Rede	Luanga
		4	413.15		Luanga (Met)
		24	5,187.15		Luanga South
Total		**256**	**50,786.74**		

Source: Bravo, 2025.

6.4 Historical Metallurgical Test Work

The Project is an exploration advanced stage project and, as a result, historical metallurgical test work was limited to first-pass (or fatal flaw) metallurgical test work.

This test work was early stage. However, it indicated that a “saleable” Pd-Pt-Au-Ni concentrate could potentially be produced.

Historical metallurgical test work focused on samples from the sulphide mineralization, since this represented the bulk of the historical PGM mineralization identified at Luanga. Work was performed at several facilities between 2002 and 2007 and can be summarised as follows:

- Mintek, 2002.
- CDM (internal Vale laboratory), 2002-2004.
- SGS Lakefield, 2003-2004.

Initial work by Mintek and CDM used a higher-grade sample (5.0 g/t Pt + Pd + Au) from the sulphide mineralization. Metallurgical test work by both companies demonstrated that recoveries to concentrates of approximately 70% could be achieved using conventional milling, griding and froth flotation, similar to other sulphide PGM deposits globally.

Test work subsequently carried out by SGS Lakefield (Canada) on a lower grade ore, 200 kg sample from the sulphide mineralization, also indicated recoveries of approximately 70%, with a concentrate from 0.78% of the fed mass of 132 g/t PGM + Au. Internal work by CDM using the same sample also supported these results.

Results of historical metallurgical work are summarised in Table 6-2.

Table 6-2: Results of historical metallurgical work

Sample	Lab	Test	Average Grade (g/t 3E*)	Mass	Concentrate Grade (g/t 3E*)	Recovery (%)
M1	Mintek	Lock Cycle Test	5.00	2.2%	137	66.2
M2	CDM	Open Circuit	5.00	3.4%	104	72.1
S1A3	CDM	Lock Cycle Test 1	1.70	1.2%	95	73.0
S1A3	CDM	Lock Cycle Test 2	1.70	0.89%	137	69.3
S1A3	Lakefield	Lock Cycle Test	1.49	0.78%	132	69.4

*3E = Pt + Pd + Au. No data is available for Rh or Ni.
Source: Mintek, 2002; CDM, 2002 - 2004; SGS Lakefield, 2003 and 2004.

GE21 Consultoria Mineral

6.5 Historical Mineral Resource

6.5.1 Historical Mineral Resource (2020)

The “Historical Estimate” of Mineral Resources for Luanga was prepared internally in 2017 by Vale and reported by Mansur et al., 2020:

142Mt@1.24g/t 3Eq (Pd + Pt + Au) + 0.11% Ni using a cut-off grade of 0.5g/t PGM + Au.

This disclosure is made as per Section 2.4 of NI 43-101, parts 1 to 7 inclusive:

1. The “2020 Historical Estimate” was prepared internally in 2017 by Vale and reported publicly by Mansur et al., 2020, with other Vale geologists as co-authors.
2. Bravo acquired the Project directly from Vale and has since conducted a significant amount of infill drilling, resampling of the historical core, metallurgical test work, geophysics and other works. Given these substantive works, the authors believe that the 2020 Historical Estimate was strongly supported by Bravo’s work and is relevant to the reader’s understanding of the status of the Project and its future potential. Further, given that this estimate was prepared by Vale, a major mining company with global operations, the authors believe it is likely to have been prepared to standards a reasonable person would use and is therefore considered reliable for the purposes of defining recommendations for future work.
3. No breakdown of the individual metals contributing to the 2020 Historical Estimate was published and no technical report related to the 2020 Historical Estimate is available to the authors. As a result, aside from the information quoted above, nothing is known about the key assumptions, parameters, and methods used to prepare the 2020 Historical Estimate.
4. The 2020 Historical Estimate used no categories to define it.
5. Until the publication of the 2023 MRE (see 6.5.2 below), there were no more recent estimates or data available to the authors.
6. The work needed to be done before the 2020 Historical Estimate could be classified as a current Mineral Resource was defined in Section 26 of the 2022 Technical Report filed in conjunction with the initial public offering of Bravo and was subsequently undertaken.
7. A QP has not done sufficient work to classify the 2020 Historical Estimate as a current Mineral Resource.
8. Bravo is not treating the 2020 Historical Estimate as a current Mineral Resource.

Bravo also cautions that the 2020 Historical Estimate was not prepared in accordance with NI 43-101 and should not be relied upon since it has been superseded by the Mineral Resource Estimate detailed in the 2023 Technical Report and, subsequent to that, the MRE contained this report. Nevertheless, Bravo believes the information is relevant to the reader. Further, readers should be aware that the assay values used to calculate the nickel content in the 2020 Historical Estimate are total Nickel and thus contain both sulphide Nickel (recoverable) and silicate nickel (unrecoverable). It is unknown to Bravo whether the nickel content in the 2020 Historical Estimate had been modified to account for this or not.

GE21 Consultoria Mineral

6.5.2 *Historical Mineral Resource (2023)*

Subsequent to the 2020 Historical Estimate, Bravo undertook extensive work programs as recommended in the 2022 Technical Report. The 2023 Technical Report contained Bravo's maiden MRE, prepared in accordance with the requirement of NI43-101, for Luanga Project, which was prepared by GE21 and disclosed by Bravo in 2023:

Table 6-3: Mineral Resource Report – 2023

Mineral Resource Classification	Weathering	Average Grades and Contained Metal Estimates												
		Tonnes	Pd Eq		Pd		Pt		Rh		Au		Ni	
		Mt	g/t	Oz	g/t	Oz	g/t	Oz	g/t	Oz	g/t	Oz	%	Tonnes
Indicated	Oxide	4.6	1.43	212,990	0.91	135,949	0.54	79,901	0.07	10,031	0.08	11,944	n/a	n/a
	Fresh rock	68.5	1.77	3,892,313	0.78	1,705,709	0.53	1,159,078	0.06	131,248	0.07	146,263	0.13	89,539
	Total	73.1	1.75	4,105,303	0.78	1,841,658	0.53	1,238,979	0.06	141,279	0.07	158,207	0.13	89,539
Inferred	Oxide	10.0	1.30	418,810	0.75	241,117	0.72	230,367	0.08	25,738	0.04	12,444	n/a	n/a
	Fresh rock	108.1	1.52	5,286,970	0.60	2,082,479	0.57	1,997,054	0.05	190,746	0.04	122,076	0.10	104,640
	Total	118.1	1.50	5,705,800	0.61	2,323,596	0.59	2,227,421	0.06	216,484	0.04	134,520	0.10	104,640

Notes:

1. The 2023 MRE was prepared by Porfirio Cabaleiro Rodriguez, Mining Engineer, BSc (Mine Eng), MAIG, director of GE21 Consultoria Mineral Ltda., an independent Qualified Persons (QP) under NI43-101. The effective date of the MRE is 22 October 2023.
2. The 2023 Mineral resources are reported using the 2014 CIM Definition Standards and were estimated in accordance with the CIM 2019 Best Practices Guidelines, as required by NI 43-101.
3. Mineral resources that are not Mineral Reserves do not have demonstrated economic viability. There is no certainty that all Mineral Resources will be converted into Mineral Reserves.
4. This MRE includes inferred Mineral Resources which have had insufficient work to classify them as Indicated Mineral Resources. It is uncertain but reasonably expected that inferred Mineral Resources could be upgraded to indicated Mineral Resources with continued exploration.
5. The Mineral Resource Estimate is reported/confined within an economic pit shell generated by Whittle software, using assumptions generated from work completed by Bravo and historical test work:
   a. Phase 1 and 2 Metallurgy test work – Metallurgical recovery in sulphide material of 80% Pd, 88% Pt, 59% Rh, 56% Au, 50% Ni to a saleable Ni-PGM concentrate.
   b. Phase 1 and 2 Metallurgy test work– Metallurgical recovery in oxide material of 73% Pd, 24% Pt, 61% Rh, 95% Au to a saleable PGM ash residue (Ni not applicable).
   c. Independent geotechnical test work – Overall pit slopes of 40 degrees in oxide and 50 degrees in Fresh Rock.
   d. Densities are based on 26,898 relative density sample measurements. Averages are 1.58 t/m^3 oxide, 2.71 t/m^3 Saprock and 2.85 t/m^3 fresh rock.
   e. External downstream payability has not been included, as the base case MRE assumption considers internal downstream processing.
   f. Payable royalties of 2%, (considering CFEM only. For future reserves the complete set of royalties must be considered).
6. Metal Pricing:
   a. Metal price assumptions are based on 10-year trailing averages: Pd price of US$1,380/oz, Pt price of US$1,100/oz, Rh price of US$6,200/oz, Au price of US$1,500/oz, Ni price of US$15,648/t.
   b. Palladium Equivalent (PdEq) Calculation
   c. The PdEq equation is: PdEq = Pd g/t + F1 + F2 + F3 + F4
   d. Where: $\boldsymbol{F1} = \frac{(Pt_p * Pt_R)}{(Pd_p * Pd_R)} \boldsymbol{Pt_t}$ $\boldsymbol{F2} = \frac{(Rh_p * Rh_R)}{(Pd_p * Pd_R)} \boldsymbol{Rh_t}$ $\boldsymbol{F3} = \frac{(Au_p * Au_R)}{(Pd_p * Pd_R)} \boldsymbol{Au_t}$ $\boldsymbol{F4} = \frac{(Ni_p * Ni_R)}{(Pd_p * Pd_R)} \boldsymbol{Ni_t}$

      P = Metal Price R = Recovery
   e. Costs are taken from comparable projects in GE21's extensive database of mining operations in Brazil, which includes not only operating mines, but recent actual costs from what could potentially be similarly sized operating mines in the Carajás. Costs considered a throughput rate of ca. 10mtpa:

i. Mining costs: US$2.50/t oxide, US$3.50/t Fresh Rock. Processing costs: US$8.50/t fresh rock, US$7.50/t oxide. US$2.50/t processed, for General & Administration. US$1.00/t processed for grade control. US$0.50/t processed for rehabilitation.

ii. Several of these considerations (metallurgical recovery, metal price projections, for example) should be regarded as preliminary in nature, and therefore, PdEq calculations should be regarded as preliminary in nature.

iii. The current MRE supersedes and replaces the Historical Estimate, which should no longer be relied upon.

iv. The QP is not aware of political, environmental, or other risks that could materially affect the potential development of the Mineral Resources other than as disclosed in the 2023 Technical Report.

v. Totals may not sum due to rounding.

Source: GE21, 2023.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Amazon Craton represents one of the main tectonic units of South America (5,600,000 km^2) and is the largest preserved block in the Brazilian Shield. The craton is separated from the Andean orogenic belt by extensive Cenozoic coverage (Colombian Llanos, Venezuelan Llanos, Paraguayan Chaco, etc.), which covers both Paleozoic basins and extensions of the craton. Several Phanerozoic basins in the northeast (Maranhão), south (Xingu and Alto Tapajós), southwest (Parecis), west (Solimões), north (Tacutu) and center (Amazonas) cover the craton area. In the eastern flank, the craton is limited to the Araguaia Orogenic Belt, part of the Tocantins Province, formed during the Neoproterozoic, as result of the convergence and collision of the Amazon Craton, to the west; São Francisco Craton, to the east; and Paranapanema Craton, to the southwest.

Currently, the Amazon Craton is subdivided into 7 geological provinces, based on geochemical and geochronological data (Figure 7-1):

- Carajás (3.0 – 2.5 Ga)
- Central Amazon (Archean?)
- Transamazonas (2.26 – 2.01 Ga)
- Tapajós–Parima (2.03 – 1.86 Ga)
- Rio Negro (1.82 – 1.52 Ga)
- Rondônia-Juruena (1.82 – 1.54 Ga)
- Sunsás & K’Mudku (1.45 – 1.10 Ga)

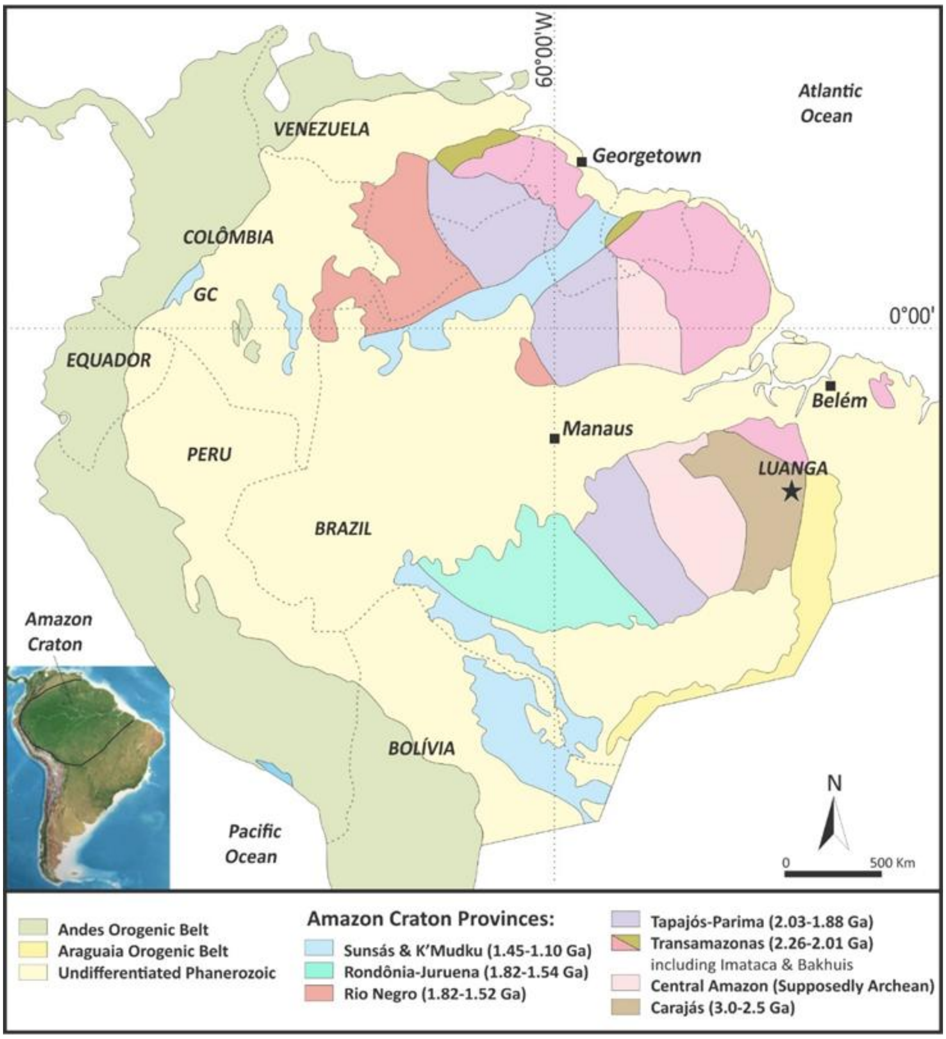


Figure 7-1: Geological provinces of the Amazon Craton

Source: Santos et al., 2008.

The Project is located within the Carajás Mineral Province (CMP), located in the southeastern margin of the Archean Amazonian Craton (Figure 7-1). The CMP is composed mostly of granites and greenstone belts and also hosts the largest gold deposits in the Amazon Craton, including Serra Pelada and the Salobo and Igarapé Bahia Cu-Au deposits. Gold deposits are concentrated in the Archean and Paleoproterozoic terranes, including the Archean Carajás Mineral Province.

7.1.1 The Carajás Mineral Province

The Carajás Mineral Province (Figure 7-2) is one of the most important mineral provinces of the South American continent, hosting several world-class Fe, Cu-Au and Ni deposits. It is in the south-eastern portion of the Amazonian Craton, bounded by the Neoproterozoic Araguaia Belt in the east and south, and overlain by Paleoproterozoic sequences generically assigned to the Uatumã Supergroup in the west (Araújo and Maia, 1991; DOCEGEO, 1988).

The Carajás Mineral Province is subdivided into two Archean tectonic domains: the older Mesoarchean Rio Maria Domain to the south and the younger Neoarchean Carajás Domain to the north (Araújo and Maia, 1991; Araújo *et al*., 1988; Dall'Agnol *et al*., 2006; DOCEGEO, 1988; Feio *et al*., 2013). A regional E–W shear zone, known as the Transition Subdomain (Feio *et al*., 2013), separates the Rio Maria and Carajás domains (Figure 7-2).

The basement of the Carajás Domain consists mainly of gneiss-migmatite-granulite terrains of the Xingu Complex (Machado *et al*., 1991; Pidgeon *et al*., 2000). These volcano-sedimentary sequences are covered by low-grade metamorphic sequences of clastic sedimentary rocks of the Águas Claras Formation.

Several mafic–ultramafic complexes intrude into both the Xingu Complex and the Archean volcano-sedimentary sequences (DOCEGEO, 1988; Ferreira Filho *et al*., 2007). These intrusions host large Ni laterite deposits (e.g., Onça-Puma, Vermelho and Jacaré) as well as PGM deposits (e.g., Luanga, Lago Grande).

GE21 Consultoria Mineral

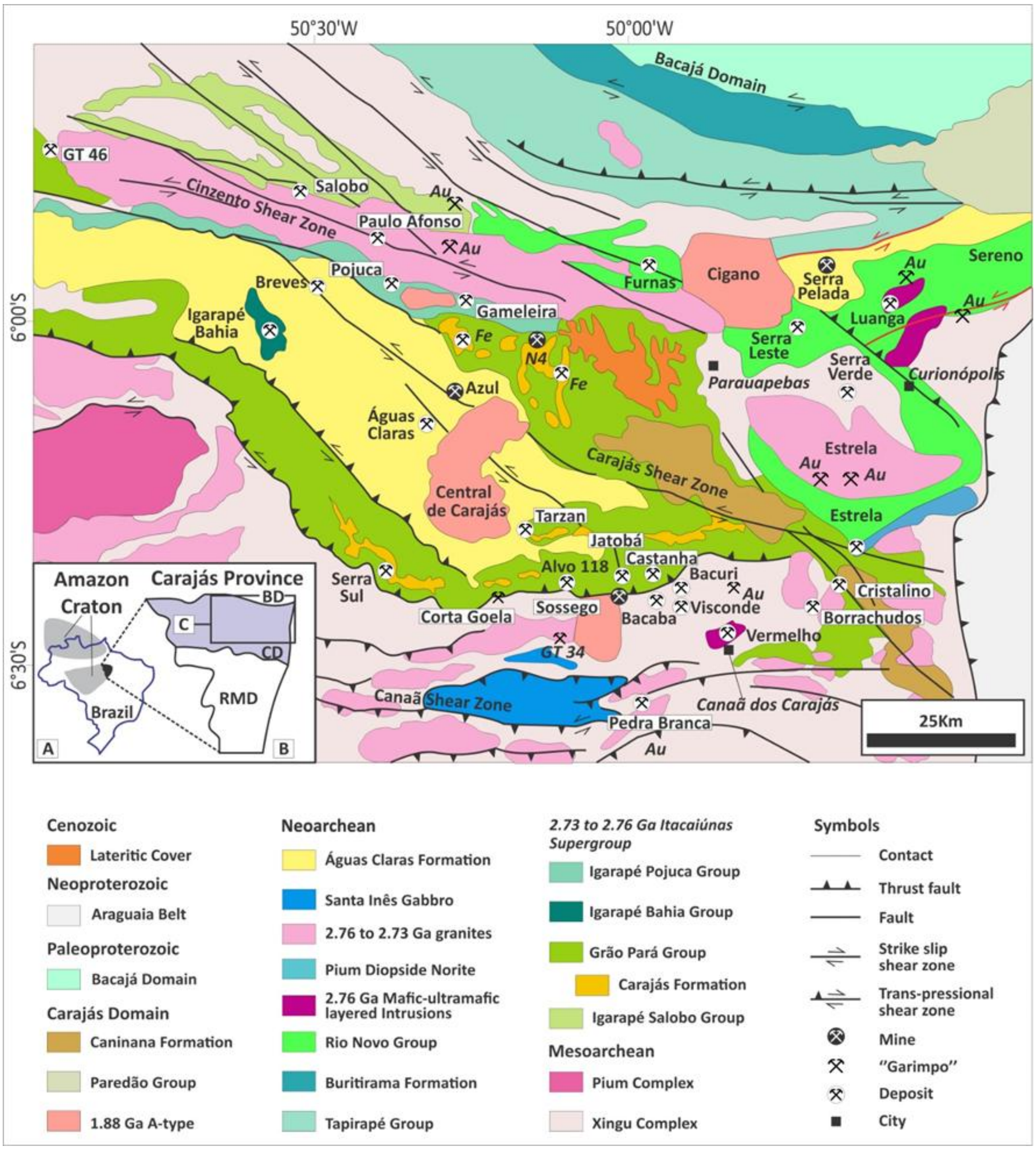


Figure 7-2: Geology and mineral deposits of the Carajás Mineral Province

Source: Trunfull et al., 2020.

7.1.2 The Serra Leste Magmatic Suite

The Serra Leste Magmatic Suite (Ferreira Filho *et al*., 2007) consists of a cluster of small- to medium-size layered mafic-ultramafic intrusions located in the north-eastern portion of the CMP. Mafic-ultramafic complexes are intruded into gneissic rocks of the Xingu Complex and/or volcanic-sedimentary rocks of the Grão Pará Group. Magmatic ages of the layered intrusions overlap with the age of the bimodal volcanism of the Grão Pará Group (2,759 ± 2 Ma, U-Pb in zircon and 2,760 ± 11 Ma, U-Pb in zircon), supporting the interpretation that they are part of a major Neoarchean magmatic event (Machado *et al*., 1991; Ferreira Filho *et al*., 2007).

The architecture of the intrusion and the crystallization sequence described in the Luanga and Lago Grande complexes indicate an overturned layered sequence (Ferreira Filho *et al*., 2007;

Teixeira et al., 2015). Even though the tectonic processes leading to the overturned sequence of layered rocks in the Lago Grande and Luanga complexes have so far not been studied in detail, regional structural studies in the Serra Leste region indicate significant tectonic transport that may have led to major overturned blocks (Holdsworth and Pinheiro, 2000; Tavares, 2015).

7.2 Regional Geophysics

The area is covered by airborne geophysical surveys. These surveys include magnetic, radiometric and electromagnetic data obtained by Geoterrex-Dighem in 1999.

Magnetic field anomalies highlight the structural framework and main geological features in the area. High signal values are associated with meta-ultramafic rocks of the mafic-ultramafic complexes and magnetite-rich shear zones related to the Serra Pelada Divergent Splay (SPDS). The meta-ultramafic rocks include dunite, meta-peridotite, serpentinite, sulphide-rich zones with pyrrhotite, and shear zones with magnetite.

The axes of the anomalies appear as anastomosed features that are ductile shear zones. Magnetite-rich, sub-parallel splays related to the Cinzento Transcurrent Shear Zone (CTSZ) crosscut the Luanga mafic-ultramafic complex (Figure 7-3). Magnetic highs are associated with magnetite-enriched meta-ultramafic rocks, such as dunite, peridotite, serpentinite, and talc-schist, as well as shear zones that truncate these complexes and remobilized the magnetite from the meta- ultramafic units. PGM mineralization in the mafic–ultramafic Luanga Complex is associated with pyrrhotite-rich meta-pyroxenite and chromitite layers close to the contact between meta-pyroxenite and peridotite/serpentinite.

Discrete circular high magnetic anomalies can be seen in the central part of the area. These anomalies are associated with shallow magnetic banded iron formation sources such as the Serra Leste iron ore deposit.

GE21 Consultoria Mineral

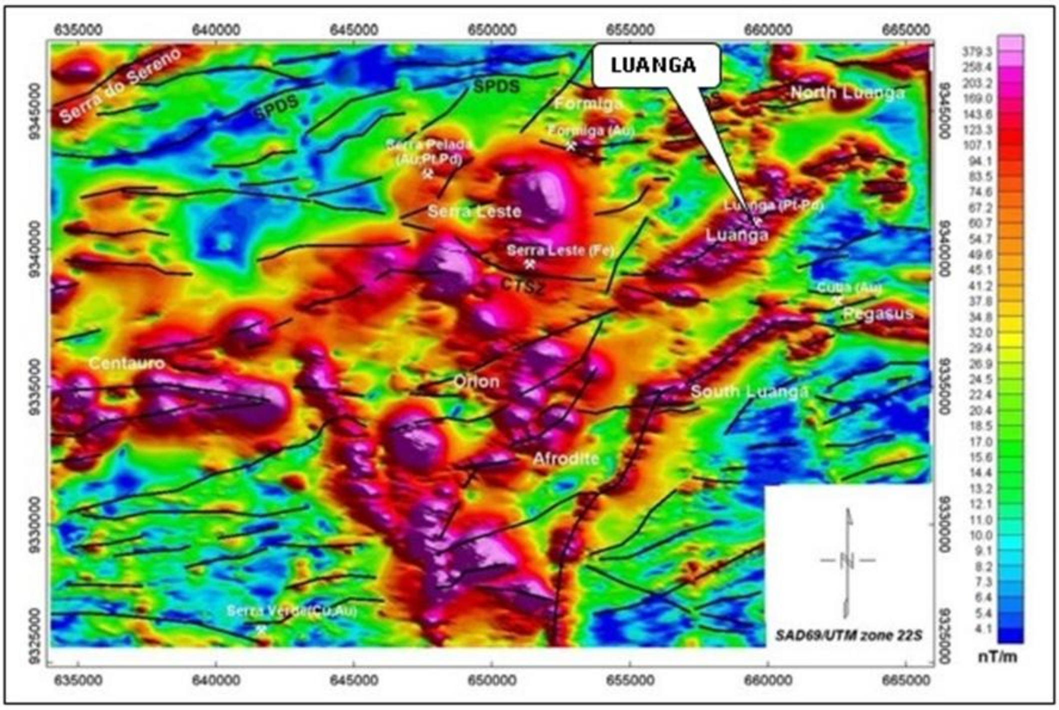


Figure 7-3: Regional aeromagnetic image with interpreted major structures shown as black lines

Source: CPRM data, 1999.

The Transient Electromagnetic (TEM) data shows conductive zones in: (a) the NE portion of the map, where there are NE-trending aligned features (part of the Serra Sereno; (b) the surroundings of the Formiga deposit, highlighting the thick alteration mantle and the meta-ultramafic rocks of the Formiga complex; (c) the mafic-ultramafic Luanga complex, where sulphide-related PGM mineralization occurs; and (d) the Serra Pelada Au-Pd-Pt deposit (Figure 7-4).

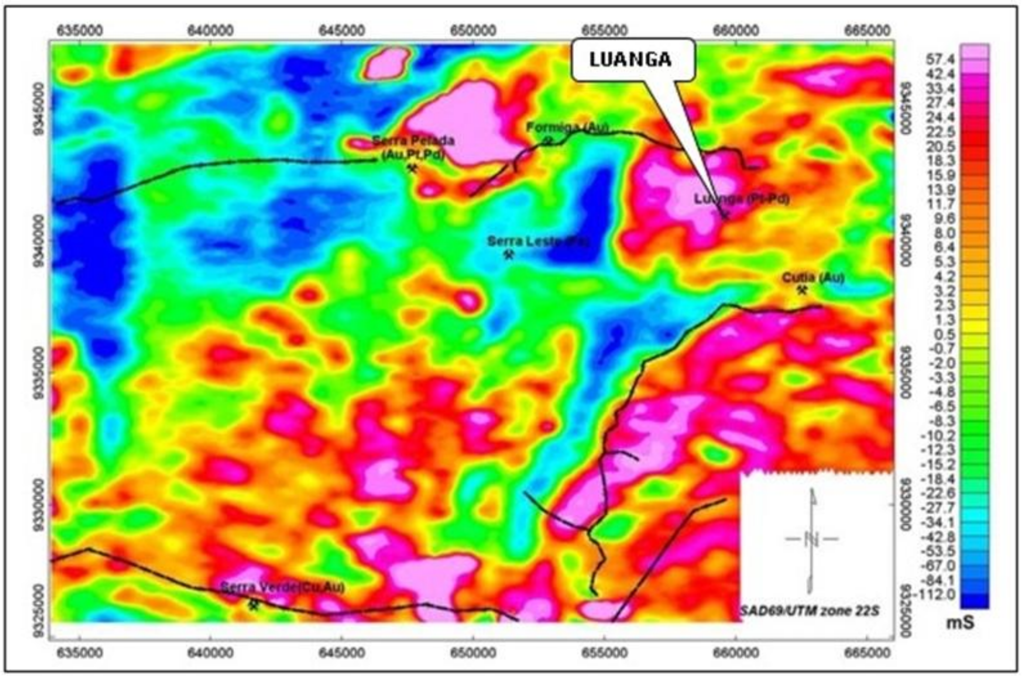


Figure 7-4: Regional airborne TEM image

Source: CPRM data, 1999.

High values of total radiation count in the image below can be related to the presence of Archean granitic rocks of the Estrela Granite, Xingu Complex. Intermediate to high total radiation count values in the central area reflects the sericite-rich metasiltstones of the Rio Fresco Group.

Low values of total gamma radiation count are associated with outcrops of the mafic–ultramafic complexes (a = Luanga, b = Luanga South) and appear as dark blue colours (Figure 7-5).

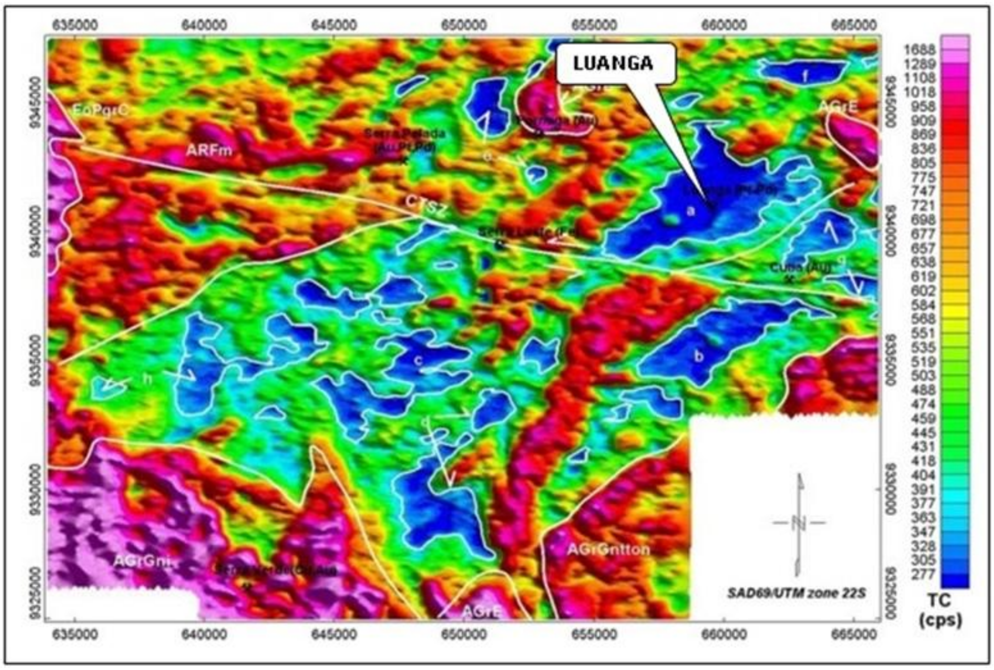


Figure 7-5: Regional air-radiometric image (total count channel) with major domains and structures in white

Source: CPRM data, 1999.

7.3 Local Geology

The principal geological unit on the mineral property is the Luanga Layered Mafic-Ultramafic Complex ("Luanga Complex"). The Luanga Complex is a 6 km long and up to 3.5 km wide (~18 km²) mafic-ultramafic layered intrusion that belong to the Neoarchean Large Igneous Province (LIP) of the Carajás Mineral Province. The intrusion is characterized by abundant unweathered outcrops, massive blocks and boulders. The layering forms an arc-shaped structure that matches the morphology. Host rocks of the Luanga Complex consist of highly foliated gneiss and migmatite of the Xingu Complex in the south/southeast, and mafic volcanics and iron formations of the Grão Pará Group in the north/west (Figure 7-6 A).

The central portion of the Luanga Complex has the thickest sequence of layered rocks. To the north and northeast, the layered sequence is truncated by granitic intrusions and to the south, it becomes progressively thinner. The Luanga Complex and host rocks are crosscuted by NNW-SSE dolerite dykes. These vertical dykes are up to several metres wide and consist of fine-to medium-grained intergranular to ophitic textured rocks. They belong to a Proterozoic swarm of magnetic mafic dykes that occurs in the Serra Leste region (Teixeira, 2013; Teixeira *et al*., 2015).

Geological sections (Figure 7-6 B) defined by drilling indicate that igneous layers have steep dips to the SE in the central and southwestern portions of the Luanga Complex. These sections indicate that the Ultramafic Zone overlies the Transition Zone, which overlies the Mafic Zone, suggesting that the layered sequence is tectonically overturned. An overturned layered sequence was previously described for the Luanga Complex (Ferreira Filho *et al*., 2007) and for the Lago Grande Complex (Teixeira *et al*., 2015). These studies suggest the existence of regional scale structures leading to large, overturned blocks in the Serra Leste region.

The Luanga layered sequence is subdivided in three zones, based on the different type and/or proportion of cumulus minerals, Ultramafic Zone (UZ), Transition Zone (TZ) and Mafic Zone (MZ). The estimated thickness of the layered sequence is some 3,500 m, as indicated in both the stratigraphic column (Figure 7-6 C), which is likely to represent the axial portion of the original magma chamber.

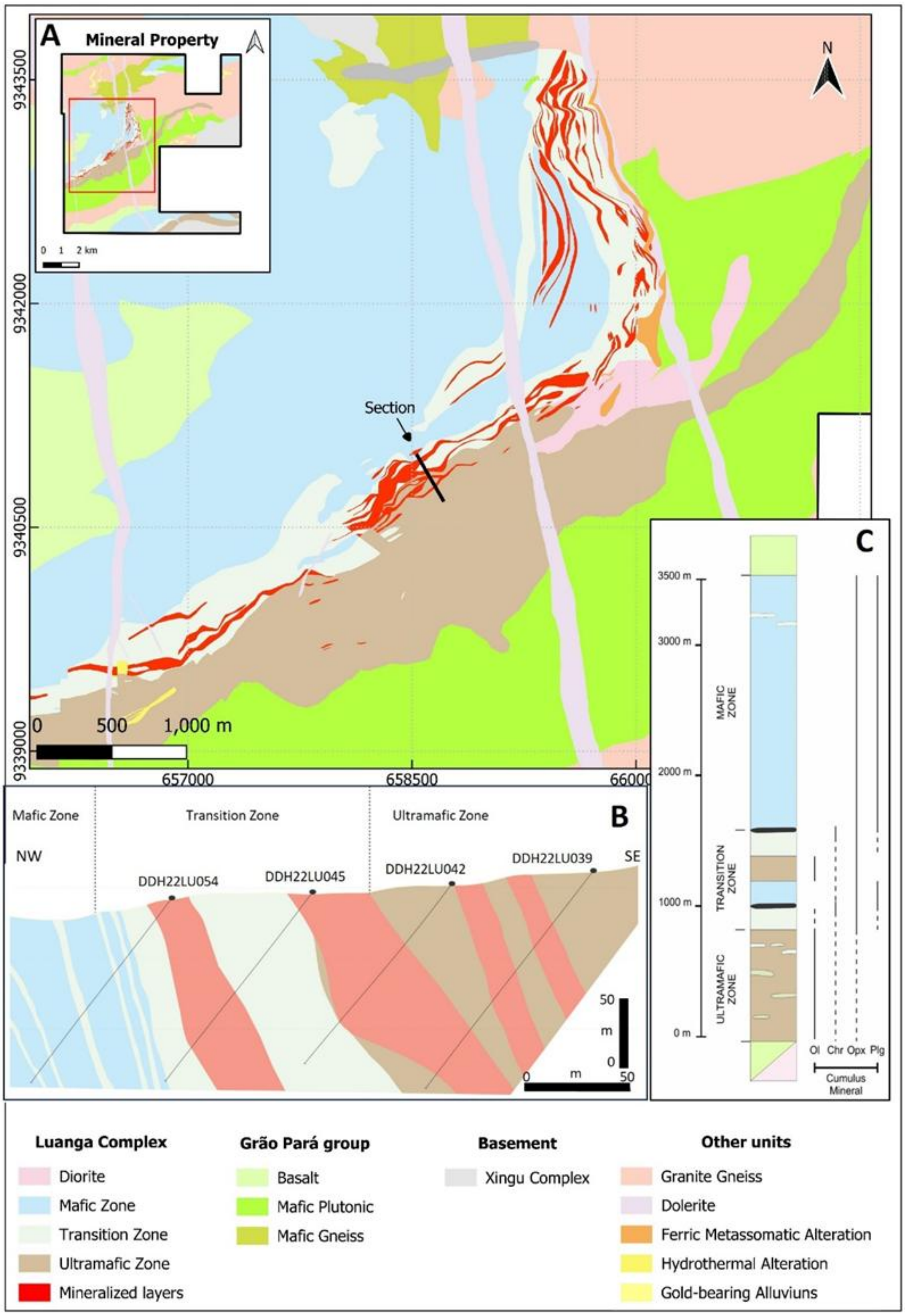


Figure 7-6: Luanga Complex A) Geological map. B) Section of the Central Sector, C) Stratigraphic column

Source: Bravo, 2025.

7.3.1 Ultramafic Zone

The Ultramafic Zone (UZ) is about 9 km long, up to 1 km wide and has an estimated thickness of 800 m. It consists of wehrlite (Olivine (Ol) + Clinopyroxene (Cpx) cumulates) and lesser dunite and lenses of clinopyroxenite. The Ultramafic Zone follows the stratigraphy of the

Luanga Complex in the southwest and central sectors but extends to the NE of the Luanga Complex as an irregular domain of ultramafic rocks (Figure 7-6 A). The lower contact of the Ultramafic Zone with the Xingu Complex and Grão Pará Group is poorly exposed and was mapped mainly by ground + air magnetic data and soil geochemistry. Typically, ultramafic rocks in the basal ultramafic zone are extensively sheared and altered, consisting of variable proportions of serpentine, amphiboles, talc and magnetite. Domains where primary magmatic textures and minerals are preserved include wehrlite (Figure 7-7 A) and clinopyroxenite (Figure 7-7 B and C). The NE extension of the Ultramafic Zone is delineated by Cr-Ni soil anomalies, blocks of serpentinite and restricted drilling data. This irregular zone of ultramafic rocks may represent the feeder conduits of the Luanga Complex.

7.3.2 Transition Zone

The Transition Zone (TZ) is about 5 km long and up to 1 km wide, comprises a pile of interlayered ultramafic and mafic cumulate rocks, which is up to 500 m thick. Interlayering of different rock types in different scales (from a few centimetres up to dozens of metres), is a distinctive feature of the Transition Zone. Cumulate rocks have variable textures and assemblages. The most common rock types are orthopyroxenite and lesser interlayered harzburgite and norite.

Orthopyroxenite is a medium- to coarse-grained orthopyroxene cumulate. The texture varies from adcumulate (Figure 7-7 E and F) to meso- and orthocumulate. Harzburgite is a medium- to coarse-grained olivine+chromite cumulate (Figure 7-7 D). Norite is a medium-grained orthopyroxene+plagiclase rock. Primary textures and minerals are variably altered to fine-grained aggregates consisting mainly of talc, serpentine and minor magnetite in the ultramafic rocks.

Chromitite layers with variable thickness (commonly 10 cm but up to 60 cm) and textures occur mainly in the upper portions of the Transition Zone and the lowermost portion of the Mafic Zone. These layers are commonly discontinuous but intercepts consisting of several chromitite layers are up to 100 m in extension. Chromitites are fine- to medium-grained chromite cumulates with variably altered intercumulus plagioclase and orthopyroxene. Chromite crystals commonly have atoll textures or abundant silicate inclusions (Figure 7-7 H and I). The upward transition from massive chromitite, to chain textured chromitite and disseminated chromitite (Figure 7-7 G) is common and provides a facing criterion for the igneous stratigraphy of the Luanga Complex.

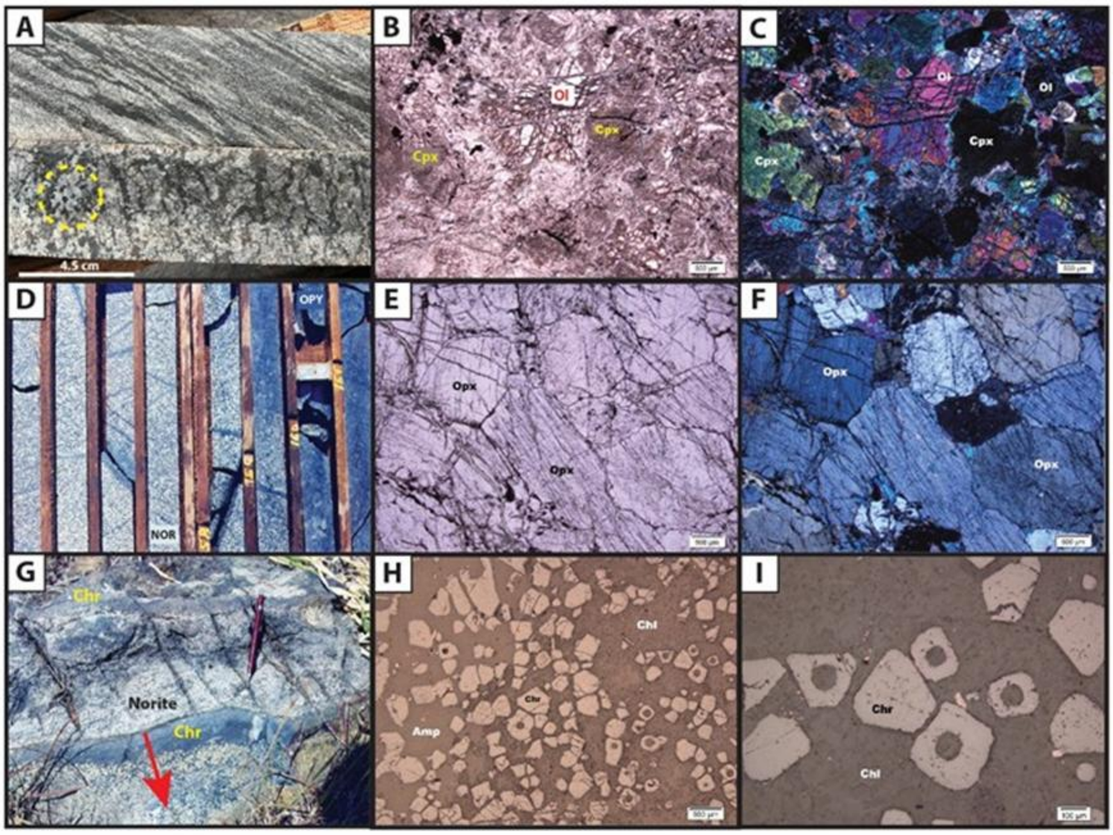


Figure 7-7: Photos and photomicrographs of representative rock types of the Luanga Complex

Legend: A) Drill core samples of the UZ showing wehrlite with primary magmatic texture (lower photo) and a sheared domain (upper photo). The dashed circle shows coarse-grained clinopyroxene (Cpx) (light color mineral) enclosing several euhedral olivine pseudomorphs (dark color mineral). B) and C) Photomicrographs of wehrlite of the UZ under parallel (N//) and crossed (NX) polarized light. D) Drill core section showing a sharp contact between norite (NOR) and orthopyroxenite (OPY) of the TZ. E) and F) Photomicrographs of adcumulate textured orthopyroxenite of the TZ (under N// and NX). G) Outcrop of interlayered chromitite and norite from the lower portion of the MZ. The arrow indicates the base-to-top gradational transition from massive chromitite to chain textured chromitite. H) and I) Photomicrograph of chromitite consisting of fine-grained euhedral chromite with atol texture (observation under reflected light). The matrix consists of an aggregate of chlorite (Chl) and amphiboles (Amp) resulting from alteration of primary orthopyroxene and amphiboles (Amp) resulting from alteration of primary orthopyroxene and plagioclase.
Source: Bravo, 2025.

The Mafic Zone (MF), about 5 km long and up to 3.5 km wide, comprises a thick monotonous pile of noritic rocks. Norite consists of medium-grained orthopyroxene + plagioclase cumulates. Primary textures and minerals are variably altered to fine-grained aggregates consisting mainly of amphiboles (hornblende-actinolite), chlorite and epidote-group minerals. Minor interlayered ultramafic rocks in the MZ, including orthopyroxenite and minor chromitite, have petrographic features like those described in the TZ.

7.3.3 Tectonic Setting

The current model of the tectonic evolution of the CMP implies that it is a product of oblique rift evolution (2.76-2.06 Ga) controlled by the Cinzento and Canaã strike-slip system (Figure 7-8). The Carajás rift is hosted in the tonalite-trondhjemite-granodiorite (TTG) terrain which includes the Rio Maria greenstone belt (3.0-2.85 Ga). The rifting stage (Igarapé Bahia Group + Grão Pará Group) began with intense tholeiitic fissure volcanism with basalt and rhyolite flows, coeval intrusions of alkali granites and mafic-ultramafic bodies, BIFs, shales and restricted tuff beds.

The oblique rifting results in dextral shearing along the Canaã dos Carajás and Cinzento shear zones (Figure 7-8). Stage 1 is responsible for mafic-ultramafic intrusions, alkalic granite emplacement, and early Iron-Oxide Copper Gold (IOCG) mineralization at the Igarapé Bahia, Bacaba, Serqueirinho, and Sossego deposits. In Stage 2 the most important IOCG deposits like Salobo, Alemão and Igarapé Bahia were formed.

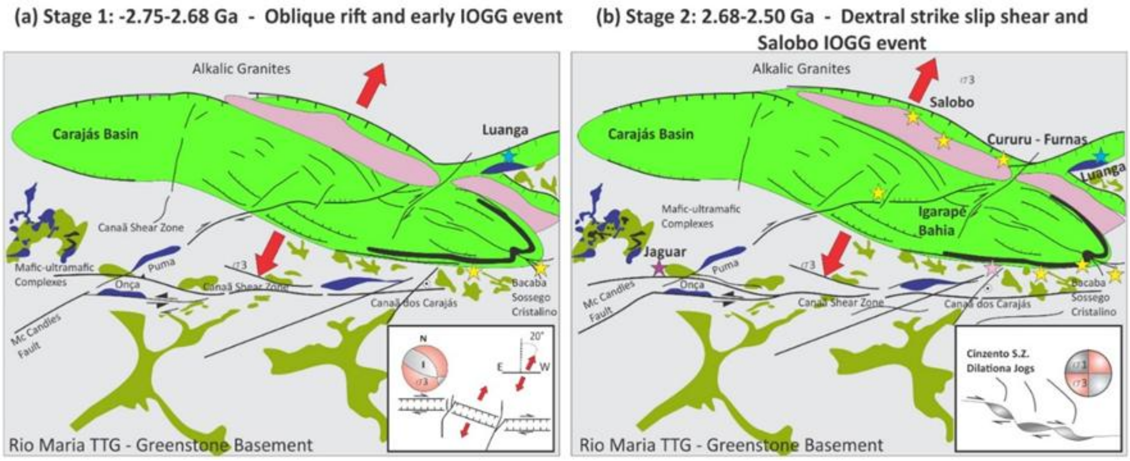


Figure 7-8: Simplified early tectonic evolution of the Carajás Basin and adjoining regions

Source: Teixeira et al., 2021.

The Luanga complex is positioned between the Cinzento and Sereno sinistral strike-slip faults. A transtensional zone of this transcurrent system could generate space and pumping (high pressure zones) or suction forces (low pressure zones) that would favor the entry of mafic-ultramafic magmas of the Luanga Complex (Figure 7-9).

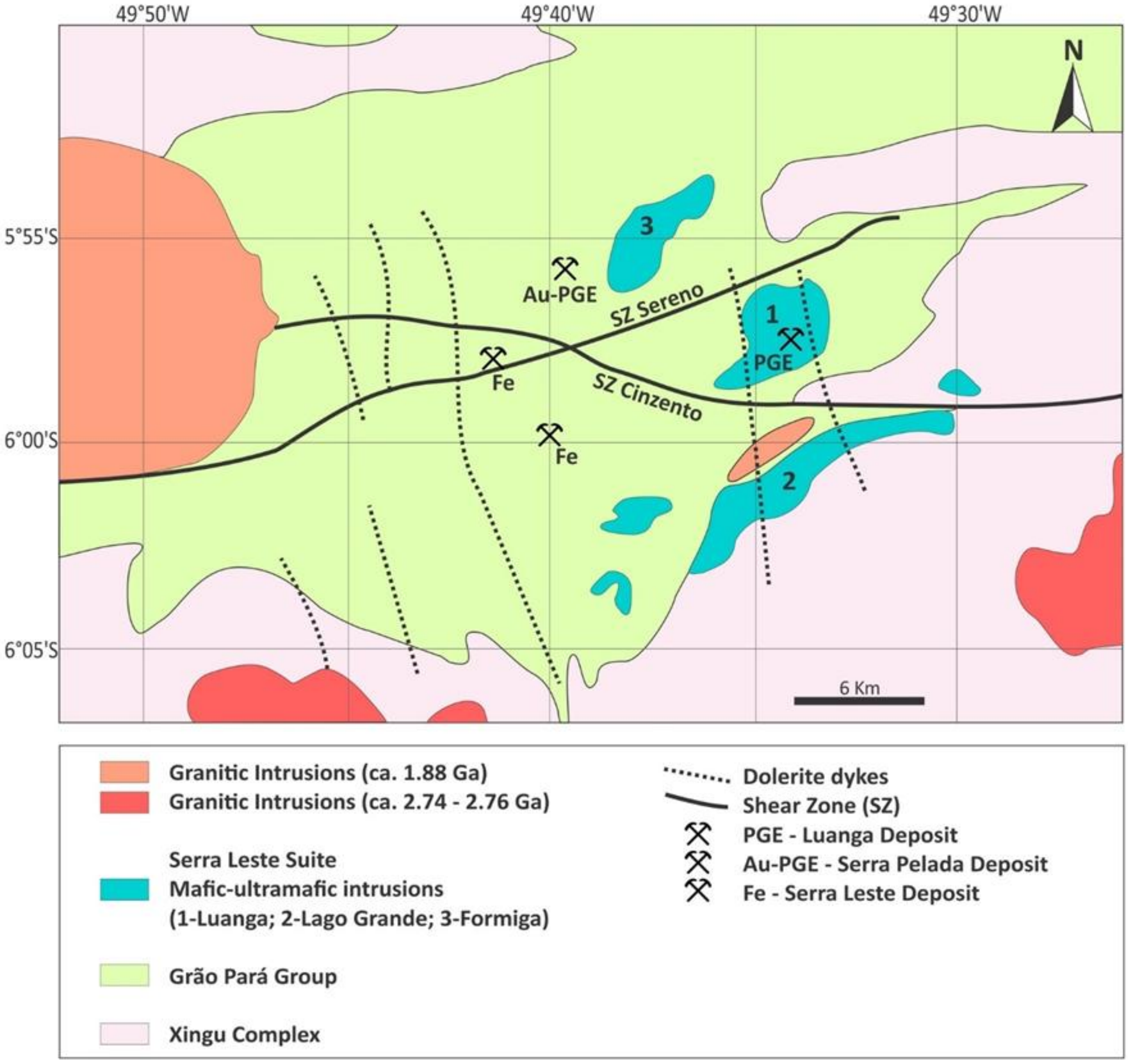


Figure 7-9: Major structures controlling Luanga intrusion

Source: Teixeira & Ferreira Filho, 2015.

In contrast, releasing structures at sites of transtension may take the form of rifts and pull-apart basins where the detailed internal geometry of motion of blocks within the rift or pull-apart has a vertical and lateral sense of displacement. Transtension generates local open space in the crust, and sub-vertically connected open spaces provide a pathway through which mantle-derived magmas can migrate from the mantle to the crust (Lightfoot & Evans-Lamswood 2015).

7.3.4 Metamorphism and Alteration

Metamorphic assemblages commonly replace the primary igneous minerals of the Luanga Complex. This widespread metamorphic alteration is heterogeneous and characterized by an extensive hydration that largely preserves primary textures, bulk rock compositions and the compositional domains of igneous minerals. The penetrative fabric is restricted to narrow domains of up to a few metres across, and igneous textures are identified in adjacent non-deformed domains. These assemblages include: (1) serpentine + talc + magnetite ± cummingtonite commonly overprinting harzburgite/orthopyroxenite, (2) serpentine + tremolite + talc + magnetite overprinting wehrlite/clinopyroxenite and (3) hornblende + chlorite + epidote overprinting norite.

Metamorphic assemblages in the Luanga Complex indicate temperatures at the transition from greenschist to the amphibolite facies of metamorphism (Ferreira Filho *et al.*, 2007; Mansur *et al.*, 2016).

Apart from the widespread metamorphism affecting the Luanga Complex and host volcanic-sedimentary sequence, the layered rocks are also impacted by hydrothermal alteration developed along crosscutting shear zones. Domains of hydrothermal rocks occur along the eastern and southeastern border of the Luanga Complex, being particularly robust at the North Sector, where drill intercepts up to 100 m thick of metasomatic rocks occur. Two main types of hydrothermal alteration were recognized in the Luanga Complex and adjacent host rocks. An older event of K-Fe-Mg-Cl metasomatism characterized by biotite-amphibole (± scapolite, ± tourmaline) assemblage, and a second event of Fe-Ca metasomatism characterized by magnetite-Ca-amphibole-grunerite-garnet assemblage. These hydrothermal events were first recognized in the Luanga Complex during the exploration program carried out by Bravo and were interpreted to belong to the regional hydrothermal system linked to IOCG deposits in Carajás Mineral Province.

7.3.5 Mineralization

The Luanga PGM + Au + Ni mineralized envelope follows the arc-shaped structure of the Mafic-Ultramafic Complex along approximately 8.1 km. The deposit is subdivided into three separate mineralized sectors, named North, Central and Southwest (Figure 7-10).

The TZ of the Luanga Complex hosts several PGM mineralized units, including the Main Sulphide Zone (MSZ) which hosts the bulk of PGM resources of the Luanga Complex. Other mineralized layers are identified within the TZ and on the UZ.

In addition, several thin chromitites layers or lenses occur in the Luanga Complex either in the upper or in lower stratigraphic portions of the TZ, the latter occurring where they are hosted by ultramafic cumulates. The upper chromitite layers are developed on the immediate contact with the overlying MZ, where they are hosted by plagioclase-bearing norite cumulates (Figure 7-11).

GE21 Consultoria Mineral

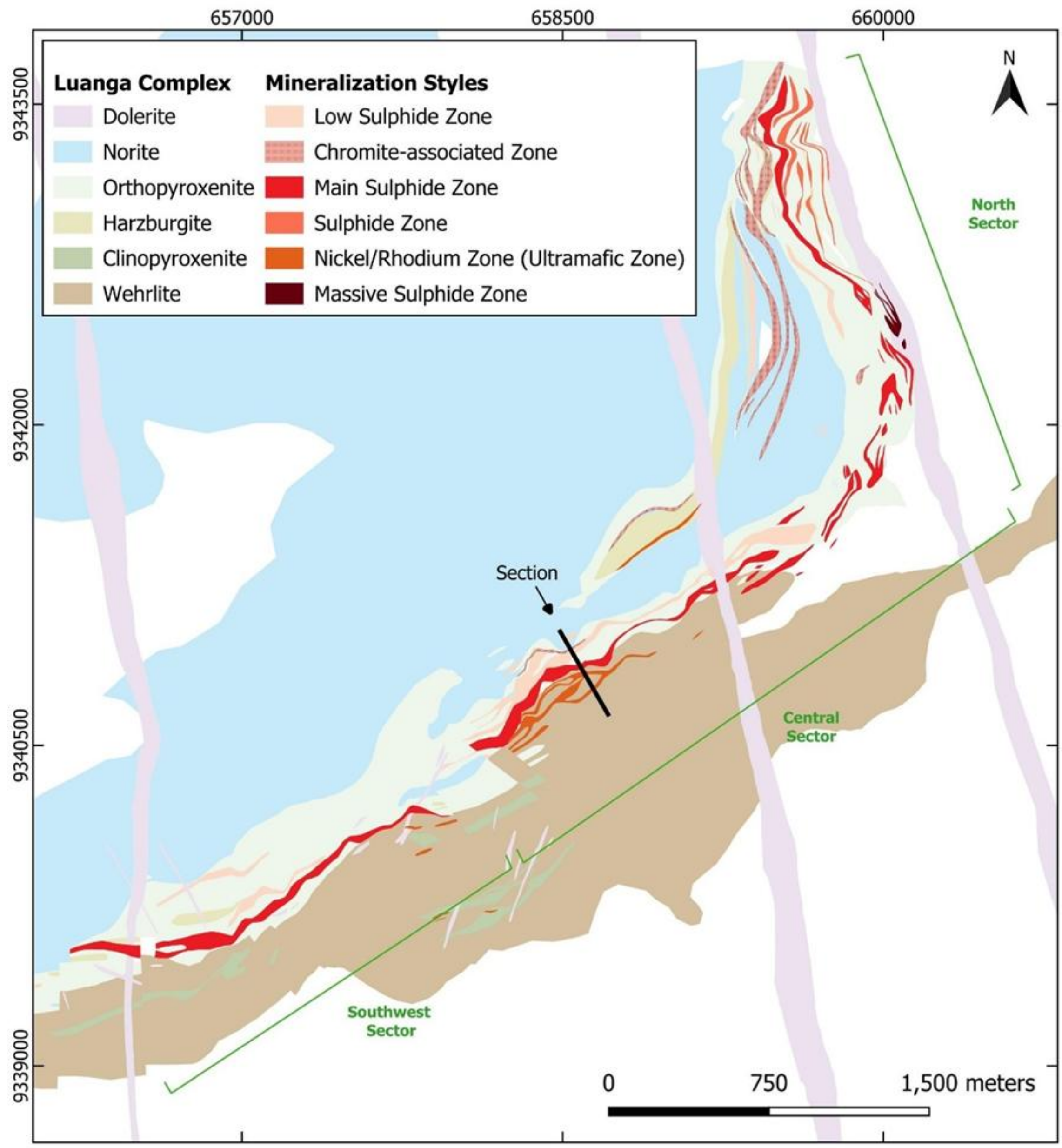


Figure 7-10: Mineralization and lithology map of the Luanga Project

Source: Bravo, 2025.

Mineralized zones of the Luanga Complex are grouped into six different styles of PGM + Au + Ni mineralization (Figure 7-1). They are named as: (i) MSZ, (ii) Low Sulphide Zone (LSZ), (iii) Chromite-associated Zone (Chr-PGM) (iv) Nickel/Rhodium Sulphide Zone (Ni-Rh), (v) Sulphide Zone (SZ) and (vi) Massive Sulphide Zone (MASU). An example of the mineralized zones in a cross section from the Central Sector is presented in the Figure 7-12, and the following topics provide an overview of the main characteristics for each style of PGM + Au + Ni mineralization.

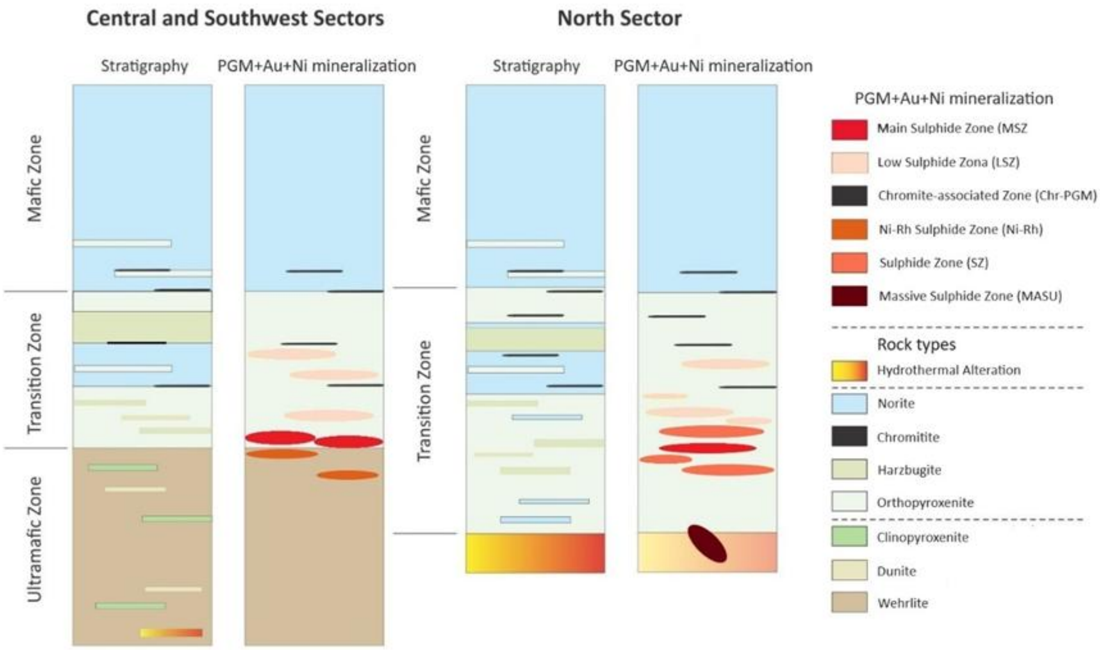


Figure 7-11: Schematic stratigraphic column locating the different styles of PGM mineralization on Luanga Complex

Source: Bravo, 2025.

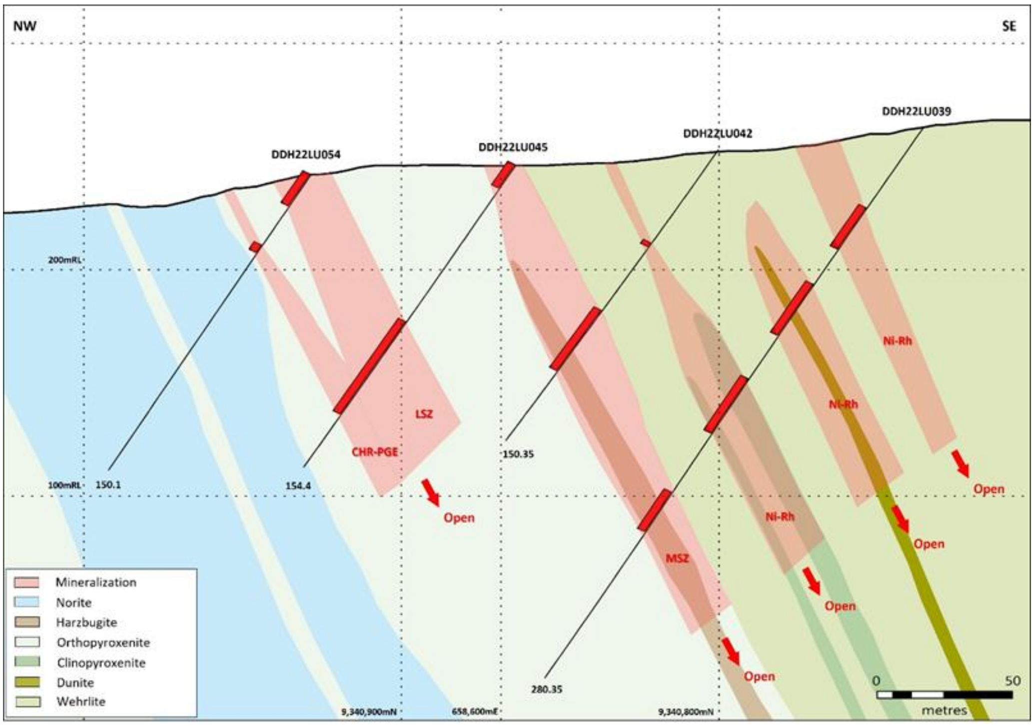


Figure 7-12: Drill section with the mineralized zones identified at Luanga

Source: Bravo, 2025.

GE21 Consultoria Mineral

Table 7-1: Styles of mineralization on Luanga deposit

PGM + Au + Ni Mineralization	Occurrence	Stratigraphy	Geology	Petrography	Chemical Composition
Main Sulphide Zone (MSZ)	Central and Southwest sectors.	Lower portion of the TZ at the contact with the UZ.	Stratabound disseminated sulphides (1-5%) with variable thickness (up to 50 m) hosted by orthopyroxenite and minor harzburgite.	Sulphide blebs interstitial to cumulus Opx-Ol. Po-Pn and minor Ccp. Pn (25-50% of Sulphide fraction) occurs as fine-grained crystals with rare exsolutions within Po.	Very high Ni (10-15%) and Pt-Pd (up to 100 ppm) tenors, variable very high Ni/Cu (~10), Pt/Pd < 0.5, and Pd/Rh ~0.05.
Low Sulphide Zone (LSZ)	All sectors	Discontinuous stratabound bodies (lenses) distributed throughout the stratigraphy of the TZ.	Several lenses of variable thickness (up to 200 m) hosted by orthopyroxenite, and minor harzburgite. No significant amounts of sulphides.	PGM mineralized lenses occur in Ol-Opx cumulates without any distinctive feature from barren cumulates.	Very low Ni-Cu contents, Pt-Pd contents < 1-2 ppm, and Pt/Pd ~ 1.0-2.0. Pt-Pd show moderate to strong positive correlation.
Chromite-associated Zone (Chr-PGM)	All sectors	Irregular lenses located predominantly at the upper portion of the TZ and lower portion of the MZ.	Chromitite pods (< 60 cm) closely associated with disseminated clusters of chromite within orthopyroxenite, harzburgite and norite. No significant amounts of sulphides.	Chromitite pods consist of fine-grained cumulus Chr commonly with abundant inclusions and/or atol texture. DIS chromite occurs as cluster of several euhedral chromite crystals.	Very low Ni-Cu contents, high Pt/Pd ratio (~4.0), and Rh/Pt ~0.3.
Ni-Rh Sulphide Zone (Ni-Rh)	Central and Southwest sectors.	Discontinuous lenses located only inside the UZ domain.	Sulphide-bearing lenses (up to 25%) with variable thickness (up to 40 m). Hosted by interlayered wehrlite and dunite, and minor clinopyroxenite.	Disseminated to net-textured sulphides consisting of Po (~60-70%), Pn (~30-40%) and minor Ccp. Pn occurs as medium-grained crystals (1-2 mm) with very rare Pn exsolutions within Po.	Pt/Pd ~0.15; Rh/Pd ~0.20. Higher % of sulphides and lower Pt-Pd contents than the MSZ, suggest relatively lower Pt-Pd tenors. Modal % of Pn (~30-40%) indicates very high Ni tenor.
Sulphide Zone (SZ)	North sector	N-S trending zones of massive to semi-massive sulphides. Stratigraphic correlation with the Central and Southwest sectors is hampered by intense tectonism and alteration.	Folded or faulted stratabound bodies of disseminated fine-grained sulphides (1-5%) with variable thickness (up to 40 m). Hosted by orthopyroxenite and minor harzburgite. Layered mafic-ultramafic sequence are crosscut by granitoids at depth.	Pervasively altered rocks with poorly preserved primary textures. Consist mainly of fine-grained aggregates of sulphides (Po-Pn and minor Ccp) interstitial to pseudomorphs of Opx.	Low Pt/Pd (commonly < 0.5, but up to 1.0) and variable Ni contents. Weak positive correlation between Pd-Ni and Pd-Rh possibly results from variably altered sulphides.
Massive Sulphide Zone (MASU)	North sector	Robust intercept of MASU (DDH22-LU047) within a hydrothermal alteration zone at the western border of the North sector.	Host rocks and the footwall sequence consist of hydrothermal rocks with variable proportions of Amp-Grt-Bt-Mag. Sulphide-bearing orthopyroxenite and interlayered norite occur in adjacent drill holes.	Sulphides consist of Po (~80-90%) and Pn (~10-20%) with local Ccp-rich domains. Amp-Bt-Qz-Mag-Bt are enclosed within MASU, with common sulphide-Amp intergrowths. Pn occurs mainly associated with Po as fine-grained crystals or exsolutions.	Variable contents of Ni-Cu-PGM, with Ni>Cu and Pd>Pt. Weak correlation between metals (Ni-Cu-Pt-Pd-Rh). Contents of Ni (< 6.0%), Pd (< 6.0 ppm), Pt (< 2.5 ppm) and Rh (0.2 ppm) indicate lower tenors than those calculated for the MSZ.

Source: Bravo, 2025.

7.3.5.1 Main Sulphide Zone (MSZ)

PGM mineralization associated with disseminated sulphides in the MSZ hosts the bulk of PGM + Au + Ni Mineral Resources of the Luanga Complex. The stratigraphic interval hosting the MSZ consists of a 10–50 m thick interval with disseminated sulphides located along the contact of the UZ and TZ (Figure 7-11).

PGM mineralization is associated with sulphide blebs (up to few cm), interstitial to cumulus Opx-Ol or their pseudomorphs. Sulphides consist of Po-Pn and minor variably distributed Ccp. Pn (~30-50% of the sulphide fraction) occurs mainly as fine-grained crystals (<1 mm) with rare Pn exsolutions within Po (Figure 7-13).

PGM mineralization in the MSZ consists predominantly of Pt-Pd-bismuth tellurides followed by stannides, arsenides and antimonides (Mansur *et al*., 2020). The PGM occurs mainly enclosed within sulphide grains, or at the contact between sulphide and silicate grains.

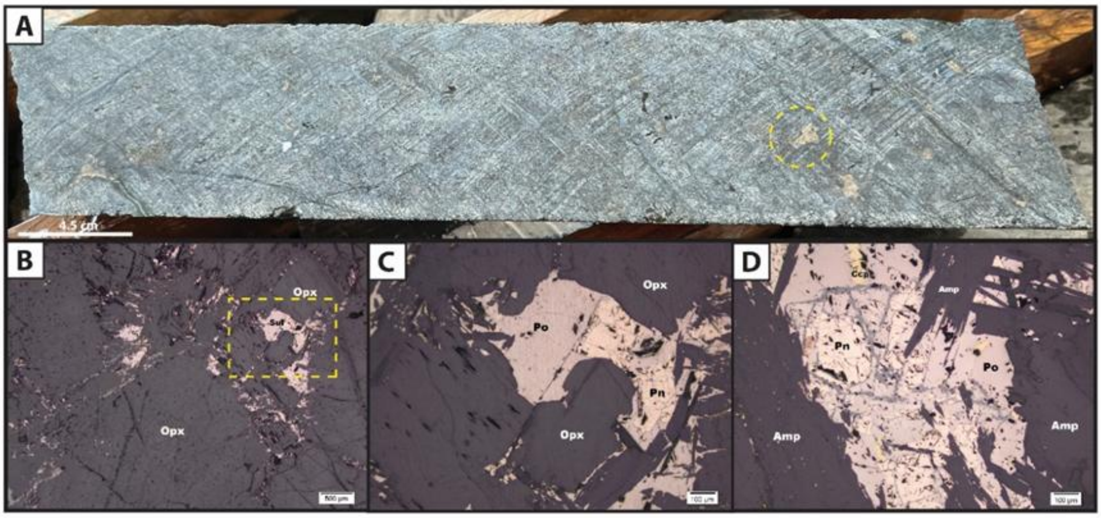


Figure 7-13: Representative photos and photomicrographs of the MSZ

Legend: A) Drill core samples of orthopyroxenite with interstitial disseminated sulphide blebs. The dashed circle shows a sulphide bleb. B) and C) Photomicrographs of sulphide blebs consisting mainly of pyrrhotite (Po) and pentlandite (Pn). Sulphides are interstitial to Opx and partially remobilized along fractures. The dashed rectangle is detailed in B. D) Photomicrograph of a sulphide bleb consisting of Po+Pn and minor Ccp. Sulphides are partially altered to magnetite (gray color lamellae) and occur in a domain where Opx is partially altered to prismatic amphibole (Amp).
Source: Bravo, 2025.

7.3.5.2 Low Sulphide Zone (LSZ)

The LSZ style of mineralization comprises PGM-mineralized rocks devoid of base metal sulphides and/or abundant chromite. The Low Sulphide mineralization of the Luanga Complex consists of up to 30 m thick stratabound zones across the TZ. The hosting rocks, mainly harzburgite and orthopyroxenite.

The LSZ occurs as several irregular stratabound bodies (lenses) located above the MSZ and throughout the stratigraphy of the TZ in the Central and Southwestern sectors, as well as in the North sector (Figure 7-11). This style of mineralization is hosted by orthopyroxenite, and minor harzburgite and plagioclase-orthopyroxenite (Figure 7-14). The LSZ lenses have a high steep dip to the southeast (Central and Southwestern sectors) and to the west (North sector).

PGM mineralization on the LSZ consist predominantly by Pt-arsenides (mainly sperrylite, $PtAs_2$), followed by Pt-Pd-stannides, antimonides, and minor bismuth tellurides (Mansur *et al*., 2020).

Figure 7-14: Representative photo of the LSZ

Legend: Drill core sample of orthopyroxenite with interstitial plagioclase. The dashed circle shows a domain with medium- to coarse-grained Opx pseudomorphs (replaced mainly by talc) with interstitial plagioclase pseudomorphs (replaced by mainly by chlorite). Whitish veinlets consist mainly of talc and minor carbonate.
Source: Bravo, 2025.

7.3.5.3 Chromite-associated Zone (Chr-PGM)

Chr-PGM mineralization occurs as chromitite pods (< 60 cm) closely associated with intercepts with disseminated clusters of chromite within orthopyroxenite, harzburgite and norite (Figure 7-15 A to C). This style of mineralization is stratigraphical located predominantly at the upper portion of the TZ and lower portion of the MZ.

Chromitite pods consist of fine-grained cumulus chromite. Disseminated chromite occurs as clusters of several euhedral chromite crystals (Figure 7-15 D to G).

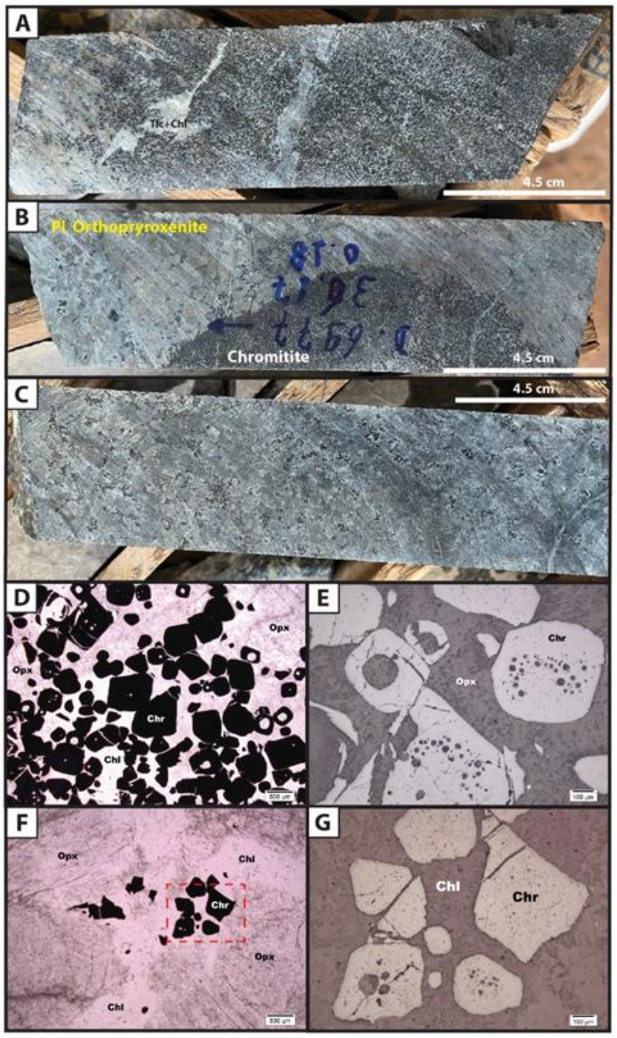


Figure 7-15: Representative photos and photomicrographs of Chr-PGM mineralization

Legend: A) Drill core sample of chromitite. Fine-grained chromite (Chr) occurs as euhedral black crystals in a light gray matrix consisting of chlorite (Chl) and talc (Tlc). B) Drill core sample of a chromitite pod hosted in a plagioclase-orthopyroxenite (pOPY) with disseminate clusters of Chr (black crystals). C) Drill core sample of a pOPY with disseminated clusters of Chr. D) Photomicrograph of a chromitite consisting of fine-grained euhedral chromite with common atol texture. Chr is associated with Opx pseudomorphs (Amp+Tlc) and fine-grained aggregates of Chl. E) Photomicrograph of euhedral Chr crystals hosted in Opx pseudomorphs (Amp+Tlc). F) Photomicrograph of pOPY with disseminated clusters of Chr The rock consists of euhedral Opx pseudomorphs (Amp+Tlc) and interstitial aggregates of Chl (plagioclase (Pl) pseudomorphs). Fine-grained Chr crystals occur mainly associated with interstitial aggregates of Chl. G) Detailed photomicrograph of the dashed red rectangle in (F).
Source: Bravo, 2025.

7.3.5.4 Ni-Rh Sulphide Zone (Ni-Rh)

The Ni-Rh mineralized zone has been identified only within the UZ on the Central and on the Southwest sectors (Figure 7-11 and Figure 7-12). It occurs as lenses of disseminated to net-textured sulphides (up to 25%) interstitial to cumulus Cpx and olivine or their pseudomorphs (Figure 7-16 A to B).

Ni-Rh zones have variable thickness (up to 40 m) and are commonly hosted by interlayered wehrlite and dunite, and minor clinopyroxenite.

Sulphides consist of Po (~60-70%), Pn (~30-40%) and minor Ccp. Pentlandite occurs mainly as medium-grained crystals (1-2 mm) (Figure 7-16 C to F).

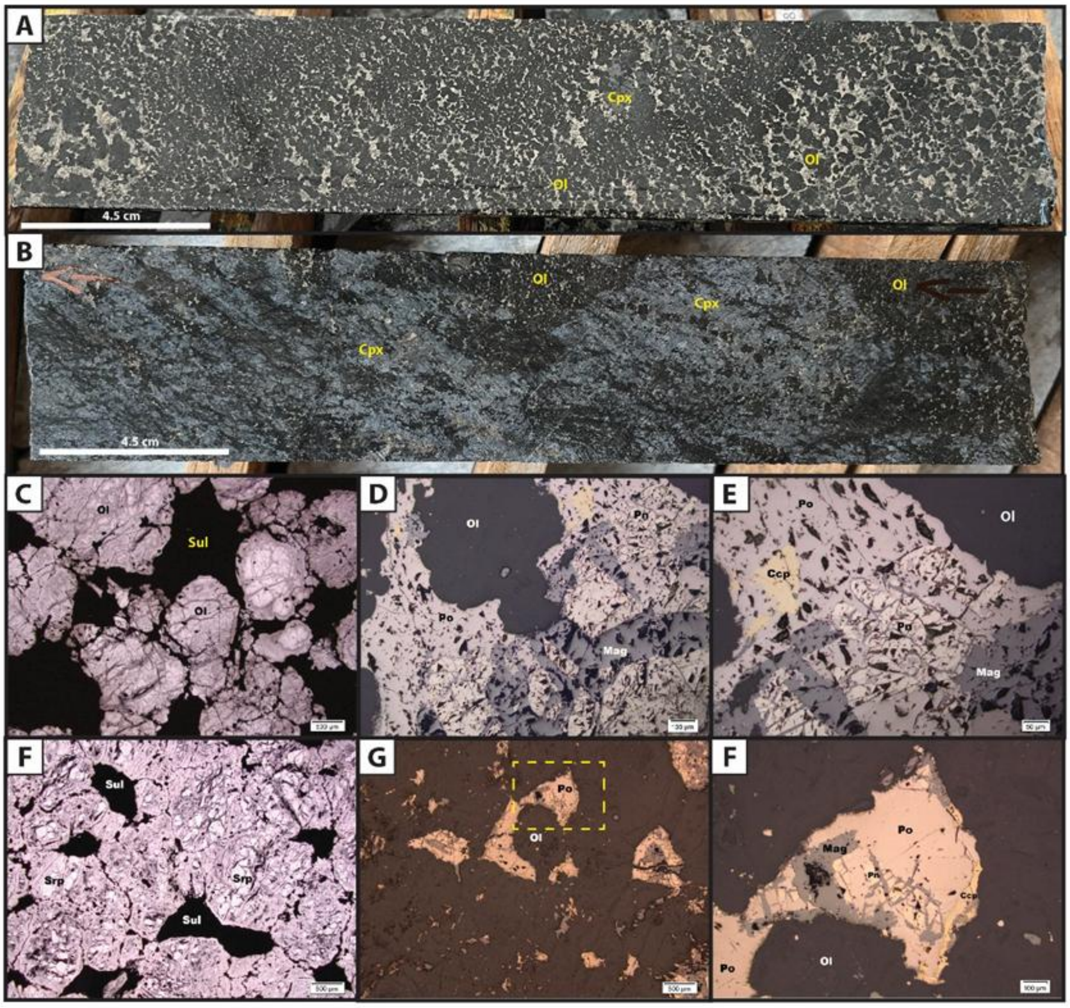


Figure 7-16: Representative photos and photomicrographs of Ni-Rh mineralization

Legend: A) Drill core sample of net-textured to matrix sulphides in dunite. Sulfides enclose euhedral olivine (Ol) pseudomorphs (black color). Minor coarse-grained anhedral Cpx crystals (dark gray color) partially enclose olivine. B) Drill core sample of disseminated interstitial sulphide blebs in wehrlite. The sample has irregular domains with variable amounts of Cpx (gray color) and Ol (black color). C) Photomicrograph of net-textured sulphides (Sul) interstitial to Ol pseudomorphs replaced by serpentine and minor magnetite. D) and E) Photomicrographs of sulphide aggregate enclosing Ol pseudomorph. Sulphides consist of pyrrhotite (Po), pentlandite (Pn) and minor chalcopyrite (Ccp), and are partially altered to magnetite (Mag). F) Photomicrograph of interstitial sulphide (Sul) blebs in a wehrlite altered to serpentinite. G) and H) Photomicrographs of sulphide blebs interstitial to Ol pseudomorph in wehrlite. Sulphides consist of pyrrhotite (Po), pentlandite (Pn) and minor chalcopyrite (Ccp), and are partially altered to magnetite (Mag).
Source: Bravo, 2025.

7.3.5.5 Sulphide Zone (SZ)

The SZ mineralization style is recognized exclusively in the Northern sector, characterized by mineralized bands of variable thickness hosted in rocks of the Transition Zone. The SZ occurs as several irregular N-S trending zones of disseminated sulphides hosted mainly in orthopyroxenite and is stratigraphically closely associated with Chr-PGM and LSZ mineralization (Figure 7-11).

The SZ occur as a ~2 km-long stacked stratabound bodies of disseminated fine-grained sulphides (1-5%) and this style of mineralization has highly variable thickness, reaching up to 40 m (Figure 7-17). This style consists mainly of fine-grained aggregates of sulphides interstitial to pseudomorphs of Opx. The sulphide paragenesis is predominantly Po-Pn and minor Ccp.

Chemically, the SZ is characterized by moderate to high PGM contents (30-40 m intercepts with 2-3 ppm of Pt + Pd) with predominantly low Pt/Pd (commonly <0.5, but up to 1.0) and variable Ni contents (up to 0.20%).

Figure 7-17: Representative photo of the SZ

Legend: Drill core sample of orthopyroxenite with interstitial plagioclase. Coarse- to medium-grained Opx pseudomorphs (replaced by talc and serpentine) are delineated by interstitial plagioclase pseudomorphs (dark color) replaced by chlorite. Fine-grained partially oxidized sulphide blebs are indicated by yellow arrows.
Source: Bravo, 2025.

7.3.5.6 Massive Sulphide Zone (MASU)

The MASU style of mineralization was identified in 2022 on the North sector where drill hole DDH22LU047 intercepted 11.0 m grading 4.24 g/t 3EqPt+Au, 2.04% Ni, 1.23% Cu within a hydrothermal alteration zone at the eastern border of the Luanga Complex (Figure 7-18 A to C).

Host rocks and the footwall sequence of the MASU consist of amphibolitite, massive rocks consisting of variable proportions of amphibole-garnet-biotite-magnetite and banded iron formation. Sulphide-bearing orthopyroxenite and interlayered norite occur above the MASU and in adjacent drill holes. Cumulate rocks close to the alteration zone are partially to pervasively replaced by the Fe-Ca-K hydrothermal minerals, including MASU to semi-MASU breccias, suggesting that the Ni-Cu-PGM mineralization originated by hydrothermal remobilization of primary sulphides. The alteration assemblage originated from a pervasive Fe-Ca-K hydrothermal alteration is possibly connected with the regional hydrothermal system associated with the IOCG deposits in Carajás.

Sulphides consist mainly of pyrrhotite (~80-90%) and pentlandite (~10-20%) with local chalcopyrite-rich domains (up to 60-70% Ccp). Variable proportions of amphibole and minor biotite, magnetite and quartz are enclosed within massive sulphides, with common sulphide-amphibole intergrowths. Pn occurs mainly associated with Po as fine-grained (<0.5 mm) crystals or exsolutions (e.g., flames, flakes) (Figure 7-18 D to G).

The MASU has variable contents of Ni, Cu and PGM, generally with Ni>Cu and Pd>Pt. Contents of Ni (<6.0%), Pd (<6.0 ppm), Pt (<2.5 ppm) and Rh (0.2 ppm) in samples of MASU (i.e., close to 100% sulphides) provide approximate tenors for these metals in this style of mineralization.

GE21 Consultoria Mineral

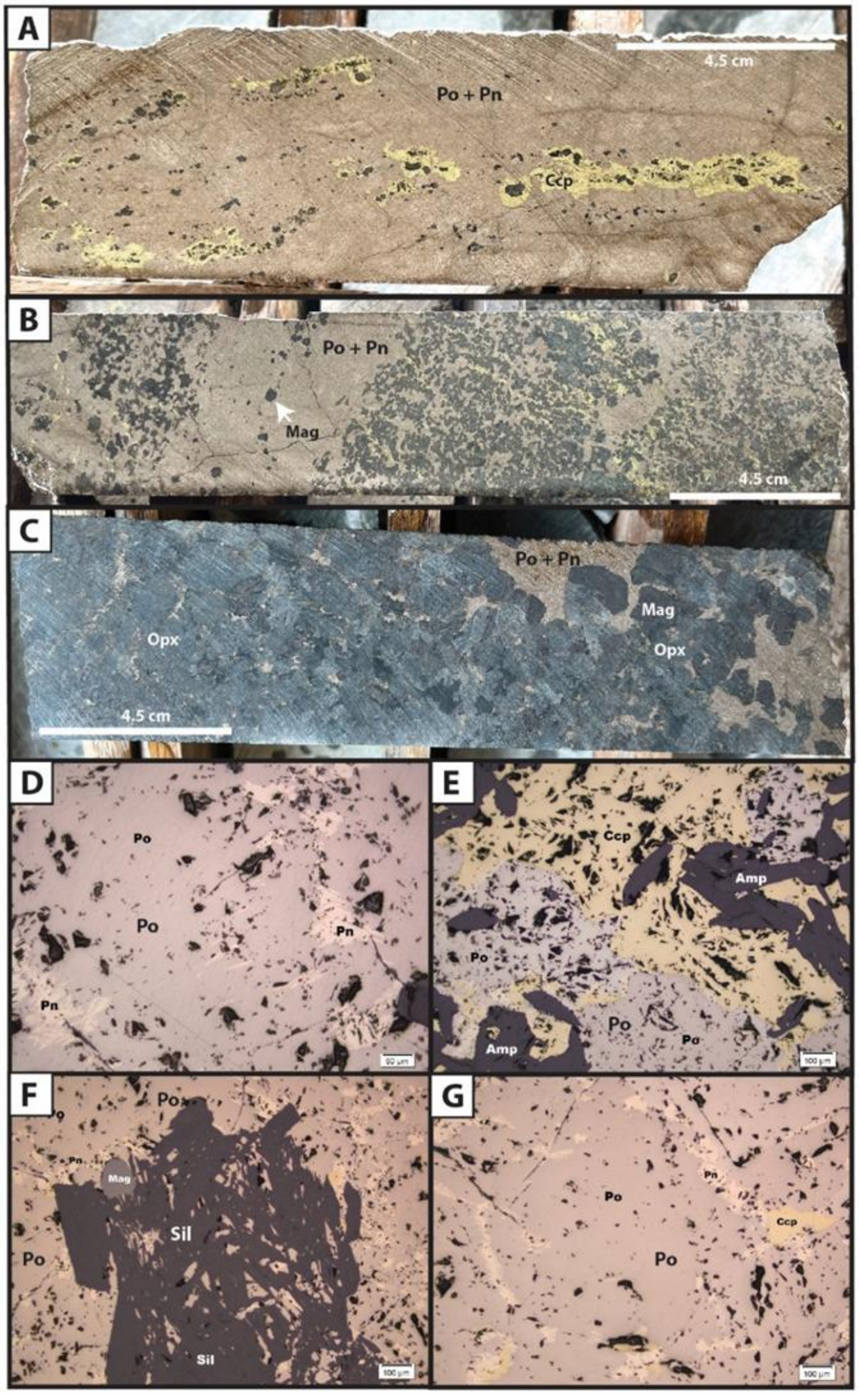


Figure 7-18: Representative photos and photomicrographs of MASU

Legend: A) Drill core sample of MASU mineralization. Sulfides consist of pyrrhotite (Po) and pentlandite (Pn), with minor chalcopyrite (Ccp) domains (yellow color). B) Drill core sample of MASU mineralization. Sulfides consist of Po and Pn, with minor Ccp (yellow color). Gangue minerals consist mainly of prismatic amphiboles (Amp) and minor magnetite (Mag).C) Drill core sample with disseminated and minor MASU mineralization interstitial or enclosing Opx pseudomorphs (replaced by Amp) and minor magnetite (Mag). D) Photomicrograph of MASU consisting of Po with Pn exsolution (flames and flakes). E) Photomicrograph of MASU consisting of Po and a Ccp-rich domain. Sulfides are associated with prismatic amphiboles. F) Photomicrograph of MASU consisting of Po with Pn exsolutions enclosing an aggregate of silicates (Amp+Chl+Bt) and minor Mag. Sulfides are associated with prismatic amphiboles. G) Photomicrograph of MASU consisting of Po with Pn exsolution and minor Ccp.
Source: Bravo, 2025.

The widespread alteration of rocks from the Luanga complex has partially disrupted their primary magmatic features. In the Luanga complex, magmatic silicates are partially altered and commonly occur as pseudomorphs. The magmatic sulphides have also been partially altered during the widespread alteration. The most common alteration of primary sulphides (pyrrhotite - pentlandite - chalcopyrite) consists of their replacement by magnetite and Fe-hydroxides. Because this alteration is heterogeneous at different scales (from mineral crystals up to several hundred meters thick zones) and largely preserves primary textures and compositions of cumulate rocks and PGM mineralized zones, magmatic features can be recognized throughout the layered intrusion.

8 DEPOSIT TYPES

A schematic model of an ideal layered intrusion is presented on Figure 8-1, showing the relative position and petrological affinities of the differing types of magmatic PGM deposits. A single layered intrusion is unlikely to host all these styles of mineralization, and that PGM deposits with differing magmatic affinities can occur in similar positions within an intrusive system.

Magmatic Ni-Cu-PGM sulfides form by the accumulation of immiscible sulphide liquid that scavenged chalcophile elements from a coexisting silicate magma (e.g., Naldrett, 2004, Barnes *et al*., 2016). Textural relationships between sulphides and their host silicates are key evidence for their origin as immiscible sulphide liquids (Barnes *et al.*, 2017, 2018). The magmatic origin of the Luanga PGM + Au + Ni deposit is supported by textural and mineralogical features described in different styles of PGM mineralization, particularly the MSZ, Ni-Rh and SZ. In these different PGM zones, sulphide blebs consisting of po+pn±cpy are interstitial to cumulus olivine and/or pyroxene. In addition, sulphide blebs enclosed in cumulate crystals, as well as their rounded/corroded faces, provide unequivocal evidence for a magmatic origin of sulphides and PGM. Variable lithochemical features in PGM zones located in distinct stratigraphic horizons of the Luanga Complex, including different metal tenors, as well Pt/Pd and Rh/Pd ratios, indicate that several events of mineralization occurred during the magmatic evolution of the Luanga Complex. The occurrence of several mineralized horizons in the Luanga Complex, including PGM mineralization hosted in chromitites, has remarkable similarity with reef-type productive deposits (e.g., Bushveld and Stillwater).

The widespread alteration of rocks from the Luanga Complex has partially disrupted their primary magmatic features. In the Luanga Complex, magmatic silicates are partially altered and commonly occur as pseudomorphs. The magmatic sulphides have also been partially altered during the widespread alteration. The most common alteration of primary sulphides (po-pn-cpy) consists of their replacement by magnetite and Fe-hydroxides. Because this alteration is heterogeneous at different scales (from mineral crystals, up to several hundred meters thick zones) and largely preserves primary textures and compositions of cumulate rocks and PGM mineralized zones, magmatic features can be recognized throughout the layered intrusion.

GE21 Consultoria Mineral

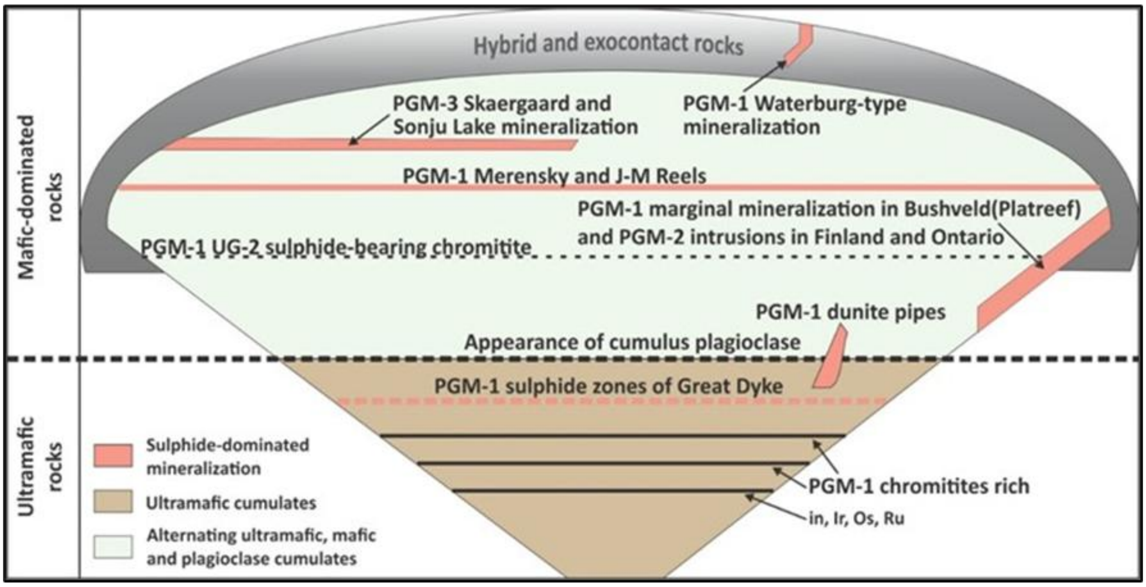


Figure 8-1: Schematic model of a Large Igneous Province (LIP) related layered intrusion

Source: Naldrett, 2010.

8.1 Mineral Deposit

The bulk of the world's PGM resources are mined from mafic-ultramafic layered intrusions, commonly from stratiform mineralized layers located near the transition from mafic to ultramafic cumulate rocks (Naldrett, 2004; Mungall and Naldrett, 2008; Zientek, 2012). This type of mineralization, known as reef-type deposits, has the Bushveld Complex as the leading model for exploration. In addition, layered mafic-ultramafic intrusions can host economic deposits of base metal sulphides (Ni-Cu-Co), chromite and magnetite-ilmenite (Fe-Ti-V). These deposits result from magmatic processes of crystallization, differentiation, and concentration during emplacement and cooling of mafic-ultramafic magmas. In particular, PGM (Ni-Cu) deposits are generally accepted to result from the concentration of variable amounts of sulphides originated as a separate immiscible liquid from the parental magma.

"PGM reefs" are stratabound enriched lode mineralization in mafic to ultramafic layered intrusions. The term "reef" is derived from Australian and South African literature for this style of mineralization and used to refer to (1) the rock layer that is mineralized and has distinctive texture or mineralogy or (2) the enriched sulphide mineralization that occurs within a rock layer.

9 EXPLORATION

Bravo has been diligently following a systematic approach in its exploration programs for the Luanga PGM + Au + Ni Project.

9.1 Preliminary Works

The earliest exploration completed by Bravo was from September 2 to 23, 2020. Bravo staff visited Vale´s core facilities where they collected five verification core samples from four historical drill holes. Samples were ¼ core from mineralized intervals previously sampled by Vale (Figure 9-1). Those samples were cut and bagged in Vale’s facilities by Bravo personnel, who were responsible for the identification (the same ID as the original sample) and shipping of the sample bags.

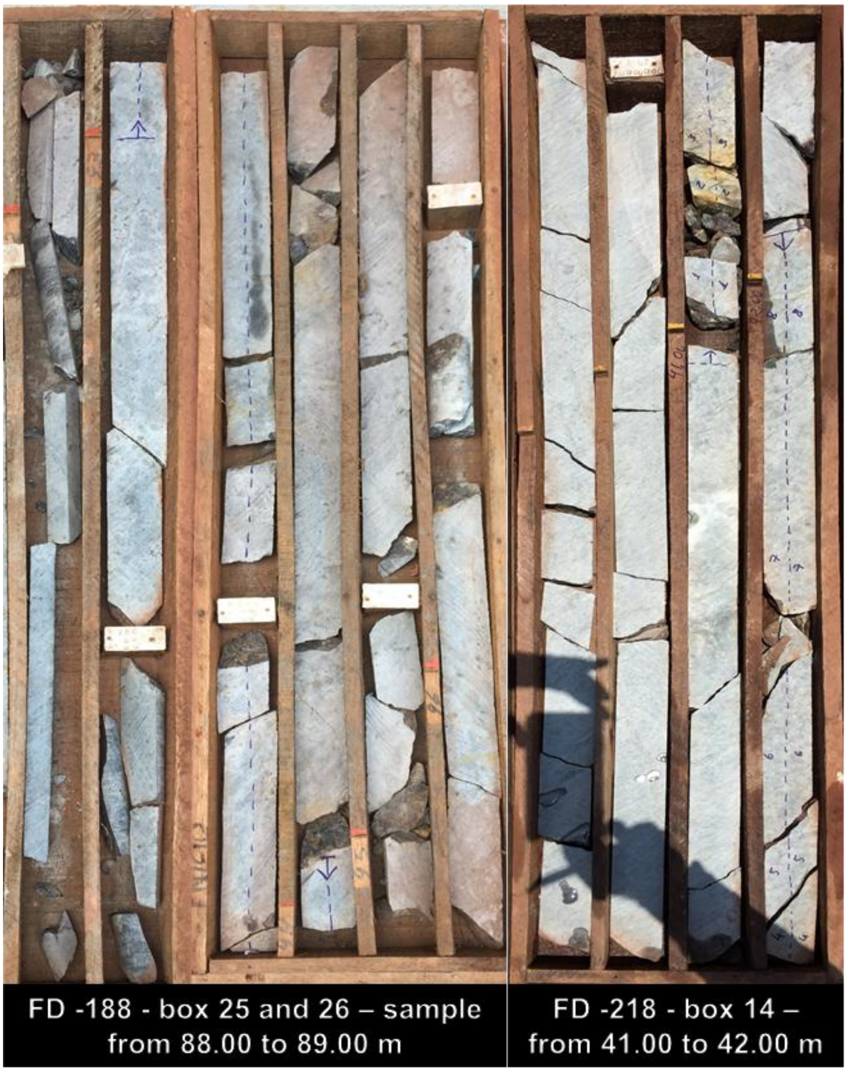


Figure 9-1: Examples of drill holes selected for independent re-sampling

Source: Bravo, 2023.

Samples chosen for assay analysis by Bravo were shipped directly to the analytical laboratory of ALS Brasil Ltda. in Belo Horizonte, Minas Gerais state. All samples were weighed, dried, crushed, split, and pulverized up to 85% <75µm.

All samples were analysed for Pt, Pd, and Au by fire assay with ICP-AES finish, and for Rh by fire assay with ICP-MS finish. The samples also were analysed for 48 elements by four acid digest with an ICP-MS finish.

In 2021, Bravo conducted a detailed review of Vale data, including results from previous Diamond Drilling campaigns.

In total, 234 complete diamond drill holes were received at Bravo core yard, representing a significant cross-section of the geology intersected by the historical drilling campaign.

Following the receipt of the drill cores, Bravo technical staff repaired and cleaned the core boxes, their markings and labels prior to relogging the core geologically (Table 9-1 and Figure 9-2). Following this relogging, the core was cleaned and photographed before the commencement of resampling.

Table 9-1: Historical drill core – quantity of relogging and resampling

Relogging & Resampling				
Received Core	Bravo Camp	Relogged	% Relogged	% Resampled
Total Historic Drill Holes	234	202 (all drilling within the Luanga deposit)	87%	87%
Total Metres	**46,029**	**40,065**	**87%**	**87%**

Source: Bravo, 2025.

Figure 9-2: Vale core now at the Bravo facilities

Source: Bravo, 2025.

For the resampling programme, half core was cut in half again by a standard industry core saw and, in cases where only quarter remains, it was sampled in its entirety. Figure 9-3 Certified Reference Materials (blanks and standards) (CRM) were inserted throughout the sample sequence at a ratio of one in every twenty samples for each, resulting in a quality control sample after every ten primary samples. Standards were purchased from both OREAS in Australia and AIMS in South Africa. These standards cover a variety of grades, while also being the best matrix match for the type of mineralization at Luanga. Samples were submitted to ALS Brasil at their sample reception facility located in Parauapebas.

Figure 9-3: Resampling program

Source: Bravo, 2025.

Historical drill core sample data and Bravo's drill core resampling to date shows an expected positive correlation for the PGM assessed. Minor variations from Vale's original assays to the re-assay are attributed to updated preparation and analytical methods and improved assaying techniques for Rh and Ni. Nickel resampling data assays presents two different populations, one probably related to the silicate and other sulphides.

9.2 Soil Samples

9.2.1 Historical Soil Sampling

A total of 5,241 soil samples were collected by Vale during the exploration works conducted at Luanga. All soil samples were submitted to chemical analyses for a suite of 16 elements (in ppb), including: Ag, As, Be, Bi, Ce, Co, Cr, Cu, La, Ni, Pb, Sb, Sn, Te, W and Zn. This suite of elements was analysed by inductively coupled plasma/mass spectrometry (ICP/MS) by the three different laboratories (Nomos, Lakefield and SGS). Soil samples were also analysed for Au, Pt and Pd (in ppb) by fire assay/atomic absorption spectrometry (FA/AAS). Information about the laboratory responsible for the FA/AAS analyses is not included in the database.

The results from historical soil sampling were reprocessed by Bravo (Figure 9-4). Chromium and nickel soil anomalies correlate well with the mafic and ultramafic rocks of the Luanga Complex and Luanga South. Palladium and platinum soil anomalies are spatially coincident with the rocks of the Transition Zone of the Luanga Complex.

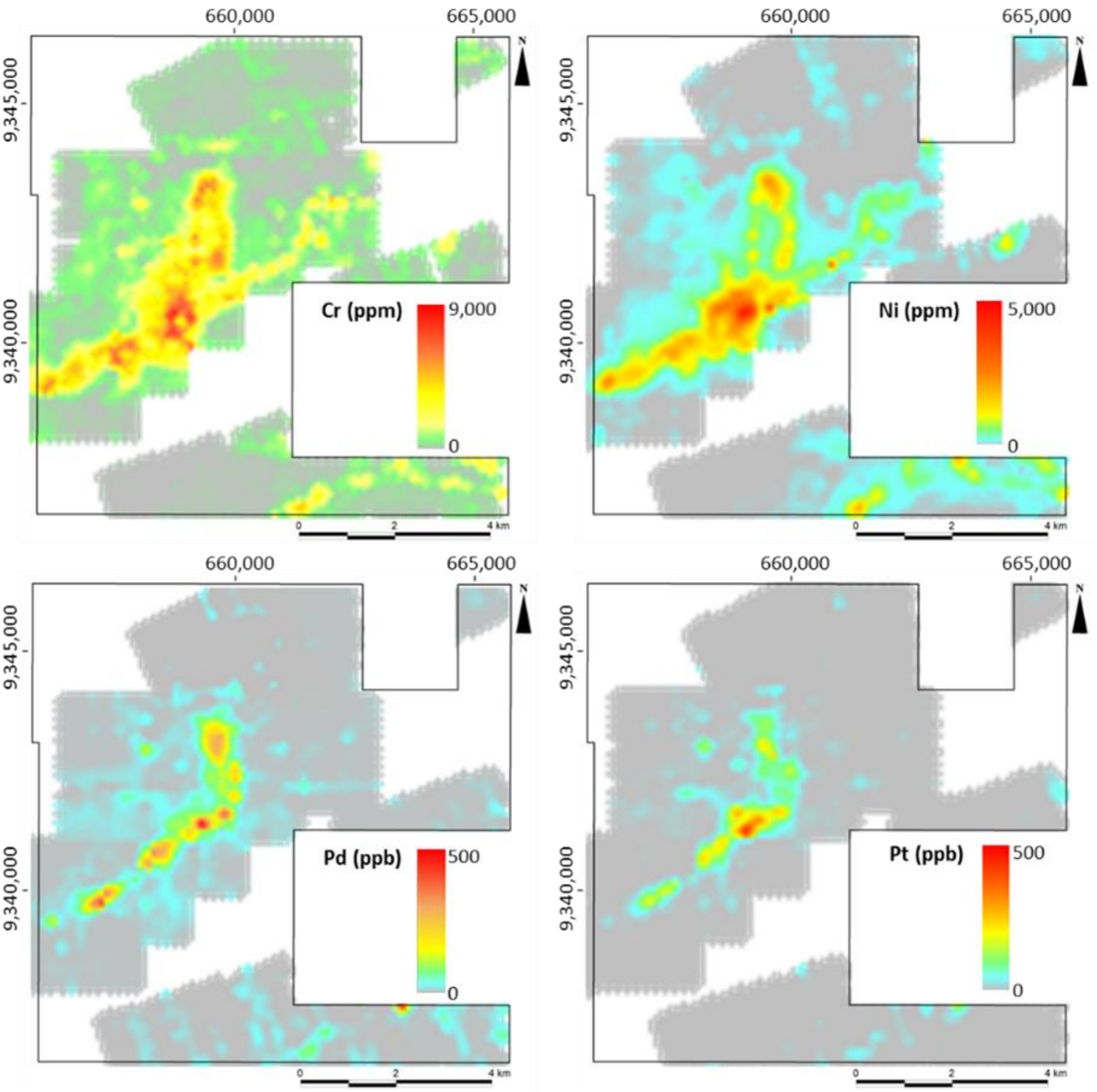


Figure 9-4: Maps with reprocessed results from historical soil sampling campaign

Source: Bravo, 2025.

9.3 Topographic Surveys

In 2022 RR Topografia & Engenharia of Brazil completed the Orthophotography and new Digital Elevation Model (DEM). Commercial drone surveying equipment was used to complete the aerial work, while ground surveying was used for control of accuracy, positioning and georeferencing. A mosaic of the ortho-imagery overlain on the 3D digital terrain model created from the DEM, is shown below in Figure 9-5.

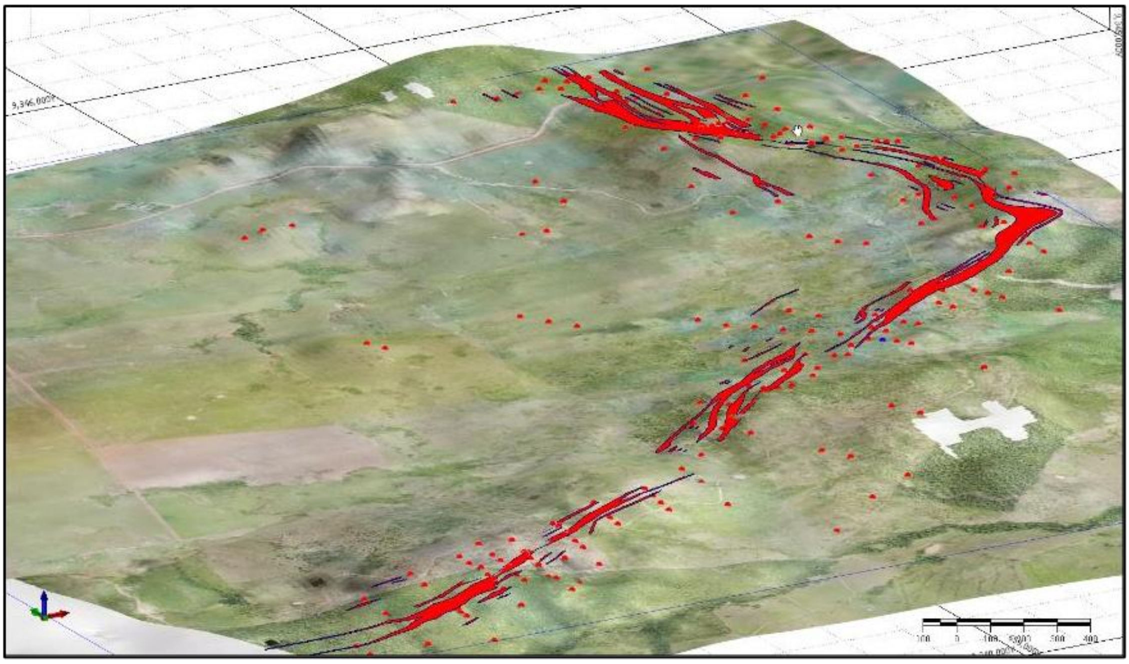


Figure 9-5: Digital elevation model, ortho-image, drill collars and mineralization zones

Source: Bravo, 2025.

In June 2023, Bravo contracted Sul Pará to conduct a new Drone Survey over the Luanga deposit and adjacent areas. The survey covered a total area of 26.96 km^2 over Luanga Complex and provided a detailed (1 m) Digital Elevation Model (Figure 9-6 A) and an ortho-mosaic image (Figure 9-6 B).

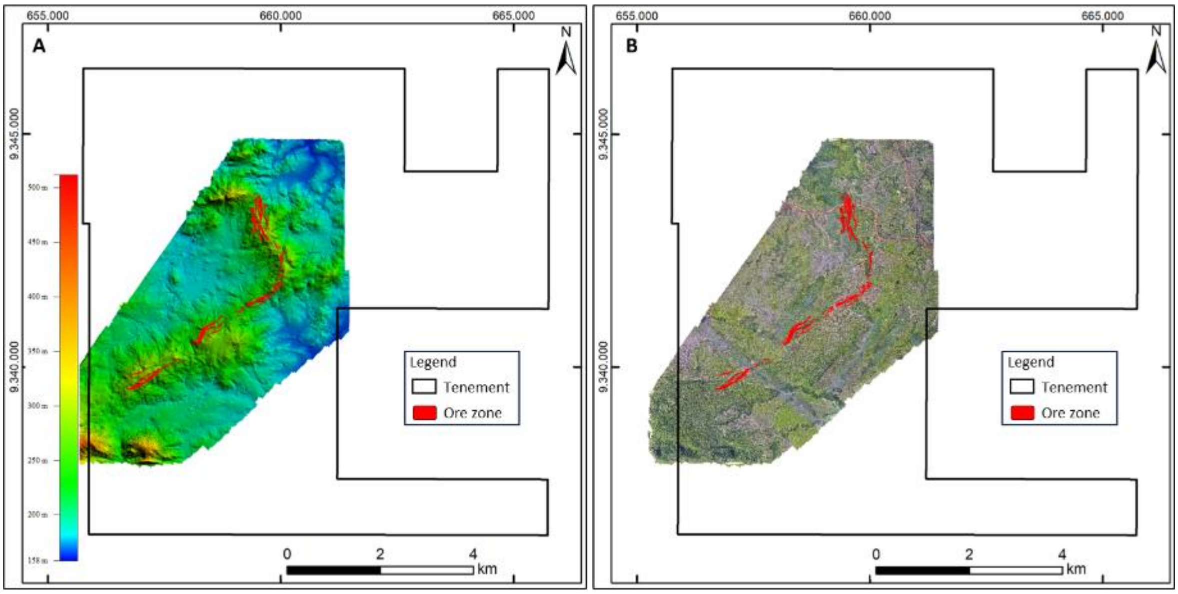


Figure 9-6: Luanga Complex digital elevation model and ortho-mosaic image

Legend: (A) digital elevation model, and (B) ortho-image. Both with identified mineralized zones.
Source: Bravo, 2025.

GE21 Consultoria Mineral

9.4 Geological Mapping

Bravo conducted a detailed geological mapping over the Luanga Complex at the 1:10,000. Mapping was performed along north-south profiles at intervals of 100 m. In more complex zones, mapping was completed at a scale of 1:5,000, also using the drilling data and ground geophysics to support interpretations.

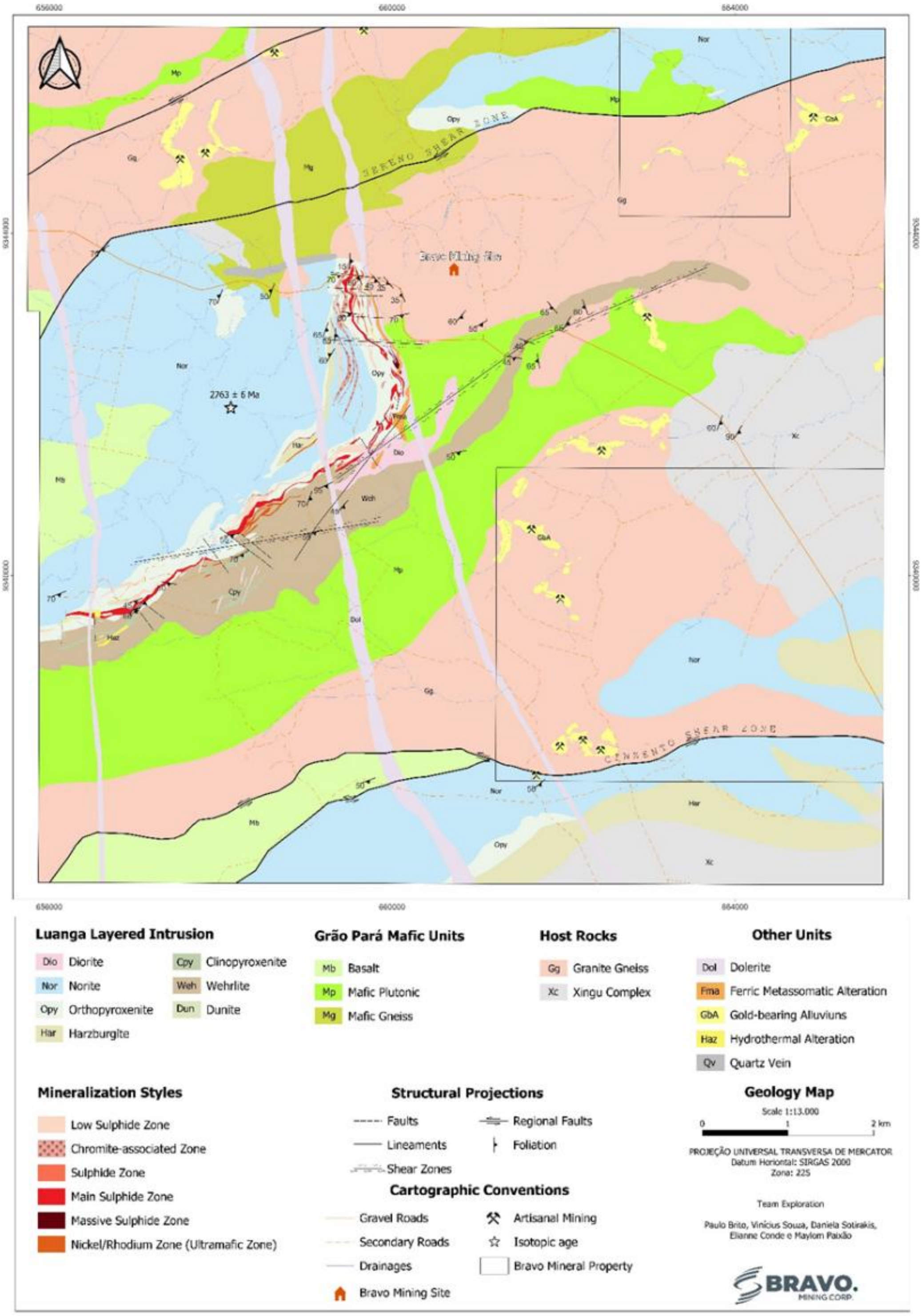


Figure 9-7: Geological map

Source: Bravo, 2025.

9.5 Geophysics

9.5.1 2021 Geophysics

Geophysical processing and interpretation was completed by both SGC and Southernrock for Bravo in 2021. Southernrock reprocessed the historical IP data, while SGC reprocessed the historical magnetic data. IP overlaid on the reprocessed magnetic image produced by SGC is shown in Figure 9-8.

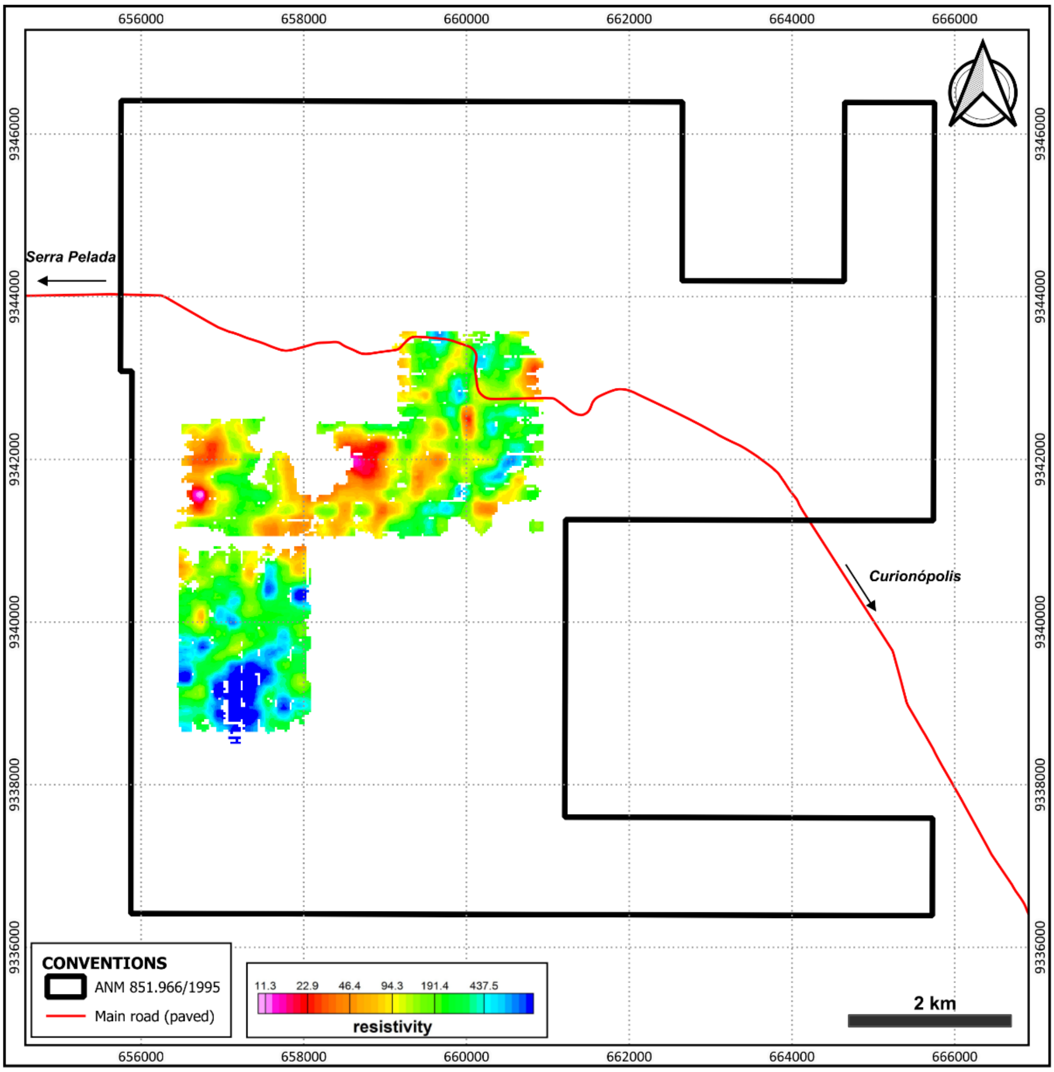


Figure 9-8: 3D inversion of IP resistivity, depth -125m

Source: Bravo, 2025.

9.5.2 Borehole Electromagnetics

Ground geophysics activities conducted during 2022, and January 2023 included borehole electromagnetics and surface electromagnetic surveys. Both surveys were conducted by Geomag S.A. (Geomag), For a total of 4,265 linear metres surveyed.

9.5.3 Fixed-Loop Transient Electromagnetics

Fixed-Loop Transient Electromagnetic (FLTEM) surveys were completed over the Central and North Sectors totaling 56 transversal lines (total of 54.62 km).

The raw data was processed, and several conductive plates modelled. Some of them have been tested and others are planned to be drilled.

9.5.4 Ground Magnetometry and Gravimetry

A ground geophysics survey consisting of magnetometry and micro-gravity, using 100 m line-spacing grid was performed over an area of approximately 18.7 km^2, covering the ultramafic and transition zones of the Luanga Complex. Ground magnetometry was performed with continuous readings along the lines and the micro-gravity survey was conducted with readings spaced 50m along the lines. The acquisition was carried out by the Bravo team during the period of 2023/2024. The raw data was delivered to SGC for quality control, corrections and processing.

In general, the magnetic data shows a linear anomalous feature trending SW-NE (Southwestern and Central Sectors) with a flank running S-N (North Sector) and another flank trending NW-SE on the eastern extreme of the survey grid (Figure 9-9 a and B).

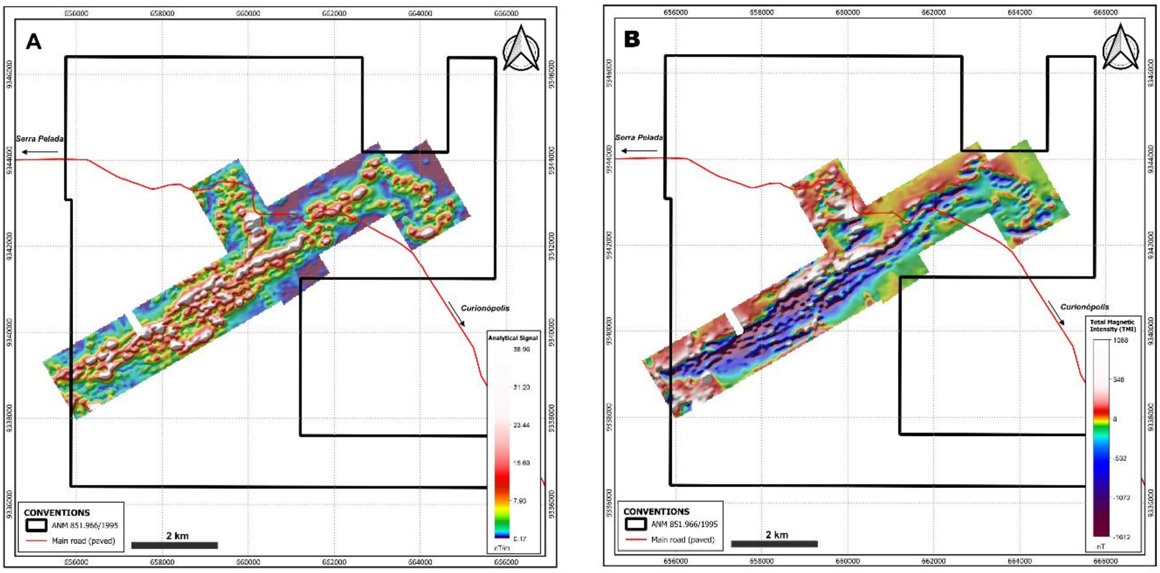


Figure 9-9: (A) Analytical signal image (B) TMI image

Source: Bravo, 2025.

The Bouguer image produced from the micro-gravity survey shows that the highest density rocks occur on the Southwestern and Central Sectors associated with the wehrlites of the basal Ultramafic Zone. Its contact with the metamafic plutonic rocks of the Grão Para Group is clear in the gravity data (Figure 9-10).

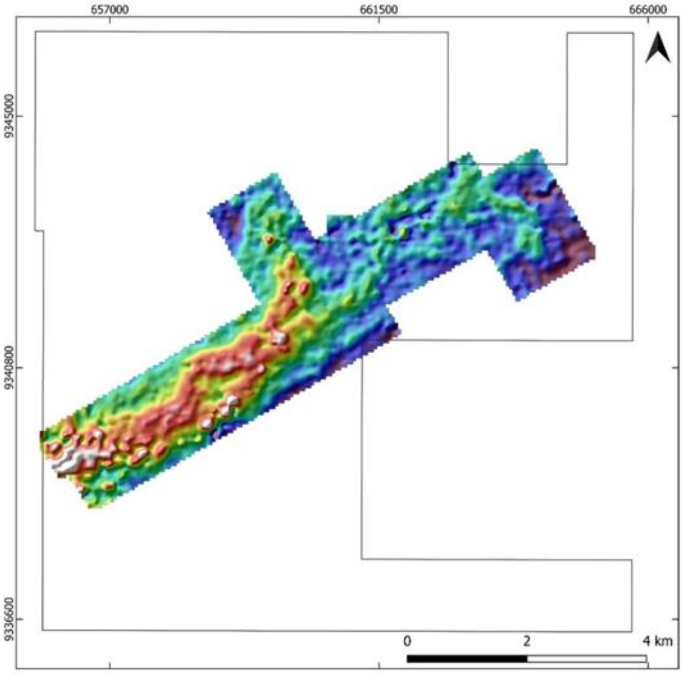


Figure 9-10: Bouguer Residual image with horizontal gradient directed to east

Source: Bravo, 2023.

9.5.5 Time Domain Electromagnetic and Magnetic Survey

Time Domain Electromagnetic and Magnetic airborne geophysical survey (HeliTEM) was carried out over an area of 99.72 km^2, covering the whole Luanga mineral property. The survey was conducted by Xcalibur Multiphysics (Xcalibur) and consisted in a total of 771.2 km of lines, being 697.1km along NW-SE transverse lines, spaced at 150 m, and 70.9 km along SW-NE tie lines, spaced at 1,500m (Figure 9-11).

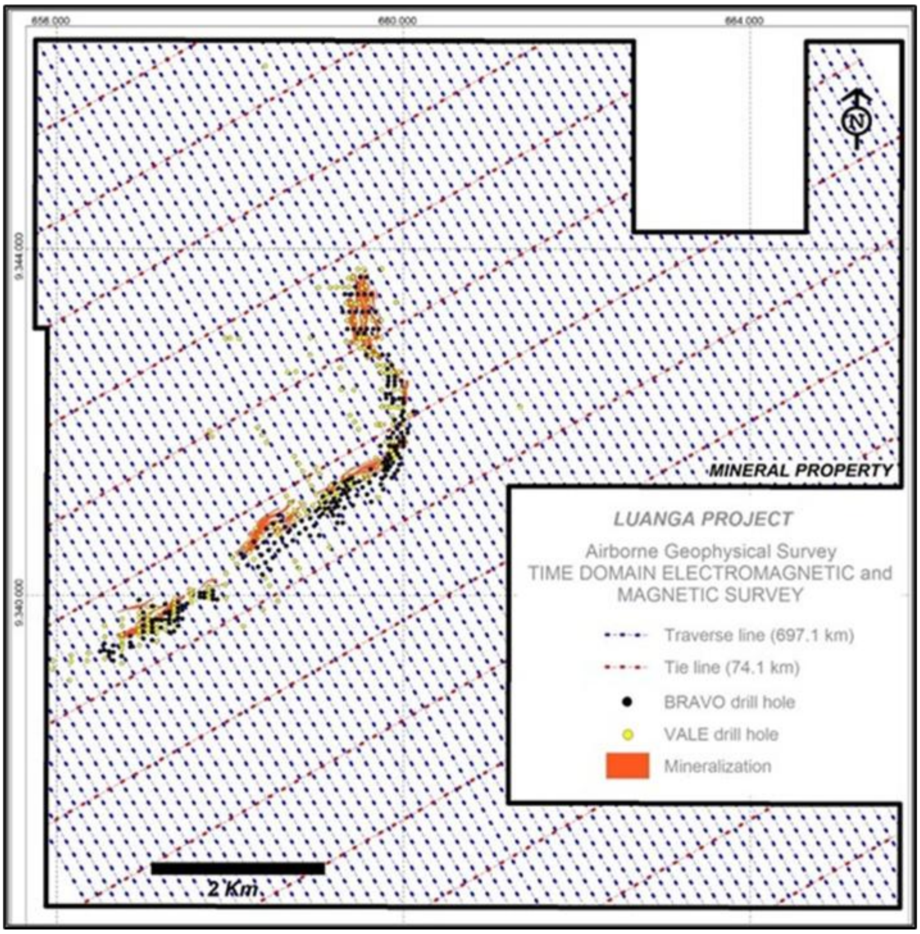


Figure 9-11: Coverage of Bravo HeliTEM survey

Source: Bravo, 2025.

Bravo engaged SGC to provide 3D modelling of the main conductors. As a result of this phase, 17 priority one (Priority 1) targets plus 16 priority two (Priority 2) targets were identified (Figure 9-12). These targets were ranked based on their electromagnetic (EM) response together with ground micro-gravity and magnetic surveys, detailed geological/structural interpretations, soil geochemistry data and the existing drilling database.

GE21 Consultoria Mineral

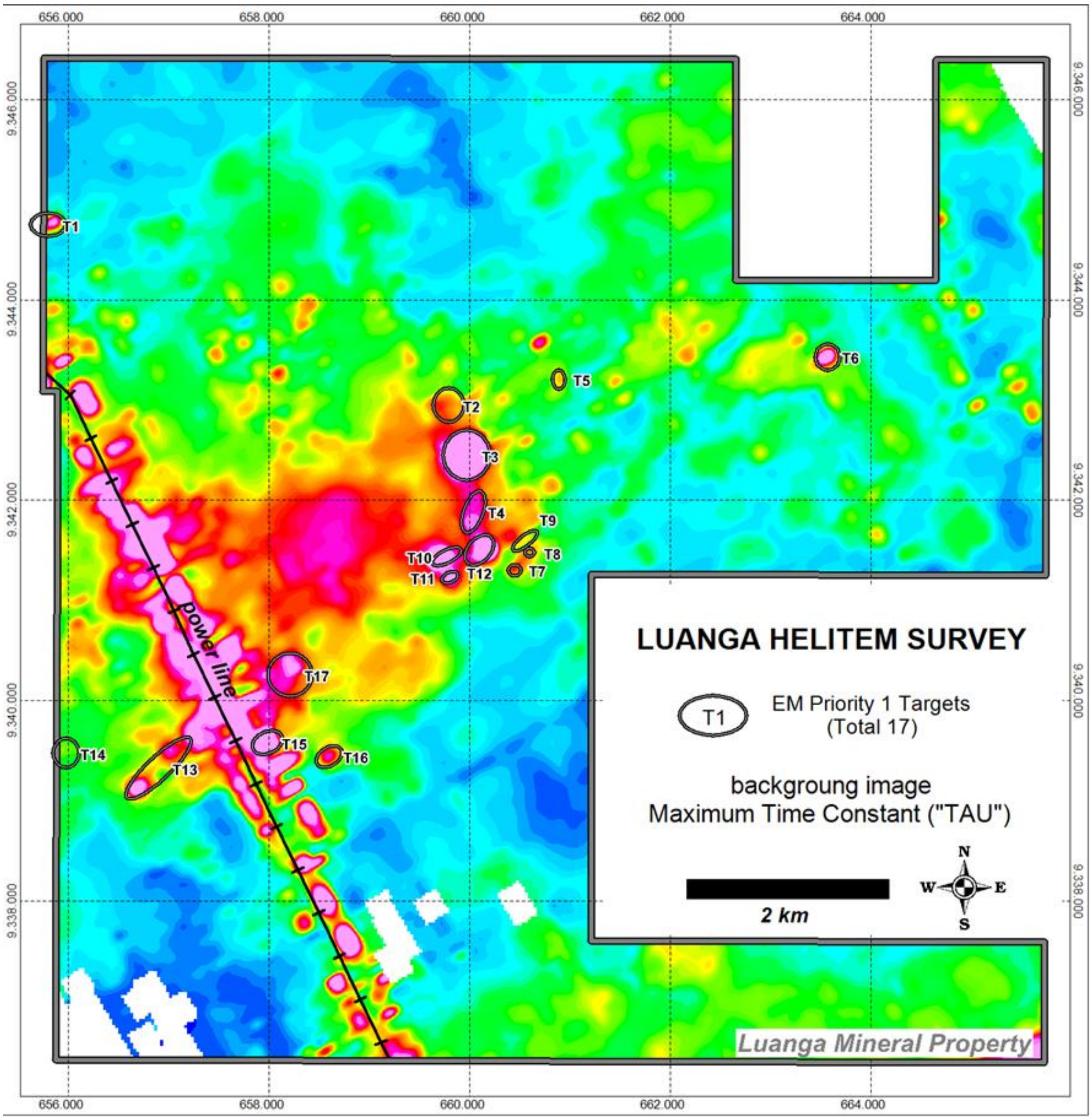


Figure 9-12: EM priority 1 targets

Source: Bravo, 2025.

9.6 In Situ Density Sampling

Bravo's field team collected 112 density samples in regolith material such as saprolite and soil. The first step for the sampling is the identification of the regolith horizon to be sampled and the pedological and geological description of the physical aspects of the material.

A sampling location is selected by a Geologist or Technician and the location is then cleaned, leaving a flat vertical surface exposed. A cylinder measuring 15 cm in height and 6 cm in internal diameter, is positioned perpendicularly to the vertical surface and then pressed against the matrix of the weathered material using a small metallic sledgehammer (Figure 9-13). After the cylinder is completely dug into the soil or saprolite, it is then removed, using a spatula to excavate the material above and next to the cylinder. The spatula is also used to remove excess material from the open ends of the cylinder, carefully avoiding sample material loss (Figure 9-14).

Figure 9-13: Cylinder insertion into weathered material

Source: Bravo, 2025.

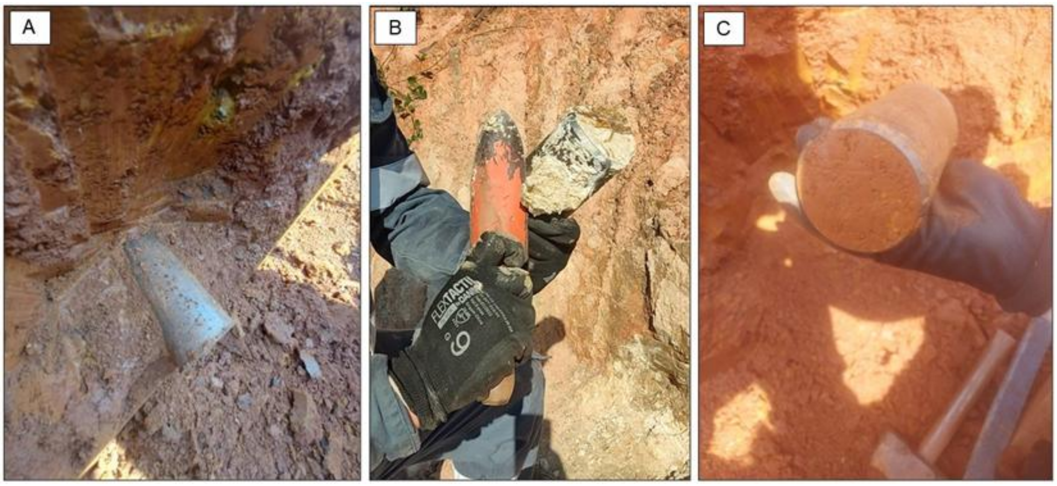


Figure 9-14: Cylinder removal

Source: Bravo, 2025.

Once the samples are collected, they are sealed in plastic bags, clearly labelled with sample identifiers (Figure 9-15). The samples are transported to the core shed, where they are weighed, dried at 105°C for 24 hours, and reweighed (Figure 9-16). Density is calculated using the formula:

$$Density\ (g/cm^3) = \frac{Dry\ Weight\ (g)}{Volume\ of\ the\ Cylinder(cm^3)}\ Volume\ of\ the\ Cylinder = 423.9cm^3$$

Moisture content is calculated using the formula:

$$Moisture\ Content\ (\%) = \frac{Wet\ Weight - Dry\ Weight}{Dry\ Weight}\ x\ 100$$

The sample location coordinates, the experimental data, and the calculated densities and humidities are all entered an Excel spreadsheet.

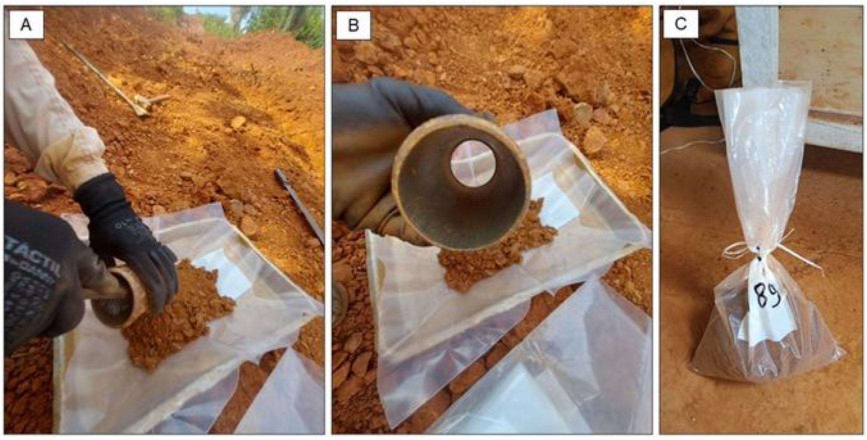


Figure 9-15: Sample material collection

Source: Bravo, 2025.

Figure 9-16: Weighing and drying procedures

Source: Bravo, 2025.

9.7 Trenching

The trenching program started in Q4/2022 aiming to provide detailed information about the mineralized zones at surface. Up to the Effective Date of this report, 45 trenches were opened totaling 9,065 linear meters. The location of the trenching program is shown on Figure 9-17.

GE21 Consultoria Mineral

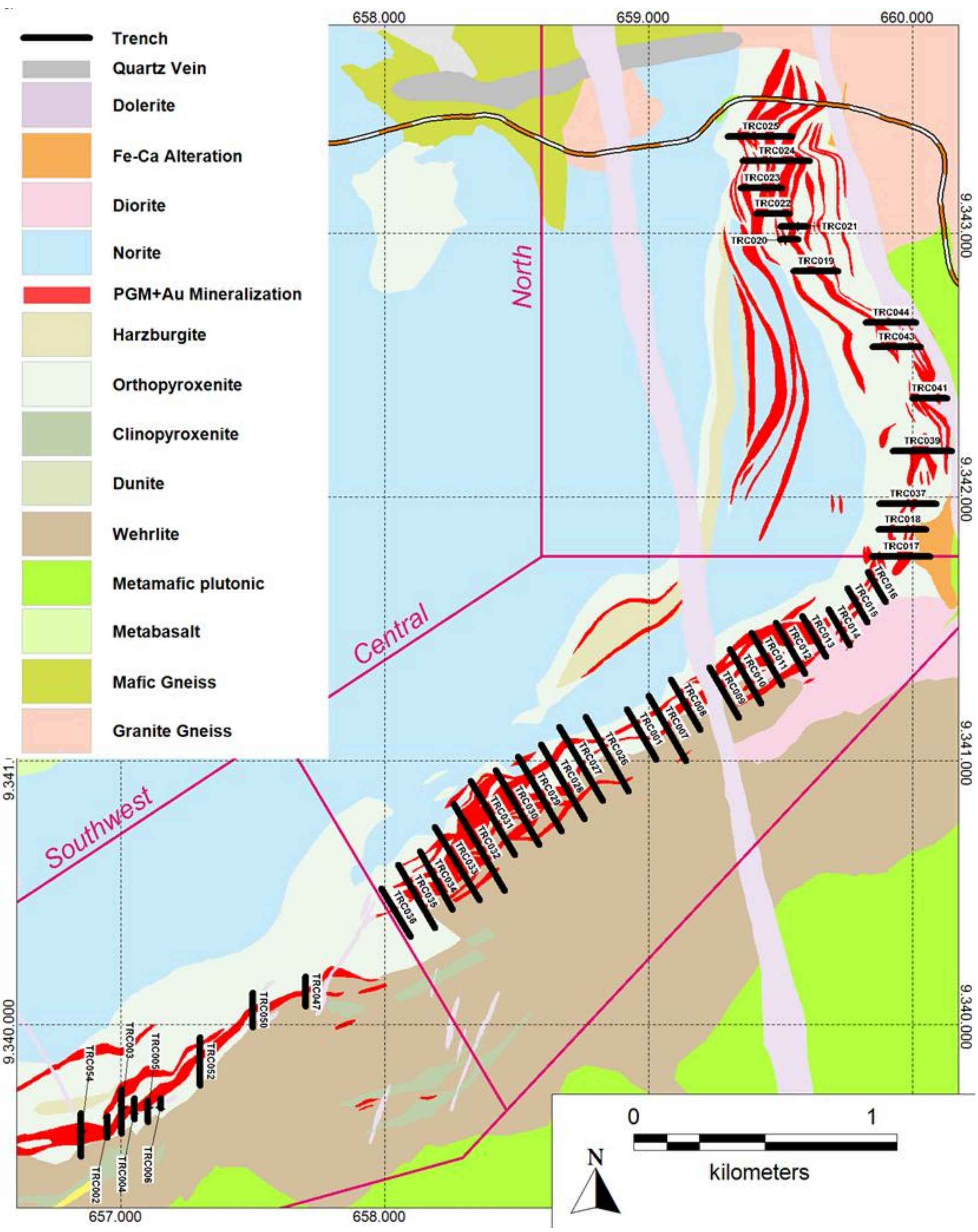


Figure 9-17: Trench opening program

Source: GE21, 2025.

The trenching procedures adopted by Bravo are based on NBR 9604, a Brazilian regulation of trenching and sampling procedures. Trenches are 80 cm wide (the size of the backhoe's rear bucket) and have a maximum height of 120 cm (Figure 9-18). All trenches were geologically described, mapped, sampled, and their channel samples were precisely surveyed with RTK (Real Time Kinetic GPS). Geologic description is logged on site into Micromine 'GeoBank For Field Teams', an application that is linked to the main Project Database.

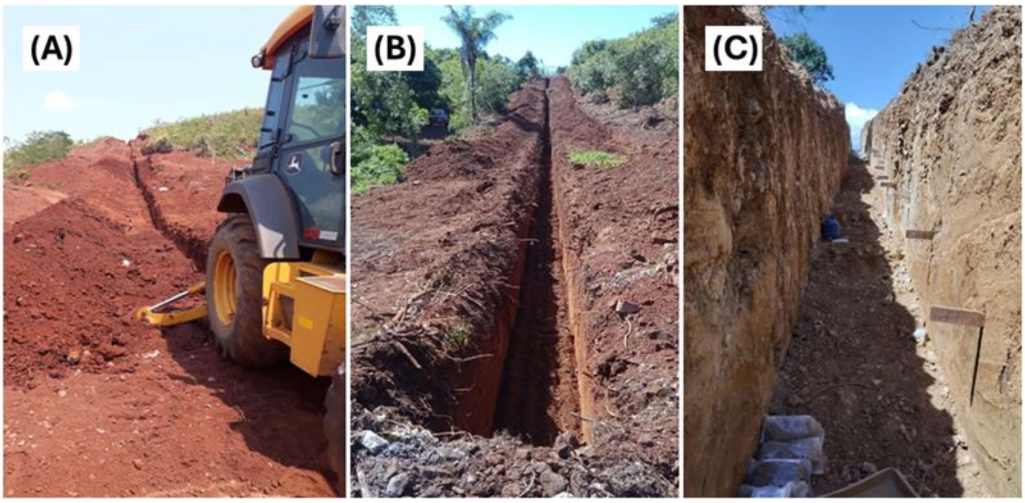


Figure 9-18: Trench opening and channel sample marking

Source: Bravo, 2025.

Sampling takes place using a 1 m standard sample size, varying from 0.7 m to 1.2 m where geological contacts are respected. In general, channel sampling is 40 to 50 cm from the trench floor, aiming to sample the saprolite and avoid sampling soil. Sample collection is performed using chisels and sledgehammers to collect samples on an aluminum tray, placed under the channel. A total of 9,521 channel samples, including QA/QC (Quality Assurance / Quality Control) samples, were collected and analyzed for 3PGM and Au at independent laboratories.

Figure 9-19 illustrates the cross section with a shallow mineralized zone of 175.7m @ 1.71 g/t PGM + Au identified in Trench TRC23LU024.

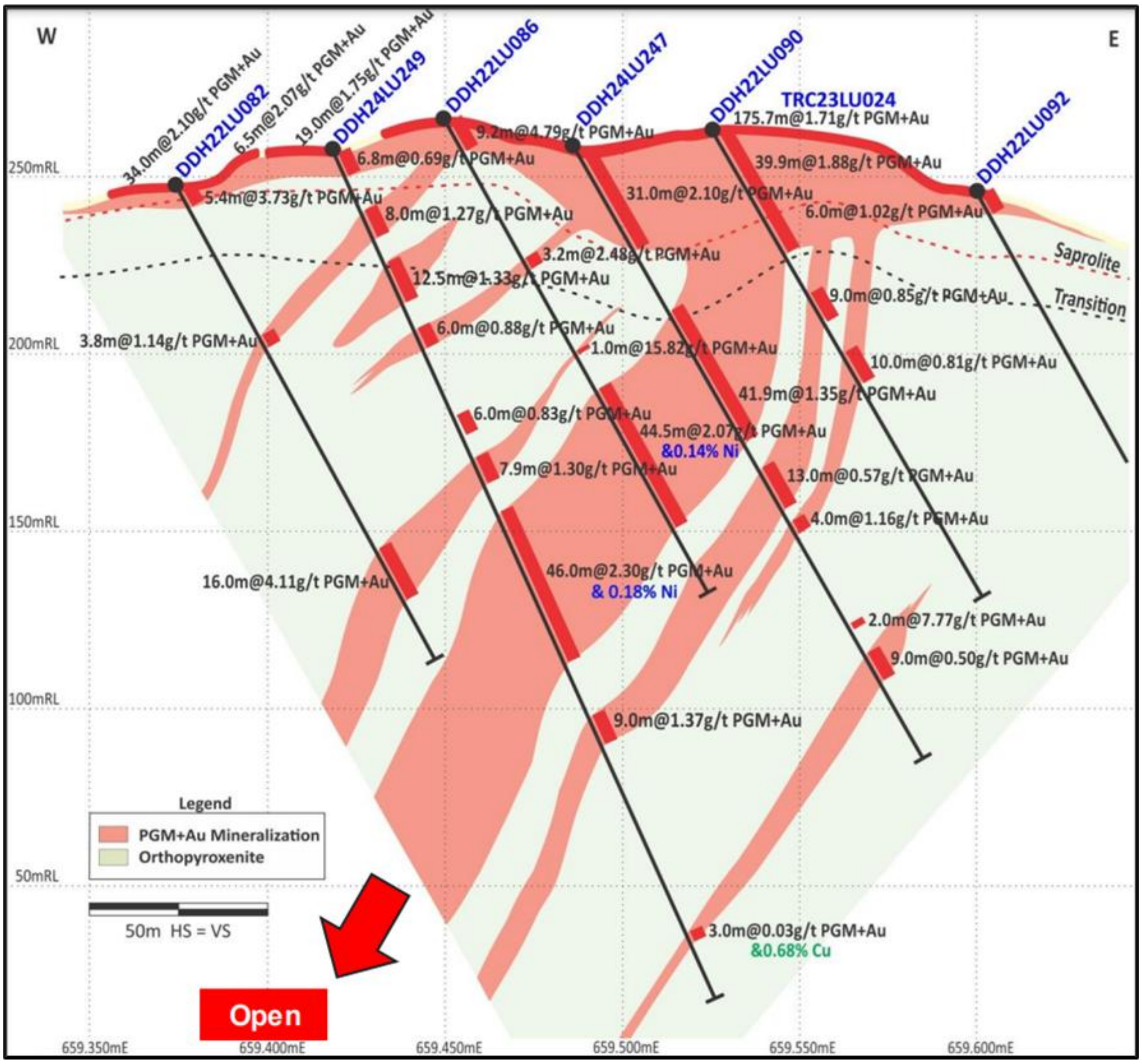


Figure 9-19: Section Bravo DDH22LU014 with trench TRC22LU003

Source: Bravo, 2025.

9.8 Other Targets

Based on the Helicopter Electromagnetic ("HeliTEM") survey carried out in 2023, together with a regional structural analysis and historic geochemical data, a set of exploration targets for Ni-Cu magmatic and/or Iron Oxide Copper Gold ("IOCG") mineralization styles was delineated within the Luanga Property. Some electromagnetic ("EM") targets were briefly tested in 2024 with scout diamond drilling, while others are being submitted to a systematic exploration program (mapping, soil geochemistry, ground geophysics) that started in 2025. Table 9-2 summarizes the current exploration targets at the Luanga Project, while their location is shown in Figure 9-20.

Table 9-2 - Summary of Cu-Au IOCG and Ni-Cu magmatic targets on Luanga Project

Target_ID	Style	Favorable Elements	Exploration Stage
T1 Target	Cu-Au IOCG	EM conductor	in progress
		Regional shear structure	
T5 Target	Cu-Au IOCG	EM conductor	advanced
		Cu-oxide mineral at surface	
T16 Target	Cu-Au IOCG	EM conductor	in progress

GE21 Consultoria Mineral

Target_ID	Style	Favorable Elements	Exploration Stage
T17 Target	Ni-Cu Magmatic	EM conductor	in progress
		Historic Cu in-soil anomaly	
Lizard	Cu-Au IOCG	Historic Cu in-soil anomaly	advanced
		Regional shear structure	
Scorpion	Cu-Au IOCG	Magnetic high	*to be started*
		Regional shear structure	
Taurus	IOCG / Magmatic	EM conductors	in progress
		Regional shear structure	
Gemini	Cu-Au IOCG	EM conductors	in progress
		Magnetic highs	
		Regional mantle-tapping structures	
Babylon	IOCG / Magmatic	EM conductor	in progress
		Magnetic high	
		Gravity anomaly	
Orion	Cu-Au IOCG	Cu-oxide mineral at surface	*to be started*
Lynx	Cu-Au IOCG	Magnetic high	*to be started*
		Regional shear structure	
Perseus	Cu-Au IOCG	Magnetic highs	*to be started*
		Regional mantle-tapping structures	
Jupiter	Cu-Au IOCG	Magnetic high	*to be started*
		Regional shear structure	
Musca	Cu-Au IOCG	EM conductors	*to be started*

Source: Bravo 2025.

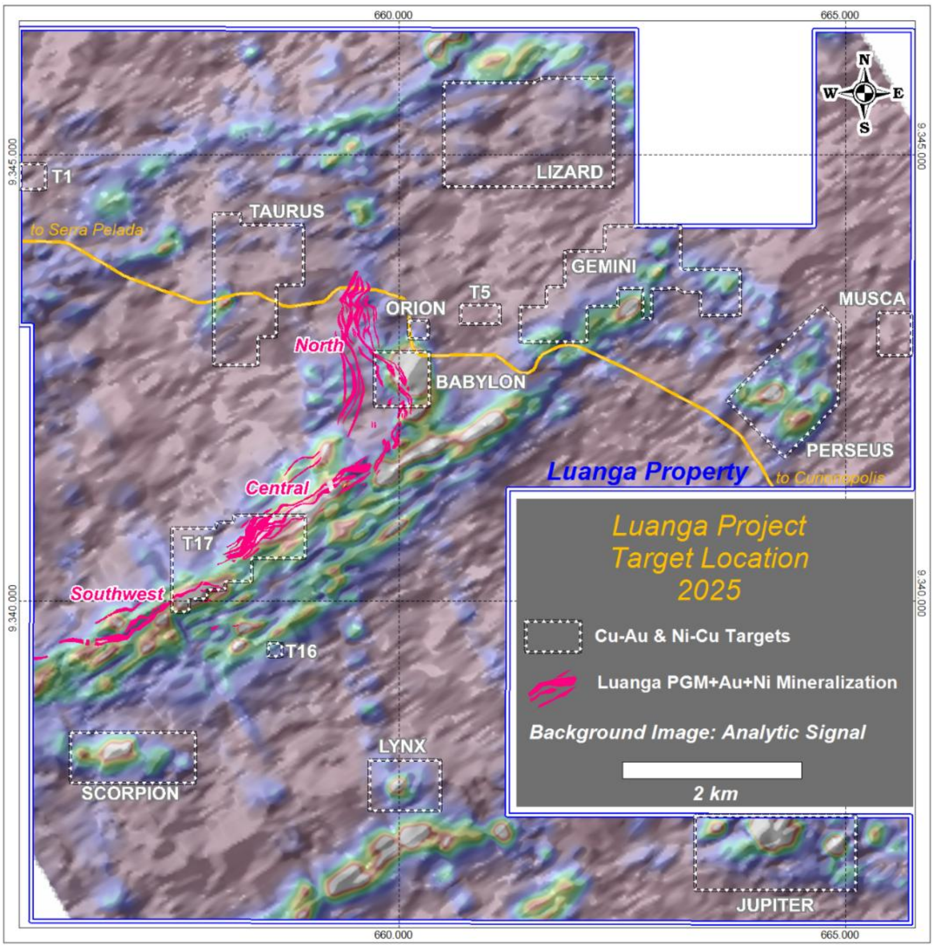


Figure 9-20 - Location of IOCG and Ni-Cu magmatic exploration targets on Luanga Project

Source: Bravo 2025.

The relevant exploration results obtained to date on the most advanced targets are briefly summarized below.

9.8.1 T5 Target

T5 Target ("T5") is located outside the Luanga Complex (~1.1km to east) and is currently an exploration stage prospect where Cu-Au mineralization was intercepted is several drill holes carried out to test EM conductors.

At the effective date of this Report, Bravo has drilled 28 drill holes, totaling 5,926.25 linear metres at T5, with sulphide mineralization intercepted in 19 of these holes. The drilling program at T5 followed the same procedures used at the Luanga PGM+Au+Ni deposit.

Table 9-3 and Figure 9-21 show mineralized intercepts obtained in drilling to date.

Table 9-3 - Significant drill intercepts on T5 Target

HOLE-ID	Target	From (m)	To (m)	Thickness (m)	Cu (%) Sulphide	Au(g/t)
DDH2305T001	T5	212.30	213.00	0.70	1.98	0.04
DDH2405T002	T5	165.62	177.10	11.48	0.11	3.33
including	*T5*	*167.50*	*170.36*	*2.86*	*22.91*	*3.62*
DDH2405T003	T5	157.91	163.10	5.19	3.10	1.12
DDH2405T004	T5	153.60	162.35	8.75	9.48	2.08
DDH2405T005	T5	43.50	48.50	5.00	0.59	0.19
DDH2405T009	T5	148.90	166.32	17.42	3.49	0.95
including	*T5*	*155.88*	*159.40*	*3.52*	*13.14*	*3.87*
DDH2405T010	T5	140.63	144.90	4.27	4.66	0.97
DDH2405T011	T5	155.50	156.50	1.00	0.30	0.10
And	T5	323.55	324.50	0.95	1.19	0.50
DDH2405T012	T5	156.00	159.20	3.20	8.00	1.83
DDH2405T013	T5	107.05	109.65	2.60	2.79	0.57
And	T5	152.95	157.85	4.90	0.93	0.68
DDH2405T014	T5	162.00	165.00	3.00	1.63	0.68
DDH2405T015	T5	121.70	135.09	13.39	2.93	0.39
including	*T5*	*132.93*	*135.09*	*2.16*	*10.23*	*1.38*
DDH2405T016	T5	174.45	178.25	3.80	1.06	0.35
DDH2405T017	T5	152.55	154.60	2.05	0.92	0.23
And	T5	180.63	182.10	1.47	0.78	0.19
DDH2405T018	T5	54.40	56.40	2.00	1.20	0.48
DDH2405T021	T5	139.25	142.55	3.30	1.09	0.26
DDH2505T024	T5	114.25	120.90	6.65	2.57	0.55
including	T5	*117.10*	*119.00*	*1.90*	*5.45*	*1.42*
DDH2505T025	T5	91.79	94.70	2.91	6.01	0.43
DDH2505T026	T5	65.55	73.86	8.31	1.65	0.15
And	T5	99.36	105.56	6.20	2.07	0.48

Note: drill holes DDH2405T006, DDH2405T007, DDH2405T008, DDH2405T019, DDH2405T020, DDH2405T022, DDH2505T023, DDH2505T027 and DDH2505T028 returned with no significant Cu-Au intersections.

Source: Bravo 2025.

GE21 Consultoria Mineral

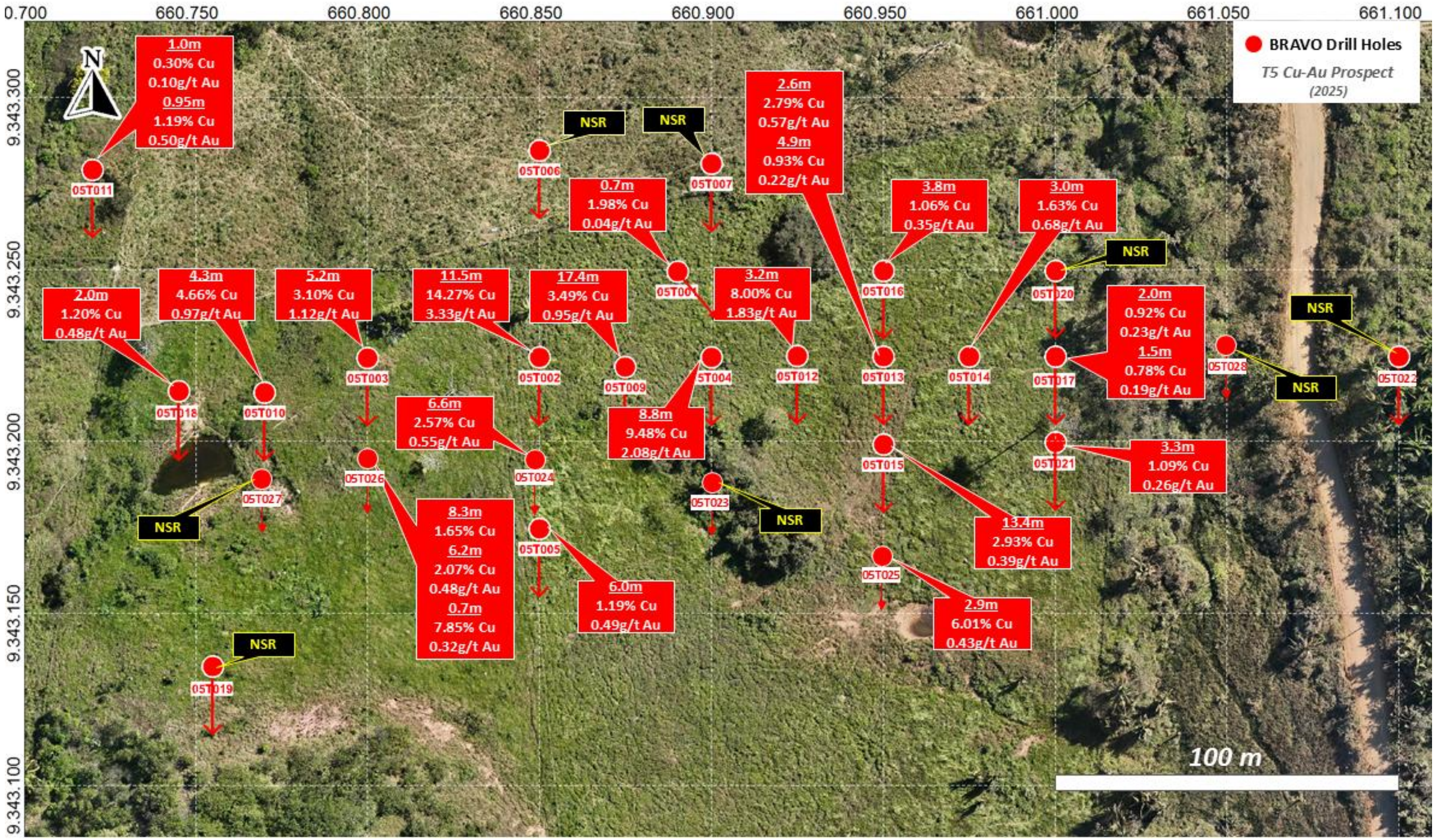


Figure 9-21 - Drill holes location and Cu-Au intercepts at T5

Note: Drill hole DDH2405T006 is not shown on the figure above because it is located far from the mineralized zone.

Source: Bravo 2025.

The Cu-Au sulphide mineralization at T5 occurs primarily as chalcopyrite with subordinate pyrrhotite plus pyrite and can be grouped into four main styles: i) massive to semi-massive sulphide, ii) disseminated, iii) brecciated and iv) vein. The sulphide mineralization is within a hydrothermal alteration zone associated with biotite, actinolite, scapolite, chlorite, apatite, tourmaline, carbonate and silica (Figure 9-22). The system is controlled by a WNW-ESE fault steeply dipping to NNE, that was generated under a brittle regime (brecciation with incipient foliation). The country host rock is a tonalitic intrusion, supported by petrographic study. The hydrothermal assemblage together with the structural control suggest the T5 Cu-Au mineralization can be postulated as an IOCG-type which is a typical style of mineralization in the Carajas Mineral Province.

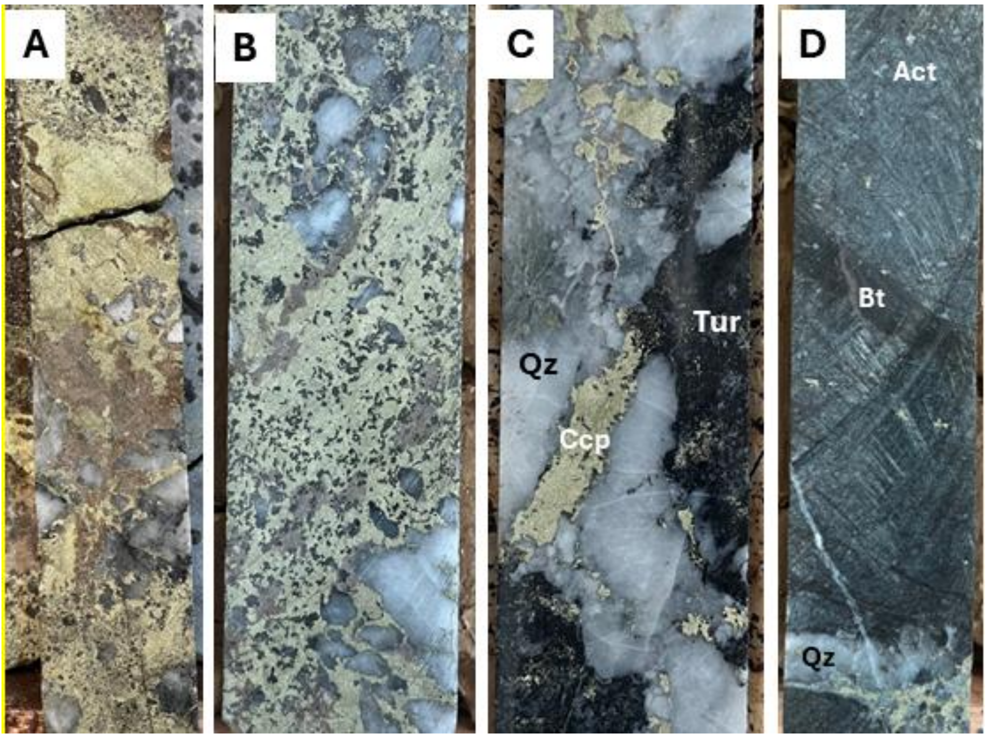


Figure 9-22 - Core samples from T5 mineralization and hydrothermal alteration

Notes: A & B) Sulphide breccia (chalcopyrite + pyrrhotite) with quartz clasts [DDH2405T002]. C) Breccia crosscutting quartz and tourmaline clusters with chalcopyrite [DDH2405T010]. D) Actinolite crosscut by biotite and quartz + chalcopyrite veinlets [DDH2405T012].

Source: Bravo 2025.

Bravo continues to explore at T5, with a detailed structural analysis supported by petrographic study before determining further drilling. To date, no Mineral Resources have been estimated to this area.

9.8.2 Lizard Target

Lizard target is located approximately 2.5 km northwest of T5, and about 3 km, from the North Sector of the Luanga PGM+Au+Ni deposit.

Soil sampling was conducted at stations spaced 25m apart along 100m-spaced gridlines (locally 50m-spaced), covering approximately 192 hectares and totaling 837 soil samples. Soil samples (~3kg) were dried and sieved to <1mm at the Luanga field camp. The <1mm fraction was analyzed with a portable XRF ("pXRF"), to assay 33 elements.

A set of 90 soil samples, representing ~10% of the total population, were selected and re-analyzed by commercial laboratory (Intertek) for Cu (ppm), Ni (ppm), Co (ppm), Ag (ppm) and Au (ppb). These samples (<1mm) were sieved to 80# and pulverized to 150#. Analyses were performed using acid digest/atomic absorption for Cu, Ni, Co and Ag and fire assay/atomic absorption for Au.

While nickel results were generally low and with a poor lateral continuity, a robust copper soil anomaly (~850 m × 450 m) was defined in NE of the target area. This anomaly (defined by Cu ≥150 ppm) is aligned EW, correlating closely with known hydrothermal and siliceous breccia zones.

Mapping at Lizard shown that the target is domained by meta-mafic plutonic rocks (meta-gabbro/meta-basalt) forming an E-W belt, while tonalites occur in a restricted area, in the southeastern portion of the geochemical grid. Between these lithotypes occurs a large corridor of hydrothermally altered, deformed rocks (hydrothermalites) and iron oxide rich siliceous breccias. Siliceous breccia zones with iron oxide are predominantly hosted within chlorite-rich hydrothermalite. Supergene Cu minerals (e.g., malachite) occur on the surface of these breccias and surrounding chlorite schists.

Rock-chip samples reported Cu grades between 0.20 to 0.78% Cu, with one sample assaying 1.39% Cu.

An auger drilling program, comprising 30 holes and totalling 269.70 metres, was carried out over the main Cu in-soil anomaly. Samples were analysed by a portable XRF equipment at the Luanga camp.

Best intercepts are:

- 6.3m @ 0.62% Cu (open to depth), including 1.0m @ 0.75% Cu on hole ADH25LZ002.
- 7.0m @ 0.56% Cu, including 1.0m @ 0.84% Cu on hole ADH25LZ010.
- 7.0m @ 0.45% Cu, including 1.0m @ 0.62% Cu on hole ADH25LZ016.
- 4.0m @ 0.37% Cu (open to depth) on hole ADH25LZ029.
- 2.0m @ 0.36% Cu (open to depth) on hole ADH25LZ028B.
- 9.5m @ 0.30% Cu (open to depth), including 1.0m @ 0.54% Cu on hole ADH25LZ023B.
- 8.0m @ 0.24% Cu (open to depth), including 1.0m @ 0.45% Cu on hole ADH25LZ015.
- 3.0m @ 0.24% Cu, including 1.0m @ 0.40% Cu on hole ADH25LZ017.

GE21 Consultoria Mineral

The location of the auger drilling best intercepts, suggests that the primary mineralization is aligned along a WNW-ESE trend

One trench (133.30 meters long) was excavated over the main Cu in-soil anomaly located on the northeast sector of Lizard target. This program generated 141 channel samples which were analyzed by fire assay (Au) and ICPOES (multi-elements) at the SGG Geosol laboratory.

Trench mapping revealed that both the mineralized zone and its schist host exhibit a consistent N70-80W strike, dipping 70-80° towards north. This structural control is corroborated by auger drilling results. Main Cu intersections on trench are shown on Table 9-4 and trench location in respect with soil geochemistry/auger drilling results presented on Figure 9-23.

Table 9-4 - Copper Intersection on trench TRC25LZ001

Trench_ID	From (m)	To (m)	Length (m)	Cu (%)	Cu Intersection
TRC25LZ001	49.55	54.85	5.30	0.25	5.30m @ 0.35% Cu
	54.85	58.35	3.50	0.12	3.50m @ 0.13% Cu
	58.35	65.10	6.75	0.33	6.75m @ 0.40% Cu

Source: Bravo 2025.

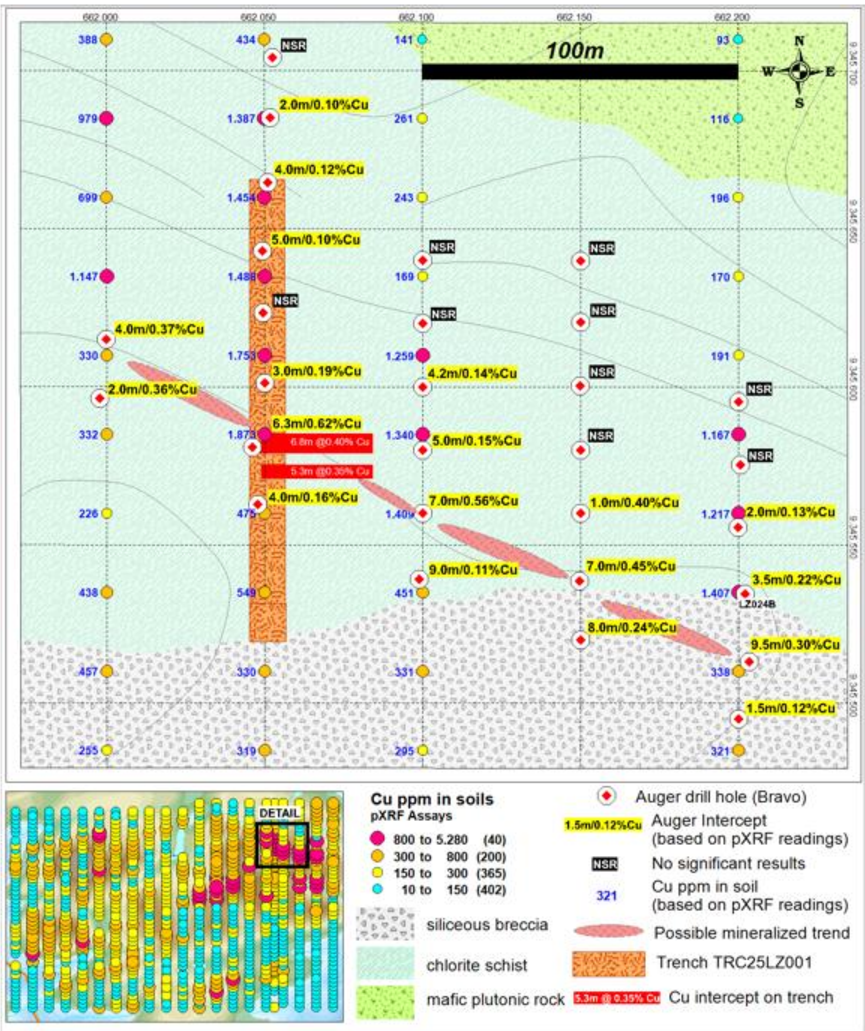


Figure 9-23 - Soil geochemistry, trenching and auger drilling on main zone at Lizard target

Source: Bravo 2025.

9.8.3 *Babylon Target*

The Babylon target was generated to evaluate the Ni-Cu potential associated with a large and prominent magnetic anomaly, together with a strong EM conductor and a gravity high. Previous drilling conducted by Bravo in this area as part of the Luanga PGM program intercepted massive sulphide mineralization with high Ni-Cu grades (Hole DDH22LU047 -11m @ 2.04% Ni, 1.23% Cu). Sulphide mineralization sits at the base of the Luanga Mafic-Ultramafic Complex, in the North Sector, and above a hydrothermal alteration zone consisting of Fe-Ca-K metasomatism. The main metasomatic alteration assemblage includes magnetite, chlorite, amphibole (actinolite, grunerite and minor hornblende), and garnet (almandine). Sulphide paragenesis is represented by pyrrhotite>>pentlandite>chalcopyrite. A ground micro-gravity survey completed by Bravo in 2024 indicates the presence of a gravity high closely associated with a high-amplitude magnetic anomaly on the vicinity of Babylon. In addition to ground geophysics, the HeliTEM survey conducted by Bravo in 2022 also highlights an EM anomaly in the target area, that partially coincides with the gravity high, and sits on the southwestern border of the magnetic anomaly (Figure 9-24).

As of the date of this Report, the Company was designing an exploratory diamond drilling program to evaluate Ni-Cu potential on Babylon target.

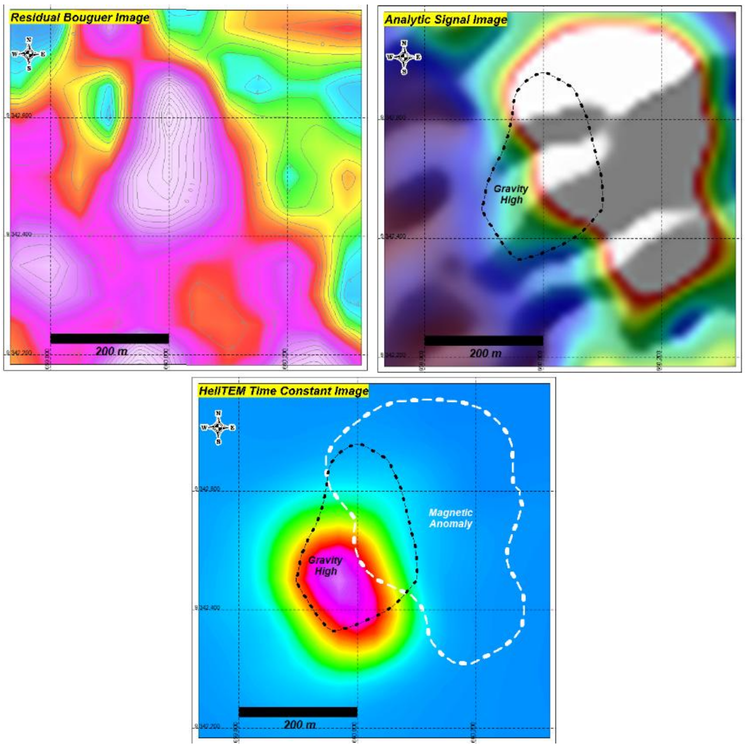


Figure 9-24 - Geophysical response over Babylon target area

Note: Residual Bouguer (Top Left), Analytical Signal (Top Right) and HeliTEM time Constant (Bottom).

Source: Bravo 2025.

9.8.4 Gemini Target

The Gemini target covers several EM conductors (T39 to T45 anomalies) defined by the 2022 HeliTEM survey, which are located 500 to 1,500 meters ENE of T5. The current exploration program comprises of an integrated soil geochemical grid covering approximately 133 hectares. Work also included detailed mapping. The main objective is to identify anomalous Cu-Au-Ni zones that warrant scout diamond drilling.

Preliminary results identified five Cu anomalous zones, above 120 ppm Cu. Notably, values above 400 ppm Cu were frequent. These zones generally showed NE-SW and E-W trends (Figure 9-25), supported by the regional HeliTEM survey. In the Carajás Mineral Province, these NE-SW structures are related Mesoarchean mantle-tapping structures that are older than the emplacement of the IOCG mineralization however, the intersection of these structures with the W-NW Neoarchean structures (like T5) create suitable conduits and trap sights for IOCG mineralized fluids.

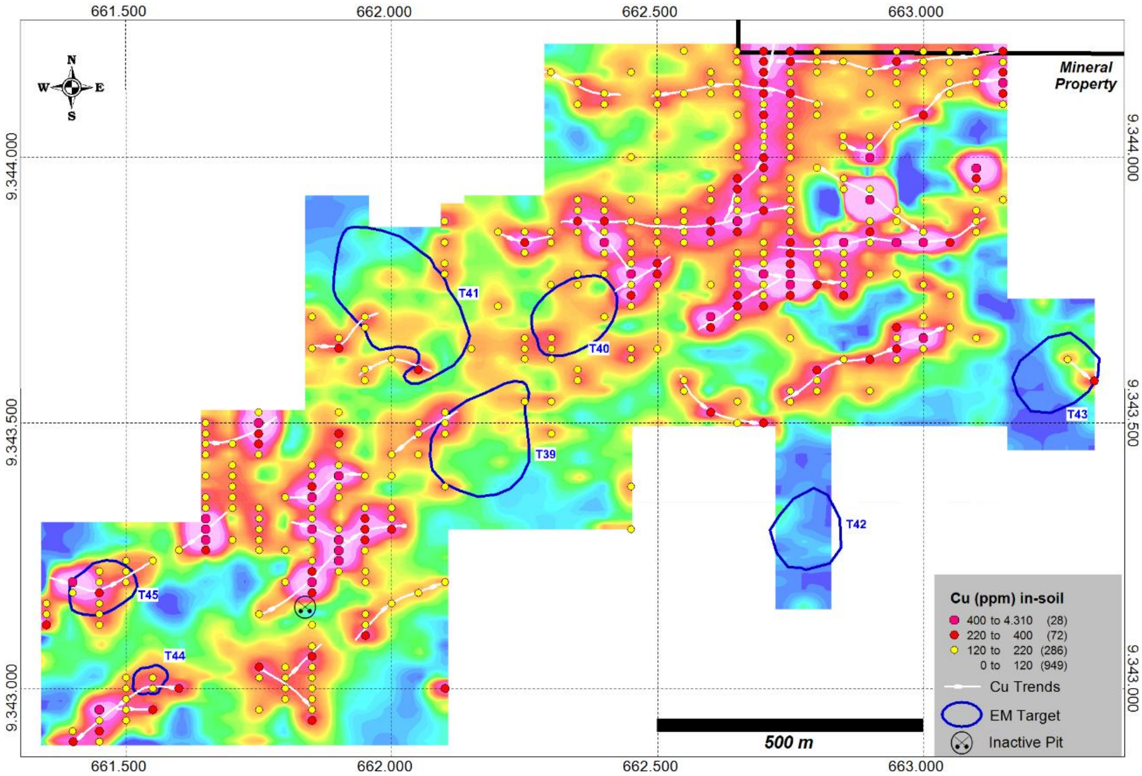


Figure 9-25 - Cu ppm (in-soil) and HeliTEM conductors at Gemini

Note: Soil results obtained from portable XRF equipment. Best Cu results replaced by independent commercial laboratory.

Source: Bravo 2025.

9.8.5 Taurus Target

Taurus is located 2.3 to 3.0 km west of T5, on the northern edge of Luanga mafic-ultramafic layered intrusion. Recently, a geochemical soil grid covering approximately 128 hectares was established to investigate three EM conductors (T48, T49 & T51). Soil samples were analysed by pXRF equipment, and anomalous Cu-Au-Ni results were verified by a commercial independent laboratory.

Currently, four areas of interest, informally named as Zones 1, 2, 3 & 4, have been delineated based on Cu in-soil values greater than 220 ppm Cu, while internal zones greater than 440 ppm Cu exist (Figure 9-26). The largest anomalies are Zone 1 (750m long by ~400m wide) and Zone 2 (750m long by 450m wide).

In terms of geochemical signature, Zones 1, 3 and 4 are defined by anomalous Cu values, while Zone 2 shows a clear Cu-Au association, where Au values are locally higher than 100 ppb (Figure 9-27).

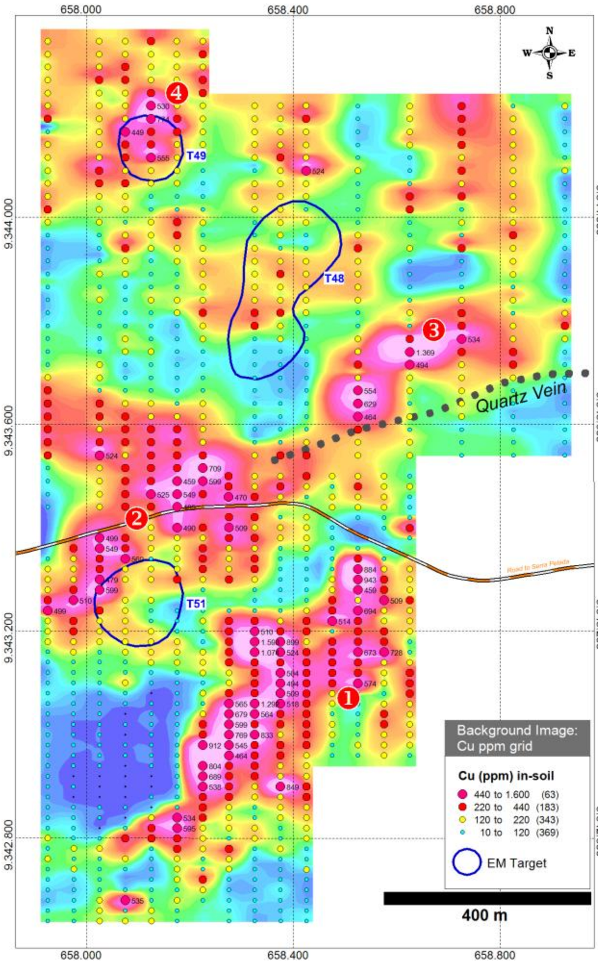


Figure 9-26 - Cu (ppm) in-soil results over Cu grid image on Gemini Target

Anomalous zones referred on the text are indicated by their respective numbers 1, 2, 3, & 4
Note: Soil results obtained from portable XRF equipment. Best Cu-Au-Ni results replaced by independent commercial laboratory results.

Source: Bravo 2025.

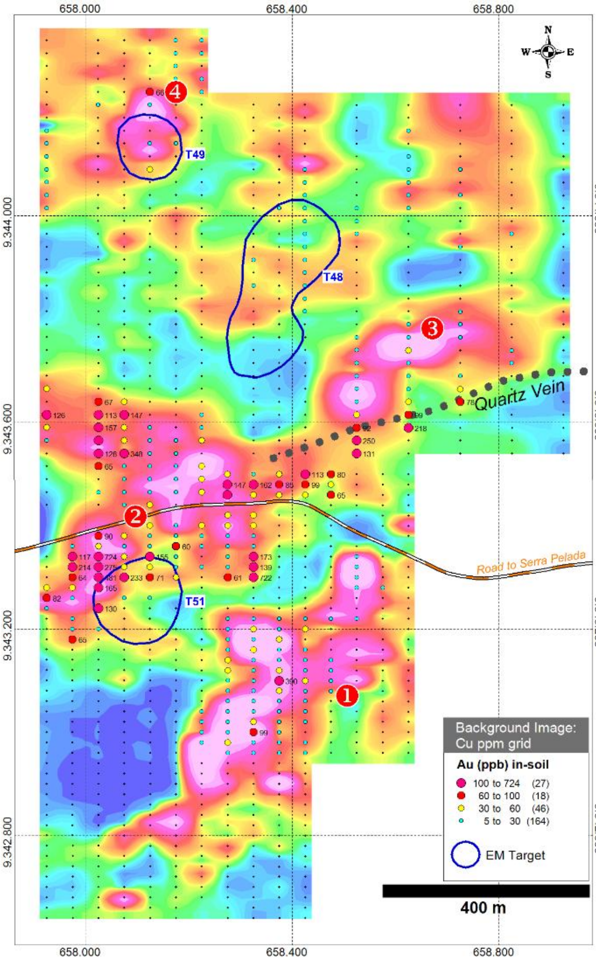


Figure 9-27 - Au (ppb) in-soil results over Cu grid image on Gemini Target

Anomalous zones referred on the text are indicated by their respective numbers 1, 2, 3, & 4
Note: Soil results obtained from portable XRF equipment. Best Cu results replaced by independent commercial laboratory results.

Source: Bravo 2025.

10 DRILLING

10.1 Introduction

Approximately 123,610 m in 601 drill holes have been drilled since 1992,. A total of 256 (50,787 m) are diamond drill holes (DDH) executed by DOCEGEO (Vale). Bravo's drilling corresponds to 72,823 m and 345 holes, representing 59% of the project's drilled metres.

Details of the various drilling programs are summarized in Table 10-1. Holes were drilled using HQ, NQ, BQ and ZWF diameters.

Drilling by Bravo started in March 2022, and has continued since then. The 2022-2024 programs were designed primarily for infill drilling, step-out and resource definition at Luanga. Bravo also performed 8 drill holes for geometallurgical purposes. In 2023/2024, Bravo´s drilling program also included some exploration drilling over several geophysical targets located outside the Luanga PGM + Au + Ni deposit. Bravo's diamond drilling was performed using a mixture of HQ and NQ2 diameters. The drill holes that were included in the Mineral Resource estimate are discussed in Section 14 of this report.

Table 10-1: Drilling summary for Luanga

<table>
<tr><th>Year</th><th>Drill Type</th><th>Drill Holes</th><th>Total Metres</th><th>Company</th><th>Contractor</th></tr>
<tr><td>1992</td><td>DD</td><td>4</td><td>643.69</td><td rowspan="4">DOCEGEO</td><td>DOCEGEO</td></tr>
<tr><td>2001</td><td>DD</td><td>86</td><td>14,584.35</td><td>Geosol</td></tr>
<tr><td>2002</td><td>DD</td><td>71</td><td>15,423.25</td><td>Geosol</td></tr>
<tr><td>2003</td><td>DD</td><td>95</td><td>20,135.45</td><td>Geosol / Rede</td></tr>
<tr><td>2022</td><td>DD</td><td>135</td><td>23,258.20</td><td rowspan="3">Bravo</td><td>Servdrill</td></tr>
<tr><td>2023</td><td>DD</td><td>116</td><td>30,296.60</td><td>Servdrill</td></tr>
<tr><td>2024</td><td>DD</td><td>94</td><td>19,268.65</td><td>Servdrill</td></tr>
<tr><td colspan="2">TOTAL</td><td>601</td><td>123,610.19</td><td></td><td></td></tr>
</table>

Source: Bravo, 2025.

10.2 DOCEGEO Drilling

The drilling conducted under DOCEGEO administration consists of 256 diamond drill holes (50,787 linear metres) at Luanga between 1992 to 2003 (Table 10-2). Most of the Diamond Drilling was carried out by two Brazilian Diamond Drilling companies Geologia e Sondagem S.A. (Geosol) and Engenharia e Sondagem Ltda (Rede). DOCEGEO was responsible for the first four drill holes at the Project.

Table 10-2: Historical drilling summary

<table>
<tr><th>Year</th><th>Drill Type</th><th>Drill Holes</th><th>Total Metres</th><th>Contractor</th><th>Target</th></tr>
<tr><td>1992</td><td>DD</td><td>4</td><td>643.69</td><td>DOCEGEO</td><td>Luanga</td></tr>
<tr><td>2001</td><td>DD</td><td>86</td><td>14,584.35</td><td>Geosol</td><td>Luanga</td></tr>
<tr><td>2002</td><td>DD</td><td>71</td><td>15,423.25</td><td>Geosol</td><td>Luanga</td></tr>
<tr><td rowspan="3">2003</td><td rowspan="3">DD</td><td>67</td><td>14,535.15</td><td rowspan="3">Geosol - Rede</td><td>Luanga</td></tr>
<tr><td>4</td><td>413.15</td><td>Luanga (Met)</td></tr>
<tr><td>24</td><td>5,187.15</td><td>Luanga South</td></tr>
<tr><td colspan="2">TOTAL</td><td>256</td><td>50,786.74</td><td></td><td></td></tr>
</table>

Source: Bravo, 2025.

GE21 Consultoria Mineral

Most of the diamond drill holes (248 holes) were drilled with inclinations varying from -55.0º to -70.0º, with the predominant inclination at -60.0º.

The maximum drill hole length in historical drilling was 497.6 m and the average hole length was 198.4 metres. The Diamond Drilling (DD) holes were drilled with a HQ (96.40 mm) diameter in the weathered zone, changing to NQ (76.20 mm) diameter in the fresh rock. There is no information about the drilling recovery in the historical database. However, from visual inspection of available core from these programs, recoveries appear to have been excellent.

The near surface portion of Luanga has been oxidized to depths of a few meters to a few tens of metres and is underlain by a thin transition zone before fresh rock is encountered. PGMs and Au are potentially recoverable from both oxide and sulphide mineralization, based on comparable deposits, whereas Ni would typically only be recovered from sulphide mineralization.

10.2.1 DOCEGEO Drill Collar Survey

The drill holes collars were sited based on the Instituto Brasileiro de Geografia e Estatística (IBGE) base datum. All the drill holes collars were surveyed at the end of each drilling campaign, using Total Station TOPCOM GTS 229 equipment with the final location entered into the drilling database. The survey Datum used was SAD69.

All the drill holes collars were capped with cement blocks, including a PVC tube and aluminium plates, including drill hole number. Information related to hole ID, coordinates, elevation, dip, azimuth and final depth data are included on the collar plugs on the aluminium plates.

Bravo re-surveyed the majority of Docegeo's drill holes, using the SIRGAS 2000 Datum. The collars that were not found on the area, their original coordinates were converted from SAD69 to SIRGAS 2000.

10.2.2 Downhole Survey

Downhole deviation surveys were carried out along the length of 240 diamond drill holes with readings collected at 3 metres intervals, covering approximately 95% of the total drill hole population.

10.3 Bravo Drilling Program

To the Effective Date, 345 drill holes have been completed within the Luanga PGM + Au + Ni deposit area and, also outside as part of the regional exploration over other targets. In total, 72,823.45 metres of diamond drilling have been completed by Bravo (Table 10-3 and Figure 10-2). Of these, 299 drill holes (63,364.1m) were drilled with the objective to intercept the Luanga PGM + Au + Ni mineralized system and this program included 8 twin holes and 8 geometallurgical drill holes. In general drill holes intercepted the same lithologies and the 2 styles of mineralization previously described by DOCEGEO on its drilling program. However, another four (4) different styles of mineralization were identified by Bravo, and some lithological units were renamed based on petrographic studies, as described in detail in Section 7 of this report.

Geometallurgical drill holes (total of 8 and 882,15 m) were drilled for the purpose of obtaining volumetric samples for metallurgical tests and did not follow the systematic assaying routine used in the drilling program. Consequently, these drill holes were not used in the Mineral Resource estimate.

As a part of the drilling program at Luanga, Bravo also carried out 46 drill holes (9,459.35 metres) over 14 geophysical targets defined by the HeliTEM survey. Some of these holes are in the proximity of the Luanga PGM + Au + Ni deposit, while others are located away from the deposit.

Table 10-3: Diamond drilling quantitative

Year	Drill Type	Drill Holes	Total Metres	Company	Contractor	Phase
2022	DD	119	21,241.55	Bravo	Servdrill	Luanga new holes
		8	1,134.50			Luanga twin holes
		8	882.15			Luanga met. holes
2023	DD	103	27,878.50		Servdrill	Luanga new holes
		13	2,418.10			Regional targets
2024	DD	61	12,227.40		Servdrill	Luanga new holes
		33	7,041.25			Regional targets
TOTAL		**345**	**72,823.45**			

Source: Bravo, 2025.

Figure 10-1: Drill rig in operation

Source: Bravo, 2025.

GE21 Consultoria Mineral

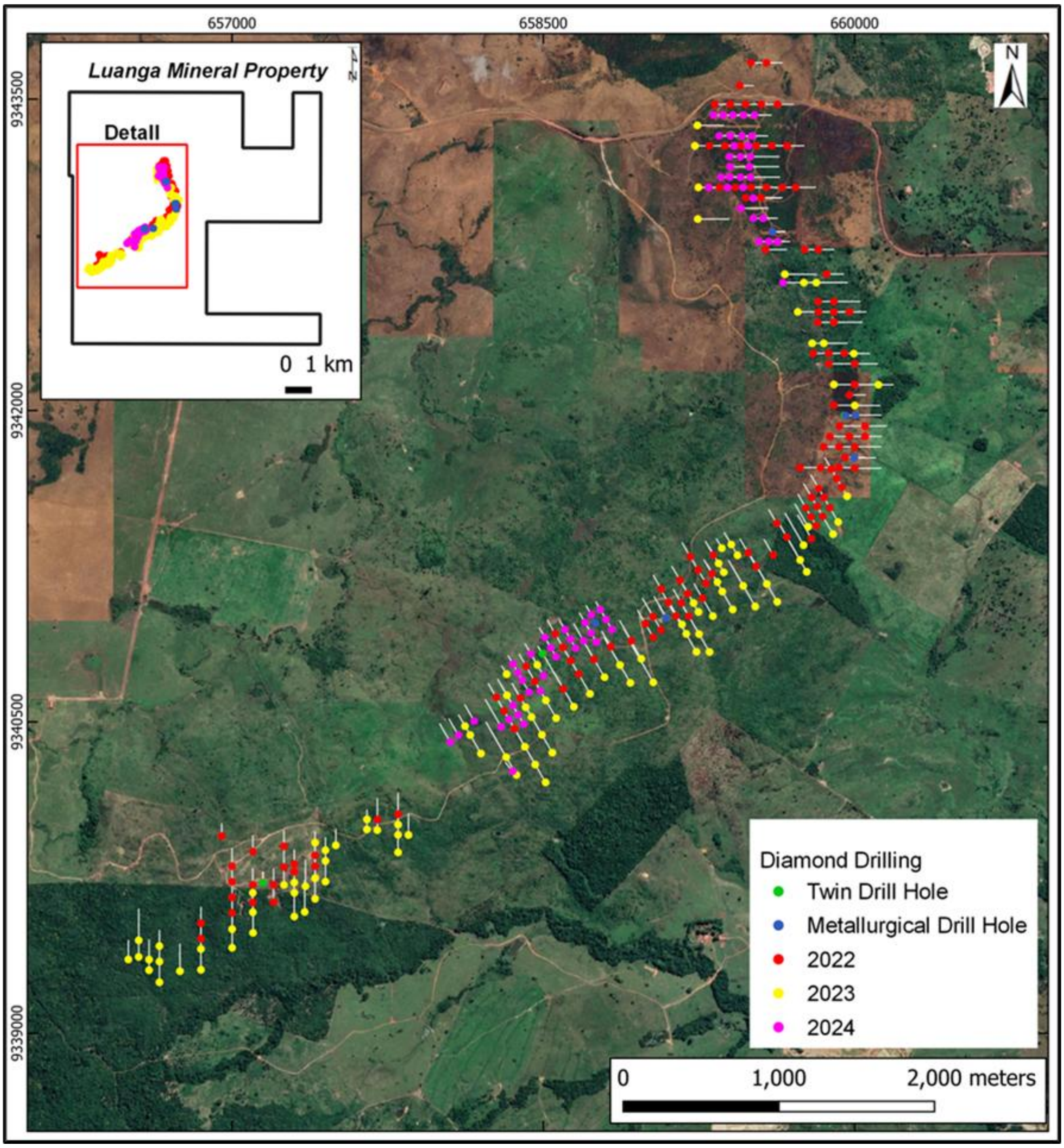


Figure 10-2: : Bravo drill hole location map for Luanga target

Source: Bravo, 2025.

The shallowest drill hole drilled by Bravo was 76.55 m and the deepest hole was 625.35 m, with an average depth of 214 m. Drilling used a HQ sized diamond core from surface, until reaching competent fresh rock, where drilling changed to NQ2 sized diamond core. Core recoveries are generally excellent, with averages of >98% in the fresh rock and 94% for oxidized rocks. Mineralization is finely and evenly disseminated, thus it is believed that there will be no nugget effects or issues affecting accuracy and reliability, as was the case in the historical core.

10.3.1 Bravo Drill Collar Survey

All drilling holes collars are geolocated with RTK with geodetic accuracy, being initially outsourced through the company RR Topo and, subsequently, done by Bravo's own team. The survey Datum used was SIRGAS 2000.

10.3.2 Bravo Downhole Survey

Deviation surveys was conducted for all holes using REFLEX GYRO SPRINT-IQ device. In addition, runs are guided whenever possible with Reflex ACT3 device (Figure 10-3).

Figure 10-3: REFLEX GYRO SPRINT-IQ device used for guided run

Source: Bravo, 2023.

10.3.3 Core Loging

In the core shed (Figure 10-4), the boxes were placed on racks for checking the depth, advance and recovery information, in addition to meter-by-meter marking. Then, magnetic susceptibility measurements was taken with the KT-20 S/C device. Marking of the oriented intervals is carried out to then proceed with structural measurements using the an IQ Logger device (Figure 10-5).

Figure 10-4: Core shed

Source: Bravo, 2023.

Figure 10-5: Marking of the oriented intervals

Legend: (A) Checking the core boxes and (B) taking structural measurements with IQ Logger.
Source: Bravo, 2023.

A geotechnical description of the holes is completed, focusing on obtaining the RQD measurement. Information about the strength and weathering of the rocks is also collected.

After the geological description, a sampling plan was drawn up (discussed in detail in Section 11). Then the numbering of the samples is marked on the boxes. A marking is also made to guide the sawing of the core into two equal halves (Figure 10-6). Photographs (wet and dry) are taken of all core boxes, which are then sawed and sampled.

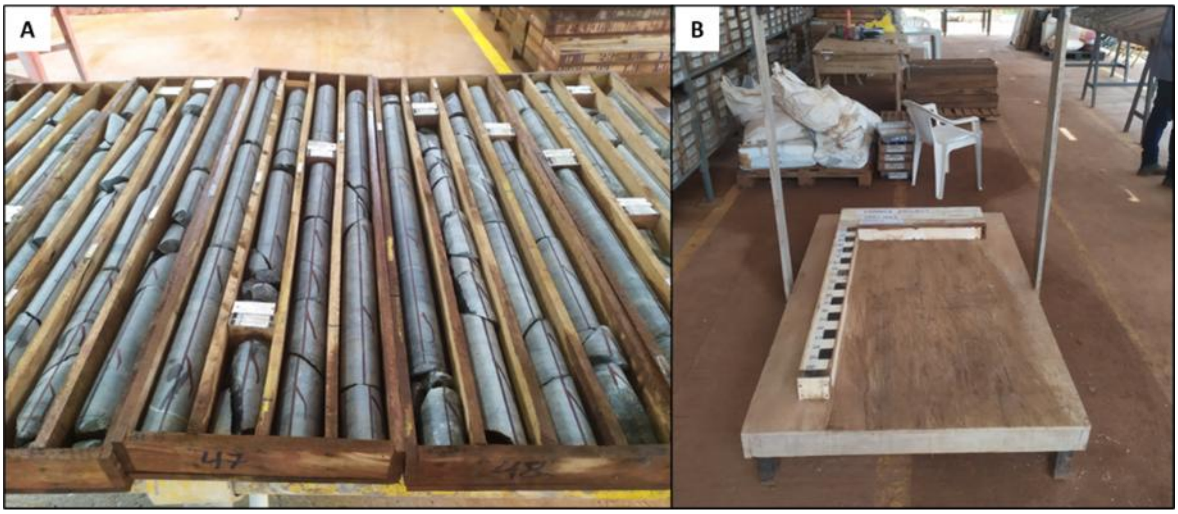


Figure 10-6: Core logging

Legend: A) Drill cores with core orientation markings and B) Core Photography Table.
Source: Bravo, 2023

10.4 Twin Holes

A twin hole program was designed to validate the historical drilling (Table 10-4). The objective was to support the use of historical drilling into Bravo's database with the same degree of quality and confidence in the geological, geochemical, and sample information. This program includes the drilling of 8 "twin holes" against historical drilling, following the coordinates surveyed in the field in SIRGAS 2000 format.

Table 10-4: Selection of results from Bravo twin hole drilling

HOLE-ID	From (m)	To (m)	Thickness (m)	Pd (g/t)	Pt (g/t)	Rh (g/t)	Au (g/t)	DOCEGEO PGM + Au (g/t)	Bravo PGM + Au (g/t)
TWIN of Historical Hole PPT-LUAN-FD0136									
DDH22LU043	0.00	16.70	16.70	15.92	16.51	3.63	0.05		36.12
PPT-LUAN-FD0136	0.00	17.00	17.00	17.36	18.36	2.94	0.17	38.73	
DDH22LU043	34.90	86.50	51.60	0.84	0.56	0.08	0.12		1.60
PPT-LUAN-FD0136	24.00	78.00	54.00	0.46	0.36	0.11	0.07	0.93	
TWIN of Historical Hole PPT-LUAN-FD0095									
DDH22LU083	0.00	93.00	93.00	1.80	1.15	0.20	0.02		3.17
PPT-LUAN-FD0095	0.00	93.00	93.00	1.60	1.01	0.10	0.01	2.71	
TWIN of Historical Hole PPT-LUAN-FD0145									
DDH22LU006	0.00	37.22	37.22	1.93	0.83	0.13	0.26		3.15
PPT-LUAN-FD0145	0.00	40.00	40.00					2.92	

Notes:

1. All ‘From’, ‘To’ depths, and ‘Thicknesses’ are downhole.
2. Intercepts are estimated to be 75% to 85% of true thickness.
3. NA: Not Applicable as intercept is oxide, or a mix of oxide and fresh rock mineralization.

Source: GE21, 2025.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

The procedures carried out by Bravo's sampling program since 2022 are summarized below.

11.1 Sample Security and Chain of Custody

The Company's QA/QC Policy outlines the key procedures implemented by Company personnel to ensure the reliability of exploration data and laboratory analytical results. These measures also maintain an accurate and secure database for the Company's exploration and development projects.

Core boxes and sample bags containing samples for analysis are transported from the drill site to the Company's core shed in a manner approved by the QPs. Each delivery is accompanied by a Chain of Custody Form, which includes the names and signatures of transferring parties, the date and time of transfer, the drill hole number and sample interval, and the box sequence with the total number of boxes and bags transferred. Upon arrival at the core logging facility, the samples received are verified against the Chain of Custody Form, and any discrepancies are reported to the QP.

The core shed and office area, where the database and records for exploration and development projects are stored, are restricted to authorized Company personnel and individuals specifically approved by the QP. These areas remain locked when not in use. Staff handling core samples must remove or tape off hand jewelry to prevent contamination or loss of sample integrity. Visitors are not permitted to handle core unless supervised by Company personnel.

Drill cores are laid out in sequential order and cleaned before geological logging and sample selection, which is conducted by authorized project geologists. Once samples are selected for cutting or splitting, a guideline is drawn on the core to ensure mineralization is equally split between the halves. If required, one half may be further split into quarters. Technicians, under the supervision of the Core Logging Supervisor, saw or split the core along these guidelines. All portions of the split core are returned to the core box in their original order and orientation.

After logging and sample selection, the core is marked, and sample numbers are stapled to the core boxes. Photographs are taken to ensure that core blocks, including length and sample numbers, are clearly visible. The portion of the core selected for analysis, typically half and always the right-side piece, is placed into pre-labeled and tagged sample bags following QP-approved procedures. Sample bags are secured with zip ties and then placed into larger bags sealed with tamperproof safety seals. These samples are sent to the assay laboratory along with assaying instructions. The Chain of Custody Form and lab submittal documents instruct the recipient to notify the sender immediately if any tamperproof seals are broken or compromised.

The remaining half of the core sample is retained in the original core box and stored in a secure location, with its location recorded. No core samples may be removed for re-assaying, metallurgical testing, or any other purpose that could lead to sample destruction without prior approval from the QP. If a sample is removed following QP approval, a record is inserted in the core box at the sample's original location, detailing the sample length and interval removed, the purpose of removal, the name of the responsible individual, and the date of removal. A digital record with the same details is maintained in the database.

These measures ensure the security and traceability of all samples from collection through analysis, maintaining compliance with industry best practices and regulatory requirements.

11.2 Sampling

The standard sample size is typically 1-meter length, with a tolerance of 0.70 m to 1.20 m length sample, respecting lithological contacts, weathering zones, and any interval with low recovery.

After drawing up the sampling plan, the samples are identified in the core boxes with the respective numbers. Then the core boxes are photographed, with drill core dry and wet (Figure 11-1).

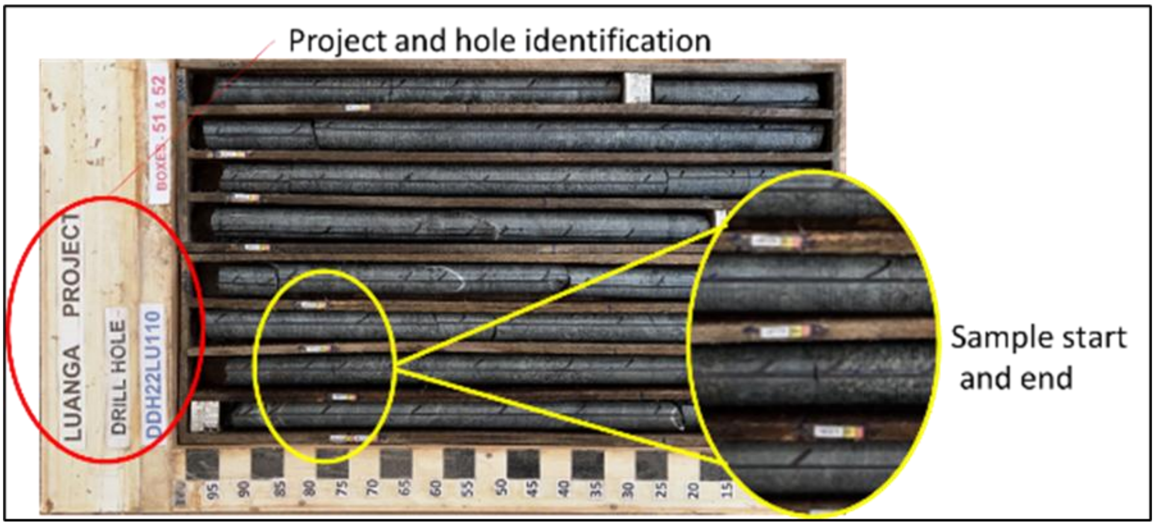


Figure 11-1: Example of photographic record of drill core box with marks and sampling ID

Source: Bravo, 2023.

Sampling takes place on the right side of the cut core and organized in plastic bags, closed and sealed with identified seals. Each sample is weighed individually, with the information registered on the sampling plan and in the database.

The assay request is prepared by the geologist responsible for the sampling plan, which is accompanied by a letter of custody with volumes sent with the list of samples and their respective seals. Each analytical batch is transported by Bravo's technical team to ALS or SGS Geosol laboratories in Parauapebas-PA (Brazil) by truck from Bravo's site. The relogging and resampling and analysis executed as of the Effective Date is described in Figure 11-2.

Drill core samples are prepared in Parauapebas-PA and analyzed in Lima (Peru) by ALS laboratories (ALS), or at Vespasiano-MG (Brazil) by SGS Geosol Laboratórios Ltda (SGS). Until December of 2022 SGS was used as the secondary laboratory but, in January 2023, Bravo started using SGS as the primary laboratory and ALS as the secondary laboratory.

Drill core samples are dried, crushed (90% passing 2mm), split (riffle splitter) and pulverized (200 mesh). Analytical methods applied are fire assay with ICP-AES finish (Pt, Pd, Au), nickel sulphide ("NiS") leach with ICP-AES finish (Ni) and fire assay with ICP-MS finish (Rh).

ALS' Fire Assays use the Pb Collection method, while SGS' Fire Assays use the NiS method for metal collection. ALS and SGS are ISO-accredited (ISO: 17025:2005) commercial laboratories, completely independent of Bravo.

Bravo's drilling and sampling campaign procedures flowchart is described in Figure 11-2.

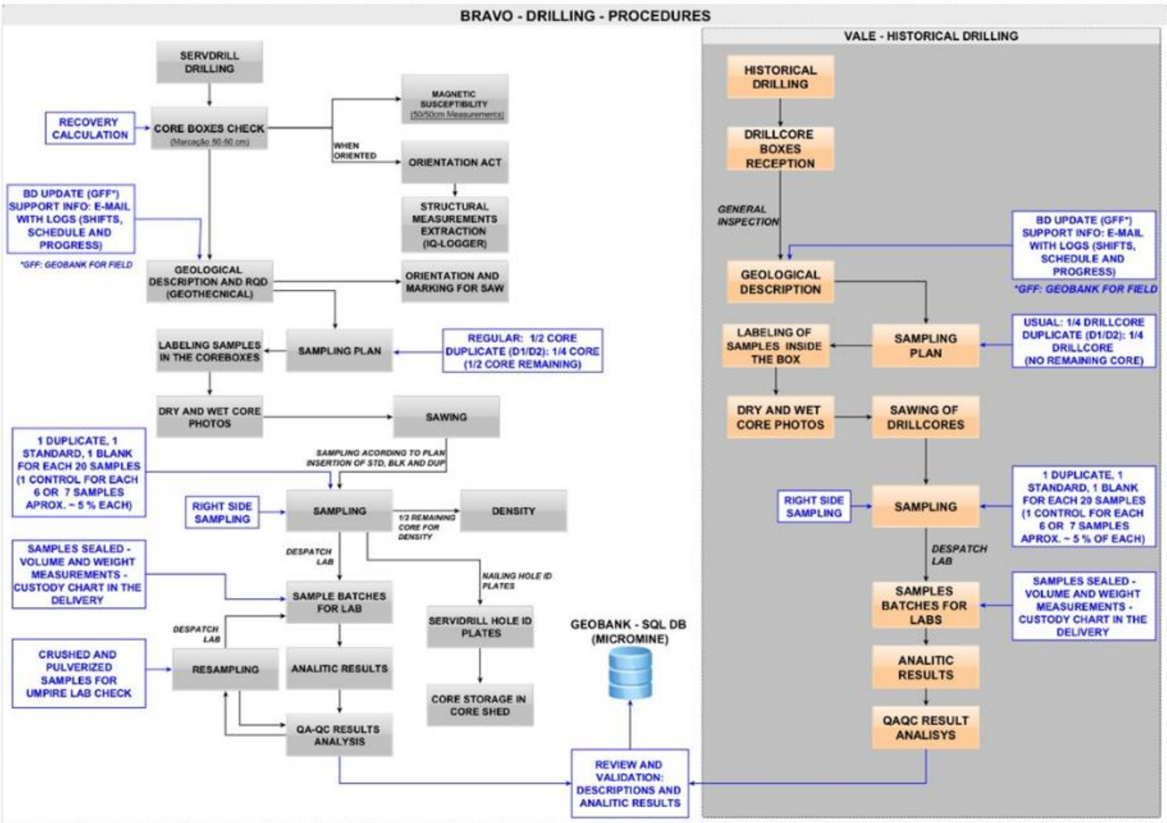


Figure 11-2: Bravo's drilling and sampling procedures

Source: Bravo, 2023.

The assay database used in the Project accounts for both Vale and Bravo samples. Vale samples which were reanalyzed by Bravo have had their original grades replaced by the new re-analyzed grades in the database.

The Table 11-1 shows the proportion of assay results present in the database from Vale and Bravo programs.

GE21 Consultoria Mineral

Table 11-1: Assays proportions by company

Company:	# Samples:	% Samples:
Bravo	75,249	63%
Vale	43,683	37%
Total	**118,932**	**100%**

Source: GE21, 2025.

11.3 Density Sampling

Density sampling of drill core is executed by Bravo's technical team, conducted after sawing and sampling for chemical analysis. A half core sample is collected every meter, corresponding to half of the remaining core in the box. The selected samples are 10 to 35 cm in length and marked with the sample number and core orientation. Subsequently, the samples are submerged in water and weighed on a 0.001kg precision scale to obtain the submerged weight.

The samples are then dried in an oven for 24 hours at 105 °C and weighed for a second time to obtain the weight in air. After the dry weight measurement, the samples are wrapped in cling film and subjected to volume measurement using the water displacement method (Figure 11-3). Finally, the samples are returned to the core box. Bravo has performed a total of 27,170 density assays as of the Effective Date of this report.

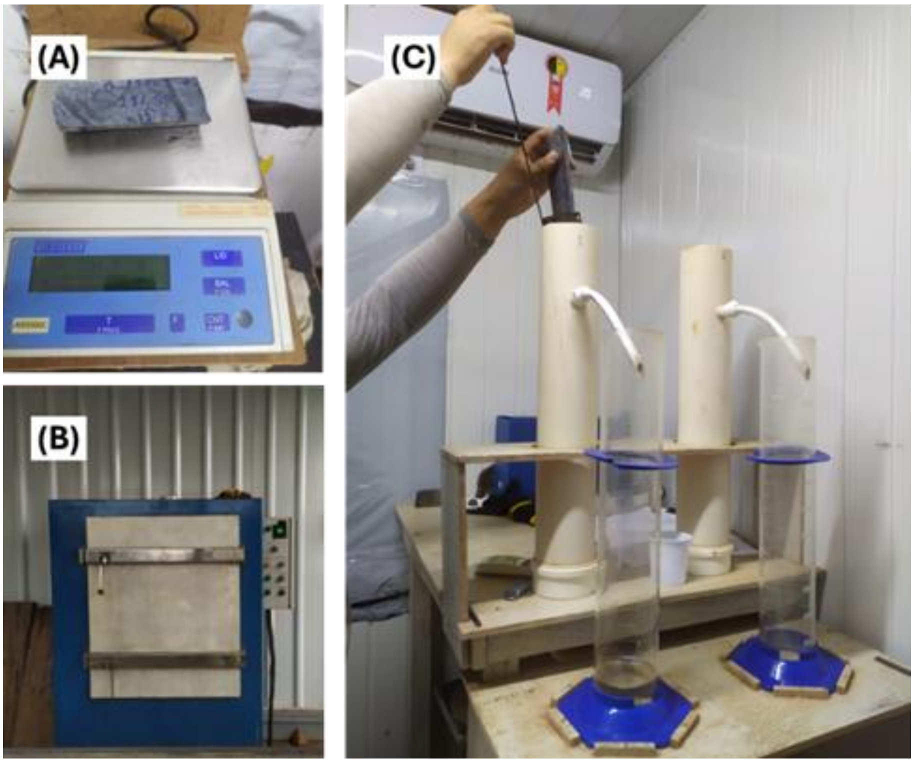


Figure 11-3: Sample density

Legend: A) Scale; Bottom left. B) Drying oven. C) Volume calculation.
Source: GE21, 2023.

Bravo's drill core density statistics are summarized in Table 11-2.

Table 11-2: Bravo's drill core density results on fresh rock

Target	Number of Samples	Length (m)	Mean	Minimum	Median	Maximum	Standard Deviation
			g/cm³				
North Mineralization	2,486	585.0	2.83	1.51	2.85	4.47	0.20
Central Mineralization	2,895	672.3	2.88	1.43	2.91	5.31	0.18
Southwest Mineralization	1,532	407.7	2.84	1.43	2.88	4.32	0.16
Host Rock	20,257	4,537.1	2.83	1.30	2.84	4.87	0.20
TOTAL	**27,170**	**6,202.1**	**2.84**	**1.30**	**2.85**	**5.31**	**0.19**

Source: GE21, 2025.

11.4 Quality Assurance and Quality Control

QA/QC procedures for assays adopted in Bravo's Diamond Drilling campaign include Field Duplicates, insertion of Certified Reference Materials (CRMs), Blank samples and Umpire Assay samples. The adopted QA/QC procedures follow the guidelines of the Company Technical Assurance Statement.

Blank and CRM samples are commercial, acquired from OREAS, AMIS and Brasil Minas suppliers. Control samples (blank, CRM and duplicate) are inserted in the analytical batch at the site at a ratio of 1:20 regular samples.

Bravo's QA/QC program accounts for 14,159 control samples, including CRMs, Blank Samples, Field Duplicates and Umpire Check Assays, representing 10.7 % of the total samples. Table 11-3 presents a summary of the QA/QC samples, including the Vale QA/QC program.

Table 11-3: Bravo's QA/QC summary

Bravo	**CRM**	**4,060**
	Blank	4,126
	Field Duplicate	4,417
	Umpire Check	1,556
	Total Bravo QC Samples	14,159
	Total Bravo Samples	75,249
	Total Bravo Database	89,408
	Total QC Samples | Bravo Database (%)	15.8%
Vale	Duplicate	2,836
	Blank	720
	Total Vale QC Samples	3,556
	Total Vale Samples	43,686
	Total Vale Database	47,239
	Total QC Samples | Vale Database (%)	8.7%
Vale – Resampled	Resampled by Bravo	2,162
	Resampled by Bravo | Vale Database (%)	4.6%
	Vale QC + Resampled by Bravo | Vale Database (%)	12.1%
Total	Total Samples	118,932
	Total QA/QC	19,877
	Total	138,809
	Total QA/QC percentage	14.3%

Source: GE21, 2025.

Bravo's QA/QC program also accounted for a resampling campaign, aiming to validate the Vale database and establish a correlation between total (silicate, oxide and sulphide) Ni grades analyzed by Vale and recoverable sulphide Ni grades assayed by Bravo. A total of 2,056 core intervals were resampled and analyzed, representing 5% of the Vale samples. The analyzed resample results were entered into the drilling database, replacing the previous total Ni grades. Twin Holes were also drilled to evaluate the quality of Vale's previous drilling, sampling, and assaying.

Bravo's team produces regular QA/QC internal reports to constantly monitor the quality of the received assay results. GE21 has accessed the reports from May 2022 to November 2024. These reports are also used as a Quality Assurance measure, specifying batches or parts of batches to be reanalyzed.

It is important to note that the selected samples are only reanalyzed if the Laboratory is available, and priority is given to unanalyzed batches. The selection and priority for reanalysis is based on the following guidelines:

Full Batch Re-assay:

- Number of failed field duplicate samples (Absolute Difference < 10% of the Pair Mean + 2x Detection Limit).
- Partial Re-assay (5 previous and 5 next samples, relative to the failed control sample):
- Number of failed CRM Samples (> 3x Standard Deviation (SD))
- Number of failed Blank Samples (>20x Detection Limit (DL))

Priority is given for the batches or sub-batches with (more) Pt and Pd failed assays. Fails in control samples do not necessarily generate a re-assay request.

11.4.1 Blank Samples

The Blank samples used are commercial materials. A total of 6 types of blanks were used. Table 11-4 presents the number of blank samples used in. Blanks AMIS0793 and OREAS 22d are CRMs, while BLK, BLK1, BLP2 and Q403 are Brasil Minas' blank materials.

Table 11-4: Bravo's blank samples summary

Blank Material	Number of Samples
Q403	1,798
BLK	9
BLK1	315
OREAS 22d	215
BLP2	930
AMIS0793	859
Total Blank Samples	4,126

Source: GE21, 2025.

GE21 has generated control charts from the Blank samples' data, using 3x the DL as the acceptance level. The data was divided into SGS and ALS analysis for the control charts construction. Considering all the Blank samples analysis for PGMs, Au and Ni, results have presented more than 98% of the assay grades below the "3x DL" acceptance level. The results indicate there was no significant or systematic contamination during the sample preparation and analysis stages.

11.4.2 Certified Reference material – CRM

Bravo's QA/QC includes a range of different Certified Reference Materials (CRM), 25 in total. Of these, 16 are from AMIS, 7 from OREAS, and 2 from CDN. The variety of Standard Materials aims to cover low, medium, and high-grade ranges of the main analyzed elements: Pt, Pd, Rh, Au. Table 11-5 presents the quantitative of Reference Materials included in the QA/QC program.

Table 11-5: Bravo's CRM samples summary

CRM	Number of Samples
AMIS0279	33
AMIS0486	407
AMIS0502	283
AMIS0504	214
AMIS0606	332
AMIS0723	49
AMIS0749	275
AMIS0759	351
AMIS0760	114
AMIS0771	277
AMIS0822	153
AMIS0853	49
AMIS0854	23
AMIS0874	150
AMIS0876	293
AMIS0890	4
CDN-ME1310	25
OREAS 085	55
OREAS 083	57
OREAS 680	35
OREAS 681	145
OREAS 682	164
OREAS 683	166
OREAS 684	37
CDN-PGMS29	23
Total CRM Samples	**4,060**

Source: GE21, 2025.

GE21 has generated Control Charts for all the Certified Materials used, dividing assay results by laboratory and by assay method. This assay method separation is essential since the SGS laboratory uses the NiS Fire Assay method while ALS uses the Pb Collection Method. Due to differences in the target metals collection, CRMs have different certified grades and SD's for Pb Collection Fire Assay and NiS Fire Assay.

Nickel control charts have shown inconsistent behavior since none of the CRMs used by Bravo have certified grades for sulphide nickel analysis, which is the current Ni assay method. Bravo's CRMs are only certified for total Ni analysis, which includes Ni present in silicates and oxides. As a result, a direct comparison between these two assay methods is not possible, making it difficult to objectively evaluate the accuracy of the nickel analysis in the database. GE21 recommends that Bravo acquire Certified or Standard Reference Materials certified for sulphide Ni assay methods.

In the QA/QC Internal Reports from December 2022 and January 2023, Bravo identified and reported to ALS a non-conformance related to an excessive number of failed AMIS0504 samples on 12 different batches. As results of this non-conformance, an internal investigation was conducted by ALS. Bravo also conducted its own investigation on the matter. Both investigations resulted in similar conclusions: the high content of refractory oxides (i.e., Al2O3, Fe2O3 and Cr2O3) in the material require higher fusion temperatures to the complete metal recovery. ALS has also reported three main possible causes for the low PGMs and gold recovery: differences in the collection methods using NiS and Pb, indicating higher recovery in the NiS method; mineralogical complexity and high refractory oxides content; the weight reduction procedure was considered below optimal, suggesting the need for further mass reduction of the sample and/or the addition of reagents to improve fusion.

GE21 analyzed the CRM data using the Certified Grades and Standard Deviations reported in each Reference Material's Certificate and found that the CRMs AMIS0279, AMIS0504, AMIS0606, AMIS0749, AMIS0760, AMIS0771, AMIS0890, OREAS085, OREAS086, OREAS680, and OREAS684 have consistently shown negative bias. GE21 has reviewed the certified values for Al_2O_3, Fe_2O_3, and Cr_2O_3 in all these CRMs and found that most of them contain a significant combined amount of these elements, which could explain the observed negative bias (as pointed out on the investigations carried out by Bravo and ALS).

The AMIS standards data used by Bravo were revised for QA/QC programs. The recommendation states that new Reference Grades and SD's should be obtained from the data acquired during the Project (after outlier treatment). After this procedure, new reference values were generated. No significant biases were found after the correction of Reference Values. Bravo discontinued the use of OREAS CRMs.

11.4.3 Field Duplicates

Bravo uses Field Duplicates to evaluate the precision of the sampling procedures. Duplicate samples are generated using ¼ drill cores. The Field Duplicates control charts were generated only for the duplicates with both original and duplicate grades above 3x the DL of the analytical method evaluated. This measure removes analytical artifacts that occur near the Detection Limit. Table 11-6 summarizes the Control Charts generated. GE21 carried out an additional Exploratory Data Analysis (EDA) of Field Duplicates.

Table 11-6: Bravo's duplicates samples summary

Laboratory	Sample Count	Element:	Assay Method
ALS	1,217	Au	PGM-ICP27
		Pd	PGM-ICP27
		Pt	PGM-ICP27
		Rh	Rh-MS25
		Ni	Ni-ICP05
		Ni	ME-ICP61
SGS	3,200	Au	FAI515
		Pd	FAI515
		Pt	FAI515
		Rh	FAA35J
		Rh	FAI30V_RH
		Ni	AAS04B
		Ni	ICP40B
		Ni	AAS04B

Source: GE21, 2025.

Duplicated samples show a good correlation with the original samples. The PGMs and gold duplicate results range between 60% and 90% below the 15% Half Average Real Difference (HARD) limit.

GE21 recommends to Bravo, the inclusion of course and fine duplicates as control samples of its QA/QC program. Coarse and fine duplicates might be collected as an aliquot from the material after the mass reduction processes employed after crushing and milling, respectively. This measure should help to evaluate the quality of the sample preparation processes.

11.4.4 Umpire Check

Umpire Check assays are used to evaluate the reproducibility of a Project's primary laboratory procedures and results. An umpire lab is used to analyze the samples and cross-check the results, like a duplicate quality control procedure. As mentioned, the Primary Laboratory was ALS until, in January 2023, Bravo decided to use SGS as the primary laboratory.

GE21 has produced control charts to evaluate the reproducibility of PGM and Au assays. The control charts were generated only for the check assays with original and duplicate grades above 3x the DL of the analytical methods evaluated. This measure is taken to remove analytical artifacts that occur near the DL. Palladium and platinum presented the best results, with practically 90% of the sample data below the 15% HARD limit.

Rhodium assays resulted in nearly 60% of samples below the 15% HARD limit. Gold check assays present the most dispersion when compared to the PGMs. This element has presented approximately 80% of the sample pairs below the 15% HARD limit.

11.5 Validation of DOCEGEO (Vale) Diamond Drilling Data

Part of Bravo`s mineral exploration campaign is aimed to check the historical Vale DD campaign. Logging, sampling, sample preparation and analysis procedures were the same as the ones employed at Bravo's infill drilling campaign.

GE21 Consultoria Mineral

11.5.1 *Twin holes*

Historical drilling is checked through relogging and resampling historical drill holes, and with 8 twin drill holes executed by Bravo (Table 11-7).

Table 11-7: Historical drill holes and their respective twin drill holes executed by Bravo

Historical					Bravo			
HOLE-ID	EASTING	NORTHING	RL		HOLE-ID	EASTING	NORTHING	RL
PPT-LUAN-FD0145	658,495.3	9,340,827.8	243.2	Vs.	DDH22LU006	658,495.8	9,340,828.1	243.0
PPT-LUAN-FD0069	659,092.5	9,341,001.7	241.3	Vs.	DDH22LU007	659,092.9	9,341,002.1	241.2
PPT-LUAN-FD0220	659,997.3	9,341,771.9	276.4	Vs.	DDH22LU026	659,998.8	9,341,772.0	254.7
PPT-LUAN-FD0136	659,950.6	9,341,976.3	290.3	Vs.	DDH22LU043	659,950.7	9,341,976.0	268.5
PPT-LUAN-FD0221	659,954.0	9,341,774.6	268.7	Vs.	DDH22LU081	659,954.1	9,341,775.1	247.5
PPT-LUAN-FD0095	659,603.1	9,342,861.4	288.4	Vs.	DDH22LU083	659,602.8	9,342,861.0	289.3
PPT-LUAN-FD0036	657149.3	9,339,723.8	272.1	Vs.	DDH22LU001	657,148.3	9,339,726.1	272.0
PPT-LUAN-FD0173	659,446.0	9,343,565.3	226.0	Vs.	DDH22LU113	659,446.0	9,343,564.9	225.9

Source: GE21, 2025.

The visual analysis of the twin pairs shows a good correlation of the PGMs, gold, and nickel grades between the holes. The spatial correlation observed indicates that the data acquired by Vale is of sufficient quality to characterize the deposit and its geochemical features.

11.5.2 *Resampling - Vale Samples*

Bravo previously relogged and resampled the historical DD holes. After the geological/structural description, the photograph of core boxes and magnetic susceptibility measurements, the geologist prepares the sampling plan respecting lithological contacts, weathering profile, drill core diameter and drilling recovery. Relogging and resampling activities follow Bravo's Diamond Drilling campaign's same operational procedures and QA/QC protocols. The historical drill core sample data and Bravo's drill core resampling of the historical core show an expected positive correlation for the PGM assessed. Correlated assay data shows spreading due to the difference in preparation and analytical laboratory methods.

The resampling campaign undertaken by Bravo aimed to validate the Vale database. Due to the big contribution of Vale samples in the database, testing the reproducibility and the precision from the Vale ownership period is of material importance. It is important to note that the resampled Vale samples have their grades updated to the reanalyzed grades in the database. Resampling is carried out on half of the stored Vale half-drill cores, resulting in a ¼ drill core new sample.

GE21 has evaluated the assay results of resampled drill cores by generating control charts for Pt, Pd, Rh, Au and Ni. The control charts were generated only for the samples with original and reanalyzed grades above 3x the DL of the analytical methods evaluated. Due to the difference in analytical methods used for Nickel readings, a strong bias is detected in the Ni charts.

The Resampling data of ALS and SGS were evaluated separately. ALS data shows a good correlation between original and reanalyzed grades. Pd and Pt resampled pairs present more than 70% of the data below the 40% HARD limit. Rhodium data is 60% below the same HARD limit, while Au data is 55% below. The SGS reanalysis data accounts only for Pt, Pd and

Rh. More than 70% of the Palladium and Rhodium sample pairs are below the 40% HARD limit. In comparison, Platinum has over 55% of samples below the limit.

11.5.3 Correlation Between Vale and Bravo Grades

GE21, at the request of Bravo, has conducted a correlation study between the grades analyzed by Bravo and Vale. This procedure aims to correct some discrepancies between the Vale and Bravo databases by transforming the Vale-analyzed grades using a linear regression equation. Linear regressions were generated using the Resampling data for Au, Pd, Pt, Rh and Ni.

For the PGMs and Gold, the grade shell modelled by Bravo was used as a spatial constraint for the regression data. Only the resampled pairs with both grades above 0.03 ppm were used for the regression. This grade limit corresponds to 3x the DL of the applied assay methods and was used to reduce the impact of analytical noise near the DL. Figure 11-4 to Figure 11-7 presents the correlation for Au, Pd, Pt and Rh after the transformation of Vale data.

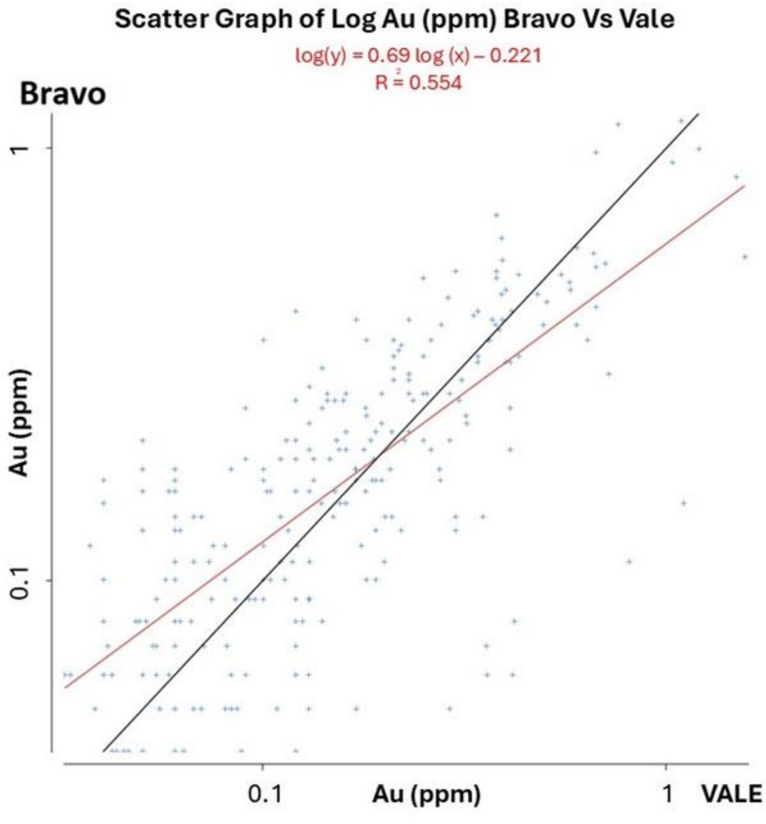


Figure 11-4: Chart correlation of Au assays from Bravo x Vale samples

Source: GE21, 2025.

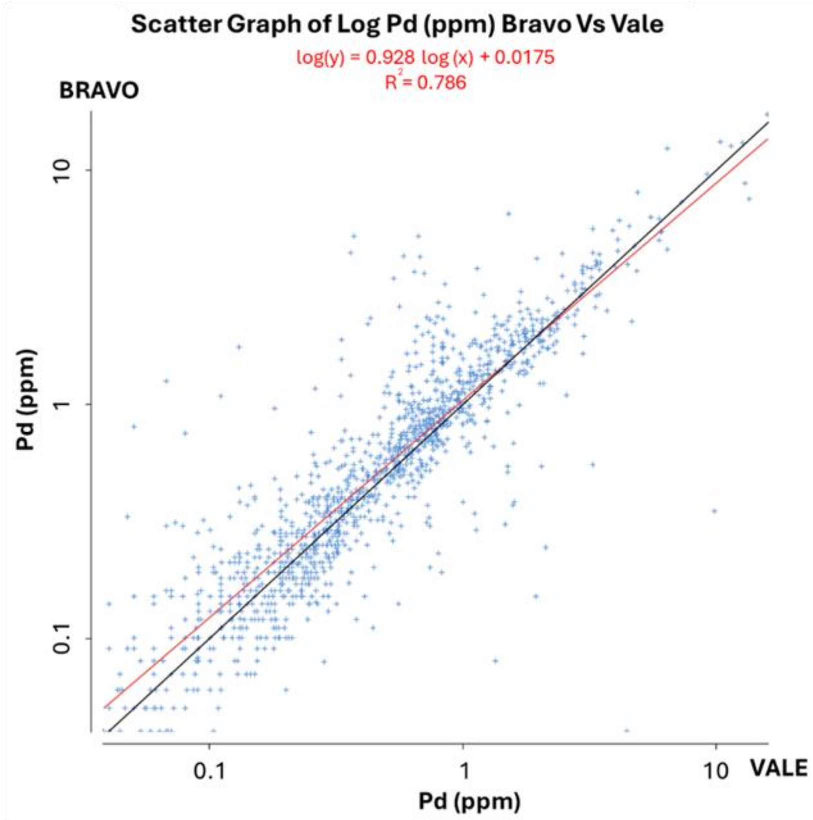


Figure 11-5: Chart correlation of Pd assays from Bravo x Vale samples

Source: GE21, 2025.

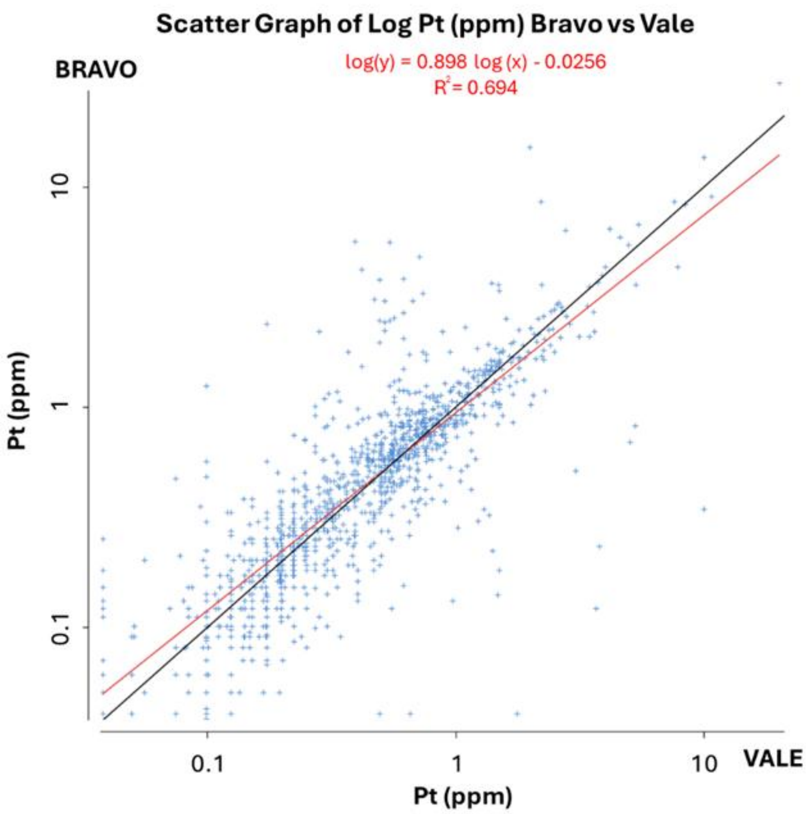


Figure 11-6: Chart correlation Pt samples Bravo x Vale

Source: GE21, 2025.

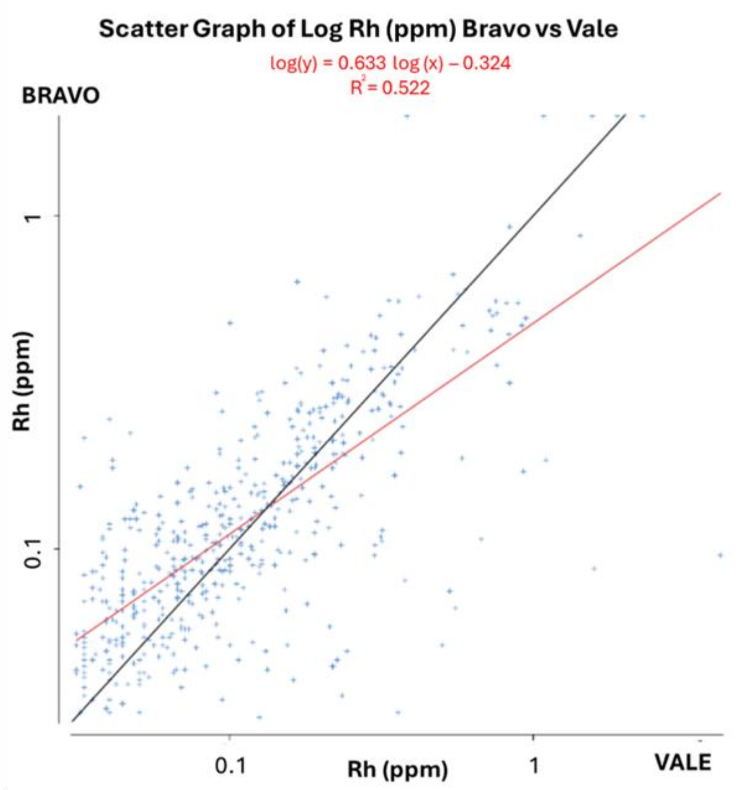


Figure 11-7: Chart correlation Rh samples Bravo x Vale

Source: GE21, 2025.

As mentioned, the Nickel grades analyzed by Vale account for total Nickel: silicate Ni, oxide Ni and sulphide Ni. In contrast, the Bravo grades correspond to the sulphide Ni only. For this reason, additional steps were taken to ensure a valid correlation between the Vale Ni and Bravo Ni grades. Those steps were:

Data was split into two subsets, one above and one below the 1:1 line. The subset below the line was named Negative Bias, and the one above was called Positive Bias.

- Linear regressions were applied in both the subsets.
- The Negative Bias regression was adjusted to make the Angular Coefficient equal to 1. This adjustment was made to reflect a fixed (constant) proportion between the total Ni and the sulphide Ni.
- Data below 100 ppm Ni was removed from the datasets. This grade corresponds to the higher DL used in the resampled pairs.
- The data constrained between the Positive Bias and the adjusted Negative Bias lines was used for a third linear regression. This regression was used to correlate the Vale and Bravo grades.

Figure 11-8 present the final linear regression for Ni and the Ni grade distribution, after the transformation of Vale data.

The Table 11-8 presents a summary of the transformations applied to the Vale grades.

GE21 Consultoria Mineral

Table 11-8: Summary of the transformations applied to the Vale grades

Element	Correlation Equation	R2
Au	[Au Bravo] = 10^(0.690*(Log10 [Au Vale]) - 0.2210)	0.554
Pd	[Pd Bravo] = 10^(0.928*(Log10 [Pd Vale]) - 0.0175)	0.786
Pt	[Pt Bravo] = 10^(0.898*(Log10 [Pt Vale]) - 0.0256)	0.694
Rh	[Rh Bravo] = 10^(0.633*(Log10 [Rh Vale]) - 0.3240)	0.522
Ni	[Ni Bravo] = 10^(1.050*(Log10 [Ni Vale]) - 0.2280)	0.941

Source: GE21, 2025.

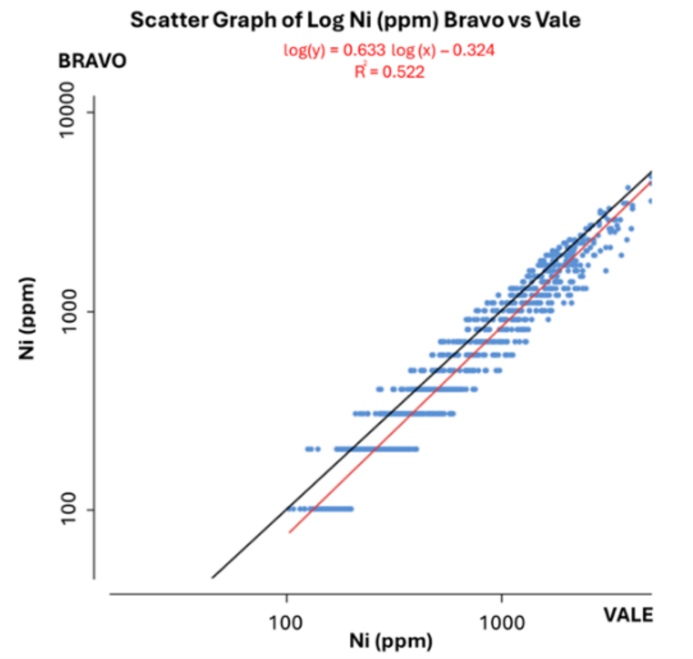


Figure 11-8: Chart correlation Ni samples Bravo x Vale

Source: GE21, 2025.

11.6 QP Opinion

Although the Vale database was historical in nature, the validation and correlation procedures applied and the results obtained, enable the QP of this Report to consider this database to be valid for estimation works. It is important to notice that the Vale QA/QC program did not include any CRM insertion. GE21 and Bravo have, unsuccessfully, tried to obtain the internal QA/QC results from the laboratory used by Vale (SGS).

QA/QC procedures, sampling methodology, and analytical methods applied by Bravo are within the industry's best practices standard. The QP responsible for this report, considering the data presented in this section, is of the opinion that the Project's Database is suited for Mineral Resource Estimation work.

Recommendations:

- Acquisition of Certified Reference Materials that are certified for sulphide Ni assay methods.
- Production of CRMs using materials from the Luanga mineralization.
- Implementation of Coarse/Crusher Duplicates and Fine/Pulverized Duplicates.

12 DATA VERIFICATION

GE21 team members have conducted several field visits since 2022 at Luanga to verify the company's infrastructure, the procedures in the course, and the results obtained from the activities that are carried out by Bravo staff.

Engineer Porfirio Rodriguez is an independent consultant and has conducted field visits to the project in 2023 and 2025. Mr. Rodriguez was accompanied by Bravo personnel in the development of the resource estimate activities.

Three site visits were conducted in the period from July 4 to 7, 2023, October 3 to 6, 2023, and January 27 to January 31, 2025. In the last two visits, GE21's QPs Team was composed by Geologist Bernardo Viana and Mr. Rodriguez.

12.1 Site Visit

The site visits included the review of the QA/QC program, field tours (Figure 12-1), exploration of the core shed, drilling in progress, review of density procedures, and discussions of the current geological interpretations with geologists of Bravo.

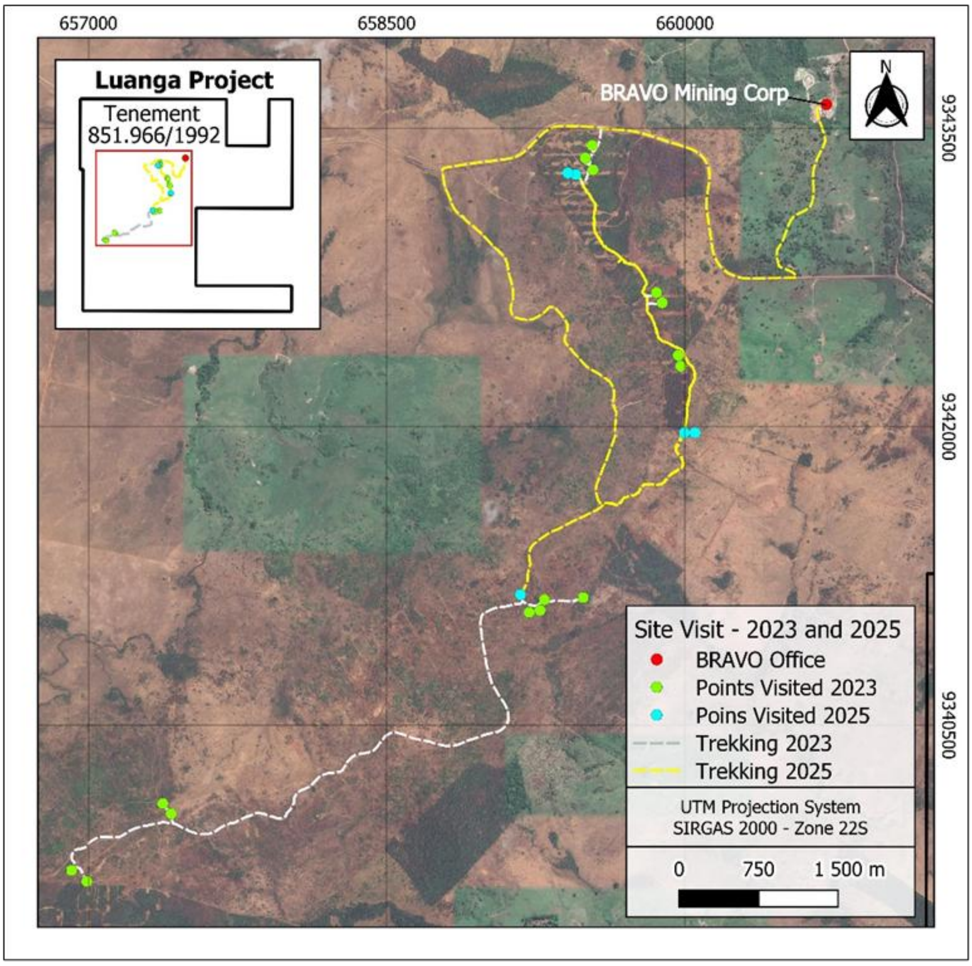


Figure 12-1: Points visited during January 2025

Source: GE21, 2025.

12.1.1 Density Test Laboratory

GE21 visited the Bravo internal density laboratory, where they observed the installation and equipment used for density measurement (Figure 12-2).

Discussions held with on-site geologists allowed to confirm procedures were adequately applied. More details about Bravo procedures are available in Sections 10 and 11 of this report.

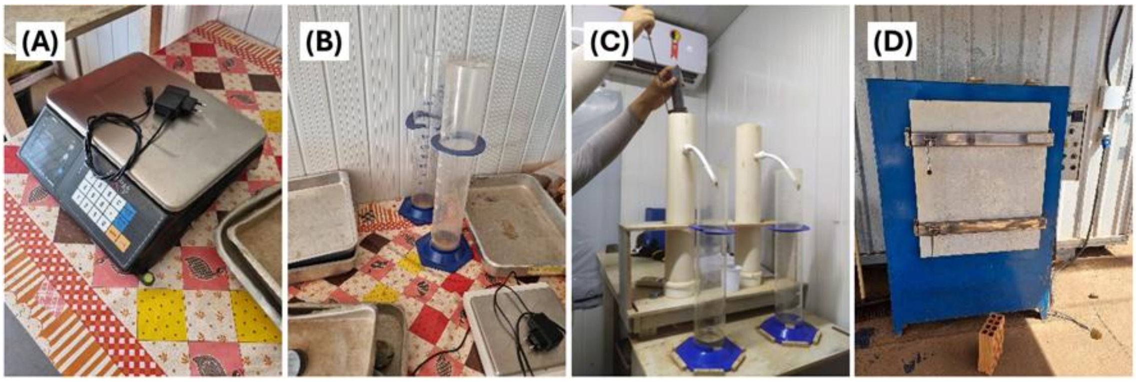


Figure 12-2: Density test equipment

Legend: A) Electronic Weight Scale. B) Test tubes and trays used in density tests. C) Water displacement method for volume calculation. Equipment for determining density by volume displacement. D) Drying Oven.
Source: GE21, 2023.

12.1.2 Drill Hole Location

GE21 verified drill hole collars in the field, checked with handheld GPS units, and compared them to the exploration database (Figure 12-3). GE21 visited the location and collar marks of:

- Metallurgic drill holes 901, 902, and 905.
- Drill holes 086, 249, 001, 007, 044, 047 and 189.
- Trench 004, 022, 024 and 037.

GE21 Consultoria Mineral

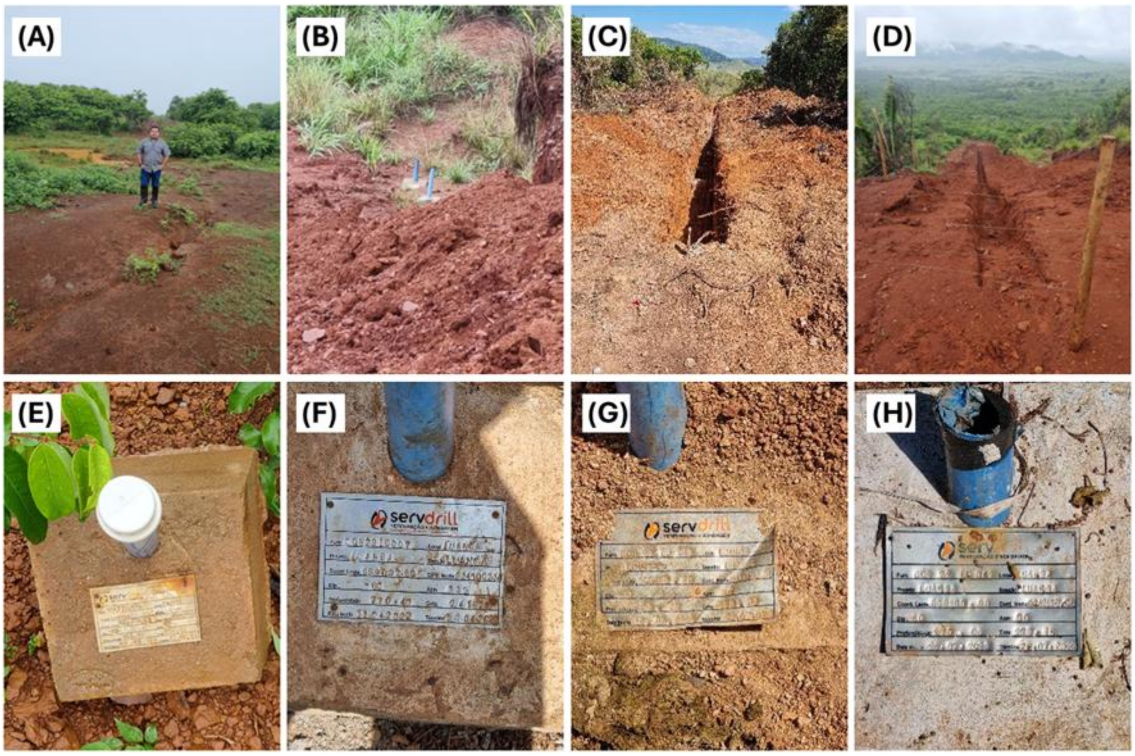


Figure 12-3: Drill hole location evidence

Legend: A) Mr. Viana ion the Field Visit B) Drilling Collar Marks. C) Trench 004. D) Trench TRC24LU037. E) Drill hole DDH22LU086 Mark. F) Drill Hole DDH22LU007 Mark. G) Drill Hole DDH22LU044 Mark. H) Drill hole DDH22LU047 Mark.
Source: GE21, 2023 and 2025.

Drill hole collars have an identification physical marker. The markers are comprised of a concrete pad with a metal plate designating the drilling contractor, drill hole number, drilling area, orientation, coordinate location, start and end date drilled, and total depth. A PVC pipe protruding from the marker provides a physical record of the drill hole orientation.

12.1.3 Core Shed

All core boxes were labelled and properly stored. Sample tags were present in the boxes, and it was possible to validate sample numbers and confirm the presence of mineralization in witness half-core samples from the mineralized zones.

During the core shed visit, the Bravo team explained in detail the entire path of the drill core, from the drill rig to the logging and sampling facility and finally to the laboratory (Figure 12-5).

GE21 visited the core shed (Figure 12-4) and did a visual inspection of the historical Vale core. The drill holes were previously selected by the GE21 team, to review sections of the mineralized core (Figure 12-6). The mineralization observations agree favourably in the extent and type of mineralization logged in the exploration database.

Figure 12-4: Core Shed installation during 1st visit, and after complete construction on 2nd visit

Legend: A) Front view of the Core Shed. B) Lateral view of the Core Shed. C) Inside the Core Shed. D) Vale Core Shed. E) Core box storage area. F) Drill core description and sampling shed.
Source: GE21, 2023 and 2025.

GE21
Consultoria Mineral

Figure 12-5: Sampling and QA/QC procedures

Legend: A) Labelled Drill core samples. B) Labelled Drill core samples batches. C) Standards for QA/QC program. D) Coarse Blank for QA/QC program.
Source: GE21, 2023.

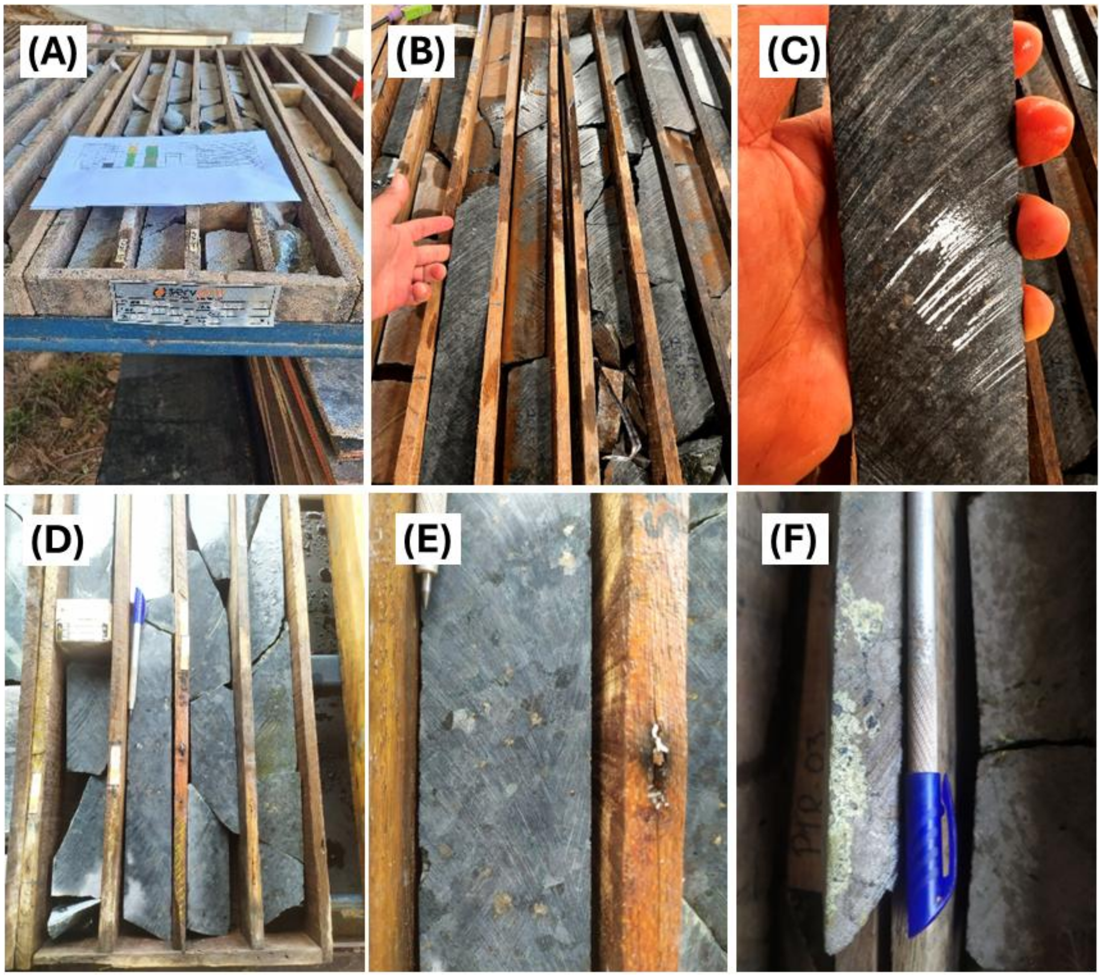


Figure 12-6: Checking drill holes

Legend: A) Personal checking on core description. B) Core box of the drill hole DDH23LU162. C) Disseminated sulphide in the drill hole DDH23LU162. D) Core box of the drill hole DDH23LU162. E) High sulphide PGM mineralization in hole DDH22LU083. F) Massive sulphide range in hole DDH22LU047 with high Ni (pentlandite) and Cu (chalcopyrite) grades. Source: GE21, 2023.

12.1.4 Witness Samples

During the visit on the first period, in 2023, Mr. Porfirio Cabaleiro collected a sample selected among the previous drill holes checked, to configure as a witness sample.

The sample on consideration was:

- Drill hole: DDH22LU007
- Interval: from 128.58 m to 128.96 m
- Sample BPGM-101433)
  - Ni: 0.31%
  - PGM + Au: 6.60 ppm

GE21 independently submitted the sample to a chemical analysis in the certified laboratory SGS Geosol in Vespasiano, receiving the result presented in Figure 12-7.

SGS GEOSOL LABORATÓRIOS LTDA.

CERTIFICADO DE ANÁLISES
GQ2308925

INFORMAÇÕES DO CLIENTE

NOME: BRAVO MINERACAO LTDA
ATTN.: Paulo Brito
ENDEREÇO: EST ESTRADA DA VILA ALTO BONITO S/N ZONA RURAL 68523000 PA CURIONOPOLIS
CPF/CNPJ: 37.262.942/0002-39

REFERÊNCIA DO LOTE DE AMOSTRAS

REF. CLIENTE: ACES-BRAVO-0050-23
QTE. AMOSTRAS: 1
RECEBIDO: 11/07/2023
PRODUTO: TESTEMUNHO
COMPLETADO: 07/08/2023
PROJETO: Luanga
EMITIDO: 07/08/2023

REFERÊNCIA ANALITICA

AA

AAS04B: Lixiviação com Citrato De Amônio e Peróxido de Hidrogênio por 2 horas.

FA

FAI30V_RH: Determinação de Ródio por Fire Assay - ICPOES - Fusão 30 g
FAI515: Determinação de Au, Pt e Pd por Fire Assay - ICP - 50g

PREP

DRY105: Secagem de amostras à 105°C
PREPQC: Controle de Qualidade - Preparação Física
PRPCLI: Preparação Física conforme definição do cliente

LEGENDA: SIGLAS

L.D. = Limite de Detecção
L.N.R. = Listado e não Recebido
L.N.F. = Não reportado devido a interferentes
BLK = Branco
I.S. = Amostra Insuficiente
REP = Replicata
N.A. = Não Analisado
OVR = Não Analisado devido ao alto teor
DUP = Duplicata
STD = Padrão

SGS GEOSOL LABORATÓRIOS LTDA.

CERTIFICADO DE ANÁLISES
GQ2308925

ANÁLISES	Peso_bruto	Ni	Cu	Au	Rh	Pd	Pt
MÉTODO	PRPCLI	AAS04B	AAS04B	FAI515	FAI30V_RH	FAI515	FAI515
UNIDADE	G	%	%	PPB	PPM	PPB	PPB
LIMITE DETECÇÃO	0,00	0,01	0,01	5	0,002	5	5
BRANCO_PREP	N.A.	<0,01	<0,01	<5	<0,002	<5	<5
BPGM-101433	1011,00	0,26	0,02	90	0,572	5039	3387
* REP BPGM-101433		0,25	0,02				
* STD HO_AM_01		0,49	0,29				
* STD AMIS0323				145		647	1083
* STD AMIS0388				8		164	356

Figure 12-7: Analysis certificate of 2023 witness sample

Source: SGS, 2023.

During the visit in January 2025, GE21 QPs Mrs. Bernardo Viana and Porfirio Cabaleiro collected three sample reserves, selected from previously sampled trenches, and sent for chemical analysis as witness samples. Table 12-1 presents the selected samples and the original results, retrieved from the Project Database. Figure 12-8 presents a graphical comparison between original grades and witness sample grades, and Figure 12-9 presents the laboratory certificate of analysis for the witness samples. GE21 concludes there is a good correlation between results on all witness samples.

GE21 Consultoria Mineral

Table 12-1: Original Witness Samples Results

Trench ID	Sample ID	Interval		Au	Pd	Pt	Rh
		From	To	ppb			ppm
TRC24LU027	BRAV-704468	124.15	125.15	22	643	203	0.044
	BRAV-704476	131.15	132.15	46	827	438	0.097
	BRAV-704491	144.05	145.05	162	5102	2958	0.749

Source: GE21, 2025.

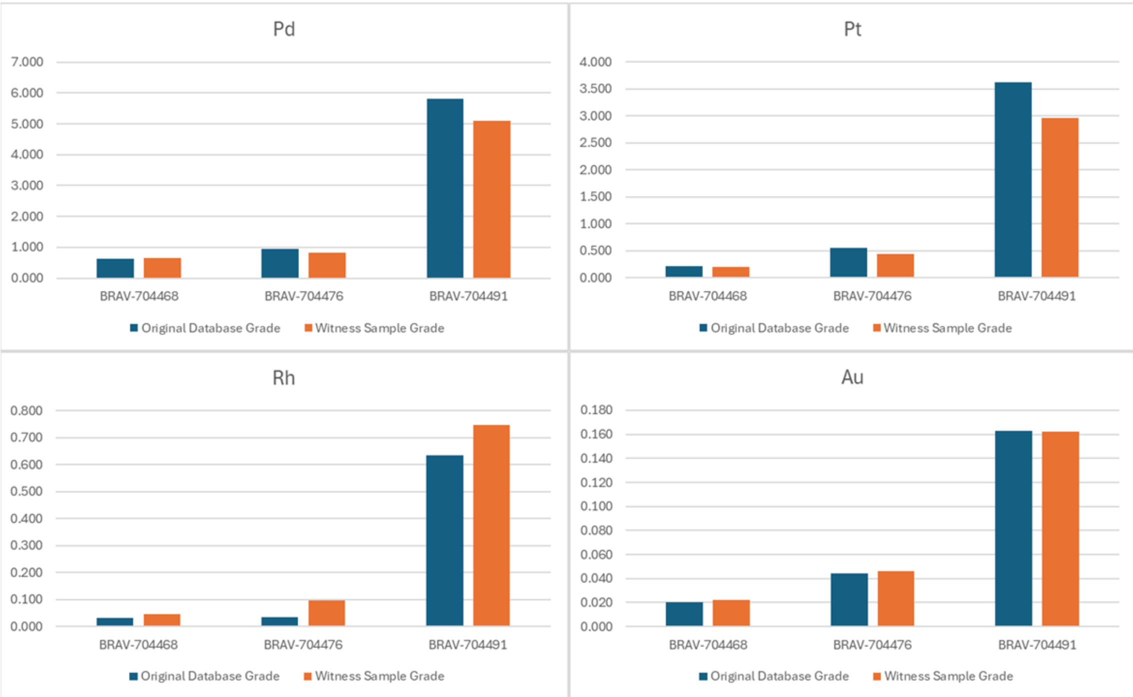


Figure 12-8: Comparison between original database grades and witness sample grades

Source: GE21, 2025.

SGS GEOSOL

SGS GEOSOL LABORATÓRIOS LTDA.

CERTIFICADO DE ANÁLISES

GQ2501058

INFORMAÇÕES DO CLIENTE

NOME: BRAVO MINERACAO LTDA
ATTN.: Paulo Brito
ENDEREÇO: ESTRADA ESTRADA DA VILA ALTO BONITO S/N ZONA RURAL 68523000 PA CURIONOPOLIS
CPF/CNPJ: 37.262.942/0002-39

REFERÊNCIA DO LOTE DE AMOSTRAS

REF. CLIENTE: ACES-BRAVO-0248-25
QTE. AMOSTRAS: 3
RECEBIDO: 04/02/2025
PRODUTO: POLPAS
COMPLETADO: 10/02/2025
PROJETO: Luanga
EMITIDO: 10/02/2025

REFERÊNCIA ANALITICA

FA

FAI30V_RH: Determinação de Ródio por Fire Assay - ICPOES - Fusão 30 g
FAI515: Determinação de Au, Pt e Pd por Fire Assay - ICP - 50g

LEGENDA: SIGLAS

L.D. = Limite de Detecção	BLK = Branco	REP = Replicata	DUP = Duplicata
L.N.R. = Listado e não Recebido	I.S. = Amostra Insuficiente	N.A. = Não Analisado	STD = Padrão
I.N.F. = Não reportado devido a interferentes		OVR = Não Analisado devido ao alto teor	

Marcos Filipe Gonçalves Silva
CRQ II 02202046
Responsável Técnico

Os ensaios foram realizados na SGS GEOSOL Laboratórios Ltda. - Rodovia MG 010, Km 24,5 - Bairro Angicos - Vespasiano - MG - Brasil - CEP: 33.206-240
Telefone +55 31 3045-0261 www.sgsgeosol.com.br
Certificados ISO 9001:2015 e ISO 14001:2015 (ABS 32982 e ABS 39911)
Os resultados expressos neste Certificado se referem somente ao material recebido. Proibida a reprodução parcial deste documento.

Relatório impresso em:10/02/2025 16:00:49 Página 1 de 2

CERTIFICADO DE ANÁLISES

GQ2501058

ANÁLISES	Au	Rh	Pd	Pt
MÉTODO	FAI515	FAI30V_RH	FAI515	FAI515
UNIDADE	ppb	ppm	ppb	ppb
LIMITE DETECÇÃO	5	0,002	5	5
BRAV-704468	20	0,032	623	210
BRAV-704476	44	0,035	956	550
BRAV-704491	163	0,636	5813	3626
* REP BRAV-704468		0,036		
* REP BRAV-704468	17		578	183
* STD AMIS0769		0,063		
* STD GPP-04	81		92	86

Figure 12-9: January 2025 witness samples laboratory certificate

Source: SGS, 2025.

12.1.5 QP Opinion

GE21 is of the opinion that the exploration data is adequate for use in the Mineral Resource estimate. What follows below are some observations that were recorded by GE21 personnel during visits as it relates to the generation, collection, control and storage of exploration data on-site at Luanga:

- Drill hole Logging: This task was considered as standard industry practice logging procedures, which has been standardized by Bravo. GE21 performed a review of logging procedures for randomly selected drill cores and verified the completeness of the logs. Considering all the evaluated content, Bravo has demonstrated that it understands the geology, and some located omissions are not considered significant.
- Database: recent data is stored in a standard commercial database. Historical Vale records are well-managed and have been migrated to the Bravo database. Data storage procedures at Bravo are considered within standard industry practice. As part of the validation process, GE21 verified 8 holes. Database validation was conducted with the help of Bravo staff according to standard validation procedures including review of collar locations, drill hole deviations and database check-assay review. No inconsistencies were found in the database.
- Density: A large database of density information has been collected during the exploration phase. The process whereby density data is obtained is considered within standard industry practice.
- Witness sample: Random samples were collected during the site visits; the results obtained from a certified laboratory were consistent with the original sample results recorded on the database.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

This section reports on the results of preliminary metallurgical test work on PGM+Au+Ni mineralization from the Project and records recommendations on metallurgical input parameters considered for the MRE and assumptions for the preliminary economic analysis in the PEA. The scope of test work completed to date includes:

- Historical test work completed for Vale (see section 13.2).
- • Extensive comminution and flotation test work on fresh (sulphide) rock samples conducted for Bravo from 2022 to 2025 including work to review and validate historical work conducted by Vale between 2001 and 2004.
- • Several test programs conducted on oxide material including exploratory leaching and gravity separation tests inclusive of parameter optimization tests conducted for Bravo between 2022 and 2025.
- • Preliminary tests to evaluate the downstream refining of Luanga concentrates.

13.2 Review of Historical Metallurgy Work

Historical Metallurgical testing on the Luanga material had been initiated at various stages of its development with the bulk of the work having been completed between 2002 and 2004. Test work was completed at bench scale and pilot plant scale on core samples from diamond drilling. The studies completed include:

- 2001/2002 – CABRI: Mineralogical Characterization Study.
- 2002 – MINTEK: Flotation Studies and Mineralogical Characterization Study.
- 2003/2004 – LAKEFIELD: Flotation Studies and Mineralogical Characterization Study.
- 2003 – HDK ENGENHARIA: Preliminary Milling Circuit Sizing Study.
- 2002/2004 – AVEC: Evaluation of the Global PGM Market.

Historical metallurgical efforts reported for Luanga were summarized in Vale's final report as follows:

"The studies of mineral processing carried out with samples of fresh sulphide mineralization have indicated that the traditional flowsheet, involving crushing/grinding and flotation, produces a concentrate with commercially accepted grades [80-154g/t] and PGM recoveries in the order of 74%. The closed-circuit (Lock Cycle Test or LCT) study obtained a (single) bulk concentrate for Luanga comprising PGM + Ni + Au".

13.2.1 MINTEK Studies

In 2002, a sulphide mineralization sample of 100kg was sent to Mintek in South Africa for the first characterization and concentration studies. The initial sample reported a feed grade of

4.8g/t PGM+Au and the mineralogical analysis showed that the main PGM-bearing minerals were Sperrylite, PGM-Bithmotellurides, and Stibiopaladinite.

Mintek defined the standard mill-float-mill-float (MF2) flowsheet as likely appropriate for preliminary tests. Primary milling was performed to p60 -75µm, followed by rougher, cleaner, and recleaner flotation stages in a “course” circuit. The rougher tailings, together with the cleaner tailings, were then reground to p80 -38µm followed by secondary rougher, cleaner and recleaner, in the “fines” circuit.

The reagents used in the flotation were NaSH (sulfidizing agent), Cu504 (activator), Depramin 267 (silicate depressant), SIBX (collector) and Dow 200 (frother). The Mintek flowchart is shown on Figure 13-1.

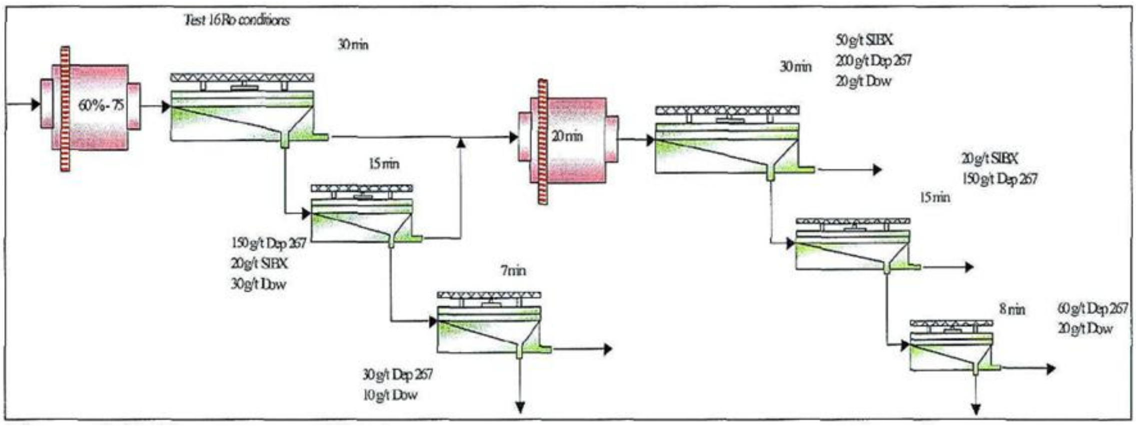


Figure 13-1: Mintek flowchart

Source: Bravo, 2023.

The final concentrate from Mintek’s open-circuit tests reported a mass pull of 3.8%, concentrate grade of 87.2 g/t PGM + Au and recovery of 70.6%. Two locked cycle tests were also carried out at Mintek and produced a concentrate content of ca. 150 g/t PGM + Au with 75% recovery.

Mintek’s grinding investigation concluded that the recovery of PGM from Luanga material into a flotation concentrate is much dependent on grind size with results demonstrating an improvement from 52% to 72% in PGM recovery by grinding finer from 40 to 60% passing 75um.

The mineralogical investigation concluded that the PGM’s occur in various modes, associated with base metal sulphides (inclusions and attached at boundaries), at silicate boundaries, and as inclusions within silicates.

13.2.2 CDM (Vale) Studies

CDM performed tests on 4 fresh sulphide samples with lower PGM+Au grades, S1 with 1.07 g/t, S2 with 2,03 g/t, S3 with 2.67 g/t and S4 with 4.14 g/t head grade.

The flotation testing in open circuit and using the MINTEK process with two stages and high reagent additions produced concentrates with grades varying between 20 and 50 g/t PGM’s for samples S-1 to S-3, with recoveries 70.4 % < 72.3 %. The S-4 composite delivered an 83 g/t concentrate with 75.4 % recovery (Table 13-1).

Table 13-1: Feed sample analysis and concentrate qualitative analysis from CDM studies

Chemical analysis samples S1 to S4 - PGM & Au (g/t)					
	Pd	Pt	Rh	Au	PGM + Au
INITIAL SAMPLE S1	0.57	0.45	<0.05	0.05	1.70
INITIAL SAMPLE S2	1.05	0.81	0.09	0.08	2.03
INITIAL SAMPLE S3	1.50	0.97	0.10	0.10	2.67
INITIAL SAMPLE S4	2.34	1.43	0.22	0.15	4.14

Chemical analysis samples S1 to S4 - Major elements (%)							
	SiO2 (%)	Mg (%)	Fe (%)	Al2O3 (%)	Ni (%)	Cr (%)	S (%)
INITIAL SAMPLE S1	47.10	14.80	7.73	4.29	0.17	0.28	0.40
INITIAL SAMPLE S2	47.00	15.30	7.00	3.86	0.19	0.29	0.34
INITIAL SAMPLE S3	47.40	15.10	7.87	4.08	0.26	0.28	0.50
INITIAL SAMPLE S4	46.50	14.60	8.67	3.37	0.31	0.31	1.18

Chemical analysis samples S1 to S4 - other metals					
	Co (%)	Cu (%)	Sb (ppm)	As (ppm)	Ag (ppm)
INITIAL SAMPLE S1	0.01	0.02	6	1	<1
INITIAL SAMPLE S2	0.01	0.02	5	2	<1
INITIAL SAMPLE S3	0.02	0.02	5	2	<1
INITIAL SAMPLE S4	0.02	0.04	5	2	<1

Feed grades PGM, Au, Ni, Co &Cu							
Pd (g/t)	Pt (g/t)	Rh (g/t)	Au (g/t)	PGM + Au (g/t)	Ni %	Co %	Cu %
1.02	0.66	<0.1	0.09	1.77	0.19	0.01	0.02

Final concentrate grades PGM, Au Ni, Co & Cu							
Pd (g/t)	**Pt (g/t)**	**Rh (g/t)**	**Au (g/t)**	**PGM + Au (g/t)**	**Ni (%)**	**Co (%)**	**Cu (%)**
54.3	42.9	3.8	3.6	105	7.5	0.35	1.66

Chemical recovery (%)					
Pd	**Pt**	**Au**	**PGM + Au**	**Ni**	**Co**
71	79	47	73	44.6	30

Source: Bravo, 2023.

CDM (Vale) increased concentrate grades by introducing additional cleaning stages – 3 in total. The tests S-1 to S-3 produced concentrates with 50 to 97 g/t PGMs with recoveries between 56 and 64%.

CDM performed two locked cycle tests (LCT). LCT1 was performed on a sample blend of S1 to S3 grading 1.77 g/t and consisted of a circuit with two stages of grinding and flotation, with a finer grind at p90 -38 µm. Rougher stages were followed by three cleaner stages, with a total of 8 flotation stages. The concentrate produced in LCT1 after 10 cycles graded 104 g/t PGM+Au with 73% recovery at a mass pull of 1.2% (Table 13-2).

LCT2 used a simpler flotation scheme, a rougher stage fed with mineralization milled to a p90 -38 µm, with concentrates cleaned in two stages. The cleaner tails fed a cleaner scavenger stage. Scavenger cleaner tails were recycled to rougher feed. Recleaner tails and cleaner scavenger concentrate returned to the cleaner feed, joining the rougher concentrate.

Table 13-2: Summary of results in LCT1 and LCT2

Element	Unit	LCT 1 grade	LCT 2 grade	Feed grade
Pd	ppm	54.3	72.8	1.02
Pt	ppm	42.9	52.1	0.66
Rh	ppm	3.8	3.8	< 0.1
Au	ppm	3.6	5.34	0.09
PGM + Au	ppm	104.6	135.03	1.77
Ag	ppm	4	< 1	< 1
Al2O3	%	1.08	0.51	4.07
As	ppm	< 1	< 1	< 1
Ca	%	0.59	0.3	2.32
Co	%	0.35	0.45	0.01
Cr	%	0.35	0.39	0.29
Cu	%	1.66	2.59	0.02
F	ppm	< 100	< 100	122
Fe	%	33.3	39.7	7.86
Mg	%	5.3	2.13	15.4
Mn	%	0.06	0.05	0.14
Ni	%	7.5	9.8	0.19
S	%	21.3	29.4	0.47
Sb	ppm	62	119	7
SiO2	%	18.7	7.85	51.8
Zn	%	0.03	0.09	0.01
Mass Recovery		1.20%	0.89%	
Metallurgical Recovery		73%	69,36%	

Source: Bravo, 2023.

CDM did 12 cycles for circuit stabilization in LCT2, resulting in a mass pull of 0.9%, concentrate grade of 134 g/t PGM+Au and 69.4% recovery.

The nickel recoveries in these tests were 46-48%. The presence of some nickel that is entrained in the crystalline structures of silicates was highlighted by a mineralogical characterization conducted at CDM.

CDM conducted mini-plant tests in February 2004 to generate concentrates for hydrometallurgical refinery tests. A total of 400 kg of sample grading about 1.64 g/t 4PGM was processed and produced a concentrate grade of 124 g/t for a mass pull of 0.8% at a recovery of 63 %.

A 200kg composite sulphide sample was sent to SGS Lakefield in Canada for replication of CDM LCT using a MF1 circuit. The results obtained include a final concentrate mass pull of 0,78% grading 132 g/t PGM+Au and a recovery of 70%. The sample grade was 1,49 g/t PGM+Au.

Vale, through CDM also attempted to test flotation efficacy on oxidized and blended fresh sulphide/oxide mineralization. These tests were generally unsuccessful reporting low recoveries and or poor concentrate quality Sample Selection.

13.3 Bravo Sample Selection

Detailed mineralogy was completed on Luanga samples from various locations by Vale. Sample locations for Bravo's 2022/2023 metallurgical test program were, for the most part, aligned with the location of the aforementioned mineralogical study. This facilitated Bravo's sample preparation, test work configuration, and results allowing interpretation to be directly supported by the historical mineralogical data. The mineralogy study had been completed on drill core samples sourced from Vale's previous drill programs. The same drill hole locations were used for Bravo's metallurgy sampling, with new sample material being generated through twin-holes, HQ, diamond core drilling completed in 2022.

Sample locations extended along the strike length of the mineralized body including the north, central and southwestern Sectors. A bulk oxide/saprolite sample was taken from the central Sector. Sample localities are shown on Figure 13-2.

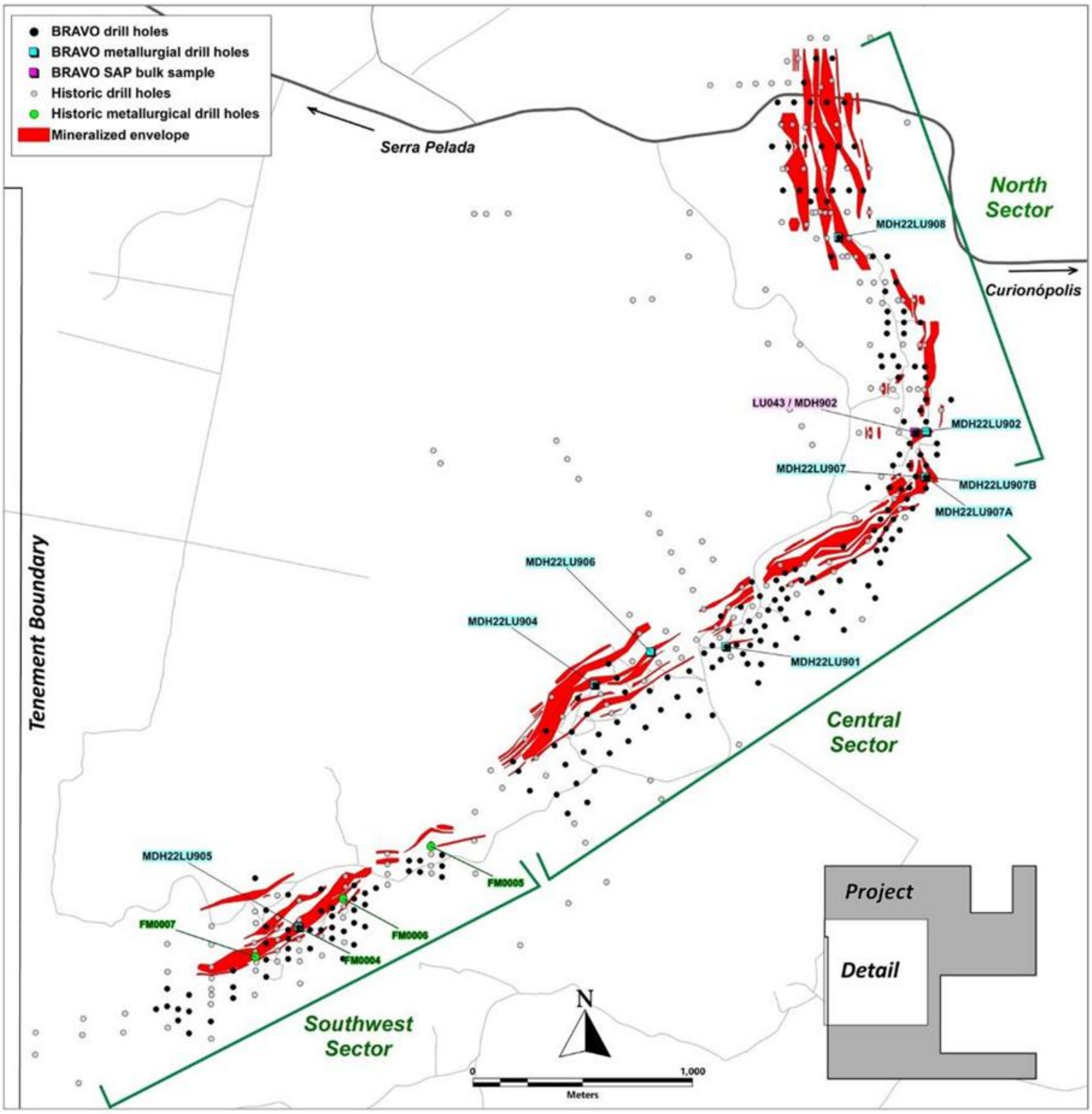


Figure 13-2: Location of metallurgical samples

Source: Bravo, 2023.

13.3.1 Sample Variance Selection

Twin drill core samples for flotation and leaching test programs were selected to be representative of the Luanga deposit. This included fresh sulphide, transition and oxide samples. Grade variance has also been taken into consideration with sample grades ranging 0.5 -11 g/t.

The bulk sample for optimization leaching tests was collected at the northern edge of the Central Sector of the Luanga deposit.

13.4 Bravo Metallurgical Program 2022/2023

At the initiation of the 2022/2023 program, Bravo submitted approximately 3 tonnes of fresh-sulphide metallurgical samples and 150kg of oxide samples to the CETEM and TESTWORK Ltd. laboratories respectively, for metallurgical studies.

13.4.1 Sulphide Material

Standard milling tests were conducted to establish comminution curves and size distribution relative to grinding times.

- Bond Ball Mill Grindability Tests

Two individual BBWi tests were performed on samples of Luanga mineralized material to determine preliminary grinding power requirements. The methodology was consistent with the standard Bond method. The tests were performed on composite blends from two major mineralized zones at Luanga, namely the central and southwest Sectors. One composite of fresh sulphide mineralized material and one sample of transitional mineralized material were submitted for analysis. The preliminary BBWi results indicate a hardness classification of “medium” for both samples (Table 13-3). Table 13-4 below shows sample composite details used on the BBWi tests.

Table 13-3: BBWi results

Test	WI
	kWh/t
1 Fresh	13.94
2 Transition	10.29

Source: Bravo, 2025.

Table 13-4: Sample composite – BBWi test

Sulphide Composite Samples	Drill hole ID	Zone	Sample Depth
600014	MDH22LU901	Central	106-112m
600015	MDH22LU901	Central	112-119m
600030	MDH22LU905	SW	49-55m
600031	MDH22LU905	SW	56-62m

Transition Composite Samples	Drill hole ID	Zone	Sample Depth
600006	MDH22LU904	Central	26-27m
600028	MDH22LU905	SW	29-34m

Source: Bravo, 2025.

13.4.1.1 Flotation Tests

Bravo's Flotation testing progressed through various series of study to establish material behaviour, characterization, equipment and reagent calibration. Testing was guided by historical data, with the earlier objectives of replicating and validating historical work. potential areas of improvement and optimization were also identified.

Flotation tests were carried out considering differing reagents, primary grind size and circuit flotation cell configurations. This included: pre-flotation of talc, fast rougher, staged roughers, rougher-scavenger, cleaner, cleaner-scavengers and recleaners.

A flotation test was also carried out on a sample of oxide mineralization. The test reported low recoveries with excessive mass pull resulting in low-grade concentrates. Its believe that the presence of clays in the oxide sample inhibits flotation selectivity and explains the poor results. No other tests were performed with oxide samples.

13.4.1.2 Preliminary Exploratory and Characterization Flotation Tests (Series T, TD)

Initial tests used coarse grinding, short flotation times, and lower additions of collector and frothing reagents relative to the historical work. Initial tests also used an amount of depressant (CMC) as required to control the amount and tenacity of talc minerals which are known to inhibit good flotation.

Only the rougher stage was tested in the first three tests of the T series. A cleaner stage was added in the final three tests. The rougher circuit was, as a first pass, a fast flotation stage to produce concentrate with higher concentrations. This represented a preference to generate saleable concentrates as early in the circuit as possible. Moreover, historical data indicated fast flotation of coarse-grained PGM and is supported by observations from the historical Mintek tests.

Results and interpretation from tests T-01 and T-02 demonstrated that talc depression should occur pre-flotation and that a finer grind is required to improve liberation. The initial grind was coarse relative to historical work, at p48 -74µm.

Tests T-03 and T-04 were carried out with a higher dosage of PAX collector, increasing from 20 to 60 g/t. A cleaning stage was also introduced. The introduction of a cleaner stage in the T-04 decreased the final concentrate mass pull dropping to ca. 1%. A similar mass recovery was achieved on the T-01 in fast rougher flotation. Test T-05 saw the introduction of a blend of two collectors, Aero 208 and Aero 3894, known for high selectivity in sulphide, PGM and gold ores.

Furthermore, the TD test series demonstrated that testing was progressing with increasing selectivity, replicating rougher results achieved by Mintek but using lower reagent addition rates. In general, the metallurgical performance of the TD series varied between concentrate grades of 28.7 – 444.0 g/t and 17 – 83.2% recovery. Feed grades varied between 1.0 g/t and 8.7 g/t (Table 13-5).

Table 13-5: Summary of the best results on flotation tests

Test	Sample	Analytical Lab	Feed		Concentrate	
			Grade (g/t PGM+Au)	Mass Recovery	Grade (g/t PGM+Au)	Final Recovery
TD-12	600015	QLS	1.0	0.3	209	79.7
TD-08	600016	QLS	2.6	1.9	443.9	50.8
TD-19	600006/28	QLS	3.3	5.9	44.8	81.1
TD-20	600006/28	QLS	4.8	12.4	28.7	74.5

Source: Bravo, 2025.

13.4.1.3 Comparative Flotation Tests (Series TC)

For the comparative tests, three bench scale flotation tests were carried out. This series aimed to compare the Bravo open circuit general arrangement with the historical closed-circuit arrangement. The comminution time for this series of tests was 80 min, targeting 80% passing -30 µm based on milling times and particle sizes obtained with partial compounds that participated in the compound used in this (TC) series. The TC-Bravo circuit comprised a 6-stage rougher, followed by 5-stage cleaner. This configuration is believed to provide for the same outcome as the circuit design applied in the historical Mintek/Vale closed circuit (three roughers followed by, cleaning, recleaning and cleaner scavenger). An Aero 65 frother was trialled but it did not perform as well as MIBC.

The above tests' best results include 82 g/t concentrate grade with 75% recovery (TC-01) and 189 g/t concentrate grade with 78% recovery (TC-02). The Feed grades were 2.6g/t and 4.6 g/t PGM respectively.

The feed material was a blend of samples 600013, 600015, and 600031.

13.4.1.4 Preliminary Process Circuit Tests (Series TB, TH)

The TB test series represented a circuit configuration of 6-stage rougher with combined rougher concentrates reporting to 5-stage cleaner. The TH circuit evaluated the configuration using reagent dosages as per the historical Vale test work (higher CMC, higher PAX, higher MIBC).

In the TH tests, two circuit options were used: a) suitable for samples with very little or no copper sulphide and b) where a first rougher flotation and a first cleaner adequate to generate a copper concentrate with commercial grade of the order of 25% Cu. This was due to sample 600031 demonstrating higher than anticipated Cu values.

TB and TH tests demonstrated increasing recovery values with the TB process, outperforming on recovery with lower reagent consumption.

Flotation tests using various reagent suits applied to samples assaying 4.4 g/t 3PGM+Au showed a weighted average concentrate grade of 138 g/t with recoveries averaging 74.5%. The performance compares well with, and constitutes a slight improvement, over results in the MINTEK and VALE historical work, where a concentrate of around 123 g/t PGM+Au with 73.8% recovery was reported.

The TB test series examined the impact of grind size on rougher recoveries applied a new flowsheet configuration. It was shown that, although a finer grind might be expected to improve recoveries slightly, the improvement was marginal and probably does not justify a p80 finer than -38um.

Previous mineralogy studies by L. Cabri indicated that the equal bimodal PGM grain size distribution was around 15-20µm and 45-50µm, whereas Mintek's study indicated that the majority of PGM are less than 15µm. Mineralogy studies indicate that average grain size likely diminishes with decreasing grade.

The preliminary conclusions from these tests are that the different grade profiles within the Luanga deposit may benefit from differing milling parameters. It is expected that a coarser grind will be required for feed grades above 2g/t and finer grind for lower grade material. Further testing is required to determine the economic trade-off between the lower grade mineralization profiles, relative to recovery and the need for a grind finer than p80 -38µm (Table 13-6).

Table 13-6: Preliminary results of Process Circuit Tests (Series TB, TH)

Test	Sample	Analytical Lab	Feed		Concentrate	
			Grade (g/t PGM+Au)	Mass Recovery	Grade (g/t PGM+Au)	Final Recovery
TH-07B	600026	SGS	0.7	1.5	39	61.3
TB-05	600015	SGS	1.57	1.9	78.2	74.0
TB-04	600030	SGS	3.7	0.67	475.2	75.3
TB-07* Rgh Only	600031	SGS	7.7	8.6	136	74.7
TB-08* Rgh Only	600032	SGS	7.4	11.1	74.1	81.8

Source: Bravo, 2025.

13.4.1.5 Fines Flotation Tests

Historical mineralogy and flotation test work on Luanga mineralization established that a significant component of the PGM grain size distribution is concentrated in the range of 3-15 µm. It is well-known that fine and ultrafine liberated valuable particles are often lost to flotation tailings due to the limitations assocaied with traditional flotation mechanisms and the passivation of partical surfaces by oxidation and slimes. Fine particle recovery is recognized as an area where potential improvement and optimization may be possible by employing different terchnologies.

Extensive research and development covering a diverse range of technologies have been undertaken over the years to find feasible solutions with the most widely adopted, extra-cellular technology being hydrodynamic cavitation devices (HCDs), causing the nucleation of ultrafine nano bubbles (NBs) on the surfaces of fine valuable particles, thus aiding their agglomeration and subsequent recovery by micro-bubbles (MBs) and normal sized (macro) flotation bubbles. The Mach Reactor HCD has found wide adoption, particularly in flotation plants recovering PGM from tailings dams through the demonstration of material improvement in flotation performance from feedstock with low grades (0.3-1g/t) and ultrafine grain size distributions (-5 to -10µm).

HCDs have been demonstrated to increase recovery (+10%) and lower capital and operating costs. The Mach reactor is now a frequent addition to commercial PGM flotation plants in South Africa.

Bravo submitted to Mintek in South Africa a 30kg sample for hydrodynamic cavitation testing using the Mach reactor (Figure 13-3). The reactor consists of an array of venturi nozzles in series or a multitude of parallel, restricted apertures in which intimate contact is achieved between the fine particles and the distribution of very fine bubbles that are formed due to cavitation and the conditions of high shear. This results in a hydrodynamic environment in the pulp zone which contrasts markedly with that observed with conventional flotation equipment.

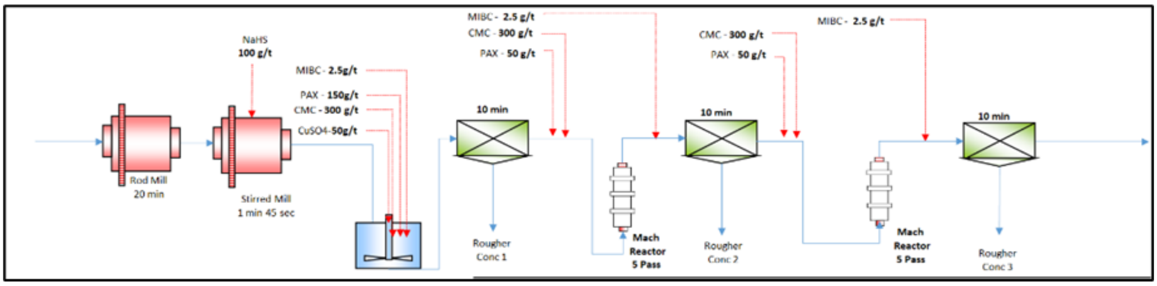


Figure 13-3: Test flowsheet and parameters used by Mintek

Source: Bravo, 2023.

Three control tests and 5 active Mach reactor tests were performed using a sample feed of 2,66 g/t. Test 4 demonstrated a 6.1% improvement in recovery with similar mass pull and concentrate grade metrics for the three stage rougher circuit.

Results below in Table 13-7 demonstrate potential improvements achievable by incorporating the Mach Reactor for Luanga mineralization, particularly lower grade mineralization where fine PGM particles predominate. Further recovery improvements are anticipated at varying grind sizes and with revisiting floatation cleaner stage performance. In commercial plant Mach Reactors are typically installed in the rougher, cleaner and scavenger stages.

Table 13-7: Results from fines flotation tests

Scenario	Description	Mass Pull (%)	4E grade (g/t)	Recovery (%)	Average Recovery (%)	Rec. Improve (%)
1	0 Pass 1A	9.17	22.27	74.74	74.5	-
	0 Pass 1B	10.2	20.9	73.9		
	0 Pass 1C	10.2	20.8	74.8		
2	10 Pass 2A	18.5	10.6	76	75.9	1.4
	10 Pass 2B	19.3	9.9	75.9		
	10 Pass 2C	18.9	9.5	75.8		
3	5 Pass + N CMC dosage	11.8	18.3	78.8	78.8	4.3
4	2 x 10 Pass + N CMC dosage	11.5	19	80.5	80.5	6.1

Source: Bravo, 2023.

13.4.1.6 Associated Base Metals

Historical Luanga test work had almost exclusively focused on the recovery performance of the platinum group metals. Generally, this is attributed to the very low mass pull on bench scale tests which produce only enough sample to analyse for PGM's. However, two locked cycle tests performed by Vale did produce adequate concentrate to examine the performance of nickel-sulphide recovery in the flotation process (Table 13-8).

Table 13-8: Nickel sulphide flotation results

CDM Vale LCT Tests 2003	Feed Grade Ni (%)	Conc Grade Ni Sulphide (%)	Ni Sulphide Recovery (%)
LTC1	0.19	7.5	47.3
LTC2	0.19	9.8	45.9

Source: Bravo, 2023.

Bravo encountered similar nickel analysis challenges due to low concentrate weights so will only be able to confirm in future testing programs.

13.4.2 Transition Material

Historically, a component of the mineralized body below the oxide horizon was classified as "transition" material based on geological observations including surficial, oxidative staining on host rock samples. Bravo investigated the metallurgical amenability of this domain by performing comparative rougher flotation tests using a coarse grind benchmarked against the abovementioned T series tests and additional comparative flotation results.

22 Transition flotation tests were performed through a grade range of 0.38 – 7.95 g/t PGM's. Samples were milled to p60 -74µm. Table 13-9 shows Test T1-2 as benchmark, fresh rock, control tests vs TBS 19, 11 and 16 as transition material flotation tests.

Table 13-9: Results of transition material on flotation tests

Rougher Comparative Test	PGM Feed grade (g/t)	Mass Pull (%)	Recovery (%)	Concentrate PGM Grade (g/t)
T1	2.18	10.9	38.0	7.2
T2	2.21	8.7	35.6	9.1
T3	2.23	12.6	47.2	8.4
TBS19	1.81	3.7	36.0	18.0
TBS11	1.36	4.4	51.0	16.0
TBS16	1.94	6.2	37.0	12.0

Source: Bravo, 2023.

The comparative results above demonstrate that the previously classified "transition" domain mineralization responds similarly to the rougher performance of fresh rock material at Luanga and thus it is concluded, from a process metallurgy perspective, that the "transition" domain be considered as fresh rock material.

13.4.3 Oxide Material

The Luanga PGM deposit is characterized at surface by an extensively weathered mineralization zone constrained from surface down to depths of 5 to 30m. It is a sensu stricto oxide horizon with most, if not all, sulphide mineralization minerals (including nickel sulphides) altered to oxide phases with the host rock altered to limonitic/saprolitic phases. Historical mineralogy and test work had led the previous owner not to consider the possibility of recovering PGM from the oxide zone. This was attributed to the oxidized state of metallic mineralization minerals and the abundance of clay material, contributing to poor flotation recoveries, low selectivity and high concentrate mass pulls.

The oxide horizon at Luanga does however account for approximately 10% of historically stated resources by tonnes. Three salient points have led Bravo to review and investigate the possibility of recovering PGMs from the oxide mineralization:

- PGM grades in the historical oxide horizon appear to be on average, higher than those reported in the fresh rock horizon. Thus, the resource ounce contribution of the oxide mineralization to the total resource is higher than observed from a resource tonnes perspective. The oxide horizon demonstrates some ultrahigh PGM grade intersections, not commonly seen elsewhere in the world.
- Due to their refractory nature, platinum group minerals tend not to respond strongly to atmospheric induced weathering, even within deeply weathered profiles. This mineral character presents an opportunity to explore recovery methods that recognise the characteristics of the preserved platinum group minerals, considering either their physiochemical properties and/or gravimetric properties.
- The oxide horizon was considered sterile and thus represents an additional waste stripping expenditure item by Vale. Should sufficient value be unlocked by establishing an economic processing route that returns capital and operating costs, potential economic benefit can be realized for the global Luanga mineralized body/project by partial stripping cost offset.

Sighter flotation tests, including pre-desliming, were also performed but were condemnatory in nature. The results re-affirmed low recoveries, high mass pulls and low concentrate grades. However, Bravo identified gravimetric separation and cyanide leaching as processing routes that may hold potential to treat oxide mineralization at Luanga.

Note that, in the current MRE estimate, the oxide component has been updated and reduced to 5% of the current MRE as opposed to ~10% of the Historical Estimate, so significantly reducing the potentially commercial importance of the oxide zone.

As a result of the relatively modest contribution to the potential plant feed and the metallurgical response of the oxide mineralization, treatment of oxides was not considered in this PEA. However, results to date suggest that continued test work is warranted as there may be potential to convert a portion of the oxide material being treated as waste in the preliminary economic analysis to material that generates positive cash flow.

GE21 Consultoria Mineral

13.4.3.1 Gravimetric Tests

Three gravity concentration tests were performed using a Knelson concentrator with sample feed grades of 1.45g/t - 1.91g/t (Figure 13-4).

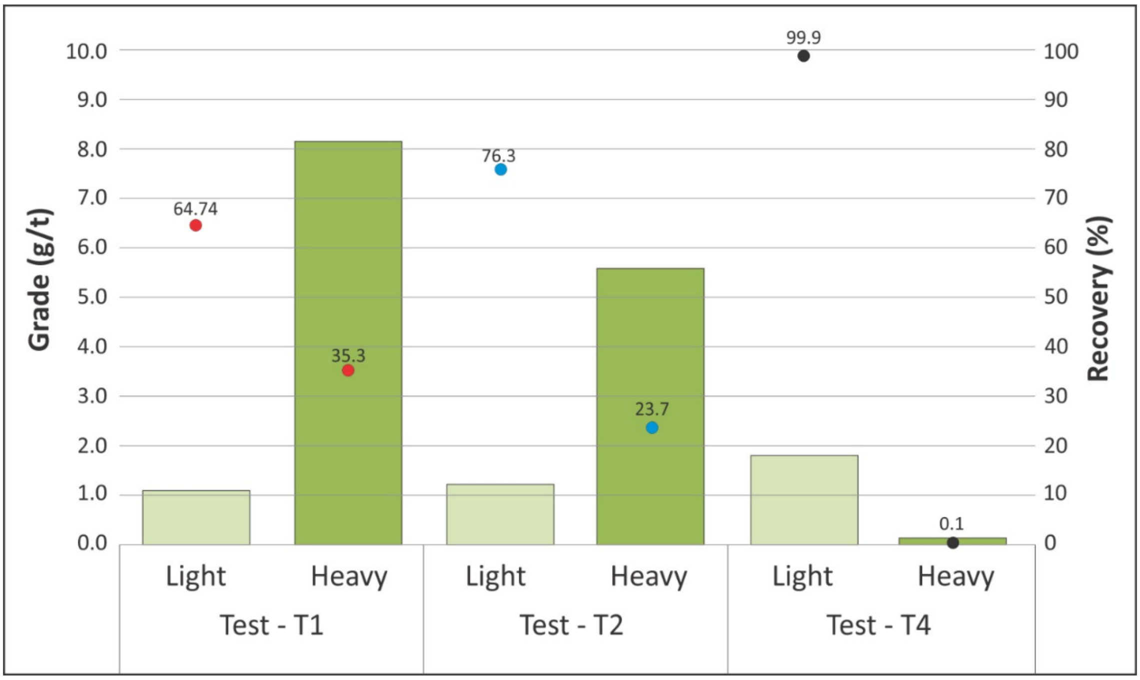


Figure 13-4: Results of gravimetric tests

Source: Bravo, 2023.

The above results demonstrate improved recoveries to the underflow but is still insufficient to envisage that an economic recovery process and saleable product can be developed using a centrifugal concentrator.

Cyclone tests were carried out using a 40mm Weir hydro-cyclone. Two tests were performed using split material with similar grades from the gravimetric sample feedstock. The hydro-cyclone apex and vortex finder diameters were 4.0 and 10.0 mm, respectively. Table 13-10 presents the operating conditions including mass and volumetric flows, percentage of solids in the streams, as well as mass partition results and desliming efficiency data.

Table 13-10: Operating conditions – hydro-cyclone test

Test	P (KPa)	t (s)	Current	Pulp (g)	Solid (g)	Water (g)	V pulp (ml)
1	250	5.8	Overflow	3,550	127.8	3,422.22	3,470
			Underflow	984	263.2	721.11	819
			Feed	4534	391	4,143.33	4,288
		Qp (L/h)	Wp (kg/h)	Ws (Kg/h)	% Ws	% sol.	ET (%)
		2153.5	2,203.4	79.3	32.7	3.6	67.3
		508.1	611	163.4	67.3	26.7	
		2661.6	2,814.4	242.7	100	8.6	

Source: Bravo, 2023.

After optimizing conditions, a mass recovery for the underflow stream in the first stage desliming of 67.3% was obtained using a feed pressure of 250 KPa. The percentages of solids obtained in the underflow and overflow in the first stage were 26.7% and 3.6%, respectively.

The overall recovery of PGM+Au was 86.7% in the underflow stream, after two desliming stages, the first with a hydro-cyclone and the second stage with siphoning using Stokes' Law of sedimentation. The overall mass recovery considering the two steps was 55.3% (Table 13-11). It is interesting to note that the global metallurgical recovery of Pt was 96.4%, being 97.2% in the first stage and 99.2% in the second stage.

Similar to the Knelson tests, the concentrate grades produced were only marginally above the feed grade and are below commercial requirements.

Table 13-11: Results of hydro-cyclone tests

Products	**Mass Balance**		**Grades (g/t)**					
1º Stage – T1	**m (g)**	**%**	**Rh**	**Pd**	**Pt**	**Au**	**PGM**	**PGM+Au**
Over 1	127.8	32.7	0.005	0.41	0.08	0	0.5	0.5
Under 1	263.2	67.3	0.006	0.45	1.4	0.05	1.86	1.91
Totals	391	100	0.006	0.44	0.97	0.03	1.41	1.45

2º Stage – T1	**m (g)**	**%**	**Rh**	**Pd**	**Pt**	**Au**	**PGM**	**PGM+Au**
Over 2 (siphoned)	1,708.1	17.87	0.003	0.14	0.06	0.04	0.21	0.25
Under 2	7,850.9	82.13	0.007	0.516	1,696	0.052	2,219	2,270
Totals	9,559	100	0.006	0.45	1.4	0.05	1.86	1.91
	Global Rec.	55.3						

	Met. Recovery (%)					
	Rh	**Pd**	**Pt**	**Au**	**PGM**	**PGM+Au**
	28.8	30.76	2.82	0	11.55	11.28
	71.2	69.24	97.18	100	88.45	88.72
	100	100	100	100	100	100
	Rh	**Pd**	**Pt**	**Au**	**PGM**	**PGM+Au**
	9.83	5.61	0.78	15.26	1.97	2.32
	90.17	94.39	99.22	84.74	98.03	97.68
	100	100	100	100	100	100
Global Rec.	**64.2**	**65.4**	**96.4**	**84.7**	**86.7**	**86.7**

Source: Bravo, 2023.

13.4.3.2 Hydrometallurgical Cyanide Leaching – Phase 1 Exploratory Program

Exploratory leaching tests were performed on oxide samples 600004 and 600005. These samples were collected from Bravo twin metallurgical drill hole MDH22LU904 (Central Sector) from approximately 9-18m depth within the oxide mineralization. Samples were homogenized and split samples from each volume were generated for head grade assay analysis at SGS Geosol. Average grades across the three samples for respective volumes and elements are presented on Table 13-12.

Table 13-12: Average grades obtained on hydrometallurgical cyanide leaching tests

Sample	Au (g/t)	Pd (g/t)	Pt (g/t)	Rh (g/t)	4E Total (g/t)
Volume A (4)	0.22	1.11	0.49	0.06	1.88
Volume B (5)	0.25	0.95	1.95	0.09	3.24

Source: Bravo, 2023.

- Direct Leaching and Carbon-in-Leach Tests

Direct leaching and Carbon-in-Leach (CIL) tests were performed to investigate the response of PGM's at differing grind sizes (p80 -1/8" and p80 -75µm). Sodium cyanide (CN) addition rates were also varied to establish concentration, influence and effective consumption rates. These tests were done at cyanide addition rates of 1,000, 5,000, and 10,000 g/t to evaluate PGM solubility. The tests were undertaken using 35% solids to promote an amenable pulp viscosity conducive for a good leaching environment. The influence of desliming and leaching with activated carbon were also evaluated.

13.4.4 Oxide Study Conclusions and Results

Generally, metal recoveries improved with CN concentration and as the grind size was decreased from p80 -1/8"to p80 -75µm. The later was credited to improved liberation, especially from remnant unoxidised rock fragments within the oxide material.

The addition of activated carbon improved the final recoveries of PGM. The following percentage improvements in recovery were observed in Table 13-13:

Table 13-13: Recovery with the addition of activated carbon

DL vs CIL	Au	Pd	Pt	Rh
Recovery CIL (onto carbon)	93.7 %	51.1 %	6.7 %	30.0 %
Recovery Improvement (above direct leach)	+1.4 %	+5.5 %	+235%	+74.4%

Source: Bravo, 2023.

Desliming by gravity prior to either direct or CIL leaching showed further improvements in global recoveries due to the purported removal of potential preg-robbing, ultrafine clays. The following better recoveries are reported below (Table 13-14):

Table 13-14: Recovery with the addition of activated carbon and desliming by gravity

DL vs CIL	Au	Pd	Pt	Rh
Recovery CIL (onto carbon)	93.7 %	51.1 %	6.7 %	30.0 %

Deslimed CIL	Au	Pd	Pt	Rh
Recovery CIL (onto carbon)	93.3 %	62.7 %	23.8 %	60.8 %

Source: Bravo, 2023.

In general, it was observed that cyanide concentration does not impact gold and rhodium recoveries but increasing cyanide concentrations improved platinum and palladium recoveries.

Effective consumption rate for cyanide varied between 900 and 4,000 g/t across tests. Lime consumption varied between 17 and 20 kg/t across tests. To reduce net cyanide consumption, future consideration should be given to the inclusion of a cyanide reclamation/recovery circuit.

Gold, palladium and rhodium leach kinetics were relatively high compared to the substantially lower kinetics seen for platinum.

The conclusion of this exploratory test program demonstrates that PGM's from the Luanga oxide zones are potentially amenable to cyanide leaching. It was strongly recommended that a more detailed and comprehensive test program be undertaken to further develop and optimize processing parameters (see Section 26 - Recommendations).

13.4.4.1 Leaching at Lower pH

Preliminary tests have been concluded investigating the impact of lower pH on PGM recoveries, particularly on palladium. Using lower pH is well documented in published technical reports which highlight improved palladium recoveries at lower pH. For these tests, lime consumption varied between 3.4 – 11.0 kg/t, lowered from the Phase 1 lime consumption range of 17 – 20 kg/t. All tests demonstrated material improvement in palladium recovery, with the best achieved recovery totalling 81.4% at a lime consumption rate or 11.0 kg/t and effective cyanide consumption of 4.3 kg/t.

Gold, platinum and rhodium recoveries under these conditions were negligible and demonstrated the potential for effective sequential PGM leaching under differing pH and cyanide concentration conditions.

13.4.4.2 Carbon Loading and Ashing Tests

Following the encouraging leaching results described above, Bravo investigated the loading potential of PGM onto carbon in a CIL circuit and the potential to produce a final "ashed" residue saleable product. Both loaded carbon and ashed carbon residue represent a saleable product. To achieve sufficient product for analysis (particularly the ashing product) a large volume solution sample was prepared to generate sufficient loaded carbon mass.

The sample was prepared in the following manner:

A tailings oxide pulp sample was analyzed and charged with dissolved gold, platinum, palladium and rhodium such that the final precious metals in-solution concentrations equalled the solution concentration of a direct leach according to observed recovery rates in tests from Phase 1. The average recoveries considered in this calculation were as follows (Table 13-15):

Table 13-15: The average recoveries considered in tests from Phase 1

Element	Au	Pd	Pt	Rh
Recovery	95 %	60 %	20 %	40 %

Source: Bravo, 2023.

Activated carbon was introduced to the system and maintained for 24 hours.

Filtered carbon was homogenized and split with duplicate samples submitted to SGS for fire assay analysis and CETEM for ICP-OES analysis and muffle furnace ashing.

PGM showed high levels of loading onto carbon with high carbon PGM grades and high adsorption recoveries. The analytical results from SGS and CETEM are summarized below (Table 13-16). Due to equipment constraints, rhodium could not be analyzed, and grades were calculated based on mass balance.

Table 13-16: Average grades obtained on Carbon Loading and Ashing Tests

Laboratory	Au (g/t)	Pd (g/t)	Pt (g/t)	Rh (g/t)	3E PGM
SGS Geosol	41.6	819.6	384.3	134.9	1380.4
CETEM	43.2	846.3	404.7	135.0	1429.2
Adsorption Recovery	99.7 %	99.7 %	92.5 %	n/d	

Source: Bravo, 2023.

13.4.4.3 Ashing Tests

In order to investigate the production of a high-grade ashed residue final product in place of a sequential elution or doré product, two ashing tests were conducted to support marketing studies.

Two 20g loaded carbon samples split from the CETEM carbon analysis samples were ashed in a bench-scale muffle furnace at CETEM. For both tests, the residual fraction mass was 0.24 grams representing a 98.8% reduction in mass. The resultant final product grade was calculated on mass balance and summarized below (Table 13-17):

Table 13-17: Average grades obtained on ashing tests

Ashed Residue	Au (g/t)	Pd (g/t)	Pt (g/t)	Rh (g/t)	4E_PGM+Au (g/t)
Grade	3 600	70 525	33 725	11 250	119 100

Source: Bravo, 2023.

The final product was highly enriched in PGM, demonstrating a total PGM grade of 119.1 kg/t or 11.91% by weight.

Note - whilst the above various oxide tests are very encouraging and warrant further investigation in the current MRE, the oxide tonnage component as a proportion of the total tonnage, has been revised down, reducing the importance of the oxide zone.

13.5 Bravo 2024/2025 Program – Fresh Sulphide

13.5.1 Sampling

A global bulk composite from the north Sector of the Luanga deposit, totalling approximately 3.6 tonnes, was constituted from diamond drill core samples. A total of 947 samples were collected from quarter-core from the MSZ unit of the deposit (Figure 13-5). A homogenized sub-sample analysis reported an average grade of 2.4 g/t 4E PGM and 0.19% Ni.

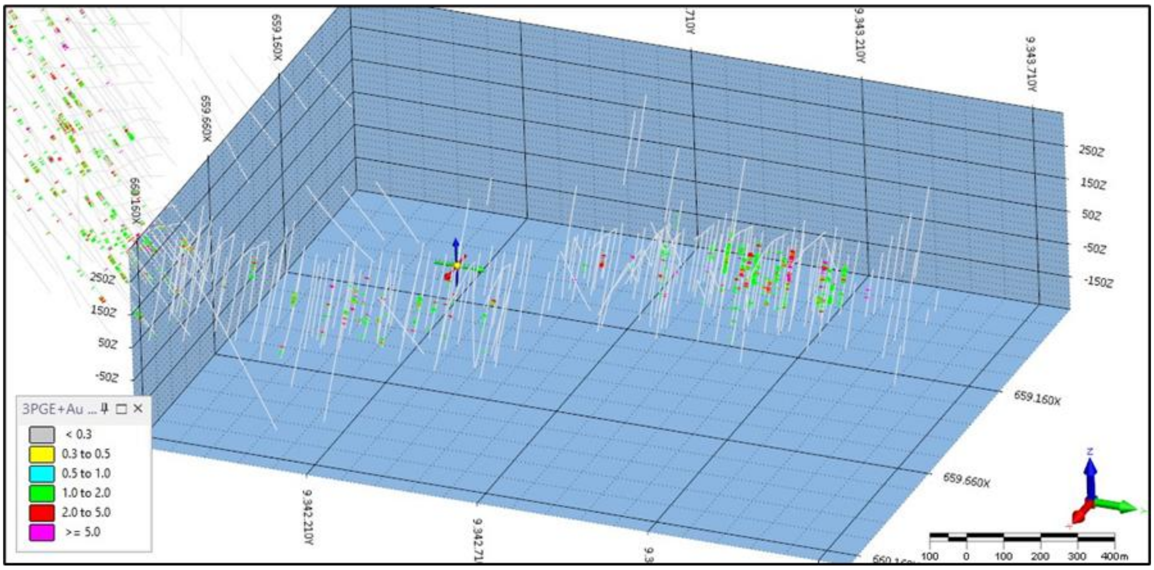


Figure 13-5: Image showing the distribution of samples collected in the north global bulk sample constitution

Source: Bravo, 2025.

Similar global composites were prepared for the Central and Southwest Sectors of the deposit. The Central and SW Sector composite was used in conjunction with the North Sector composite for detailed comminution test work, while only the North composite was used in further minerals processing tests thus far. It is envisioned that a similar scope of work will be subsequently applied to the Central and Southwest Sector global composites (see Section 26 - Recommendations).

13.5.2 Comminution

Metso Brazil was contracted to perform detailed comminution tests on North, Central and Southwest Sector global composite material. A sub-sample of 200kg from each composite was selected and sent to Metso in Belo Horizonte. The scope of work was to conduct industry-standard laboratory tests to obtain physical material characteristics that will assist in process route definition and equipment sizing. Tests completed on samples from each deposit Sector include:

- Bond Abrasion Index
- Macon Abrasion Index
- Macon Crushability
- Bond Work Index
- Bulk Density
- Specific Gravity
- Volumetric Capacity Index
- Strength Index
- Product Flakiness Index

The results of tests 1-9 are summarised in the table below (Table 13-18).

Table 13-18: Comminution test results

Test	Deposit Sector					
	North	Comment	Central	Comment	SW	Comment
Bond Abrasion Index (Ai)	0.07	Low Abrasive	0.091g	Low Abrasive	0.044g	Low Abrasive
Macon Abrasion Index (Ai)	228 g/t	Low Abrasive	502 g/t	Low Abrasive	148 g/t	Low Abrasive
Macon Crushability (Cr)	32.40%	Medium	29.10%	Medium	37.70%	Medium
Bond Standard Ball Mill (Wi)	14.98 kW/t	Med-Hard	18.91 kWh/t	Hard	14.43 kW/t	Med-Hard
Bond Standard Rod (Wi)	16.4 kWh/st	Hard	19.3 kWh/st	Very Hard	16.5 kWh/st	Hard
SAG Circuit Specific Energy	10.56 kWh/t		11.75 kWh/t		9.44 kWh/t	
Bulk Density (Y)	1.68 t/m3		1.73 t/m3		1.71 t/m3	
Specific Gravity (Yr)	2.95 t/m3		3.03 t/m3		2.91 t/m3	
Volumetric Capacity Index (C)	83.13%	Minimum	77.89%	Minimum	75.46%	Minimum
Strength Index (R)	190.64%	Minimum	248.31%	Minimum	161.97%	Minimum
Product Flakiness Index (L)	1.69%	Cubical Material	14.16%	Cubical Material	14.16%	Cubical Material

Source: Bravo, 2025.

The North and Southwest Sectors exhibit material which has low abrasion qualities, moderate crushability and medium to hard indices for ball milling. The Central Sector exhibits a higher material hardness, with higher abrasion, crushability and milling indices. All three sectors exhibit very similar SAG (Semi-Autogenous Grinding) Circuit Specific Energy values.

13.5.3 Flotation

Bravo conducted several flotation optimization programs: including testing at CETEM, independent verification and optimization program at Base Metal Laboratories (BML) in Canada and a series of mini-plant flotation tests at CETEM.

13.5.3.1 Parameter Optimization Program – CETEM

Basic rougher kinetic style tests were conducted investigating conditioning time, cell energy input, pH and pulp density impact on rougher performance (Figure 13-6).

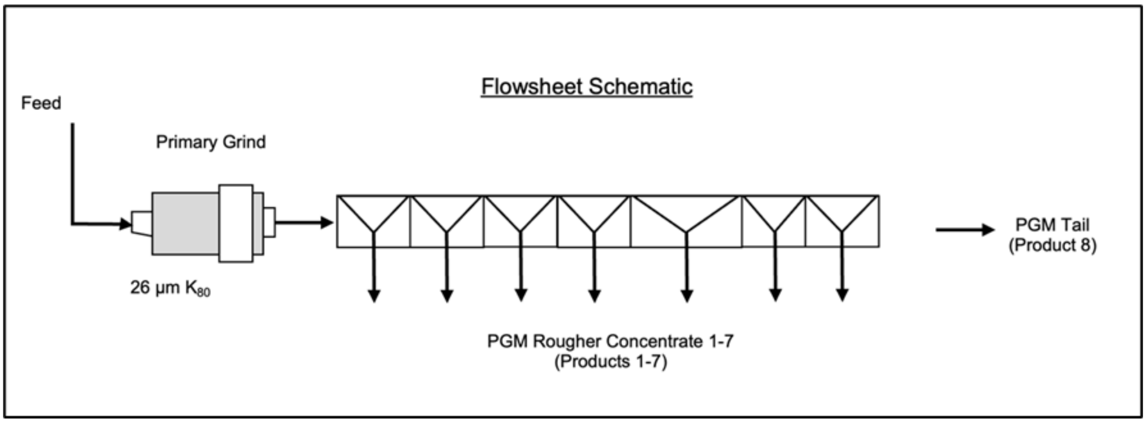


Figure 13-6: Image showing the distribution of samples collected in the north global bulk sample constitution

Source: Bravo, 2025.

Base line conditions included a 60 min grind time to achieve p80 -28µ grind size. CMC addition was 400 g/t with a 1 min conditioning time. Collector, frother, individual conditioning and flotation times are demonstrated in the table below (Table 13-19). Tests were conducted in a 3l cell using a Denver flotation machine with base line impeller speed at 1,050 rpm.

Table 13-19: Collector, frother, individual conditioning and flotation times

Stage	Reagents - g/tonne				Time Minutes	
	CMC	MIX AERO 3894 + SENXOC 2	PAX	MIBC	Condition	Float
Primary Grind					60	
Rougher Condition	400				1	
Rougher 1		20		30	30''	2
Rougher 2			20		30''	3
Rougher 3			20		30''	5
Rougher 4			20		30''	5
Rougher 5			20		30''	5
Rougher 6			20		30''	5
Rougher 7			20		30''	5

Source: Bravo, 2023.

Longer conditioning times demonstrated improvements in fast rougher recovery, fast rougher concentrate grade and final concentrate grade but only marginal improvement in total rougher recovery.

Higher cell rotation velocities improved performance in fast rougher recovery, concentrate grade and total rougher recovery, while the final rougher concentrate was lower than base line. Nickel recovery improved materially.

Highly alkaline cell conditions adversely affected performance while acidifying the naturally alkaline pulp to a pH of 7.5 marginally improved total recovery for PGM while adversely affecting Ni recovery.

Higher pulp density positively impacted total rougher recovery for PGM and Ni but adversely affected selectivity with a drop in concentrate grade relative to a lower 20% pulp density, which showed improved selectivity in both the fast and total rougher concentrate grade (Table 13-20.

Table 13-20: Average grades obtained on mini-plant flotation tests at CETEM

Test	Parameter	FRgh Rec %	Fast Rgh Conc g/t	Total Rgh Grade	Total Rgh Rec % PGM	Total Rgh Rec % Ni	Tail Grade
Baseline K		16.60	19.70	8.90	79.60	58.80	0.79
Test 1	Cond 1'30"	8.06	125.47	8.20	81.49	57.82	0.69
Test 2	Cond 5'	9.05	158.13	10.10	77.73	53.84	0.81
Test 3	Cond 10'	10.02	252.02	12.20	80.02	50.34	0.72
Test 4	Rot 1200 rpm	16.41	101.93	6.90	87.07	64.64	0.54
Test 5	Rot 900 rpm	3.70	52.03	9.50	71.89	47.00	0.98
Test 6	pH 11.5	8.12	35.99	6.50	73.20	51.46	0.99
Test 7	pH 7.5	17.00	116.52	8.20	83.12	52.08	0.71
Test 8	pH 4.5	13.10	91.13	7.90	82.10	54.07	0.69
Test 9	Sol 20%	14.09	120.43	12.80	79.54	48.83	0.74
Test 10	Sol 40%	18.77	49.24	6.80	86.88	61.40	0.57

Source: Bravo, 2025.

GE21 Consultoria Mineral

13.5.4 Base Metal Laboratories

Base Metal Laboratories in Kamloops, Canada, were contracted to independently perform flotation test work, principally on the North Sector global composite. Initial exploratory tests were also conducted on the Central Sector global composite.

13.5.4.1 Sample Characteristics

Analyses were performed to measure feed characteristics. These included chemical analysis using standard analytical techniques as well as fiber analysis on the North Sector composite. A summary of the chemical data is shown in Table 13-21.

Table 13-21: Average feed samples before the tests

Sample	Assays - percent or g/t					
	Pd	Pt	Au	3E	Ni	S
North Zone Composite	1.45	1.09	0.08	2.61	0.21	1.28
Central Zone Composite	1.73	0.56	0.05	2.35	0.25	0.64

Source: Bravo, 2025.

The North Sector composite measured 1.45 g/t Pd, 1.09g/t Pt, and 0.08 g/t Au for a combined content of 2.61g/t (denoted as 3E throughout this program). The Central Sector composite measures higher Pd and lower Pt at 1.73 and 0.56 g/t, respectively, resulting in a slightly lower 3E of 2.35g/t. Nickel was about 0.21 to 0.25 in both composites. The sulphur content was quite different between the two, measuring 1.28 percent in the North Sector composite and 0.64 percent in the Central Sector composite.

The ratio of sulphur to other elements may be an important consideration when considering concentrate grade from a bulk sulphide flotation process; higher PGM and nickel concentrate grades may be possible from the Central Sector composite, provided the non-sulphide gangue can be reasonably rejected.

13.5.4.2 Metallurgical Tests

Metallurgical testing was conducted on the samples to evaluate currently considered processing options, and to conduct optimization of the conditions and flowsheet. Development testing focused on the North Sector composite and investigated various parameters. Two locked cycle tests were conducted on the North Sector composite. The Central Sector composite was tested with the same (North Sector) flowsheet, including a single locked cycle test. The metallurgical test data is discussed in the following sub-sections.

13.5.4.3 Rougher Test Work

Rougher kinetic tests were conducted to determine the mass recovery versus metal recovery for various conditions. Historical testing on the project indicated fine-grained PGMs and sulphides were present, and a fine primary grind was required. A primary grind series of tests and some alternative flowsheet configurations, including multistage grinding (MF2 – mill float), were tested. Talc, a naturally hydrophobic gangue mineral, was present and will need to be controlled to avoid dilution of the final concentrate. PE26, a CMC starch-based depressant was used in testing. Talc pre-flotation was also briefly evaluated in select tests. A summary of the rougher flotation test conditions and results are shown in Table 13-22 and Table 13-23.

Table 13-22: Rougher test work parameters

Composite	Test	General Description	PG	g/t		Time
			µm K80	PE26	PAX	Min
North Zone	T01	Effect of PG*	46	425	88	20
	T02	Prefloat	46	0	98	25
	T03	Grav	46	400	88	20
	T04	Effect of PG	30	550	88	20
	T05	Effect of density	30	550	88	20
	T06	Effect of PG	82	400	88	20
	T08	Effect of PG	20	750	215	25
	T10	H2S04 pH 7.5	30	750	165	20
	T12	Calgon	30	550	328	30
	T13	MF2	46/18	900	228	23
	T14	Prefloat	30	200	118	20
	T21	Effect of density/Collector	30	650	328	24
	T22	Effect of density/Collector	30	700	328	24
	T23	800g/t Calgon	30	650	328	24
	T25	Fine/Coarse Circuit	46/43	770	106	31
Central Zone	T28	Baseline	33	475	88	25
	T31	Increase PE 26	33	825	88	25
	T32	Effect of PG	27	725	88	25

*PG = primary Grind

Source: Bravo, 2025.

Table 13-23: Average grades obtained on rougher test work

Composite	Test	Mass %	Assay - percent or g/t						Distribution - percent					
			Pd	Pt	Au	3E	Ni	S	Pd	Pt	Au	3E	Ni	S
North Zone	T01	36.3	3.2	2.1	0.2	5.5	0.4	1	80	73	84	77	60	29
	T02	14.6	5.8	5	0.2	11.1	0.9	5.6	57	59	39	57	59	63
	T03	34.2	2.7	1.4	0.7	4.8	0.3	1.2	72	61	97	71	56	32
	T04	22.6	5	3.4	0.2	8.7	0.6	1.9	77	76	77	77	65	34
	T05	12.2	7.7	7.8	0.5	15.9	1.2	3.5	73	77	62	74	66	33
	T06	37.1	3.1	2.3	0.2	5.6	0.3	0.8	74	74	77	74	54	22
	T08	16.7	5.5	5	0.3	10.8	1	6.4	80	81	72	80	76	82
	T10	19.4	5	4.7	0.2	10	0.8	3.4	77	81	72	79	70	53
	T12	18	8.3	5.4	0.3	14	0.9	3.5	87	81	71	84	71	51
	T13	15.3	8.4	5.3	0.3	14	1	3.4	90	75	62	83	70	40
	T14	17.3	6.7	4.1	0.3	11.1	0.8	3.4	78	70	48	73	64	45
	T21	29.9	4.9	3.1	0.1	8.1	0.5	1.9	86	85	76	86	70	46
	T22	11.9	11.7	8.4	0.4	20.6	1.2	4.7	84	80	75	82	70	43
	T23	15.6	8.6	6.5	0.4	15.4	0.9	3.7	85	83	77	84	75	47
	T25	14.6	10	5.6	1.2	16.7	1	5.4	85	78	92	83	70	56
Central Zone	T28	16.5	8.3	3.1	0.8	12.2	1.1	2.1	74	67	94	73	66	49
	T31	9.6	13.1	5.3	1	19.4	2.1	4	78	76	91	78	71	63
	T32	13.6	9.3	4.1	0.8	14.1	1.4	2.6	80	76	93	79	68	55

Source: Bravo, 2025.

The 3E rougher recovery ranged from 57 to 86 percent, at mass recoveries of 10 to 37 percent. The mass versus recovery for 3E is shown in Figure 13-7. In general, there was a significant component of the metals of interest that was fast floating. Past a certain point, the incremental recovery was quite low, with only a marginal increase in recovery with significant mass recovery. Production of a flash concentrate, producing final concentrate from rougher 1 may be possible. Jameson flotation technology may be a suitable means to recover a portion of the final concentrate material from the rougher.

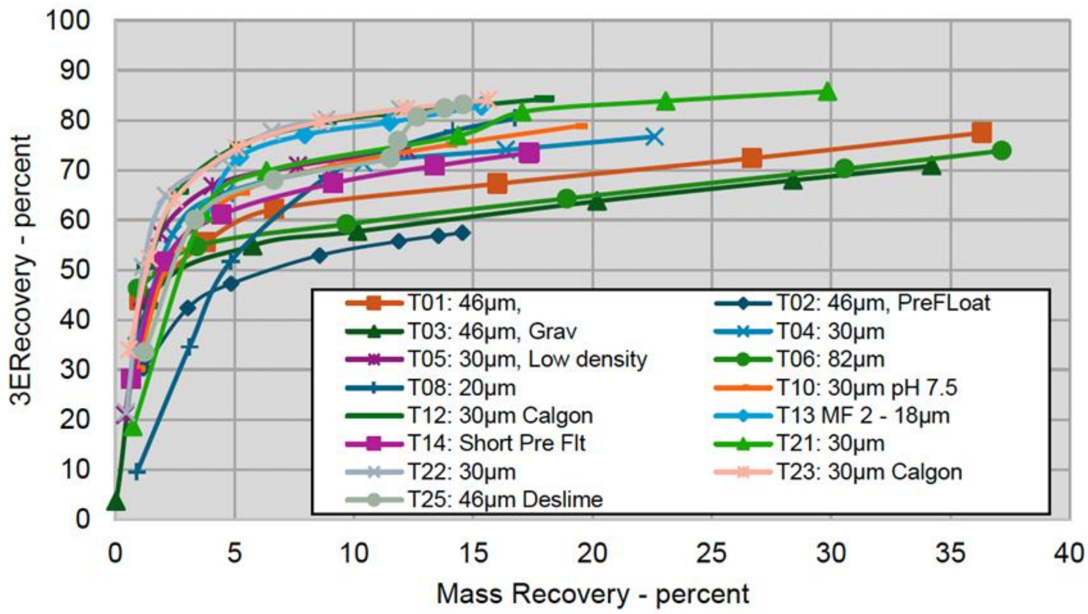


Figure 13-7: The mass versus recovery for 3E

Source: Bravo, 2025.

The primary grind was confirmed to be a significant contributing factor to recovery. The highest recoveries were generally achieved at 30µm K80 or finer primary grinding. Figure 13-8 displays the grade of the rougher tailings versus the primary grind sizing for Pd, Pt, and 3E. Although there is some noise in the data from other changing variables and inherent test variation, there is a clear trend of decreasing PGM tails grade with finer grinding. The tails data is also presented in Table 13-24.

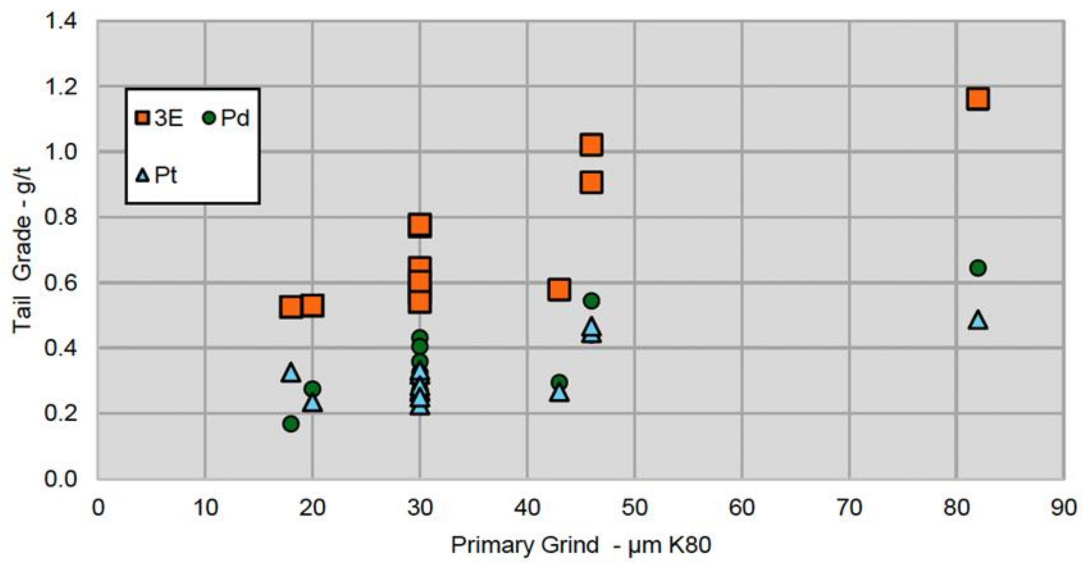


Figure 13-8: Grades of the rougher tailings versus the primary grind sizing for Pd, Pt, and 3E

Source: Bravo, 2025.

Table 13-24: Average grades obtained on rougher test work in tails

Composite	Test	General Description	Sizing	Rougher Tail Assay - percent or g/t					
			µm K80	Pd	Pt	Au	3E	Ni	S
North Zone	T08	Effect of Primary Grind	20.00	0.27	0.24	0.02	0.53	0.06	0.28
	T04		30.00	0.43	0.32	0.02	0.77	0.10	1.08
	T01		46.00	0.44	0.45	0.02	0.91	0.14	1.42
	T06		82.00	0.64	0.49	0.03	1.16	0.14	1.58
	T05	Effect of density (2kg/8L)	30.00	0.40	0.33	0.04	0.78	0.08	0.97
	T21	Effect of density/Collector (2kg/4L)	30.00	0.32	0.23	0.02	0.57	0.09	0.95
	T22	Effect of density/Collector (2kg/8L)	30.00	0.30	0.28	0.02	0.60	0.07	0.84
	T12	Calgon	30.00	0.27	0.27	0.03	0.57	0.08	0.72
	T23	800g/t Calgon	30.00	0.27	0.25	0.02	0.54	0.06	0.80
	T03	Gravity Included	46.00	0.54	0.47	0.01	1.02	0.14	1.35
	T10	H2S04 pH 7.5	30.00	0.36	0.27	0.02	0.65	0.08	0.74
	T13	MF2	18.00	0.17	0.33	0.03	0.53	0.08	0.94
	T25	Fine/Coarse Circuit	43.00	0.29	0.27	0.00	0.58	0.08	0.70
Central	T28	Baseline	33.00	0.56	0.30	0.01	0.87	0.1 1	0.43
	T31	Increase PE 26	33.00	0.40	0.18	0.01	0.59	0.09	0.25
	T32	Effect of Primary Grind	27.00	0.37	0.20	0.01	0.59	0.10	0.33

Source: Bravo, 2025.

13.5.4.4 Cleaner Test Work

Cleaner testing was conducted to determine the concentrate quality and recovery in the open circuit and to guide conditions for locked cycle testing. Multiple flowsheets were tested with various reagent dosages and with and without regrinding. The main flowsheets tested include a split high-grade (HG) and low-grade (LG) circuit and a single circuit (Bulk). Table 13-25 and Table 13-26 show a summary of conditions and results (Figure 13-9). The following points are of note when reviewing the data:

- For the North Sector, recovery of 3E to the final concentrate measured between 46 and 71 percent at a grade of 70 to 682 g/t 3E. Nickel performance was variable, with recoveries ranging from 5 to 63 percent.
- Only two cleaner tests were conducted on the Central Sector composite due to mass constraints. In both tests, 63 percent of the 3E was recovered to the concentrate at a grade of between 154 and 198 g/t 3E.
- There is some indication performance is related to the dosage of PE26. PE26 is primarily used to depress talc to reduce dilution in the concentrate. While this is a key reason, higher PE26 dosages are also assisting with recovery, indicating there may be other interactions of the PE26 beyond just talc depression; the exact mechanism is unknown. Calgon was evaluated as an additional gangue control mechanism. Calgon is a detergent that is efficient at complexing calcium and magnesium ions in solution, potentially reducing the slime coating of the sulphide surfaces. The impact of Calgon was inconclusive and did not provide a significant boost to performance.
- Higher density resulted in higher mass recovery due to more talc floating under similar conditions. The increased talc recovery could be controlled with increased PE26 addition. A trade-off evaluating the differences in low-density versus higher reagents, among other parameters, should be considered.
- When significant regrinding was used in Test 17, significant increases to concentrate grade were measured; however, recovery was reduced. Given the saleable grades without regrinding, it is not required for the tested samples.
- A portion of the PGMs and nickel is fast floating. However, long flotation times are required for high recovery.

Table 13-25: Cleaner test work parameters

Conditions											
Composite	Test	Flow Sheet	General Description	µm K80		g/t					Ro Cell L
				PG	Rgd	PE26	Calgon	PAX	3894	A208	
North Zone	T09	Bulk	Baseline	30	29	660	-	153	-	-	8
	T11	HG/LG	HG/LG	30	30	695	-	133	-	-	8
	T15	HG/LG	HG/LG	21	21	695	-	133	-	-	8
	T16	HG/LG	HG/LG - Calgone	21	15	695	750	133	-	-	8
	T17	HG/LG	T16 - no Rgd	21	-	695	750	133	-	-	8
	T18	HG/LG	T11 - no Rgd	30	-	695	-	133	-	-	8
	T19	HG/LG	T11 Closed Clnr	30	32	695	-	133	-	-	8
	T24	Bulk	T09 No Rgd, add Scav	30	-	805	-	173	-	-	8
	T26	Bulk	T24, Rgd	30	17	875	-	173	-	-	8
	T27	Bulk	Higher Density	30	-	930	-	173	-	-	4
	T29	Bulk 1 clnr	Clnr 1 kinetic, selective collectors	30	-	650	-	105	25	25	8
	T33	Bulk	Low er PE26, Mag sep on Tls	30	-	400	-	153	-	-	4
Central	T31	Bulk	Preliminary Test	33	-	930	-	173	-	-	4
	T32	Bulk	Finer Primary Grind	27	-	830	-	173	-	-	4

Source: Bravo, 2025.

Table 13-26: Average grades obtained on cleaner test work

Composite	Test	Mass	Assay - percent or g/t						Distribution- percent					
		Final Concentrate												
		%	Pd	Pt	Au	3E	Ni	MgO	Pd	Pt	Au	3E	Ni	MgO
North Zone	T09	0.5	192	75.5	14.4	282	6.8	12.8	66	41	73	57	17	0.3
	Til	1.7	71.2	47.7	2.66	122	6.1	17.6	73	69	67	71	47	1.2
	T15	1.1	95.2	33.9	3.25	132	2.9	20.7	70	36	62	56	15	1
	T16	1.3	75.5	33.4	2.61	111	2.4	20	69	42	60	57	14	1.1
	T17	1.2	78.9	37	3.4	119	3.4	21.6	68	46	57	59	20	1.2
	T18	2.5	45.3	29	1.06	75	5.4	12.6	78	61	46	70	63	1.5
	T19	1.8	62.1	39.2	8.16	110	6.5	11.7	71	64	74	69	54	1
	T24	0.9	109.5	71.5	4.68	186	10.3	2.1	72	66	80	69	47	0.1
	T26	0.2	460.1	187	35	682	4.6	3.9	60	36	81	51	5	0
	T27	0.9	122.9	95.8	5.3	224	10.3	3.6	72	71	55	71	45	0.2
	T29	1.6	60.8	39.3	1.63	102	4.2	17.9	63	55	41	59	33	1.3
	T33	1.9	43.3	25.9	1.2	70	3.5	13.6	49	41	57	46	34	1.1
Central	T31	1	103.5	42.9	7.99	154	14.2	7.8	62	62	77	63	49	0.3
	T32	0.8	128.2	57.4	12.4	198	14.9	6.9	62	60	82	63	42	0.2

Source: Bravo, 2025.

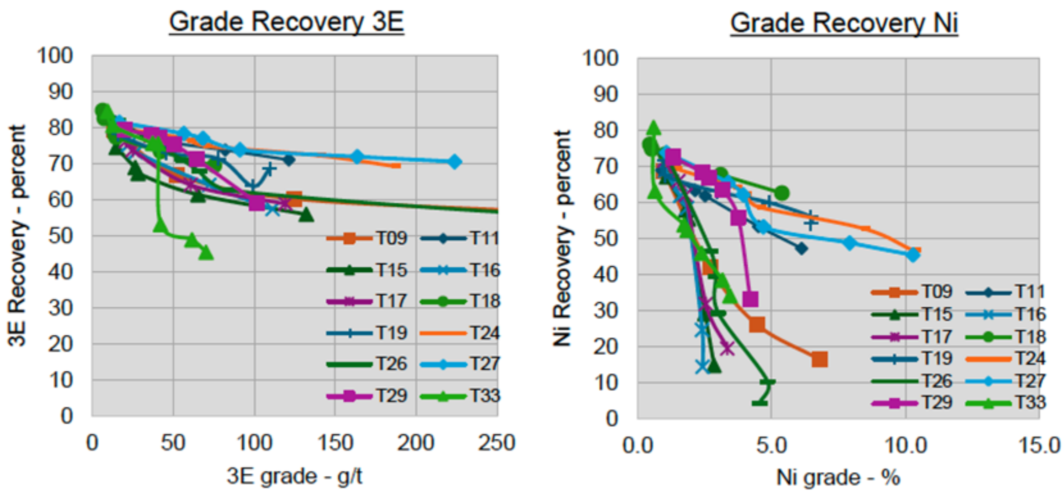


Figure 13-9: Recovery obtained on cleaner test work

Source: Bravo, 2025.

13.5.4.5 Locked Cycle Tests

Locked cycle tests were conducted to determine the overall performance in closed circuit operation. Two locked cycle tests were conducted on the North Sector composite and a single locked cycle test on the Central Sector composite. The flowsheet, conditions, and results are shown in Table 13-27 to Table 13-29, and Figure 13 11 and Figure 13-11.

For the North Sector, two flowsheet configurations were used, including a split feed circuit with lower density, higher collector, and Calgon in Test 20. Following this test, additional batch optimization was conducted, and a subsequent locked cycle test with a simplified circuit, including only one circuit, higher rougher flotation density, no Calgon and lower collector. Slightly higher PE26 was used as established by higher density batch tests. The measured overall performance for these tests was quite similar. In both tests, the 3E recovery measured 78 percent for the North Zone composite. The concentrate graded 121 g/t 3E for the split circuit (Test 20), and 154g/t 3E for the simplified flowsheet (Test 20). Nickel recovery measured 62 percent, resulting in a concentrate grade of 9.4 % nickel for Test 30.

The simplified circuit was also used for the Central Sector composite, Test 30. For this test, 3E was 68 % recovered to the bulk concentrate at a 3E grade of 202 g/t. The rougher recovery was similar to the North Sector composite, but higher cleaner tailings losses were measured. Nickel recovery measured 44 %, at a concentrate grade of 16 % nickel.

An opportunity exists to further improve recovery, particularly for the Central Sector composite. According to client provided targets, a concentrate grade closer to 80 g/t 3E is still favourable, so there may be potential to improve recovery, while slightly lowering concentrate grade with more aggressive conditions. A Trace Mineral Search is in progress on Test 20 rougher tailings. It is likely that some of the losses are liberation constrained, but fine liberated particles may be a further target to exploit.

A single cleaner flotation test was conducted after locked cycle tests, evaluating a reduced PE 26 dosage, as well as magnetic separation from the flotation tailings. The lower PE26 significantly reduced both recovery and grade; further optimization with a systematic evaluation of various PE26 dosages and other gangue depressant is recommended. The magnetic concentration recovered a significant amount of mass, at some upgrading of the PGM and significant recovery of Ni and S. Magnetic separation warrants further investigation, possibly on a feed sample which could have the potential to reduce the mass to fine grinding and flotation, provided the metals of interest are adequately recovered to the magnetic concentrate.

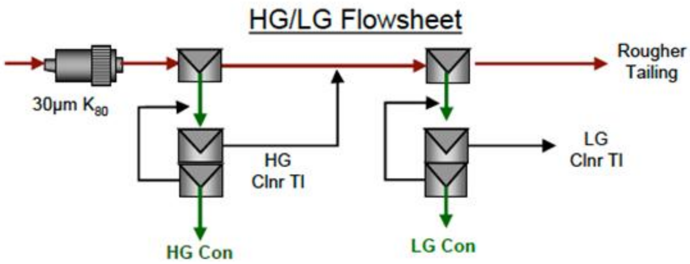


Figure 13-10: HG and LG flowsheet

Source: Bravo, 2025.

Table 13-27: HG and LG flowsheet parameters

Test	Product	Stage	Cell Size	g/t					pH	Time
			L	PE26	PAX	Calgon	MBC	W31		min
T20	HG Rougher	1	8	550	48	-	84	-	8.9	9
	HG Clnr	2	2.5/ 1.5	40	25	-	21	10	8.9	6/4
	LG Rougher	1	8	150	200	500	-	-	8.6	15
	LG Clnr	2	2.5/1.5	135	35	-	14	-	8.6	7/4

Source: Bravo, 2025.

GE21 Consultoria Mineral

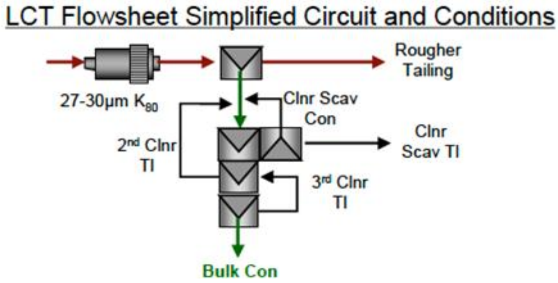


Figure 13-11: HG and LG flowsheet simplified

Source: Bravo, 2025.

Table 13-28: HG and LG Flowsheet simplified parameters

Test	Product	Stage	Cell Size	g/t					pH	Time
			L	PE26	PAX	Calgon	MIBC	W31		Min
T30	Rougher	1	4	825	88	-	49		9.0	25
	Cleaner	3	4/2.5/1.5	105	75	-	56	60	8.7	12/7/9
	Clnr Scav	1	4	-	10	-	-		8.3	4
T34	Rougher	1	4	725	88	-	49		9.2	25
	Cleaner	3	4/2/1	105	75	-	56	10	9.0	12/7/9
	Clnr Scav	1	4	-	10	-	-		8.7	4

Source: Bravo, 2025.

Table 13-29: Average grades obtained on locked cycle tests

LCT Results Cycle D+E														
Test / Composite / Flow sheet	Product	Mass	Assay Results						Distribution - percent					
		%	Pd g/t	Pt g/t	Au g/t	3E g/t	Ni%	S%	Pd	Pt	Au	3E	Ni	S
T20: North Zone HG/LG Flow Sheet	Cycles D + E													
	Feed	100	1.64	1.2	0.07	2.9	0.21	1.2	100	100	100	100	100	100
	Comb. Con	1.8	71.7	47	2.5	121	6.9	19.1	81	74	69	78	62	29
	HG Con	0.6	190.8	115.3	6.89	313	13.1	30.4	72	61	65	67	39	15
	LG Con	1.2	11.4	12.8	0.24	24.4	3.7	13.4	8	13	4	10	22	14
	LG Clnr Tis	7.8	0.7	0.7	0.03	1.47	0.16	1.4	3	5	3	4	6	9
	Ro Tls	90.4	0.29	0.3	0.02	0.59	0.07	0.8	16	21	27	18	32	62
T30: North Zone Simplified Flow sheet	Cycles D + E													
	Feed	100	1.4	1.1	0.05	2.54	0.2	1.2	100	100	100	100	100	100
	Bulk Con	1	86.3	65.4	2.74	154	3.39	29.1	79	76	78	78	62	31
	Clnr Scav Tis	12	0.54	0.54	0.01	1.1	0.004	1.4	5	6	3	5	11	15
	Ro Tls	86.7	0.27	0.22	0.01	0.5	0.002	0.7	16	18	19	17	27	54
T30: Central Simplified Flow sheet	Cycles D + E													
	Feed	100	1.62	0.64	0.1	2.36	0.29	0.7	100	100	100	100	100	100
	Bulk Con	0.8	137.2	54.3	10.49	202	15.95	29.4	67	67	86	68	44	34
	Clnr Scav Tis	13.4	1.98	0.68	0.04	2.69	0.55	1.2	16	14	6	15	25	23
	Ro Tls	85.8	85.8	0.14	0.01	0.46	0.1	0.3	16	19	9	17	30	43

Source: Bravo, 2025.

13.5.5 Mini Plant Program

A mini plant test program was also initiated to:

1. Produce larger quantities of concentrate to evaluate potential downstream processing opportunities.
2. Further evaluate parameters determined from the bench scale program.
3. Evaluate detailed concentrate chemistry and mineralogy.

Downstream processing of Luanga concentrates is an important consideration since the size of the Luanga deposit is anticipated to support a large tonnage, long life operation. Sample concentrates will be used in a range of pyrometallurgical and hydrometallurgical smelting and refining tests to inform technical and strategic options.

The sample used for this program originated from a large-diameter diamond core (ZW) acquired from the previous project owner. The core was sourced from holes FM004, FM005, FM006 and FM007, drilled in the SW sector of the deposit. These holes were drilled into a localized, high talc area and the core material had subsequently been archived and stored for approximately 20 years.

The drill core was sampled and composited to achieve a target grade of 2g/t 4E PGM.

13.5.6 Phase 1 Concentrate Production Run

Bravo Mining contracted CIT SENAI Laboratory and Frank Rezende Consulting to conduct a concentrate production program using the mini pilot plant equipment at CIT SENAI in Belo Horizonte, Brazil.

Six-grade profile bulk samples were prepared from the historical ZW drill core acquired from Vale. A total of 2.5t of material was prepared for processing with the intention of producing a wide arrangement and volume of concentrate grades for further downstream processing test work (Table 13-30).

Table 13-30: The feed samples' average grades do Phase 1

COMPOSITE	Au(g/t)	Pt (g/t)	Pd(g/t)	Rh(g/t)	4 E (g/t)	Ni (%)
A	0.041	0.310	0.764	0.033	1.148	0.138
B	0.046	0.373	0.907	0.041	1.377	0.147
c	0.045	0.470	1.171	0.058	1.744	0.179
D	0.066	0.645	1.610	0.090	2.410	0.192
E	0.125	0.879	2.181	0.155	3.339	0.182
F	0.128	1,609.000	3.467	0.240	5.444	0.176

Source: Bravo, 2025.

The circuit configuration composed 3 mechanical rougher cells. The first cell was used as a fast, rougher float, producing a high-grade concentrate. The remaining rougher cells produced a lower-grade combined concentrate, while tailings were treated with a single-stage scavenger to produce a third concentrate.

The feed material was milled to p90 -38 μm while general rougher cell reagent conditions were maintained as per the Bravo standard conditions as determined in the 2023 flotation program, which included the addition of 500 g/t of CMC as depressant, 40g/t of A208/A3894 (50/50) and 80g/t staged collector addition and 40g/t MIBC frother.

Resultant concentrates were composited for trace and deleterious element analysis. All trace elements were found to be below smelter thresholds including As, Sb, Bi, Cd, Pb, Te, Zn, F, Hg, and Cl (Table 13-31).

Table 13-31: Typical smelter threshold for trace elements

Element	Bravo Analysis (%)	Typical Smelter Threshold (%)
As	0.015	0.2
Sb	0.0018	0.05
Bi	0.0048	0.01
Cd	0.00007	0.02
Pb	0.0007	0.6
Te	<0.001	0.03
Zn	0.0005	2.5
F	0.0135	0.03
Hg	<0.05	0.001
Cl	0.014	0.05
Cr2O3	0.22	0.90

Source: Bravo, 2025.

A concentrate with a target grade of 60 g/t was also prepared from the produced material for pyrometallurgical test work.

13.5.7 Phase 2 Parameter Circuit Run

Due to equipment sizing and low mass pull from the Phase 1 mini-plant program, sufficient cleaner stage investigations could not be undertaken. A remnant feed sample (Sample B) was delivered to CETEM in Rio de Janeiro to conduct further test work, including cleaners, into the processing circuit. The process configuration is shown in Figure 13-12.

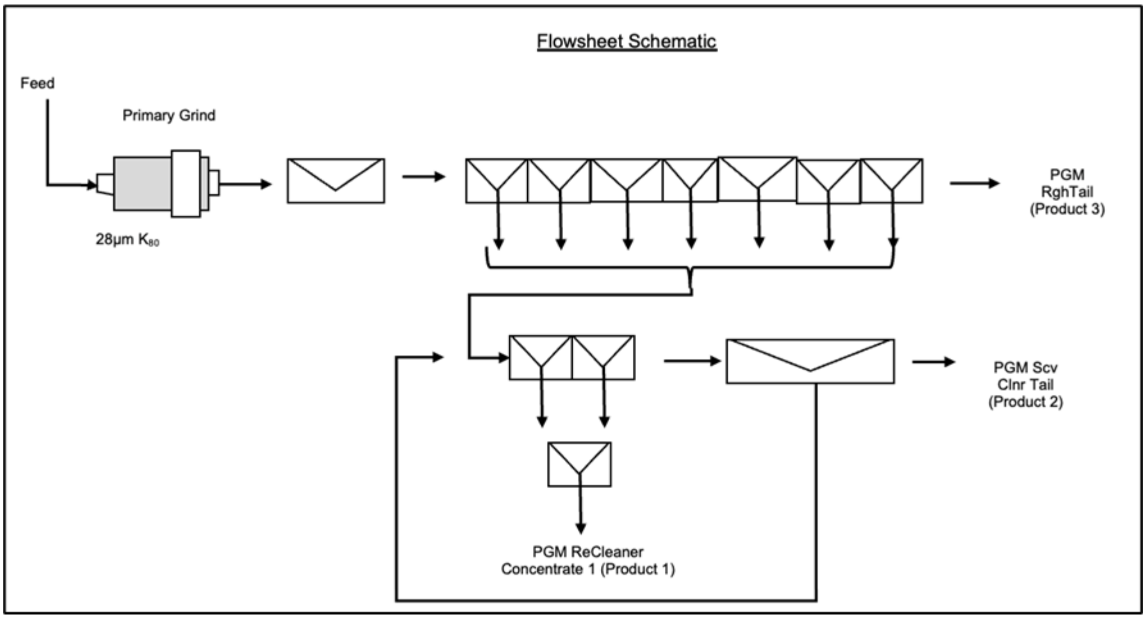


Figure 13-12: Flowsheet schematic of the Phase 2

Source: Bravo, 2025.

The circuit configuration is a primary grind followed by a 7-stage rougher, 2-stage cleaning and a single-stage recleaner and scavenger.

The primary grind was performed to p80 -28 μm. Additional reagent conditions are presented in the test sheet below (Table 13-32).

Table 13-32: Phase 2 parameter circuit run

Stage	Reagents - g/tonne			
	CMC	MIX (AERO 3894 + SENKOL 2)	PAX	MIBC
Primary Grid				
Talc Condition	700			
Rougher 1 MIX		20		30
Rougher 2 PAX			90	
Rougher 3 to 4 PAX				
Rougher 5 PAX	200		90	
Rougher 6 to 7 PAX				
Cleaner 1 to 2 PAX			20	
ReCleaner				
Scavenger Cleaner			5	

Source: Bravo, 2025.

Metal recoveries are shown in Table 13-33. Recoveries were broadly in line with anticipated results from a localized, high-talc historical sample. Target concentrate grade was not achieved, with a final concentrate grade of 36 g/t 4E and 3.7% Ni. The lower concentrate grade was attributed to excessive gangue contamination and lower selectivity.

Table 13-33: Metal recoveries in Phase 2

Element	Recovery
Pd	73%
Pt	77%
Rh	39%
Au	87%
Ni	59%

Source: Bravo, 2025.

13.5.7.1 Oxide Material Leaching

13.5.7.1.1 Leaching – Phase 2 Parameter Definition and Optimization Program

TESTWORK Process Development were contracted to perform a Phase 2 metallurgical study on Luanga oxide samples.

The Phase 2 sample material was collected as a bulk excavated sample of approximately 2 tonnes, collected at sample location MDH902. The location coincides with the collar location of metallurgical twin diamond drill hole MDH22LU902, situated at the northern edge of the Central Sector of the Luanga mineralized body (refer to sampling map).

The sample was delivered to TESTWORK, where it was homogenized and split into workable sample sizes. Representative split samples were sent to SGS Geosol in Belo Horizonte for head grade analysis (Table 13-34).

Table 13-34: Head grade analysis results – leaching Phase 2

Head Grade	Au (g/t)	Pd (g/t)	Pt (g/t)	Rh (g/t)	4E_PGM+Au (g/t)
Split Sample Average	0.03	1.32	1.58	0.28	3.20

Source: Bravo, 2025.

The scope of the Phase 2 program investigated (Table 13-35):

1. Initial Exploratory Priority Tests
2. Influence of NaCN Initial Concentration
3. 2-Stage pH Augmentation Influence
4. Temperature Influence
5. Influence of Contact Time
6. Size-by-Size Leaching
7. Influence of Additives
8. Oxide/Altered/Fresh Blend Leach
9. Additional Standardized Tests

Table 13-35: Parameters and recoveries of Phase 2

Regime	Conditions	Tests	Best Elemental Recoveries			
			Pd	Pt	Au	Rh
Exploratory	pH 9.5 – 10.5 NaCN 2500 – 5000 ppm ZTPOLY, H_2O_2, $NaCO_3$	LT1 – LT11	82%	5%	74%	2%
NaCN Concentration	500 – 10000 ppm ZTPOLY, H_2O_2	LT 12 – LT21	78%	7%	89%	5%
2-Stage pH	12h pH 9.5 + 12h pH 11.0	LT22 – LT23	74%	*	65%	10%
Temperature	pH 10.5 35°C and 50°C	LT24 – LT25	55%	4%	73%	*
Contact Time	2, 4, 8, 24, 48, 72 hr	LT26 – LT27	84%	*	65%	*
Size by Size	>1/4", >1mm, <1mm 5000ppm NaCN	LT28 – LT33	71% 78% 67%	3% 3% 7%	84% 65% 73%	1% 2% 15%
Additives	Glycine 2g/L Glycine 2g/L + H_2O_2 1g/L NaOH Turbo 2g/L Leach Aid 2 g/L	LT34 – LT45	83%	15%	79%	5%
Blend Leach	NaCN 9281 pH 10.5 24/48hr Direct Leach 48hr CIL	BRF LT1 – LT4	75%	8%	87%	95%
Supplementary Standardized	NaCN 11670 ppm pH 10.5 24 hr	SP1 – SP12	66%	21%	90%	25%

Source: Bravo, 2025.

13.5.8 Palladium

Palladium generally showed good recoveries across a range of conditions with best recoveries at pH 10.5 and NaCN dosages of 5000 ppm. Lime consumption was 5 kg/t to maintain pH. Longer leaching times favoured improved recoveries with the leaching curves showing potential for further recovery improvement at extended leach times.

Glycine as an additive improved Pd recoveries, even at low dosages. Further exploratory tests to investigate optimised dosages should be investigated.

Other additives, temperature or high NaCN dosages did not further improve recovery.

13.5.9 Gold

Like Pd, gold showed best recoveries in the range of 10.5 pH and NaCN dosage of 5,000 ppm. Gold recoveries were neutral to extended leaching times with leach curves stabilizing after 24hr. Additives, temperature augmentation and high NaCN dosages did not further improve Au recoveries.

13.5.10 Platinum

Like the initial exploratory tests conducted in 2022/2023, Pt showed both lower and highly variable recovery during leaching test work. Due to low liquor Pt tenors, analytics were less stable compared to Pd and Au.

Best recoveries for Pt were achieved with high dosages of NaCN with shorter leach times (24hr). Glycine additive improved recoveries, even at low dosages and warrant further investigation.

13.5.11 Rhodium

Rhodium showed both lower and highly variable recovery during leaching test work (2 – 95%). Due to low liquor Rh tenors, analytics were less stable compared to Pd and Au, similar to platinum.

In general, best recoveries were achieved at the base line conditions of pH10.5 and 5,000 ppm NaCN dosage.

13.5.12 PGM Speciation

Over 38 individual platinum group mineral species have been identified at Luanga. Differences in PGM speciation, specifically as it relates to platinum and rhodium recovery may play an important role in understanding metallurgical performance.

This is evidently demonstrated in the stark performance difference between the Phase 1 core composite sample (18 – 61%), Phase 2 bulk sample rhodium recovery performance (2 – 15%), the standardized supplementary tests (2 – 25%), and the Blend tests recovery performance (81 – 95%).

As previously described, the bulk Phase 2 sample consists of excavated soil-oxide material. The standardized supplementary test samples originate from composites within the oxide material below the soil horizon. The blend sample consists of a composite from the lower part of the oxide and the upper part of the fresh rock horizon in the Southwest Sector of the Luanga deposit. The blend sample showed very high levels of alteration (+60% talc).

The differences in Rh recovery could potentially be attributed to varying PGM speciation within the different domains within the oxide horizon. More detailed domaining and complimentary metallurgical within discrete oxide horizons will be required to gain a more complete understanding of metallurgical performance.

13.5.13 Size-by-Size Analysis

A granulo-chemical assessment was done of the oxide bulk sample used for the Phase 2 leaching program to better understand the deportment of metals by size fraction within the soil oxide domain (Table 13-36). A 20kg split sample was deagglomerated in a steel concrete mixer drum for 15min after which the sample was screened to varying size fractions from +1/4" to -38 µm

It was found that 77% Pd, 81% Pt, 75% and 51% Au by contained metal is in the +1mm fraction while the +1mm fraction accounts for 41% of the mass. Furthermore, the grade of +1mm fraction is approximately double that of the total sample.

These results demonstrate that there exists potential to optimize the envisaged oxide processing route by incorporating screening into the circuit. This could have a potential positive impact on equipment sizing, plant capital expenditure, feed grade and recovery. A trade-off investigation will be required to consider the benefit of this circuit relative to the potential loss of metal in the finer fraction. The high clay content of the oxide zone will likely be an impediment to screening operations and should be included in future investigation.

Table 13-36: Average grades on granulo-chemical of the bulk sample used for the Phase 2 leaching program

Sample	Mass	Mass	Grade Au	Au	Au	Grade Pd	Pd	pd
	(kg)	(%)	(g/t)	(mg)	(%)	(g/t)	(mg)	(%)
BRV01 - 1/4	3.79	21.91%	0.04	0.14	13.42%	6.52	24.70	28.06%
BRV01 - 1 mm	4.00	23.12%	0.04	0.17	16.05%	10.23	40.93	46.50%
BRV01 - 300 µm	1.18	6.82%	0.05	0.05	5.01%	4.78	5.64	6.41%
BRV01 - 150 µm	0.54	3.11%	0.04	0.02	2.03%	5.57	3.00	3.40%
BRV01 - 75 µm	0.57	3.29%	0.03	0.02	1.61%	3.08	1.75	1.99%
BRV01 - 45 µm	0.80	4.60%	0.25	0.20	18.78%	2.19	1.74	1.98%
BRV01 - 38 µm	0.24	1.41%	0.35	0.09	8.07%	2.04	0.50	0.57%
BRV01 - < 38 µm	6.18	35.74%	0.06	0.37	35.03%	1.58	9.77	11.10%
Total	**17.3**	**100.00%**	**0.06**	**1.06**	**100.00%**	**5.09**	**88.03**	**100.00%**

Sample	Mass	Mass	Grade Pt	Pt	Pt	Grade Rh	Rh	Rh
	(kg)	(%)	(g/t)	(mg)	(%)	(g/)	(mg)	(%)
BRV01 - 1/4	3.79	21.91%	6.13	23.23	25.45%	1.67	6.35	36.67%
BRV01 - 1 mm	4.00	23.12%	10.79	43.16	47.28%	2.03	8.11	46.83%
BRV01 - 300 µm	1.18	6.82%	4.46	5.26	5.76%	0.38	0.45	2.61%
BRV01 - 150 µm	0.54	3.11%	5.90	3.17	3.48%	0.59	0.32	1.84%
BRV01 - 75 µm	0.57	3.29%	4.24	2.41	2.64%	0.39	0.22	1.27%
BRV01 - 45 µm	0.80	4.60%	0.52	0.42	0.46%	0.37	0.29	1.70%
BRV01 - 38 µm	0.24	1.41%	0.14	0.03	0.04%	0.34	0.08	0.47%
BRV01 - < 38 µm	6.18	35.74%	2.20	13.60	14.90%	0.24	1.49	8.61%
Total	**17.30**	**100.00%**	**5.28**	**91.29**	**100.00%**	**1.00**	**17.3**	**100.00%**

Source: Bravo, 2025.

GE21 Consultoria Mineral

13.5.14 Tailings and Concentrate Thickening and Filtration

13.5.14.1 Westech – Fresh Rock Tailings and Concentrate

Bravo Mining contracted Westech (Swire Water) to conduct vacuum sedimentation and filtration tests on fresh rock tailings and concentrate material. Sample material was sourced from the mini plant test work program which provided sufficient material for such test work (Table 13-37). Based on these results vacuum filtration of flotation concentrates is unlikely to achieve acceptable shipping moisture contents. Future work will instead focus on pressure plate and frame filtration.

Table 13-37: Vacuum sedimentation and filtration test material

Description	Tailings Sample	Concentrate Sample
Solids Concentration (%)	35	17
Solids Density (t/m^3)	2.50	2.86
Liquid Density (t/m^3)	1.00	1.00
Pulp Density (t/m^3)	1.26	1.12
P80 (μm)	Ca. 25	Ca. 20

Source: Bravo, 2025.

Tests included dilution optimization, flocculant dosage optimization, sedimentation, and filtration tests. A summary of the results is presented in Table 13-38 and Table 13-39.

Table 13-38: Vacuum sedimentation and filtration test results

Description	Tailings Sample	Concentrate Sample
Optimum Dilution	7.0 % Solids	4.6 % Solids
Best Flocculant	Magnafloc 10 (BASF)	Magnafloc (BASF)
Best Dosage	50 g/t	40 g/t
Concentrate Underflow	46.85 % Solids	45.81 % Solids
Unit Area	0.45 m^2/tpd	0.67 m^2/tpd

Source: Bravo, 2025.

Table 13-39: Vacuum sedimentation and filtration test results

Description	Tailings Sample	Concentrate Sample
Fabric Selection	7μm aperture	20μm aperture
Cake Moisture	32.0 %	35.0 %
Vacuum Pressure	21 – 22 “Hg	18 – 20 “Hg
Cake Thickness	15.0 mm	15.0 mm
Filtration Rate	0.440 t/h.m^2	0.700 t/h.m^2

Source: Bravo, 2025.

13.5.14.2 Watershed Laboratories – Oxide Tailings

Watershed Laboratories in Brazil conducted thickening, water recovery and density profile evaluation tests on oxide tailings material. Tailings material was composited from available tailings at TESTWORK Laboratories from Phase 2 test tailings.

Free drainage, compressive drainage and compressive densities were evaluated in three tests and various flocculant dosages (300/500/800 g/t) (Nalco 7763, emulsion polymer).

Samples were tested at the treatment pulp density of 29-30%. Total water recovery of 70% was achieved with a thickened product density of 2.80 t/m^3 and water turbidity of 316 NTU of suspended solids (or 100 mg/l).

The best results were achieved with 500 g/t and the 300 g/t test. Dosage at 800 g/t negatively affected all parameters.

Test parameters and results are summarised in Table 13-40.

Table 13-40: Watershed in oxide tailings tests parameters

Sample - FEED				
Test	**#**	**1**	**2**	**3**
Original slurry (as received)	solids %	28.6%	28.6%	28.6%
Product	code	7,763	7,763	7,763
	concentration %	1.00	1.00	1.00
Dilution	V/V	40%	40%	40%
Dose	g/t	300	500	800
Free Drainage	solids %	38.8%	41.7%	30.0%
Compressive Drainage	solids %	54.2%	57.3%	43.3%
Compressive Density	g/cm3	2.63	2.8	
Water Recovery based on Water in the Feed Sample				
Free Drainage Water Recovery	%	37.0%	44.1%	6.8%
Compressive Drainage Water Recovery	%	29.2%	26.1%	40.9%
Total Water Recovered	%	66.2%	70.2%	47.7%
Feed Sample	g	640	640	640
Feed Solids	g	182.9	182.9	182.9
Feed Water	g	457.1	457.1	457.1
Free Drainage Water Recovery Free Drainage on screen	g	169.13 470.87	201.64 438.36	31.08 608.92
Compressive Drainage Water Rec Compressive Drainage - Plug	g g	133.51 337.36	119.34 319.01	186.84 422.08
Ballance checking		640	640	640
		0.00	0.00	0.00

Source: Bravo, 2025.

13.5.15 Pyrometallurgy

A 68 g/t concentrate sample from Luanga was sent to Arxo Metals in South Africa for roasting and smelting tests. Final report pending while assay results have been received.

The concentrate sample was roasted at 1000°C for 120 min which effectively reduced sulphur and other volatile content to 1.23% (target below 2% for reductive smelting).

Bravo roasted concentrate was bulked with a chemically matched concentrate from Tharisa Minerals to meet the furnace charge requirement.

The smelting test in a 35 kVA DC furnace produced a Ni-Fe alloy with 716 g/t 4E PGM, while slag PGM concentration was below the detection limit for all precious metals, suggesting +99% recovery. Slag Ni was below the detection limit, while Cu concentration was reported as 1%, suggesting base metal recoveries of +99% and 99%, respectively and no alloy entrapment in slag.

Some results are shown in Table 13-41, Table 13-42, and Figure 13-13.

Table 13-41: PGM summary of results

Summary of Results:										
	4E (g/t)	Fe	Al	Cr	Ca	Cu	Ni	Si	Mg	S
Bravo PGM conc	68.15	11.35%	1.72%	0.21%	2.86%	1.12%	3.47%	22.94%	14.90%	5.11%
Roasted Bravo PGM Conc	70.99	11,37%	1.96%	0.23%	3.00%	1.43%	3.82%	23.29%	16.47%	1.23%
Tharisa PGM Conc	71.84	7,37%	2.33%	2.13%	1.99%	o. 29%	0.38%	27.51%	18.16%	0.73%
Combined PGM Conc	71.66	8,50%	2.59%	1.73%	2.40%	0,47%	0.86%	23.33%	15.25%	0.84%

Source: Bravo, 2025.

Table 13-42: Smelting (35kVA DC R&D Furnace)

Smelting (35kVA DC R&D Furnace)						
	% Alloy fall	Upgrade	Assays			
				Feed	Alloy	Slag
PGM Combined Conc 10kg (Bravo:Tharisa 2.1:7.9)	8.0	9.0	4E (g/t)	82.1	716.8	ND
			% Cu	1.7	2.3	0.5
			% Ni	0.4	9.7	0.1
			% Fe	8.8	77.5	7.5
			% Cr	2.2	0.6	2.1
			Total mass (kg)	10	0.8	9.3

Source: Bravo, 2025.

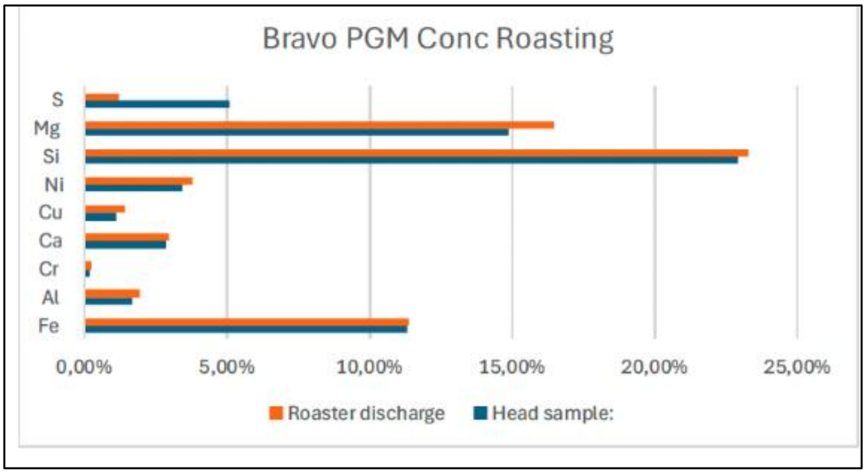


Figure 13-13: Bravo PGM conc roasting

Source: Bravo, 2025.

13.6 MRE Recommendation

The basis of recommendations for metallurgical inputs into the determination of Mineral Resources are from validated, reproduced historical metallurgical test results, newly generated tests results and external comparable results, where deemed “reasonable” for Fresh Material Flotation.

Vale, through external independent laboratories (SGS Lakefield, Mintek) and internal development work, reported their conclusion, that 74% of PGM could reasonably and economically be recovered at a LOM average feed grade of 1.24 g/t PGM+Au with demonstrated concentrate grades of 80 – 150 g/t.

Bravo, through its 2022/2023 metallurgical program has reproduced and validated the nature of the historical results achieved by Vale and elucidated further areas of potential improvement.

For the purposes of Mineral Resource estimation, the recovery numbers were simplistically modelled for tests across the grade profile and achieved concentrates of saleable grades (80 - 90 g/t). Concentrate quality considerations were based on concentrates from one operating and one development project and their qualities and terms for delivery to two separate Southern African smelters.

In the 2023 MRE, recovery estimate values were adjusted to accommodate for improvement demonstrated on Luanga recoveries through exploratory ultrafine hydrodynamic cavitation tests. An improvement of 6.1% was achieved at rougher stage flotation. This result replicated improved recoveries reported from Rustenburg tailings retreatment facilities at materially lower grades (0.69 g/t) of 5% at rougher stage. Global recovery improvement stabilized at 10% under these reported results.

For the 2025 MRE and PEA, the Luanga metallurgical model has removed any benefit previously attributed from additional fines flotation technology application assumptions and relied only on unadjusted performance recovery numbers. Bravo was not able to complete additional circuit tests aside from the initial rougher exploratory tests due to equipment availability and may update the metallurgical model at a later stage when such benefit can be demonstrated throughout the circuit, especially into the cleaner cycle. Previous investigation of cleaner tails at Luanga has demonstrated that +50% of fines losses to tails were liberated PGM or PGM attached to liberated base metal sulphides underscoring the potential to further improve flotation performance through the recovery of this fine fraction.

The average concentrate grade of tests under consideration equaled 174 g/t PGM+Au. Maintaining a saleable concentrate grade target of 80-90 g/t PGM+Au, additional recovery benefits may be realized by adjusting flotation times and/or reagent aggressiveness. For the 2023 MRE, an average recovery benefit was attributed to tests that achieved above-target concentrate grades. For the 2025 MRE, a more quantitative approach was taken, with high grade concentrates linearly regressed to target grade and improved recoveries measured accordingly. Concentrates that were near target grade and up to approximately 120 g/t were not adjusted.

The current metallurgical model shows recoveries of ca. 75 – 84% across a feed grade of 0.9 – 7.0 g/t PGM+Au for concentrates above 80 g/t, slightly lower than the 2023 MRE estimate of 76 – 85% (Figure 13-14).

Results from the mini-plant have been appropriately incorporated into the metallurgical assumptions by regressing recoveries generated to target concentrate grade. A qualitative talc abundance map was modelled for the Southwest Sector, which represents a minor portion of the overall MRE (see Section 14 below), and treated as a unique geological domain, to which the regressed recoveries were applied directly and separately from the global recovery assumptions generated for the rest of the Luanga deposit. The resultant recoveries for the localised high-talc domain are assumed to be Pd 51%, Pt 56%, Rh 27%, Au 27%.

The graph below demonstrates the modelled metallurgical input parameters for the purposes of the 2025 Luanga Mineral Resource Estimate. The logarithmic model trend line has a formula of:

$$y = 4.317\ \ln(x) + 77.656$$

where, y = (recovery) and x = (material grade)

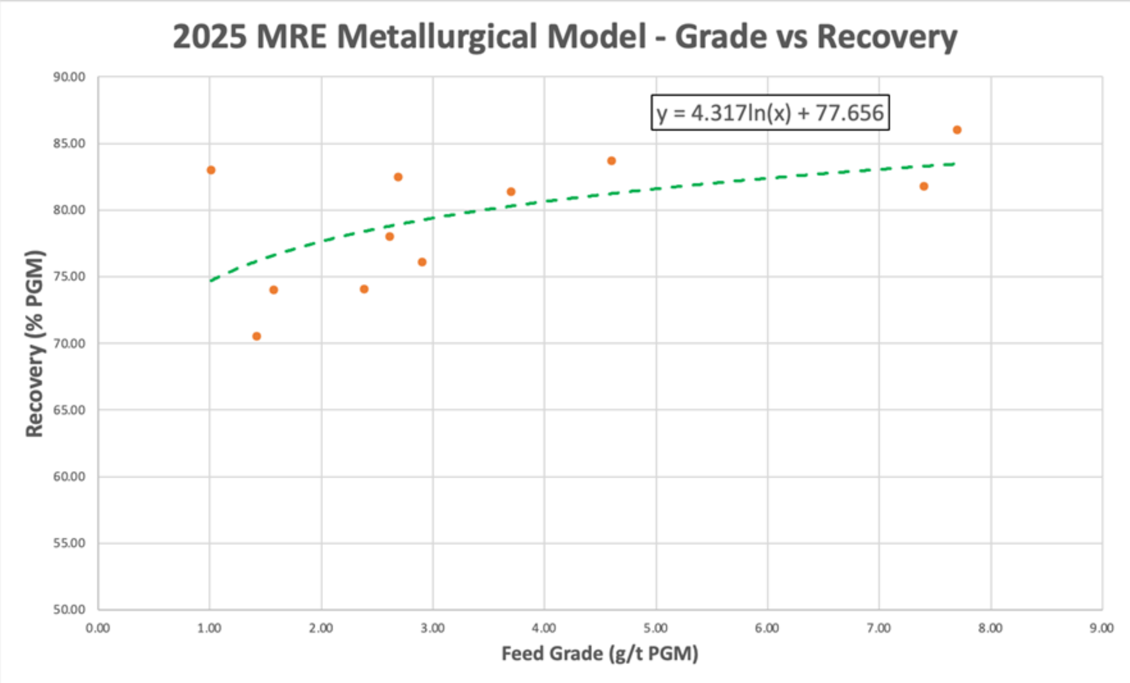


Figure 13-14: MRE metallurgical model – grade vs recovery

Source: Bravo, 2025.

Nickel recoveries have been demonstrated through Vale locked cycle tests to be achievable with between 45 and 47% recoveries at a feed grade of 0.19 %. Recent locked cycle test work performed by Bravo at Base Metal Labs demonstrated 62% Ni recovery in both sets of North Sector tests at feed grades of 0.21% Ni. Locked cycle test 34 on the Central Sector global composite demonstrated a 44% recovery from a 0.3% Ni feed grade while producing the highest Ni grade in concentrate yet seen at Luanga at 16% Ni in concentrate (plus 202 g/t PGM in concentrate). While these results clearly demonstrate the further potential to improve Ni recoveries at Luanga, in the 2025 MRE and PEA, the Company has maintained the 50% recovery assumption from the 2023 MRE until these results can be demonstrated more conclusively (Table 13-43).

Table 13-43: MRE Recommendation fresh rock global recovery

MRE Recommendation Fresh Rock Material	4E PGM	Pt	Pd	Rh	Au	Ni
Global Recovery (2 g/t PGM feed)	78 %	81 %	77 %	51 %	48 %	50 %
MRE Recommendation High-Talc Domain	**Pt**	**Pd**	**Rh**	**Au**	**Ni**	
Global Recovery (2 g/t PGM feed)	51 %	56 %	27 %	27 %	0 %	

Source: Bravo, 2025.

13.6.1 Oxide Material Leaching

The hydrometallurgical recoveries achieved to date demonstrate relatively high and consistent recoveries for Pd and Au, with lower and more variable recoveries for Pt and Rh. The recoveries have been verified through multiple leaching tests and the conceptual processing flowsheet has been validated with tests including PGM solubility in the presence of cyanide at ambient temperature and pressure, a PGM adsorption onto carbon, and final product generation as saleable, high grade, PGM residue. The current data demonstrates the probability for economic recovery of PGM+Au from oxide material at Luanga through conventional sodium cyanide leaching and carbon-in-leach extraction.

For the purposes of the 2023 MRE, individual best element recoveries based on exploratory tests were used to inform the metallurgical input assumptions.

Input assumptions for the 2025 MRE and PEA have relied on data from extensive follow-up parameter investigation test work, which has resulted in improved assumptions for Pd, similar assumptions for Au and more conservative assumptions for Pt and Rh.

Bravo has relied on average recoveries from relevant parameter tests instead of individual best element recoveries as the project can now rely on a larger, more comprehensive data set of results from which to evaluate potential recoveries. (Table 13-44).

Table 13-44: MRE recommendation oxide global recovery

	Au	Pd	Pt	Rh
Global Recovery (1-3 g/t)	90 %	81 %	23 %	54 %

Source: Bravo, 2025.

14 MINERAL RESOURCE ESTIMATES

GE21 carried out the 3D geological modelling, statistical and geostatistical studies, and grade estimate for the Project, assessing a set of factors, including the amount, and spacing of available data, interpreted mineralization controls, mineralization style, and quality of used data.

Geological modelling and estimation were performed using Leapfrog 2024.1 and Isatis.Neo, respectively. The UTM Projection – Zone 22 South, Datum: SIRGAS 2000 was adopted as a reference for the database of this work.

14.1 Drilling Database

The drilling database supplied by Bravo was visually validated, considering the relationship between tables, gaps, overlaps, and the absence of essential information. Using Leapfrog Geo software, GE21 also validated the Collar, Survey, Assay, and Lithology tables. No relevant inconsistencies were identified in this stage of the work since this was verified in the Data Verification stage.

GE21 used Trench channel (TRC) samples and Diamond drill hole (DDH) core samples from both Vale and Bravo campaigns, available on the Effective Date. Table 14-1 summarizes the drilling database used in this project stage. The map shown in Figure 14-1 shows the spatial distribution of the holes used.

Table 14-1: Drill holes summary

Company	Type	Count	Length (m)
Bravo	DDH	295	62,952
	TRC	45	8,714
	TOTAL Bravo	340	71,666
Vale	DDH	191	36,677
Overall Total		**531**	**108,343**

Source: GE21, 2025.

The database has geochemical results of the variables: Pd (ppm), Pt (ppm), Rh (ppm), Au (ppm), and Ni (ppm). Apart from these five elements of interest, an additional 35 elements were analyzed (Ag, Al, As, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, K, La, Li, Mg, Mn, Mo, Na, P, Pb, S, Sb, Sc, Sr, Sn, Th, Ti, Tl, U, V, W, Y, Zn, and Zr).

The database has analysis from Vale and Bravo campaigns, including Vale assays resampled by Bravo. The Mineral Resource Estimation (MRE) database contains data from the Vale campaign. This data was transformed using the correlation correction presented in Section 11.5.3. The assay results are quantitative for the presented variables, separated by sampling method and campaign, and are quantified in Table 14-2.

GE21 Consultoria Mineral

Table 14-2: Assays summary

Company	Variable	Diamond drill holes		Trenches	
		Number of Samples	Length (m)	Number of Samples	Length (m)
Bravo	Pd	62,606	62,223	9,355	9,070
	Pt	62,606	62,223	9,355	9,070
	Rh	62,606	62,223	9,355	9,070
	Au	62,606	62,223	9,355	9,070
	Ni	62,606	62,223	169	168
Vale	Pd	36,259	36,223	-	-
	Pt	36,259	36,223	-	-
	Rh	36,259	36,223	-	-
	Au	36,259	36,223	-	-
	Ni	36,259	36,223	-	-

	Number of Samples	Length (m)
Total PGM + Au	108,220	107,516
Total Ni	98,644	98,231

Source: GE21, 2025.

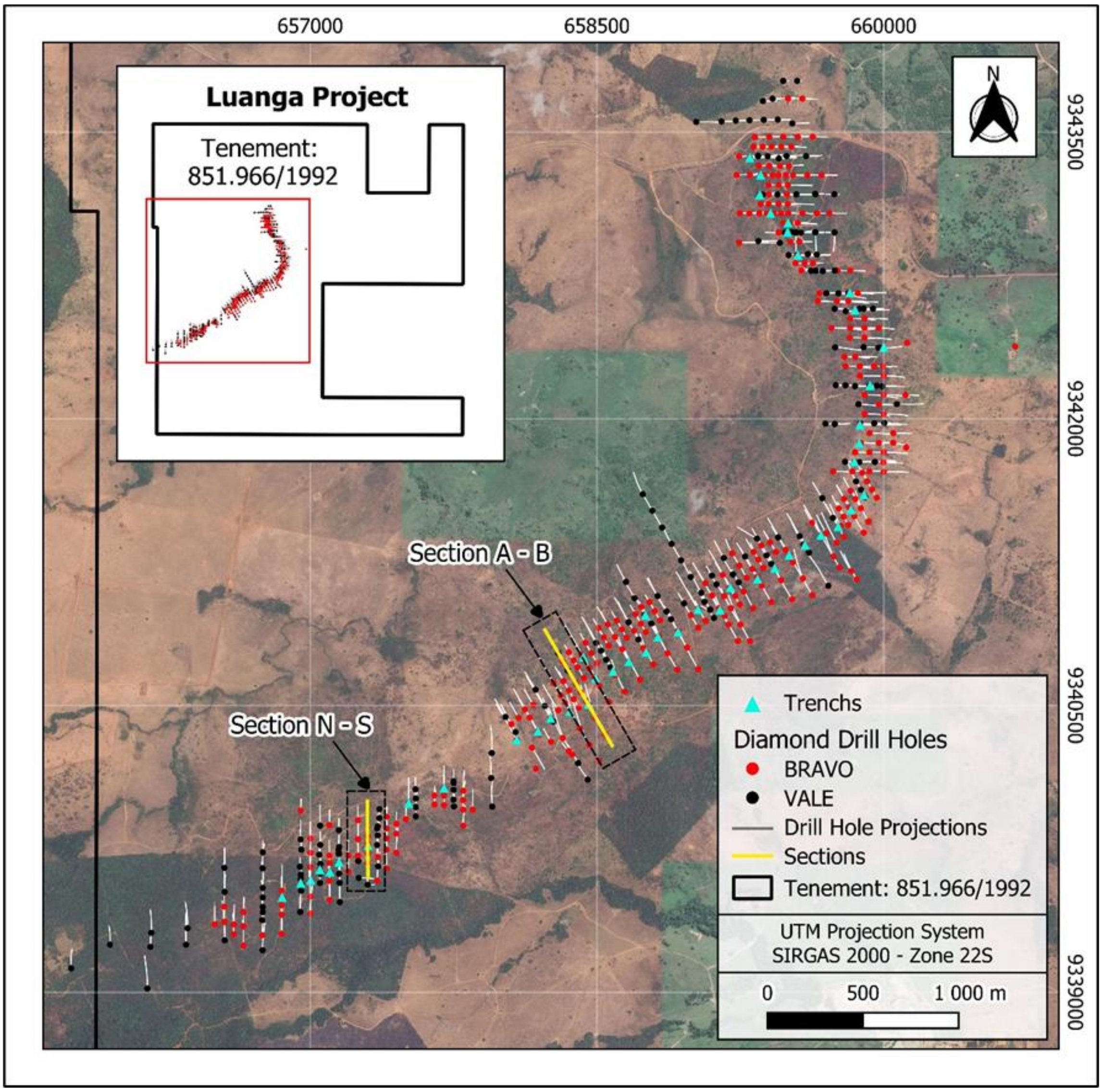


Figure 14-1: Map presenting the MRE database and the location of geologic models' sections

Source: GE21, 2025.

14.2 Geological Modeling

Four 3D models were generated. Each model spatializes different information regarding the mineralization and the host rocks. These models are:

- **Grade Shell Model**: defines mineralized grade shells of low grade (> 0.3 ppm PGM + Au) and high grade (> 1.0 ppm PGM + Au)
- **Regolith Model**: represents the weathering stages of the rock in the project area.
- **Estimation Domains**: combines the Grade Shell and the Regolith Models for a grade estimation.
- **Metallurgical Recovery Model**: defines zones with different metallurgical responses.

14.2.1 Grade Shell Model

Bravo's geologists interpreted the mineralized grade shell and lithology model along the 118 drill sections. The mineralized zone interpretation was modelled using continuity of PGM + Au grade. Two types of iso-grade surfaces were generated:

- Low Grade (LG): 3PGMs + Au > 0.3 ppm
- High Grade (HG): 3PGMs + Au > 1.0 ppm

The Mineralization Model was split into nine regions, generating 18 grade shell groups (9 LG + 9 HG). These groups are comprised of a total of 157 individual lenses. Figure 14-2 presents a section view of the Grade Shell Model. The regions of the mineralization are:

- North N
- North S
- Central HW
- Central Main
- Central FW
- Southwest N
- Southwest HW
- Southwest Main
- Southwest FW

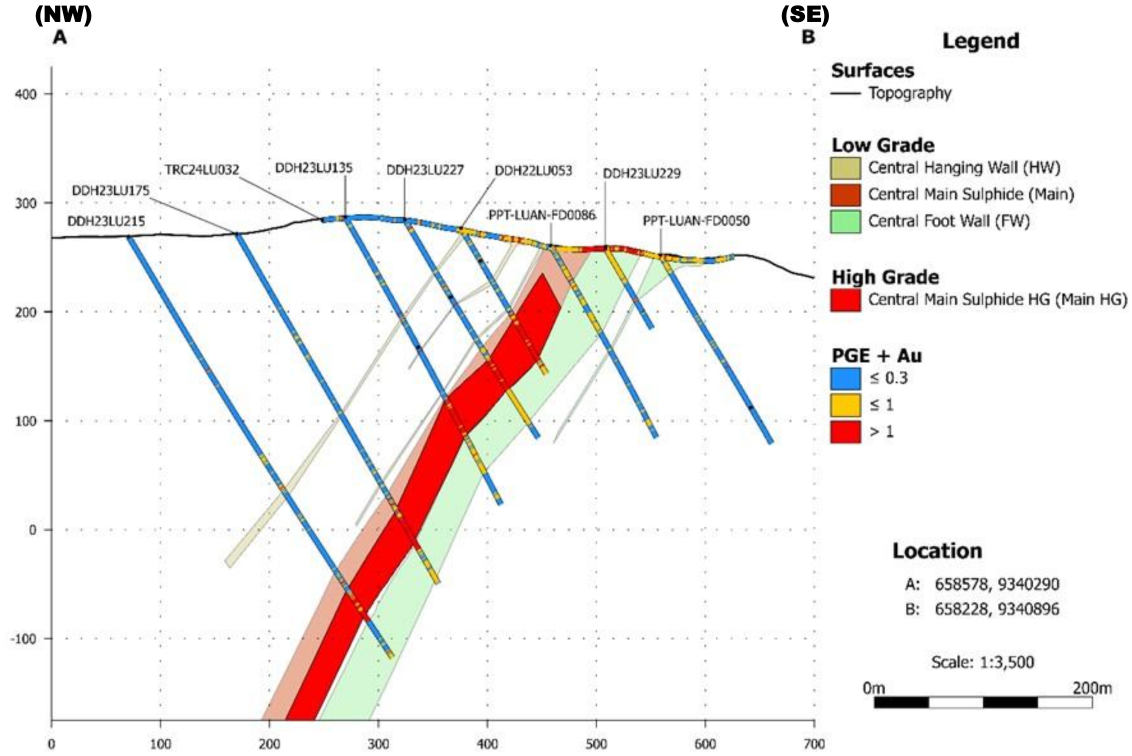


Figure 14-2: Grade shell model – section A-B – view of the central area

Source: GE21, 2025.

14.2.2 Regolith Model

The Regolith Model was built using the information on drill hole logs, RQD, and weathering types defined in the geological description. Given the metallurgical similarities and very small volumes of some, this was distilled to two defined zones (Figure 14-3):

- Oxidized Zone (Soil and saprolite): SO_SAP
- Fresh (Unweathered) Rock: FR

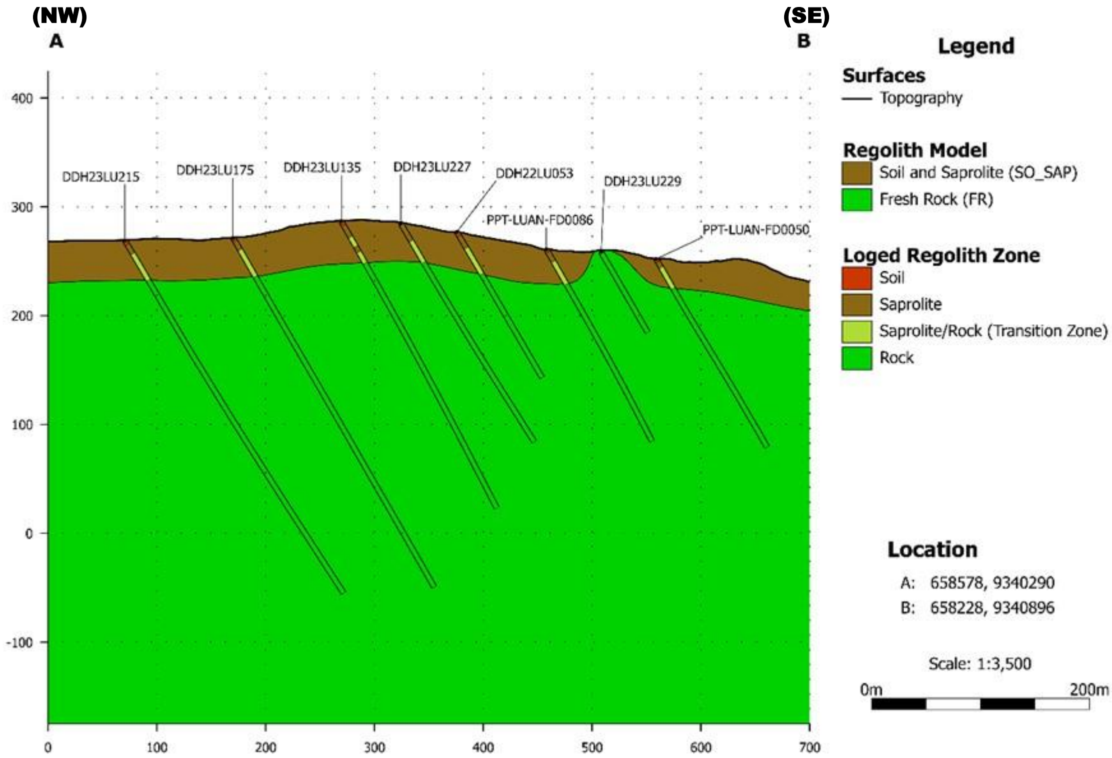


Figure 14-3: Regolith Model – Section A-B – View of the Central area

Source: GE21, 2025.

14.2.3 Estimation Domains

After the construction of all 3D models, the estimation domains were defined. The intersection of the Grade Shells with the Weathering Model generated 157 fresh rock lenses (HG + LG), that were grouped in main zones (Central, Southwest, North N and North S) with respect to the stationary domain (HG and LG). These zones were estimated separately (a total of 8 domains).

The upper zone of the Regolith Model, Soil and Saprolite (SO_SAP), intersection with the Grade Shells generated three additional Estimation Domains (Oxidation zones):

- SO_SAP_North
- SO_SAP_Central
- SO_SAP_Southwest

Chemical elements are estimated using different estimation methodologies according to the Weathering Model. Ordinary Kriging (OK) was applied to the Oxidized domain, while the Turning Bands Simulation was applied to fresh rock. The Stationary Domains (High grade and Low grade), presented in Table 14-3, were estimated individually.

GE21 Consultoria Mineral

Table 14-3: Summary of estimation domains and relationships between weathering model, stationary domain and estimation methodology

Domain Number	Weathering Model	Position	Stationary Domain	Estimation Method
1	Oxidized	Soil + Sap North	Soil + Sap North	Ordinary Kriging
2		Soil + Sap Central	Soil + Sap Central	
3		Soil + Sap Southwest	Soil + Sap Southwest	
4	Fresh	Central	LG (Low Grade)	Turning bands simulation
5			HG (High Grade)	
6		Southwest	LG (Low Grade)	
7			HG (High Grade)	
8		North S	LG (Low Grade)	
9			HG (High Grade)	
10		North N	LG (Low Grade)	
11			HG (High Grade)	

Source: GE21, 2025.

14.2.4 Metallurgic Recovery Model

The Metallurgic Recovery 3D model was constructed to define different recovery factors for zones with different mineralogical and geochemical setups. A total of three domains were individualized (Figure 14-4):

- High Talc Material (High Talc)
- Oxidized Material (Ox)
- Fresh Material (Fresh)

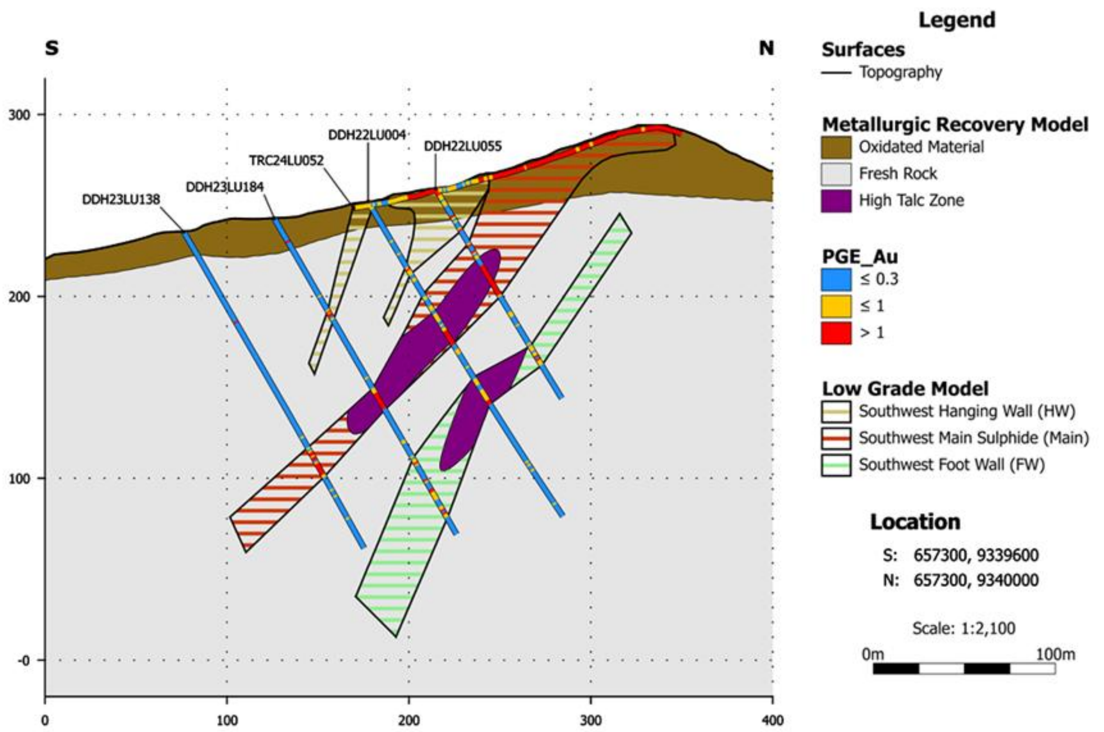


Figure 14-4: : Metallurgic Recovery Model and Low-Grade Shells – Section N-S – View of the Southwest area

Source: GE21, 2025.

14.2.5 QP Opinion

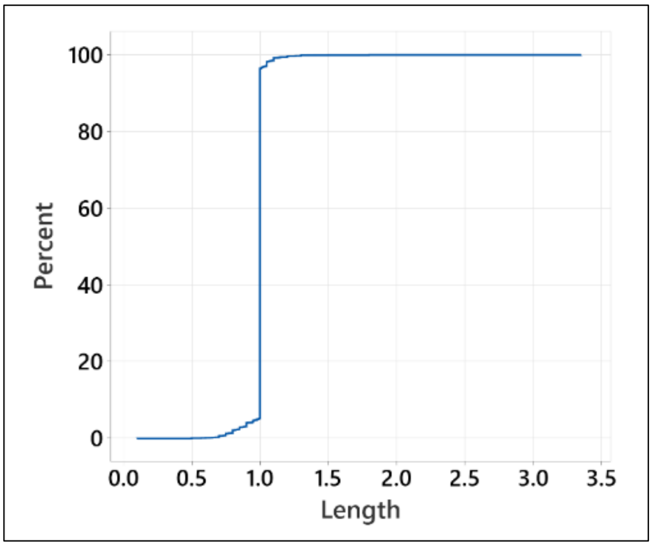
The QPs considers the geological interpretations and modelling to be adequate for a Mineral Resource Estimation study. The mineralized 3D bodies honour the mineralized intervals and have adequate continuity. Weathering and Oxidation models are consistent with drill hole logs and assays. QA/QC procedures follow the industry's practices.

The QP recommends, in further geological models, adopting an approach with implicit modelling methods and reducing domain internal dilution.

14.3 Statistical Analysis

14.3.1 Regularization of Samples

The analysis of the sample database showed that more than 97% of the drilling samples have a length of less than or equal to 1 meter. GE21 performed the regularization of samples inside the modelled domains. The chosen composite size is 1 meter. This composite was used for the complementary studies of statistics and geostatistics (Figure 14-5).



Parameter	Value
Count	107,449
Mean	0.995
SD	0.055
CV	0.056
Variance	0.003
Minimum	0.10
Q1	1.00
Q2	1.00
Q3	1.00
Maximum	3.35

Figure 14-5: Un-composited assay interval length statistics

Source: GE21, 2025.

14.3.2 Support Correction of Trench Dataset

The geostatistical concept of support is associated with average values in volumes, positions, areas and lengths that have the same statistical population. Different analytical and sampling methodologies could represent different support, since present differences in variability due to the precision of these methods. To merge the databases of different sampling methodologies, a support correction was applied to trench data, converting it to the same population as diamond drill holes. Figure 14-6 shows the process for support correction of the trench dataset using the Discrete Gaussian Model. Transformation is performed in Pd, Pt, Au and Rh values. Ni values are not present in trench samples.

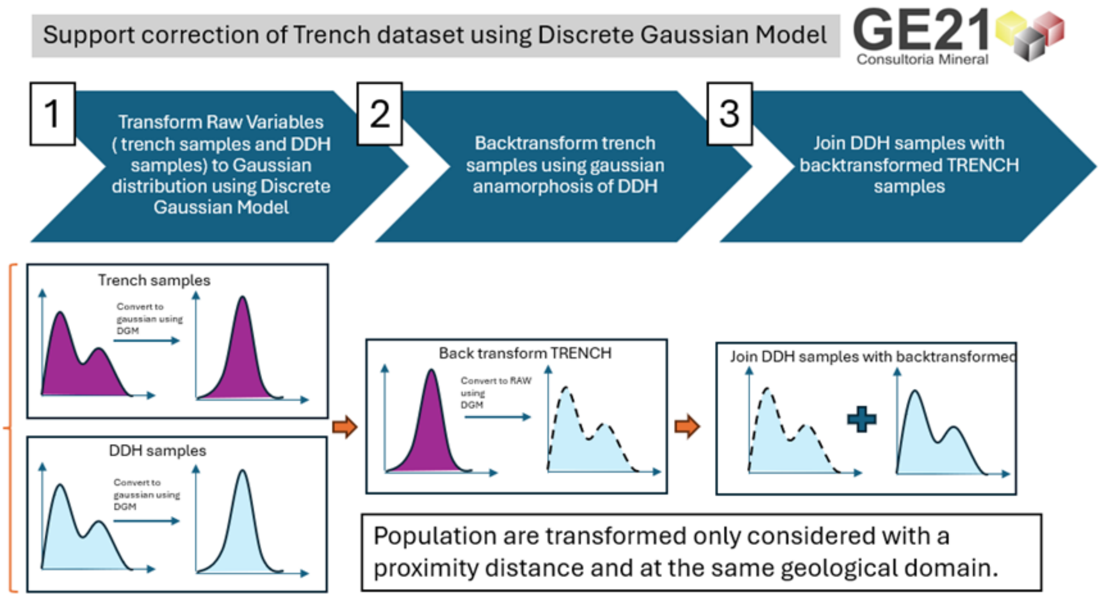


Figure 14-6: Process of support correction of trench dataset using discrete Gaussian model

Source: GE21, 2025.

A boundary of 50m from Trench data and DD data was used to transform the trench variables. Only samples inside soil and saprock (SO_SAP) were considered for this transformation.

The dataset used for support correction of drill holes, and trenches are shown in Figure 14-7.

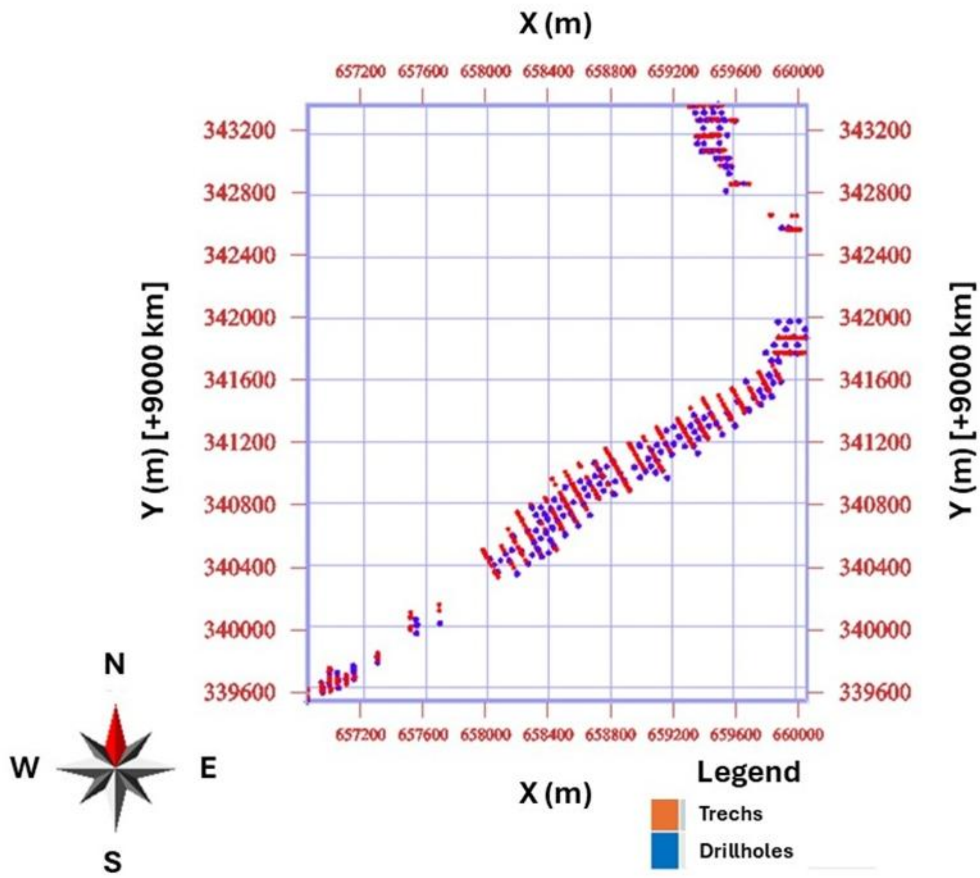


Figure 14-7: Dataset used for transformation of trench data and DDH

Source: GE21, 2025.

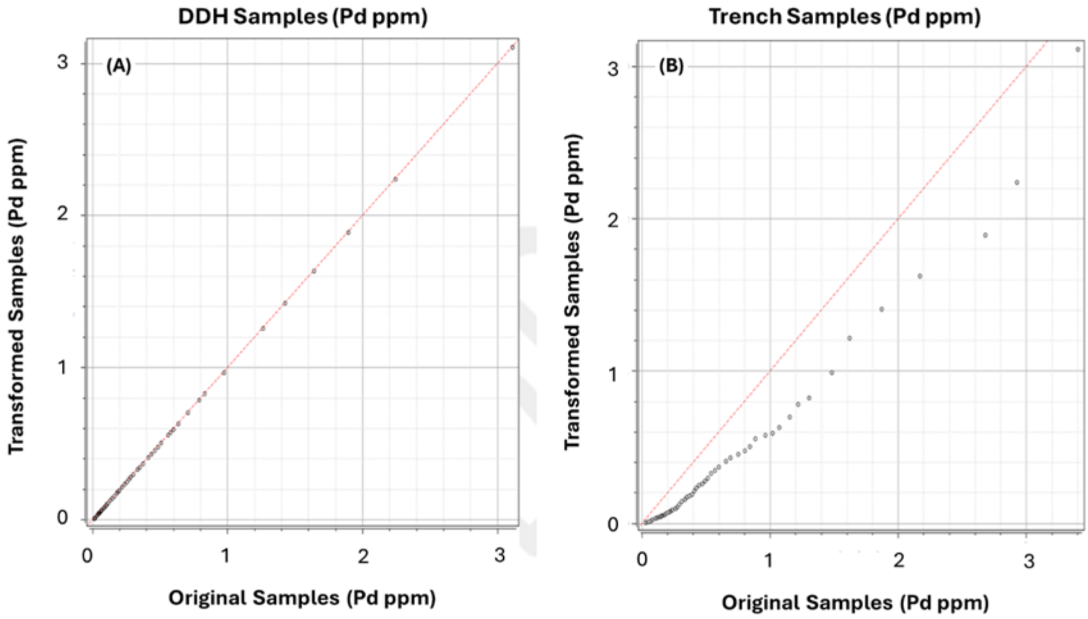


Figure 14-8: QQ plot of transformed and raw Pd values (trench data and DDH data)

Legend: A) QQ-Plot of DDH raw and transformed samples. B) QQ-Plot of TRENCH raw and transformed samples.
Source: GE21, 2025.

Table 14-4 shows the statistics for transformed and original composite data.

Table 14-4: Statistical for transformed and composite dataset

Variable	Unit	Count	Mean	Standard Deviation	Coefficient of variation	Minimum	Maximum	Quantile 25%	Quantile 50%	Quantile 75%
Au	ppm	30,035	0.04	0.09	2.49	0.00	3.91	0.00	0.01	0.03
Au transformed	ppm	30,035	0.04	0.10	2.51	0.00	3.91	0.01	0.01	0.03
Pd	ppm	30,035	0.66	1.70	2.57	0.00	156.29	0.18	0.33	0.69
Pd transformed	ppm	30,035	0.66	1.80	2.75	0.00	156.29	0.18	0.33	0.67
Pt	ppm	30,035	0.47	1.83	3.92	0.00	158.09	0.13	0.24	0.47
Pt transformed	ppm	30,035	0.47	1.95	4.10	0.00	158.09	0.13	0.23	0.46
Rh	ppm	30,035	0.06	0.25	4.21	0.00	17.33	0.00	0.02	0.05
Rh transformed	ppm	30,035	0.06	0.29	4.81	0.00	17.33	0.00	0.02	0.05

Source: GE21, 2025.

14.3.3 Exploratory Data Analysis (EDA)

Drilling samples composite statistical analysis was performed for Pd (ppm), Pt (ppm), Rh (ppm), Au (ppm) and Ni (ppm) variables. The EDA was performed using the sample composites and considering the estimation domains presented in Section 14.2.3. Table 14-5 shows the basic statistics for these variables.

Figure 14-9 to Figure 14-13 show the box plots for the Pd (ppm), Pt(ppm), Rh (ppm), Au (ppm) and Ni (ppm) variables.

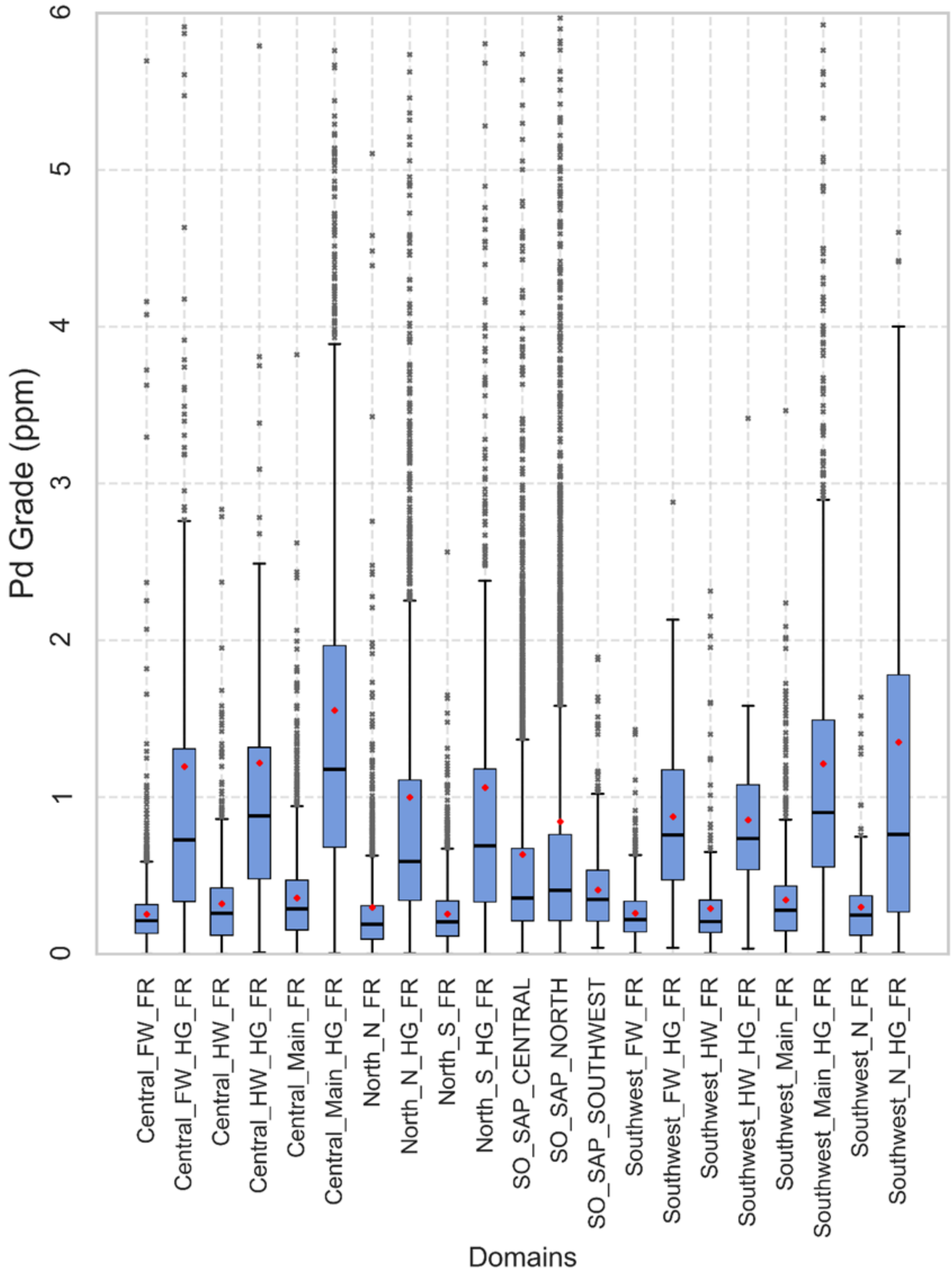


Figure 14-9: Pd ppm box plot chart by domains

Source: GE21, 2025.

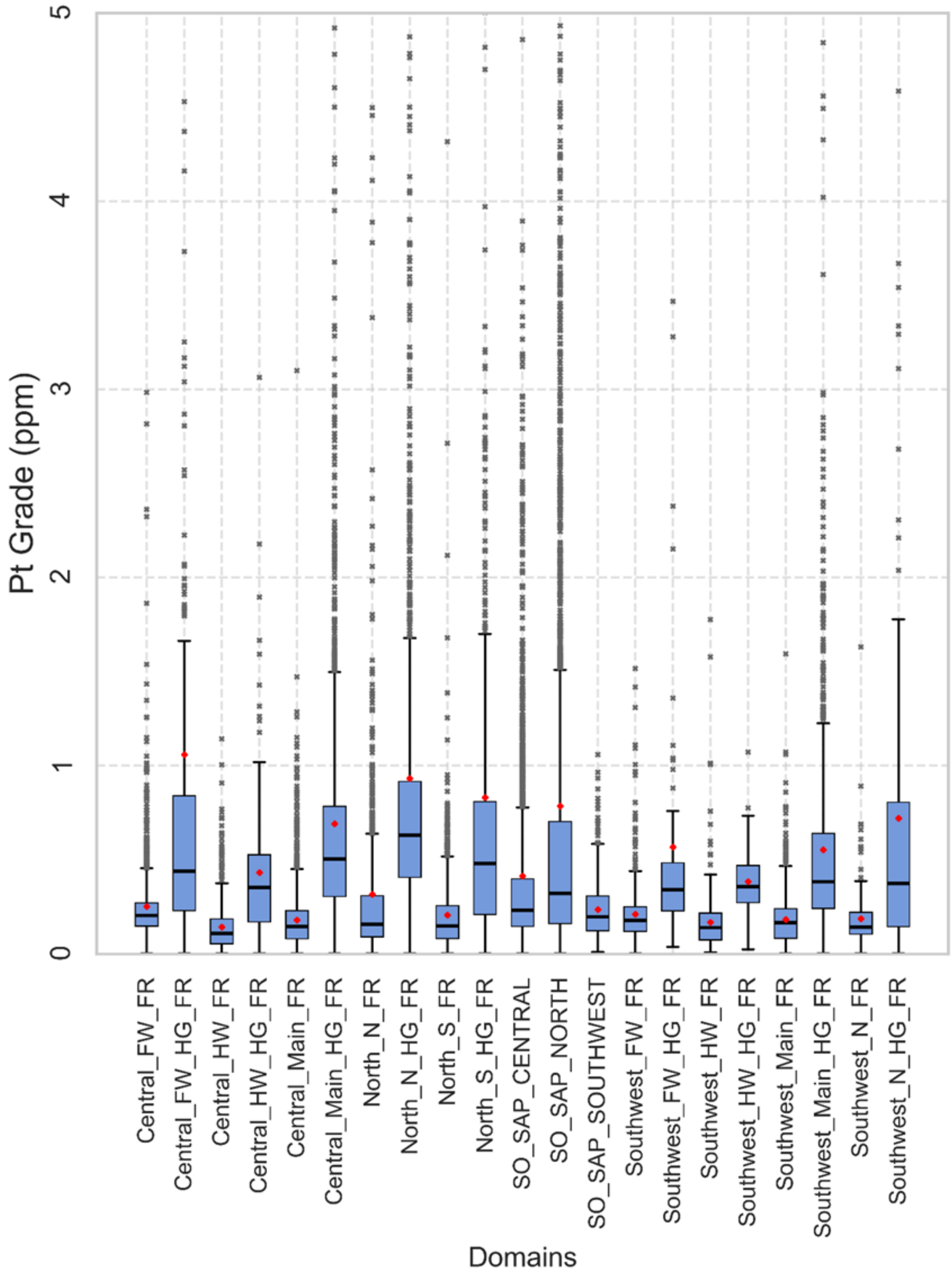


Figure 14-10: Pt ppm box plot chart by domains

Source: GE21, 2025.

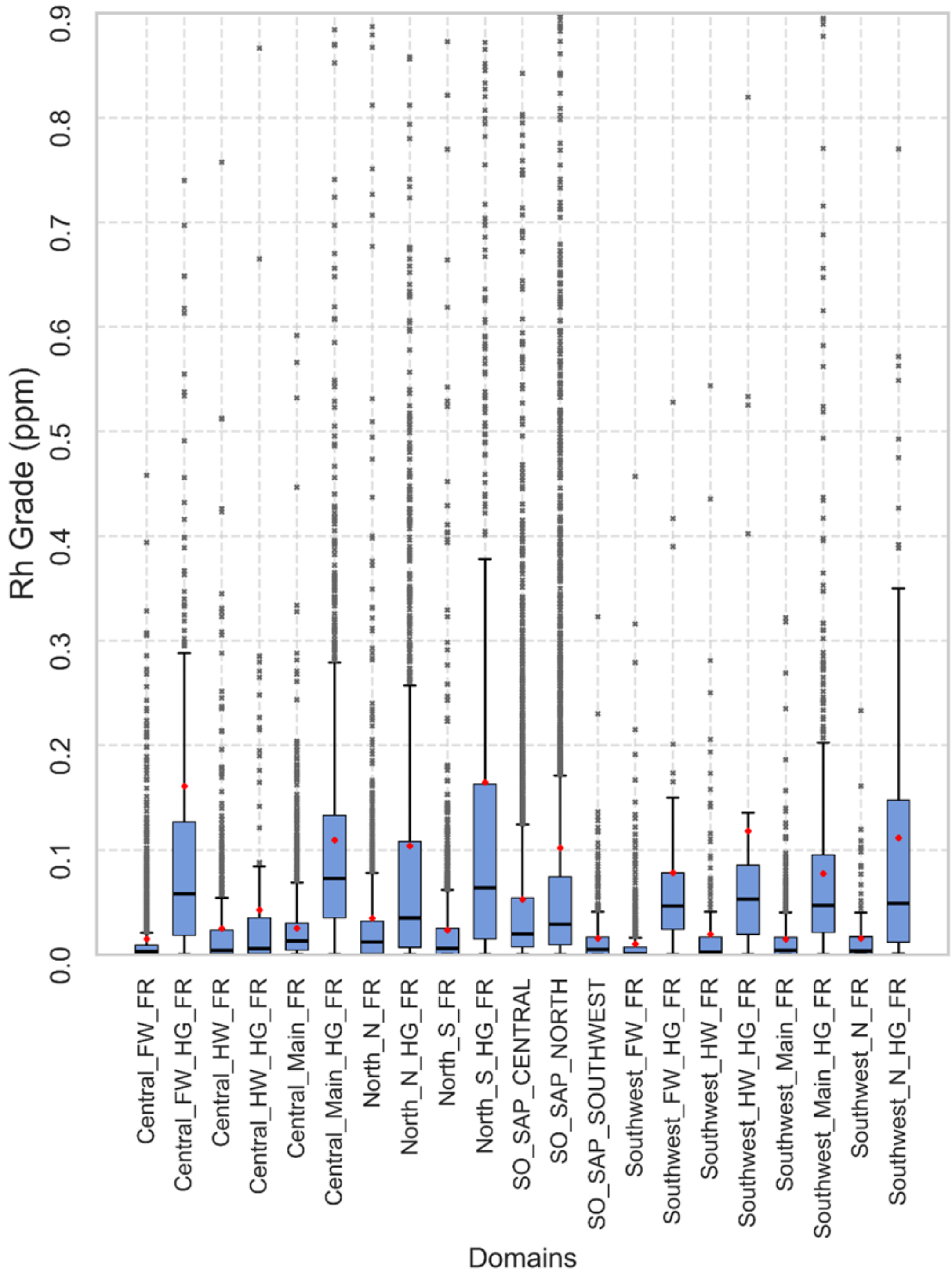


Figure 14-11: Rh ppm box plot chart by domains

Source: GE21, 2025.

GE21 Consultoria Mineral

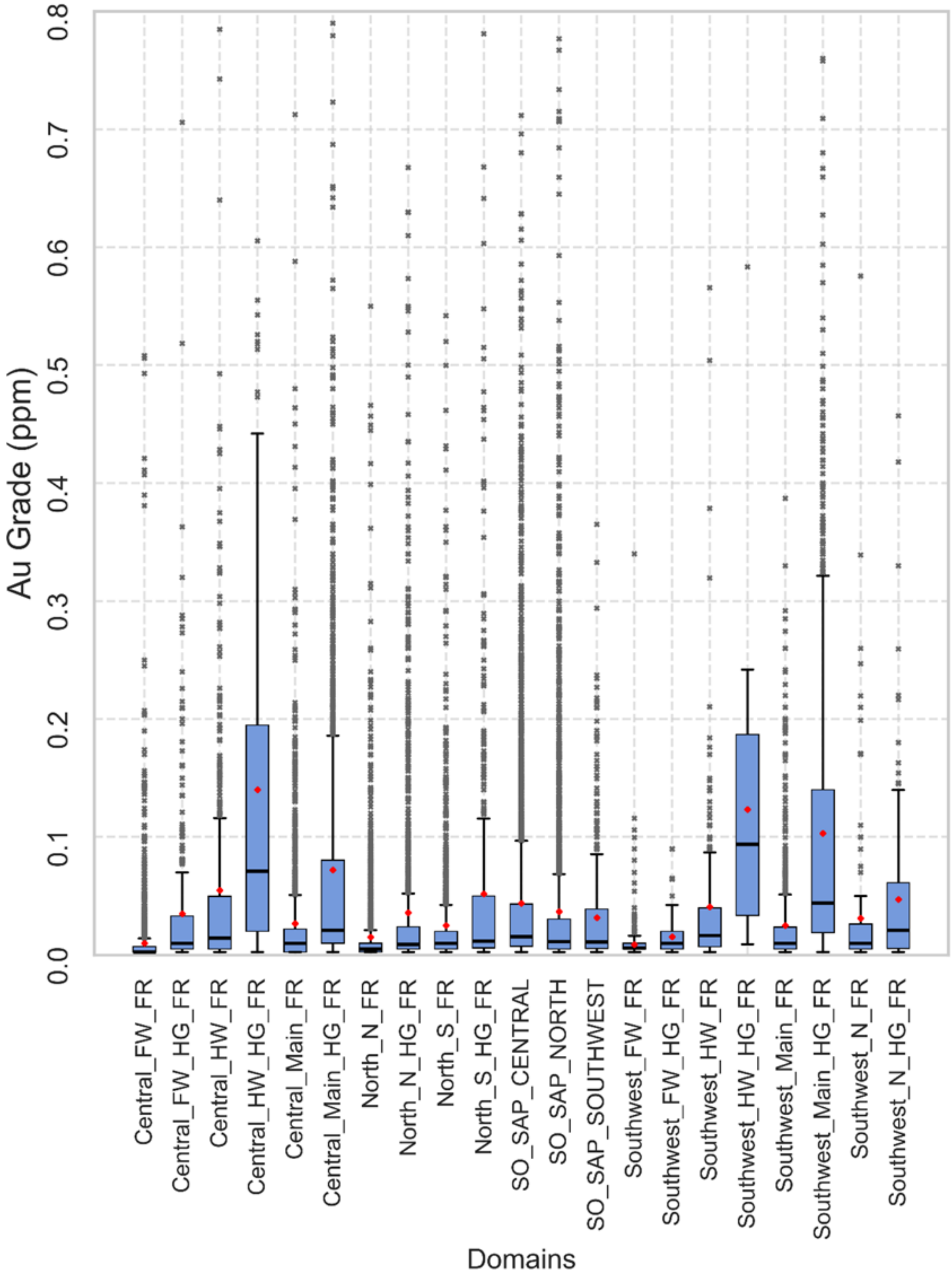


Figure 14-12: Au ppm box plot chart by domains

Source: GE21, 2025.

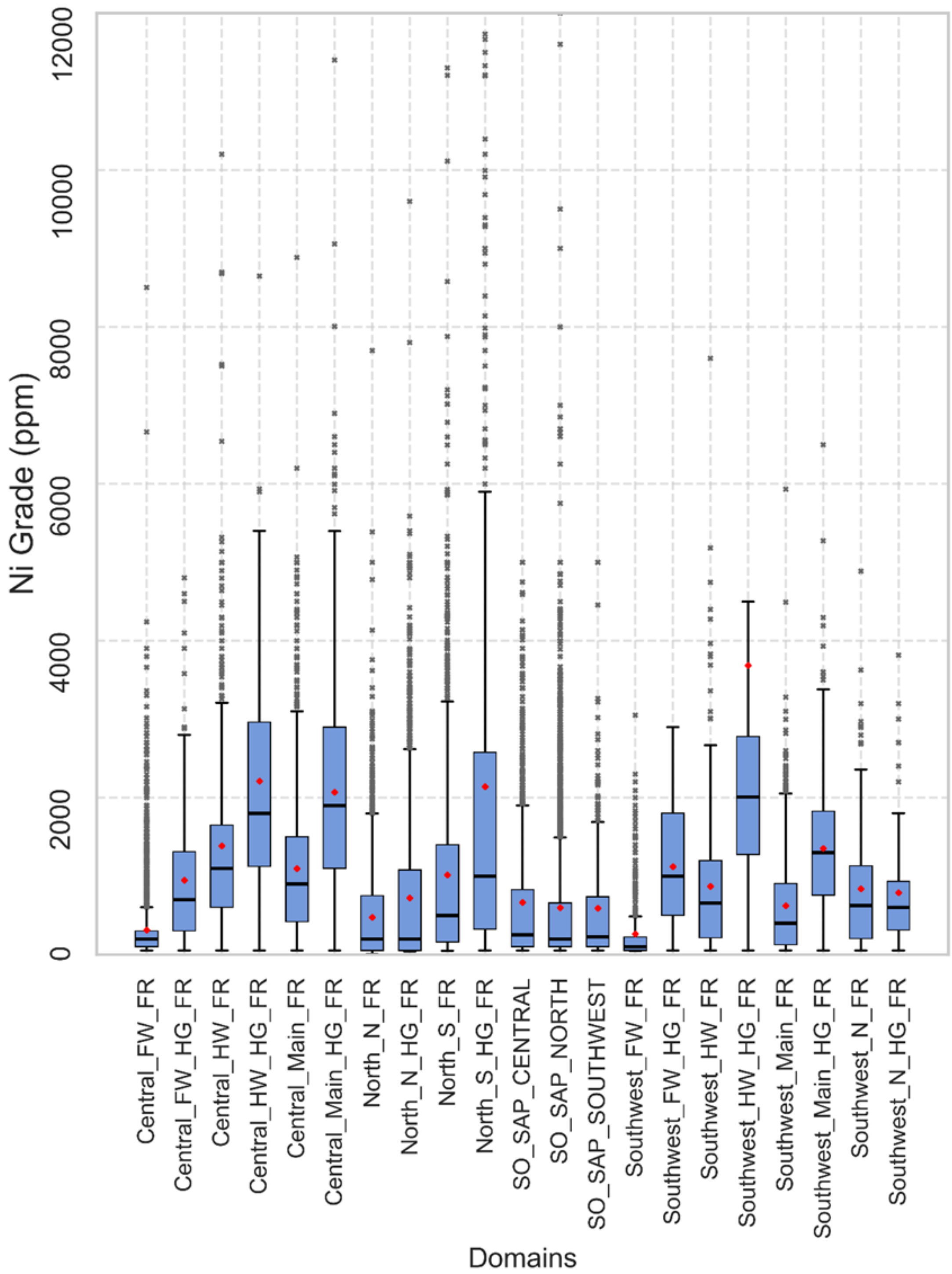


Figure 14-13: Ni ppm box plot chart by domains

Source: GE21, 2025.

Table 14-5: Basic statistics of Pd, Pt, Rh, Au, and Ni in the domains

Variable	Domain	Length (m)	Mean	S.D.	C.V.	Var.	Min.	Q25	Q50	Q75	Max.
Pd (ppm)	Central_FW_FR	2,986.3	0.253	0.328	1.297	0.108	0.0005	0.132	0.213	0.315	12.436
	Central_FW_HG_FR	374.3	1.189	2.864	2.410	8.204	0.0005	0.334	0.727	1.304	50.965
	Central_HW_FR	883.0	0.320	0.305	0.953	0.093	0.0005	0.119	0.260	0.420	2.835
	Central_HW_HG_FR	162.6	1.225	1.446	1.180	2.092	0.0100	0.479	0.882	1.317	8.764
	Central_Main_FR	1,895.0	0.357	0.347	0.972	0.121	0.0005	0.153	0.289	0.469	7.250

GE21 Consultoria Mineral

Variable	Domain	Length (m)	Mean	S.D.	C.V.	Var.	Min.	Q25	Q50	Q75	Max.
	Central_Main_HG_FR	1,915.7	1.553	1.750	1.126	3.061	0.0025	0.679	1.180	1.970	38.374
	North_N_FR	2,990.5	0.296	2.870	9.681	8.239	0.0010	0.093	0.190	0.307	156.291
	North_N_HG_FR	1,779.4	0.998	1.425	1.428	2.031	0.0025	0.341	0.591	1.109	20.233
	North_S_FR	1,776.3	0.255	0.209	0.821	0.044	0.0005	0.113	0.206	0.338	2.563
	North_S_HG_FR	739.7	1.060	1.360	1.283	1.850	0.0005	0.330	0.690	1.175	11.333
	SAP_SO_CENTRAL	5,545.7	0.745	1.367	1.834	1.868	0.0025	0.224	0.405	0.820	36.293
	SAP_SO_NORTH	4,820.5	0.846	2.570	3.037	6.604	0.0025	0.212	0.407	0.762	97.162
	SAP_SO_SOUTHWEST	891.3	0.420	0.317	0.754	0.100	0.0051	0.222	0.350	0.530	3.299
	Southwest_FW_FR	771.6	0.260	0.181	0.698	0.033	0.0025	0.139	0.220	0.337	1.430
	Southwest_FW_HG_FR	116.4	0.878	0.535	0.609	0.286	0.0400	0.472	0.760	1.174	2.880
	Southwest_HW_FR	327.7	0.290	0.306	1.057	0.094	0.0050	0.136	0.208	0.346	2.314
	Southwest_HW_HG_FR	28.5	0.859	0.663	0.771	0.439	0.0344	0.553	0.728	1.047	3.415
	Southwest_Main_FR	938.5	0.345	0.323	0.937	0.104	0.0005	0.145	0.280	0.433	3.465
	Southwest_Main_HG_FR	1,058.8	1.214	1.153	0.949	1.329	0.0079	0.554	0.902	1.499	9.913
	Southwest_N_FR	171.3	0.299	0.275	0.920	0.076	0.0025	0.119	0.248	0.374	1.637
	Southwest_N_HG_FR	135.6	1.358	1.704	1.255	2.903	0.0067	0.260	0.750	1.799	8.614
Pt (ppm)	Central_FW_FR	2,986.3	0.252	1.062	4.215	1.127	0.0025	0.148	0.205	0.271	54.012
	Central_FW_HG_FR	374.3	1.045	3.236	3.096	10.470	0.0028	0.227	0.440	0.840	39.685
	Central_HW_FR	883.0	0.143	0.135	0.947	0.018	0.0025	0.053	0.110	0.188	1.143
	Central_HW_HG_FR	162.6	0.430	0.416	0.969	0.173	0.0025	0.170	0.354	0.528	3.064
	Central_Main_FR	1,895.0	0.181	0.171	0.947	0.029	0.0025	0.080	0.146	0.230	3.100
	Central_Main_HG_FR	1,915.7	0.691	0.921	1.333	0.848	0.0025	0.305	0.505	0.784	18.189
	North_N_FR	2,990.5	0.317	2.937	9.260	8.624	0.0025	0.090	0.158	0.309	158.089
	North_N_HG_FR	1,779.4	0.930	1.697	1.825	2.880	0.0025	0.404	0.631	0.914	42.494
	North_S_FR	1776.3	0.206	0.448	2.173	0.201	0.0043	0.082	0.150	0.257	17.055
	North_S_HG_FR	739.7	0.832	1.782	2.141	3.176	0.0027	0.210	0.482	0.810	33.432
	SAP_SO_CENTRAL	5,545.7	0.446	1.770	3.966	3.134	0.0025	0.154	0.255	0.451	104.310
	SAP_SO_NORTH	4,820.5	0.788	2.868	3.640	8.228	0.0025	0.162	0.323	0.704	94.405
	SAP_SO_SOUTHWEST	891.3	0.230	0.167	0.727	0.028	0.0050	0.122	0.194	0.292	1.608
	Southwest_FW_FR	771.6	0.211	0.262	1.238	0.068	0.0025	0.120	0.176	0.250	5.960
	Southwest_FW_HG_FR	116.4	0.569	1.294	2.276	1.674	0.0375	0.228	0.342	0.484	13.270
	Southwest_HW_FR	327.7	0.168	0.177	1.054	0.031	0.0081	0.074	0.140	0.217	1.776
	Southwest_HW_HG_FR	28.5	0.386	0.233	0.604	0.054	0.0237	0.277	0.362	0.476	1.073
	Southwest_Main_FR	938.5	0.183	0.144	0.787	0.021	0.0025	0.083	0.168	0.244	1.594
	Southwest_Main_HG_FR	1,058.8	0.554	0.570	1.029	0.325	0.0026	0.241	0.383	0.644	5.334
	Southwest_N_FR	171.3	0.183	0.159	0.868	0.025	0.0025	0.102	0.141	0.221	1.632
	Southwest_N_HG_FR	135.6	0.700	0.930	1.329	0.865	0.0025	0.144	0.371	0.796	7.280
Rh (ppm)	Central_FW_FR	2,986.3	0.015	0.070	4.708	0.005	0.0005	0.001	0.003	0.009	2.607
	Central_FW_HG_FR	374.3	0.159	0.484	3.043	0.234	0.0005	0.018	0.058	0.124	6.062
	Central_HW_FR	883.0	0.025	0.059	2.358	0.003	0.0005	0.001	0.004	0.024	0.757
	Central_HW_HG_FR	162.6	0.043	0.105	2.430	0.011	0.0005	0.001	0.006	0.036	0.866
	Central_Main_FR	1,895.0	0.025	0.042	1.667	0.002	0.0005	0.004	0.013	0.030	0.592
	Central_Main_HG_FR	1,915.7	0.109	0.148	1.353	0.022	0.0005	0.035	0.073	0.133	2.011
	North_N_FR	2,990.5	0.035	0.224	6.458	0.050	0.0005	0.001	0.012	0.032	10.988
	North_N_HG_FR	1,779.4	0.103	0.294	2.843	0.086	0.0005	0.007	0.035	0.108	8.369
	North_S_FR	1,776.3	0.024	0.061	2.566	0.004	0.0005	0.001	0.006	0.025	0.873
	North_S_HG_FR	739.7	0.165	0.345	2.095	0.119	0.0005	0.015	0.064	0.162	4.770
	SAP_SO_CENTRAL	5,545.7	0.057	0.123	2.144	0.015	0.0005	0.008	0.024	0.065	3.202
	SAP_SO_NORTH	4,820.5	0.102	0.496	4.846	0.246	0.0005	0.009	0.029	0.074	17.334
	SAP_SO_SOUTHWEST	891.3	0.017	0.027	1.631	0.001	0.0005	0.001	0.007	0.019	0.323
	Southwest_FW_FR	771.6	0.010	0.031	3.024	0.001	0.0005	0.001	0.001	0.007	0.457
	Southwest_FW_HG_FR	116.4	0.078	0.201	2.569	0.040	0.0005	0.024	0.047	0.078	2.073
	Southwest_HW_FR	327.7	0.019	0.052	2.681	0.003	0.0005	0.001	0.002	0.017	0.544
	Southwest_HW_HG_FR	28.5	0.122	0.203	1.673	0.041	0.0005	0.020	0.053	0.087	0.820
	Southwest_Main_FR	938.5	0.015	0.029	1.961	0.001	0.0005	0.001	0.004	0.016	0.322
	Southwest_Main_HG_FR	1,058.8	0.077	0.111	1.436	0.012	0.0005	0.021	0.047	0.096	1.535
	Southwest_N_FR	171.3	0.015	0.030	1.965	0.001	0.0005	0.001	0.003	0.017	0.233

Variable	Domain	Length (m)	Mean	S.D.	C.V.	Var.	Min.	Q25	Q50	Q75	Max.
	Southwest_N_HG_FR	135.6	0.110	0.140	1.278	0.020	0.0010	0.011	0.048	0.145	0.770
Au (ppm)	Central_FW_FR	2,986.3	0.010	0.029	2.931	0.001	0.0025	0.003	0.003	0.007	0.508
	Central_FW_HG_FR	374.3	0.035	0.068	1.950	0.005	0.0025	0.005	0.010	0.033	0.706
	Central_HW_FR	883.0	0.055	0.182	3.293	0.033	0.0025	0.005	0.014	0.050	3.896
	Central_HW_HG_FR	162.6	0.140	0.173	1.233	0.030	0.0025	0.020	0.071	0.195	1.023
	Central_Main_FR	1,895.0	0.027	0.054	2.015	0.003	0.0025	0.003	0.010	0.022	0.712
	Central_Main_HG_FR	1,915.7	0.072	0.131	1.816	0.017	0.0025	0.010	0.021	0.081	2.354
	North_N_FR	2,990.5	0.015	0.053	3.464	0.003	0.0025	0.003	0.005	0.010	2.158
	North_N_HG_FR	1,779.4	0.036	0.089	2.485	0.008	0.0025	0.005	0.009	0.024	2.058
	North_S_FR	1,776.3	0.025	0.050	2.018	0.003	0.0025	0.005	0.010	0.020	0.542
	North_S_HG_FR	739.7	0.052	0.099	1.891	0.010	0.0025	0.006	0.012	0.050	1.040
	SAP_SO_CENTRAL	5,545.7	0.053	0.107	2.034	0.011	0.0025	0.008	0.018	0.051	2.563
	SAP_SO_NORTH	4,820.5	0.037	0.106	2.874	0.011	0.0025	0.005	0.011	0.030	3.912
	SAP_SO_SOUTHWEST	891.3	0.028	0.040	1.419	0.002	0.0025	0.007	0.012	0.033	0.365
	Southwest_FW_FR	771.6	0.009	0.015	1.700	0.000	0.0025	0.005	0.006	0.010	0.340
	Southwest_FW_HG_FR	116.4	0.016	0.017	1.103	0.000	0.0025	0.005	0.010	0.020	0.090
	Southwest_HW_FR	327.7	0.041	0.094	2.326	0.009	0.0025	0.007	0.016	0.040	1.083
	Southwest_HW_HG_FR	28.5	0.124	0.120	0.969	0.014	0.0091	0.035	0.114	0.194	0.583
	Southwest_Main_FR	938.5	0.025	0.043	1.702	0.002	0.0025	0.005	0.010	0.024	0.387
	Southwest_Main_HG_FR	1,058.8	0.103	0.152	1.476	0.023	0.0025	0.019	0.044	0.140	2.400
	Southwest_N_FR	171.3	0.031	0.067	2.140	0.004	0.0025	0.005	0.010	0.026	0.576
	Southwest_N_HG_FR	135.6	0.047	0.072	1.530	0.005	0.0025	0.005	0.020	0.062	0.457
Ni (ppm)	Central_FW_FR	2,986.3	312	455	1.5	206575	50	100	200	300	8503
	Central_FW_HG_FR	374.3	949	815	0.9	664598	50	300	700	1312	4800
	Central_HW_FR	883.0	1,379	1787	1.3	3194888	50	600	1098	1649	32063
	Central_HW_HG_FR	162.6	2,221	1914	0.9	3663130	50	1124	1800	3000	18767
	Central_Main_FR	1,895.0	1,094	942	0.9	888094	50	417	900	1500	17410
	Central_Main_HG_FR	1,915.7	2,072	1284	0.6	1649055	50	1100	1900	2900	14207
	North_N_FR	2,863.4	476	596	1.3	355278	5	50	200	750	7700
	North_N_HG_FR	1,737.5	722	1030	1.4	1061372	40	50	200	1088	9600
	North_S_FR	1,776.3	1,018	1346	1.3	1811107	48	161	500	1400	17544
	North_S_HG_FR	739.7	2,121	3157	1.5	9966045	50	317	1000	2572	25269
	SAP_SO_CENTRAL	1,777.3	612	886	1.4	785780	50	100	213	725	5000
	SAP_SO_NORTH	2,951.7	597	1069	1.8	1143613	50	100	200	658	20995
	SAP_SO_SOUTHWEST	465.8	686	940	1.4	883853	50	100	250	886	5000
	Southwest_FW_FR	771.6	263	388	1.5	150880	46	50	101	225	3051
	Southwest_FW_HG_FR	116.4	1,127	809	0.7	654313	50	500	999	1800	2900
	Southwest_HW_FR	327.7	869	908	1.0	824057	50	215	663	1200	7600
	Southwest_HW_HG_FR	28.5	3784	5567	1.5	30989448	52	1302	2020	3402	25018
	Southwest_Main_FR	938.5	624	647	1.0	418231	50	126	399	905	5930
	Southwest_Main_HG_FR	1,058.8	1,355	813	0.6	660621	50	751	1300	1830	6500
	Southwest_N_FR	171.3	839	812	1.0	659308	50	203	649	1108	4886
	Southwest_N_HG_FR	135.6	791	658	0.8	433066	50	315	600	997	3818

Source: GE21, 2025.

14.4 Grade Estimation

14.4.1 Simulation Approach and Kriging Estimation

Geostatistical simulation and kriging are both spatial interpolation methods abroad by Geostatistics, but they have different purposes and methodologies. The kriging estimation produces a smooth Best Linear Unbiased Estimator value at unsampled locations (BLUE), with a single interpolated map. This map represents the expected value (mean value) in block models. Geostatistical simulations generate multiple realizations (possible scenarios) of spatial variability, capturing uncertainty. Figure 14-14 represents the difference between the two methods. The geostatistical simulation is represented by Figure 14-14 (a) which each Selective Mining Unit (SMU) is characterized by a probabilistic distribution of some feature, while Figure 14-14 (b) represent the estimation (kriging) process which a single average value is associated with the SMU. The geostatistical simulations allow quantifying spatial uncertainty, risk assessment, and decision-making in resource estimation.

The E-type post processing consists of given the average of overall simulations in some support and is close to kriging estimation. The importance of simulations in Mineral Resource Estimation is defined by the possibility of measurement of the production uncertainties in the advanced steps in the mining project, the calculation of recoverable resources and dimension definition of sampling strategies. These steps form the foundation of the mining planning and future activities in mining project.

A valid conditional simulation in geostatistics ensures that simulated values honour both spatial continuity and data distribution:

- The **variogram** characterizes the spatial dependence of the data by defining how values at different locations are correlated. A valid conditional simulation reproduces the input modeled variogram.
- The **histogram** (or probability distribution function – "PDF") represents the statistical distribution of values in the dataset. A valid conditional simulation preserves the histogram of the data.

If these conditions are satisfied, the simulations ensuring statistical realism and spatial realism of mineral deposit.

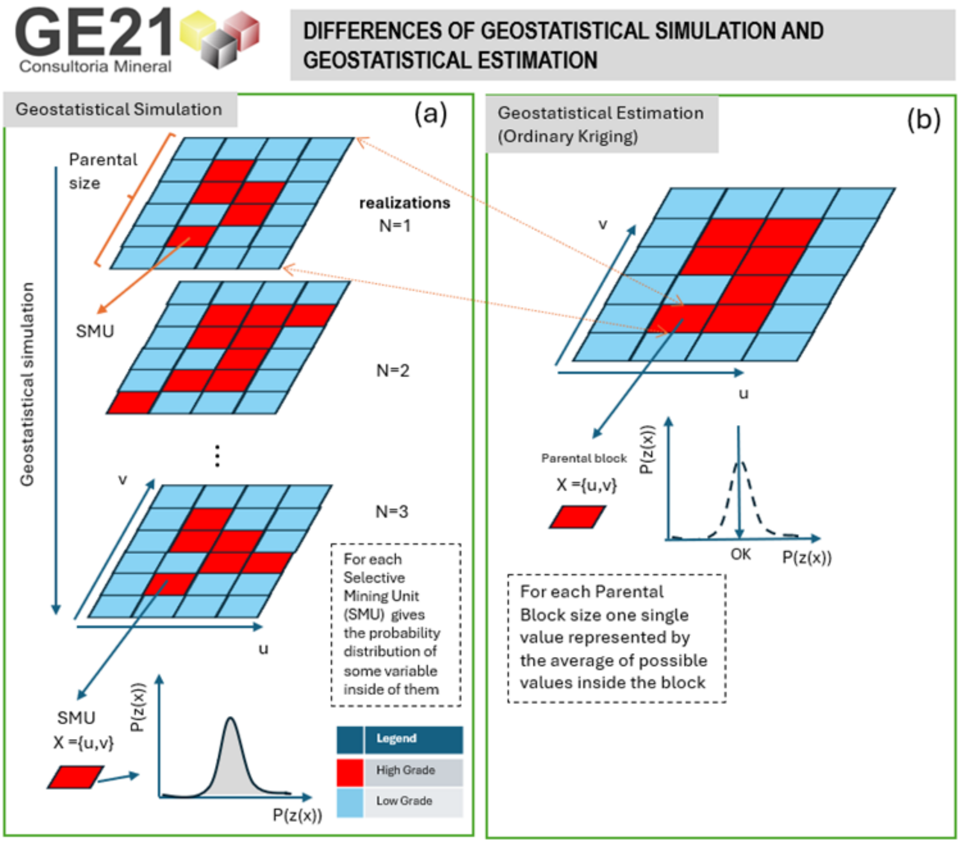


Figure 14-14: Differences between geostatistical simulations and geostatistical estimation

Source: GE21, 2025.

14.4.2 Simulation and Kriging Strategies

A test for defining the number of simulations was conducted in Pd in Central Domain, for High and Low Grade. Figure 14-15 and Figure 14-16 indicate that 50 simulations are sufficient to account for the full range of ergodic variations in the simulations.

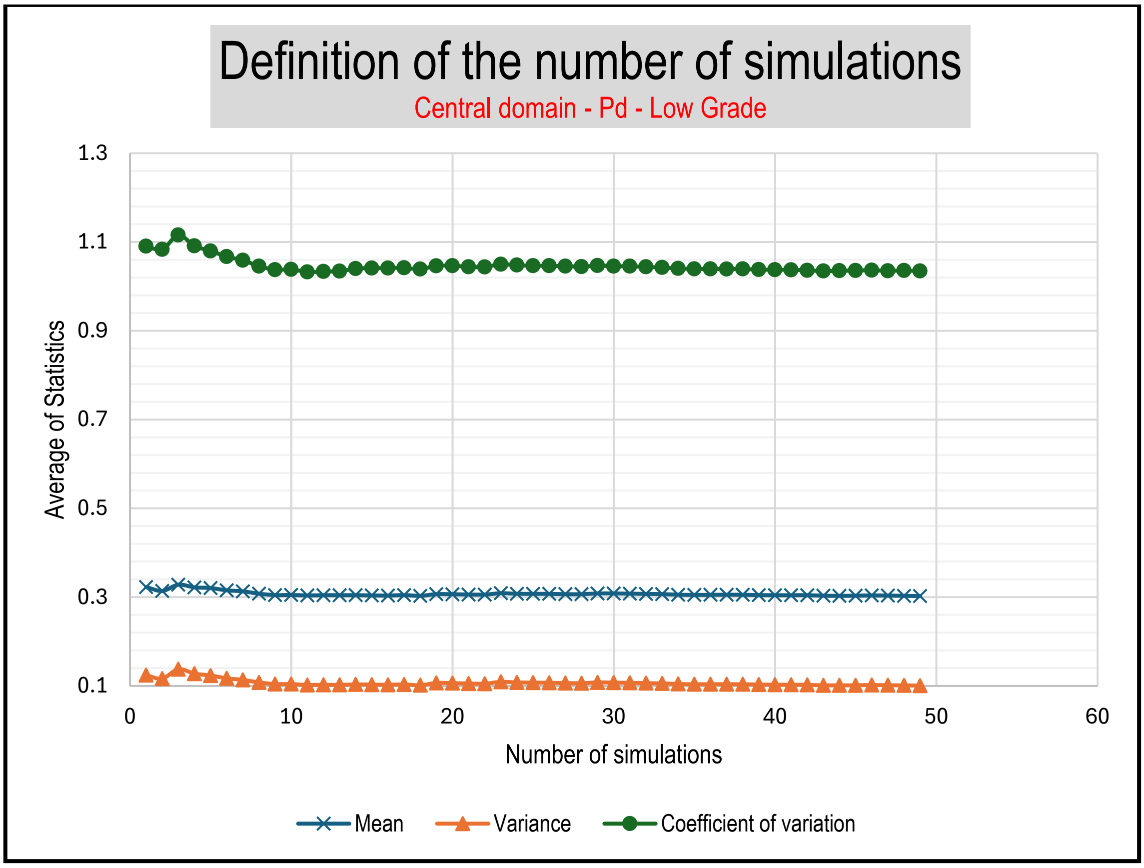


Figure 14-15: Average of Statistics according to the number of simulations (Pd Central Low Grade)

Source: GE21, 2025.

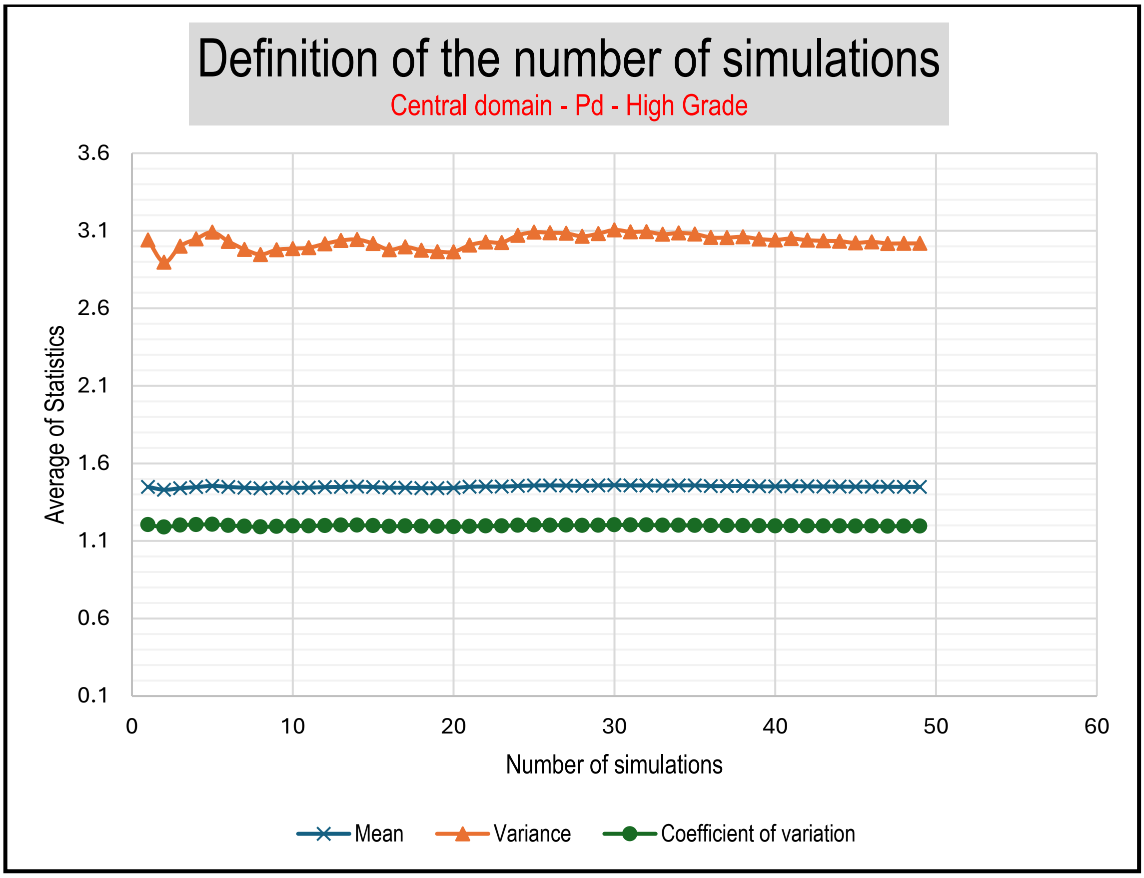


Figure 14-16: Definition of the number of simulations (Pd Central High Grade)

Source: GE21, 2025.

Table 14-6 shows the simulation strategy used for all domains.

Table 14-6: Simulation strategy for all domains and chemical elements

Region	Variable	Orientation			Ranges			Min number of samples	Max number of samples	Nested
		Dip	Azim	Rake	Max	Med	Min			Ellipsoid factor
		Degrees	Degrees	Degrees	m	m	m			
Central	Pd	60	140	90	40	40	10	3	20	2x, 3x and infinite
	Pt	60	140	90	20	20	10	3	20	2x, 3x and infinite
	Au	60	140	90	100	100	10	5	20	2x, 3x and infinite
	Ni	60	140	90	100	100	10	5	20	2x, 3x and infinite
	Rh	60	140	90	100	100	10	5	20	2x, 3x and infinite
	Cu	60	140	90	100	100	10	5	20	2x, 3x and infinite
Southwest	Pd	65	150	90	40	40	10	5	20	2x, 3x and infinite
	Pt	65	150	90	100	100	10	5	20	2x, 3x and infinite
	Au	65	150	90	100	100	10	5	20	2x, 3x and infinite
	Ni	65	150	90	100	100	10	5	20	2x, 3x and infinite
	Rh	65	150	90	100	100	10	5	20	2x, 3x and infinite

GE21 Consultoria Mineral

Region	Variable	Orientation			Ranges			Min number of samples	Max number of samples	Nested
		Dip	Azim	Rake	Max	Med	Min			Ellipsoid factor
		Degrees	Degrees	Degrees	m	m	m			
	Cu	65	150	90	100	100	10	5	20	2x, 3x and infinite
North N	Pd	80	260	90	40	40	10	5	20	2x, 3x and infinite
	Pt	80	260	90	100	100	10	5	20	2x, 3x and infinite
	Au	80	260	90	40	40	10	5	20	2x, 3x and infinite
	Ni	80	260	90	100	100	10	5	20	2x, 3x and infinite
	Rh	80	260	90	100	100	10	5	20	2x, 3x and infinite
	Cu	80	260	90	100	100	10	5	20	2x, 3x and infinite
North S	Pd	64	260	90	100	100	10	5	20	2x, 3x and infinite
	Pt	64	260	90	100	100	10	5	20	2x, 3x and infinite
	Au	64	260	90	40	40	10	5	20	2x, 3x and infinite
	Ni	64	260	90	100	100	10	5	20	2x, 3x and infinite
	Rh	64	260	90	40	40	10	4	12	2x, 3x and infinite
	Cu	64	260	90	100	100	10	5	20	2x, 3x and infinite

Source: GE21, 2025.

Table 14-7 shows the estimation strategy for oxide domain.

Table 14-7: Search strategy for oxide domain

Region	Orientation			Ranges			Min number of samples	Max number of samples first factor	Max number of samples second factor	Max number of samples third factor	Maximum number per drill hole	Nested factors
	Dip	Azim	Rake	Max	Med	Min						
	Degrees	Degrees	Degrees	m	m	m						
Central	60	140	90	40	40	10	3	8	10	15	2	2x, 3x and infinite
Southwest	65	150	90	40	40	10	3	8	10	15	2	2x, 3x and infinite
North N	80	260	90	40	40	10	3	8	10	15	2	2x, 3x and infinite
North S	64	260	90	40	40	10	3	8	10	15	2	2x, 3x and infinite
Soil	0	0	90	40	40	10	3	8	10	15	2	2x, 3x and infinite

Source: GE21, 2025.

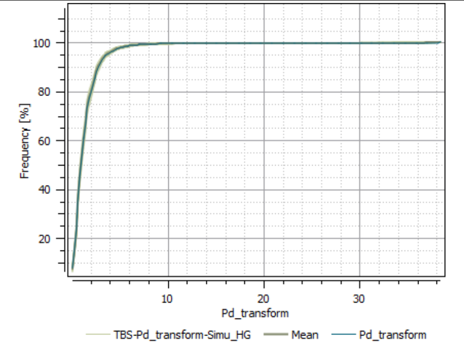
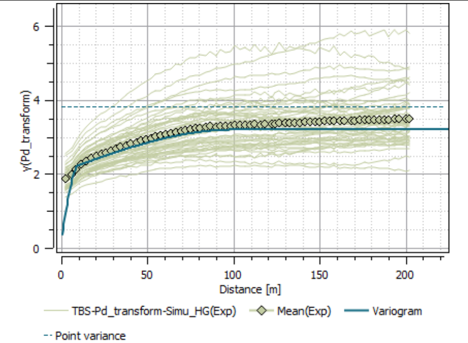
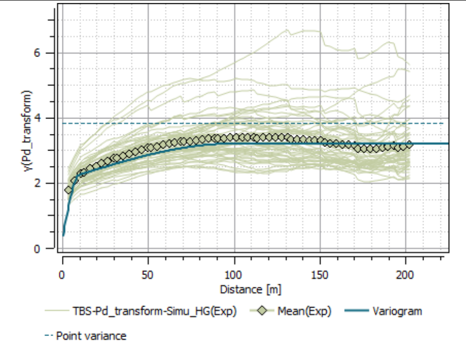
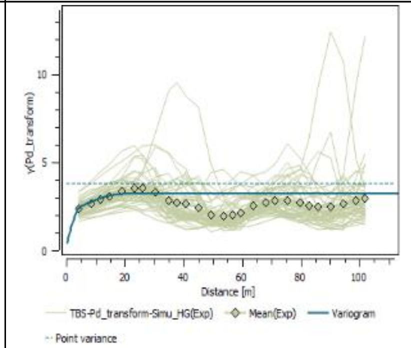
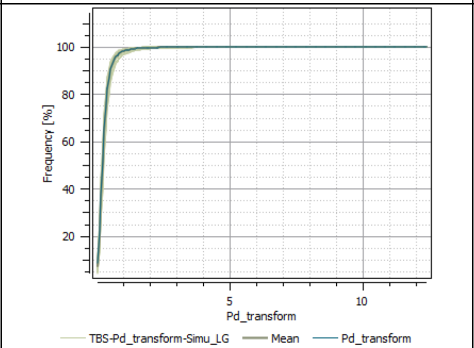
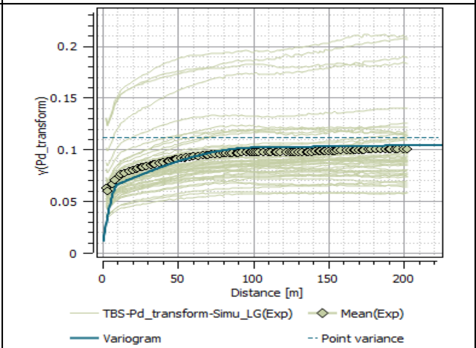
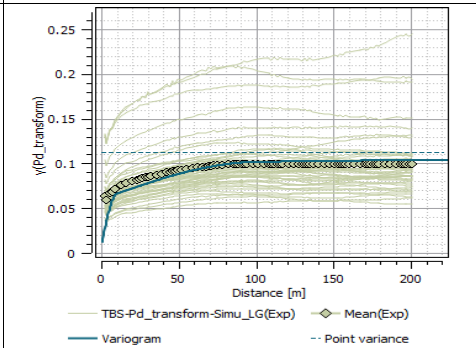
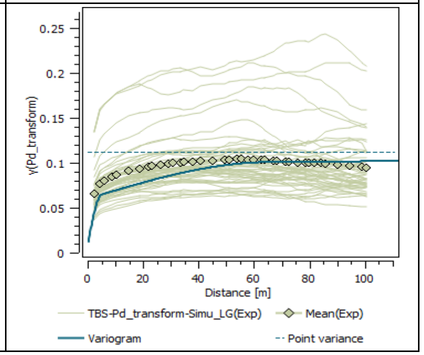
The OK strategy shown on Table 14-7 was used for estimation of the variables in fresh domain, which purpose for comparison with E-type simulation values.

14.4.3 Variograms and Simulation Validation

Simulation was calculated using Isatis.Neo Mining (Version 2024.04.2), performing 50 realizations with 800 bands (Seed: 165426). Simulations are compared with raw variogram and histogram values. Table 14-8 shows an example of simulation validation in Central Domain, variable Pd. It can be observed that ergodic fluctuations encompass all raw variograms and histograms, representing valid realizations.

Table 14-9 shows the variogram models used for simulation of all domains and Table 14-10 shows the variogram models for oxide domain.

Table 14-8: Example of simulation validation for Pd in the central area domains

D	Histogram	Major Axis	Intermediate Axis	Minimum Axis
Pd Central HG				
Pd Central LG				

Source: GE21, 2025.

GE21 Consultoria Mineral

Table 14-9: Variogram models for each element and domain

Geostatistical Set	Variable	Dip (°)	Dip Azimuth (°)	Pitch (°)	Nugget	Structure 1					Structure 2					Structure 3				
						Model	Sill	Range			Model	Sill	Range			Model	Sill	Range		
								Major (m)	Semi (m)	Minor (m)			Major (m)	Semi (m)	Minor (m)			Major (m)	Semi (m)	Minor (m)
Ag_Central_HG	Ag	60	140	90	0.30	Sph	0.15	40	40	10	Sph	0.25	100	100	20	Exp	0.30	∞*	∞*	∞*
Ag_Central_LG	Ag	60	140	90	0.30	Sph	0.70	50	50	10	-	-	-	-	-	-	-	-	-	-
Ag_NN_HG	Ag	80	260	90	0.33	Sph	0.67	50	30	10	-	-	-	-	-	-	-	-	-	-
Ag_NN_LG	Ag	80	260	90	0.33	Sph	0.67	50	30	10	-	-	-	-	-	-	-	-	-	-
Ag_NS_HG	Ag	64	260	90	0.32	Sph	0.12	45	25	5	Sph	0.56	90	45	10	-	-	-	-	-
Ag_Southwest_HG	Ag	65	150	90	0.20	Sph	0.20	40	40	40	Exp	0.10	100	100	80	Exp	0.50	∞*	∞*	∞*
Ag_Southwest_LG	Ag	65	150	90	0.31	Sph	0.69	40	20	10	-	0.00	-	-	-	-	-	-	-	-
Au_Central_HG	Au	60	140	90	0.30	Sph	0.25	40	40	10	Sph	0.45	100	80	20	-	-	-	-	-
Au_Central_LG	Au	60	140	90	0.39	Exp	0.27	20	20	5	Sph	0.34	60	60	25	-	-	-	-	-
Au_NN_HG	Au	80	260	90	0.33	Sph	0.67	50	30	20	-	-	-	-	-	-	-	-	-	-
Au_NN_LG	Au	80	260	90	0.34	Exp	0.56	40	40	20	Sph	0.10	∞*	∞*	∞*	-	-	-	-	-
Au_NS_HG	Au	64	260	90	0.32	Sph	0.12	45	25	5	Sph	0.56	90	45	10	-	-	-	-	-
Au_NS_LG	Au	64	260	90	0.33	Sph	0.05	100	25	5	Sph	0.51	150	45	10	Exp	0.10	∞*	∞*	∞*
Au_Southwest_HG	Au	65	150	90	0.22	Exp	0.73	100	100	40	Sph	0.05	∞*	∞*	∞*	-	-	-	-	-
Au_Southwest_LG	Au	65	150	90	0.32	Exp	0.38	100	50	50	Exp	0.30	∞*	∞*	100	-	-	-	-	-
Ni_Central_HG	Ni	60	140	90	0.30	Sph	0.25	40	40	10	Sph	0.45	90	90	20	-	-	-	-	-
Ni_Central_LG	Ni	60	140	90	0.30	Sph	0.70	40	40	10	-	-	-	-	-	-	-	-	-	-
Ni_NN_HG	Ni	80	260	90	0.20	Sph	0.25	10	10	10	Sph	0.55	50	50	40	-	-	-	-	-
Ni_NN_LG	Ni	80	260	90	0.30	Exp	0.10	20	20	15	Exp	0.60	60	60	50	-	-	-	-	-
Ni_NS_HG	Ni	64	260	90	0.25	Sph	0.20	80	25	5	Sph	0.55	90	45	10	-	-	-	-	-
Ni_NS_LG	Ni	64	260	90	0.20	Sph	0.20	80	25	5	Sph	0.60	100	45	15	-	-	-	-	-
Ni_Southwest_HG	Ni	65	150	90	0.28	Sph	0.11	50	50	50	Exp	0.33	200	200	100	Sph	0.28	∞*	∞*	∞*

Geostatistical Set	Variable	Dip (°)	Dip Azimuth (°)	Pitch (°)	Nugget	Structure 1					Structure 2					Structure 3				
						Model	Sill	Range			Model	Sill	Range			Model	Sill	Range		
								Major (m)	Semi (m)	Minor (m)			Major (m)	Semi (m)	Minor (m)			Major (m)	Semi (m)	Minor (m)
Ni_Southwest_LG	Ni	65	150	90	0.25	Sph	0.25	80	80	50	Exp	0.50	250	250	150	-	-	-	-	-
Pd_Central_HG	Pd	60	140	90	0.07	Sph	0.57	10	10	5	Sph	0.35	100	100	20	-	-	-	-	-
Pd_Central_LG	Pd	60	140	90	0.07	Sph	0.47	10	10	5	Sph	0.35	100	100	65	Exp	0.10	∞*	∞*	∞*
Pd_NN_HG	Pd	80	260	90	0.20	Sph	0.45	8	8	8	Sph	0.35	80	30	15	-	-	-	-	-
Pd_NN_LG	Pd	80	260	90	0.30	Sph	0.50	10	10	10	Sph	0.20	inf	inf	inf	-	-	-	-	-
Pd_NS_HG	Pd	64	260	90	0.31	Sph	0.33	10	10	5	Sph	0.36	55	40	25	-	-	-	-	-
Pd_NS_LG	Pd	64	260	90	0.31	Sph	0.33	10	10	8	Sph	0.36	55	55	25	-	-	-	-	-
Pd_Southwest_HG	Pd	65	150	90	0.31	Sph	0.38	8	8	8	Sph	0.31	50	30	30	-	-	-	-	-
Pd_Southwest_LG	Pd	65	150	90	0.31	Sph	0.40	5	5	5	Sph	0.29	45	35	35	-	-	-	-	-
Pt_Central_HG	Pt	60	140	90	0.27	Sph	0.49	10	10	3	Sph	0.24	70	70	30	-	-	-	-	-
Pt_Central_LG	Pt	60	140	90	0.27	Sph	0.48	10	10	3	Sph	0.13	70	70	30	Exp	0.13	∞*	∞*	∞*
Pt_ddh	Pt	0	90	90	0.82	Sph	0.09	66	66	66	Sph	0.09	367	367	367	-	-	-	-	-
Pt_NN_HG	Pt	80	260	90	0.18	Exp	0.60	4	4	4	Sph	0.17	20	10	10	Sph	0.05	∞*	∞*	∞*
Pt_NN_LG	Pt	80	260	90	0.18	Sph	0.53	3	3	3	Sph	0.10	22	10	10	Exp	0.19	∞*	∞*	∞*
Pt_NS_HG	Pt	64	260	90	0.33	Sph	0.50	5	5	5	Sph	0.10	10	10	10	Sph	0.07	∞*	∞*	∞*
Pt_NS_LG	Pt	64	260	90	0.33	Sph	0.40	5	5	5	Exp	0.15	10	10	10	Exp	0.11	∞*	∞*	∞*
Pt_Southwest_HG	Pt	65	150	90	0.19	Exp	0.66	5	3	3	Sph	0.15	20	10	10	-	-	-	-	-
Pt_Southwest_LG	Pt	65	150	90	0.22	Sph	0.62	5	5	3	Sph	0.11	40	10	10	Exp	0.04	∞*	∞*	∞*
Rh_Central_HG	Rh	60	140	90	0.24	Sph	0.33	10	10	3	Sph	0.34	100	50	16	Exp	0.09	∞*	∞*	∞*
Rh_Central_LG	Rh	60	140	90	0.36	Sph	0.47	50	10	3	Sph	0.17	134	55	16	-	-	-	-	-
Rh_NN_HG	Rh	80	260	90	0.32	Sph	0.37	10	10	5	Sph	0.21	100	50	30	Exp	0.11	∞*	∞*	∞*
Rh_NN_LG	Rh	64	260	90	0.50	Exp	0.25	20	10	5	Sph	0.10	60	30	30	Sph	0.15	∞*	∞*	∞*
Rh_NS_HG	Rh	60	260	90	0.10	Sph	0.35	15	5	2	Sph	0.55	55	20	20	-	-	-	-	-
Rh_NS_LG	Rh	60	260	90	0.15	Exp	0.79	20	20	3	Sph	0.06	100	100	15	-	-	-	-	-
Rh_Southwest_HG	Rh	65	150	90	0.30	Exp	0.55	10	10	5	Sph	0.15	80	30	30	-	-	-	-	-
Rh_Southwest_LG	Rh	65	150	90	0.25	Exp	0.65	8	8	5	Sph	0.10	50	30	30	-	-	-	-	-

* Infinite (Range greater than 1000m)

Source: GE21, 2025.

GE21 Consultoria Mineral

Table 14-10: Variogram models for oxide domain

Geostatistical Set	Variable	Dip (°)	Dip Azimuth (°)	Pitch (°)	Nugget	Structure 1					Structure 2				
						Model	Sill	Range			Model	Sill	Range		
								Major (m)	Semi (m)	Minor (m)			Major (m)	Semi (m)	Minor (m)
Au_Solo	Au	0	90	90	0.16	Sph	0.33	5	5	5	Sph	0.51	20	20	20
Ni_Solo	Ni	0	90	90	0.08	Sph	0.00	22	22	22	Sph	0.92	37	37	37
Pd_Solo	Pd	0	90	90	0.50	Sph	0.18	5	5	5	Sph	0.32	20	20	20
Pt_Solo	Pt	0	90	90	0.65	Sph	0.17	6	6	6	Sph	0.18	27	27	27
Rh_Solo	Rh	0	90	90	0.61	Sph	0.16	2	2	2	Sph	0.23	16	16	16

Source: GE21, 2025.

14.4.4 Block Model

The simulation approach is commonly performed in point support. The QP considers that a block 10 times discretized is sufficient for represent the point variability of the data. The simulation block has the support of 2.5 m x 2.5 m x 2.5 m. The parameters of simulation block model are related to Table 14-11.

Table 14-11: Grid geometry of simulation block size

	X	Y	Z
Number of nodes	1920	2240	320
Mesh size	2.5 m	2.5 m	2.5 m
Grid origin (center)	655775.00 m	9338780.00 m	-241.49 m
Grid origin (corner)	655773.75 m	9338778.75 m	-242.74 m

Source: GE21, 2025.

After simulation the realizations are upscaled for a parental block model with the dimensions of 25 m x 25 m x 5 according to Table 14-12. The value of the block is considered by the average mean of all upscaled realizations.

Table 14-12: Grid geometry of parental block size 25 x 25 x 5

	X	Y	Z
Number of nodes	192	224	160
Mesh size	25 m	25 m	5 m
Grid origin (center)	655,786.25 m	9,338,791.25 m	-240.24 m
Grid origin (corner)	655,773.75 m	9,338,778.75 m	-242.74 m

Source: GE21, 2025.

The parental block model data was migrated to a subblock with 8m x 8m x 4m divisions, for reporting the feature average values with a valid reproduction of geological models and volumes. The final block model attributes are present in Table 14-13.

Table 14-13: Block model attributes

Attribute Name	Type	Decimals	Background	Description
Au	Float	2	-99	Au ppm E-type estimated grade
Pd	Float	2	-99	Pd ppm E-type estimated grade
Pt	Float	2	-99	Pt ppm E-type estimated grade
Ni	Float	2	-99	Ni ppm E-type estimated grade
Rh	Float	2	-99	Rh ppm E-type estimated grade
Pd_Eq	Float	2	-99	Pd equivalent – grade calculation
Density	Float	2	-99	Density g/cm3
ANM	Character	-	-	Mineral Right (ANM Process)
Mineralization	Character	-	0	Mineralization Grade Shell
Rec_class	Integer	-	0	1 = measured, 2 = indicated, 3 = inferred, 4 = exploratory potential, 0 = not classified
Weathering	Float	2	-99	Weathered zones

Source: GE21, 2025.

14.4.5 Density

GE21 has applied an Inverse of Quadradic Distance (IQD) to estimate density samples in the fresh domain, using an 80m x 80m x 50m ellipsoid aligned to the main directions of the mineral lens. A fixed value of 1.46 g/cm^3 was defined for the oxide domain according to the analysis of the *in-situ* sampling dataset. The density averages for block model attribution are shown in Table 14-14.

Table 14-14: : Bravo's density values by all domains

Density (g/cm^3)										
Name	Block Count	Mean	Standard Deviation	Coefficient of Variation	Variance	Minimum	Lower quartile	Median	Upper quartile	Maximum
Oxide Domain	1253506	1.46	0.01	0.01	0.00	1.46	1.46	1.46	1.46	2.96
Central FW HG	424558	2.85	0.14	0.05	0.02	2.47	2.74	2.81	2.95	3.25
Central FW LG	820882	2.89	0.13	0.05	0.02	2.46	2.82	2.88	2.97	3.43
Central HW HG	188528	2.83	0.18	0.06	0.03	2.58	2.71	2.81	3.02	3.43
Central HW LG	267722	2.82	0.13	0.05	0.02	2.47	2.73	2.80	2.88	3.44
Central Main HG	1970980	2.86	0.13	0.05	0.02	2.52	2.78	2.85	2.93	3.54
Central Main LG	293494	2.94	0.17	0.06	0.03	2.58	2.80	2.90	3.10	3.42
North N HG	1015581	2.75	0.11	0.04	0.01	2.19	2.67	2.73	2.82	3.36
North N LG	370692	2.77	0.10	0.04	0.01	2.56	2.69	2.74	2.88	2.97
North S HG	507543	2.85	0.18	0.06	0.03	2.50	2.74	2.81	2.91	3.53
North S LG	303666	2.92	0.13	0.04	0.02	2.59	2.81	2.92	3.04	3.23
Southwest FW HG	178859	2.78	0.05	0.02	0.00	2.46	2.74	2.81	2.81	2.96
Southwest FW LG	312514	2.84	0.12	0.04	0.02	2.56	2.71	2.93	2.94	2.99
Southwest HW HG	12454	3.00	0.30	0.10	0.09	2.64	2.79	2.83	3.46	3.47
Southwest HW LG	102013	2.80	0.12	0.04	0.01	2.58	2.72	2.78	2.90	3.20
Southwest Main HG	647558	2.81	0.09	0.03	0.01	2.61	2.74	2.80	2.91	3.01
Southwest Main LG	127113	2.79	0.12	0.04	0.01	2.53	2.69	2.76	2.90	3.13
Southwest N HG	49404	2.74	0.05	0.02	0.00	2.67	2.71	2.72	2.76	2.86
Southwest N LG	40546	2.71	0.11	0.04	0.01	2.58	2.59	2.71	2.78	3.00

Source: GE21, 2025.

14.5 Estimates Validation

The QP carried out the validation of the estimate through visual verification and by the Global and Local bias verification. The global and local bias checks used the Nearest Neighbour (NN) and OK as the comparison with the Simulation (E-type) estimate.

NN-Checks plots, (Figure 14-17 and Figure 14-20) show the results for global bias analysis of the estimated Pd ppm, which allowed verifying the occurrence of expected smoothing of the estimation by E-type within the acceptance limits. The comparison showed that E-type globally respected the average grades, and the global bias in the estimated grades is within the limits of acceptance.

The local bias assessment by the swath plot method aims to analyze the occurrence of local bias by comparing the average grades for the model through E-type, Ordinary Kriging, and the Nearest Neighbour method in swath coordinate intervals graphs along the X, Y, and Z axes. Figure 14-23 shows the swath plot validation results of the estimated Pd ppm.

Figure 14-21 and Figure 14-22 show the swath plot comparison of ordinary kriging, simulation E-type and Nearest Neighbour values. The E-type post-processing shows smoother estimations than OK since it accounts for the average of the possibility of high and low values since kriging is too much affected by the presence of local sample values. Because simulation is conditional, the effect of conditioning bias is much more expressed in OK results. The QP considered the result obtained from the estimation by the E-type of simulations to be acceptable, noting that there was no local or global bias outside the acceptance limits.

Nearest Neighbour

Variable	Pd Grades (ppm)
Domain	HG
N. of Samples	5084775
Minimum	0.001
Maximum	50.965
Average	1.224
Variance	2.634
Std Deviation	1.623

NN Quantiles

Quantile	Value
2.5%	0.116
5%	0.198
25%	0.625
Median	0.889
75%	1.39
95%	3.051
97.5%	4.117

Ordinary Kriging

Variable	Pd Grades (ppm)
Domain	HG
N. of Samples	5084775
Minimum	0.021
Maximum	18.129
Average	1.258
Variance	0.74
Std Deviation	0.86

OK Quantiles

Quantile	Value
2.5%	0.293
5%	0.374
25%	0.787
Median	1.061
75%	1.552
95%	2.607
97.5%	3.123

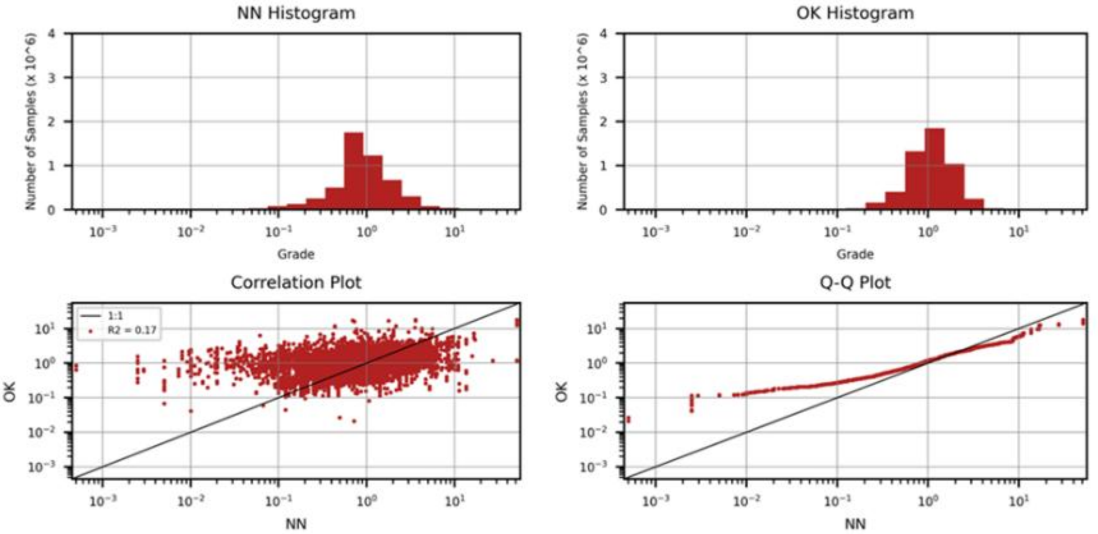


Figure 14-17: NN vs OK (HG)

Source: GE21, 2025.

GE21 Consultoria Mineral

Nearest Neighbour

Variable	Pd Grades (ppm)
Domain	LG
N. of Samples	2577592
Minimum	0.001
Maximum	156.291
Average	0.319
Variance	3.885
Std Deviation	1.971

NN Quantiles

Quantile	Value
2.5%	0.039
5%	0.061
25%	0.158
Median	0.248
75%	0.36
95%	0.68
97.5%	0.914

Ordinary Kriging

Variable	Pd Grades (ppm)
Domain	LG
N. of Samples	2577592
Minimum	0.005
Maximum	67.891
Average	0.318
Variance	1.17
Std Deviation	1.082

OK Quantiles

Quantile	Value
2.5%	0.107
5%	0.126
25%	0.198
Median	0.27
75%	0.357
95%	0.529
97.5%	0.602

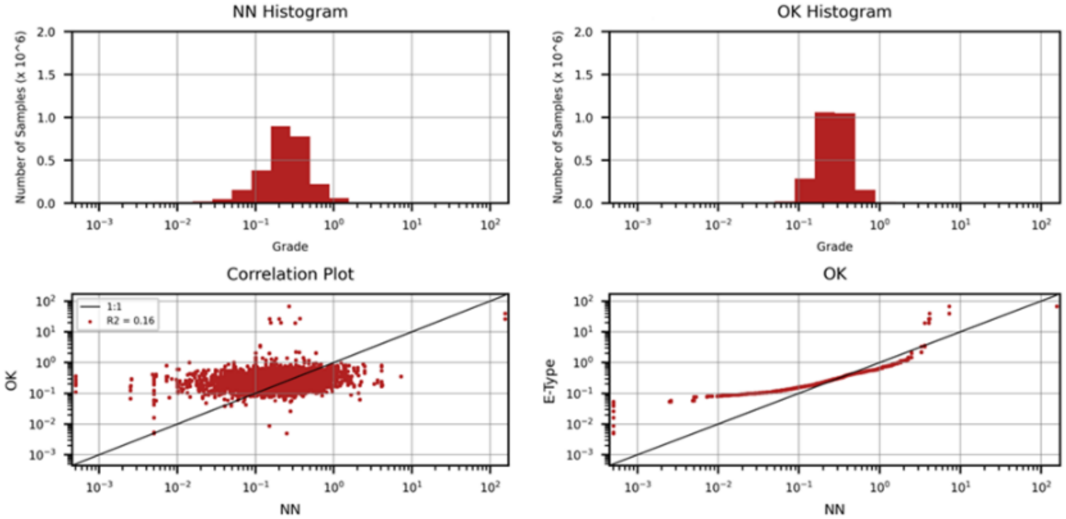


Figure 14-18: NN vs OK (LG)

Source: GE21, 2025.

Nearest Neighbour

Variable	Pd Grades (ppm)
Domain	HG
N. of Samples	5084775
Minimum	0.001
Maximum	50.965
Average	1.224
Variance	2.634
Std Deviation	1.623

NN Quantiles

Quantile	Value
2.5%	0.116
5%	0.198
25%	0.625
Median	0.889
75%	1.39
95%	3.051
97.5%	4.117

Simulation E-Type

Variable	Pd Grades (ppm)
Domain	HG
N. of Samples	5084775
Minimum	0.386
Maximum	3.834
Average	1.244
Variance	0.097
Std Deviation	0.311

E-Type Quantiles

Quantile	Value
2.5%	0.796
5%	0.858
25%	1.025
Median	1.187
75%	1.414
95%	1.809
97.5%	2.004

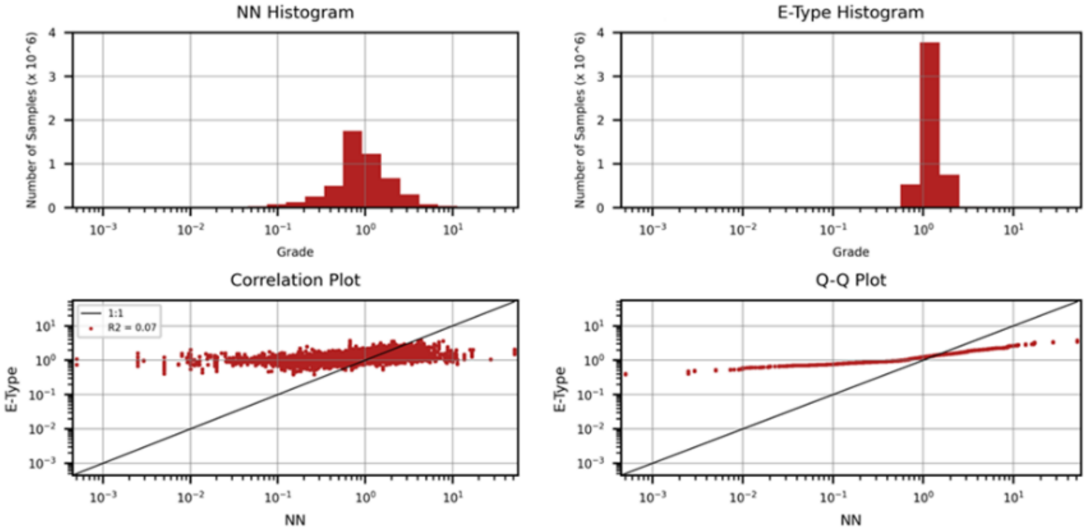


Figure 14-19: NN check vs E-type (HG)

Source: GE21, 2025.

GE21 Consultoria Mineral

Nearest Neighbour

Variable	Pd Grades (ppm)
Domain	LG
N. of Samples	2577592
Minimum	0.001
Maximum	156.291
Average	0.319
Variance	3.885
Std Deviation	1.971

NN Quantiles

Quantile	Value
2.5%	0.039
5%	0.061
25%	0.158
Median	0.248
75%	0.36
95%	0.68
97.5%	0.914

Simulation E-Type

Variable	Pd Grades (ppm)
Domain	LG
N. of Samples	2577592
Minimum	0.124
Maximum	0.592
Average	0.294
Variance	0.002
Std Deviation	0.041

E-Type Quantiles

Quantile	Value
2.5%	0.204
5%	0.22
25%	0.271
Median	0.298
75%	0.317
95%	0.356
97.5%	0.374

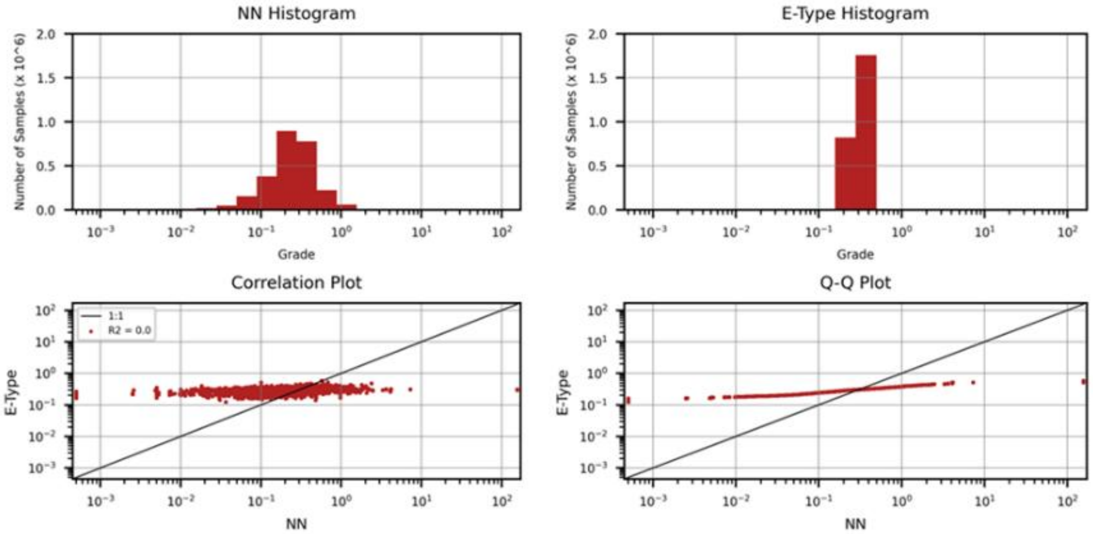


Figure 14-20: NN vs E-type (LG)

Source: GE21, 2025.

Ordinary Kriging

Variable	Pd Grades (ppm)
Domain	HG
N. of Samples	5084775
Minimum	0.021
Maximum	18.129
Average	1.258
Variance	0.74
Std Deviation	0.86

OK Quantiles

Quantile	Value
2.5%	0.293
5%	0.374
25%	0.787
Median	1.061
75%	1.552
95%	2.607
97.5%	3.123

Simulation E-Type

Variable	Pd Grades (ppm)
Domain	HG
N. of Samples	5084775
Minimum	0.386
Maximum	3.834
Average	1.244
Variance	0.097
Std Deviation	0.311

E-Type Quantiles

Quantile	Value
2.5%	0.796
5%	0.858
25%	1.025
Median	1.187
75%	1.414
95%	1.809
97.5%	2.004

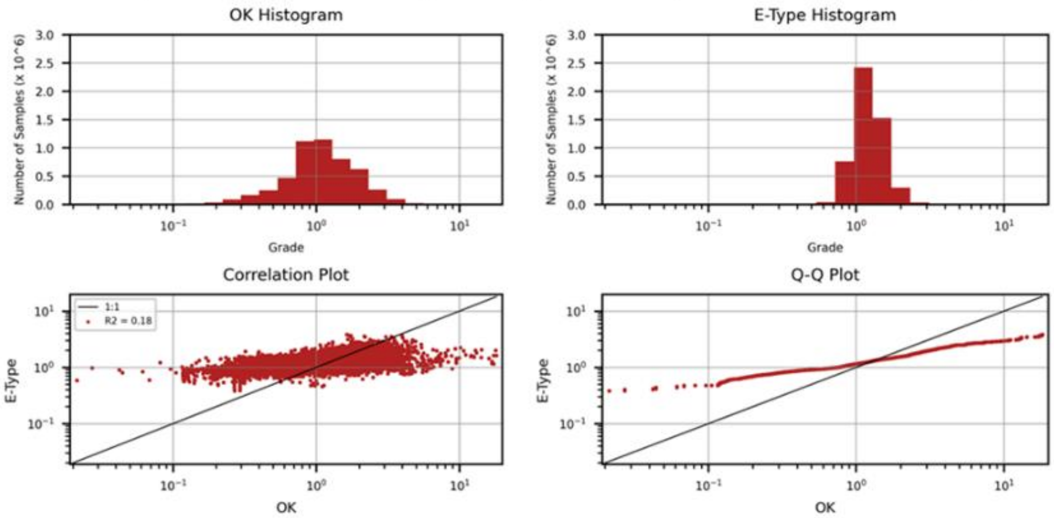


Figure 14-21: OK vs E-type (HG)

Source: GE21, 2025.

Ordinary Kriging	
Variable	Pd Grades (ppm)
Domain	LG
N. of Samples	2577592
Minimum	0.005
Maximum	67.891
Average	0.318
Variance	1.17
Std Deviation	1.082

OK Quantiles	
Quantile	Value
2.5%	0.107
5%	0.126
25%	0.198
Median	0.27
75%	0.357
95%	0.529
97.5%	0.602

Simulation E-Type	
Variable	Pd Grades (ppm)
Domain	LG
N. of Samples	2577592
Minimum	0.124
Maximum	0.592
Average	0.294
Variance	0.002
Std Deviation	0.041

E-Type Quantiles	
Quantile	Value
2.5%	0.204
5%	0.22
25%	0.271
Median	0.298
75%	0.317
95%	0.356
97.5%	0.374

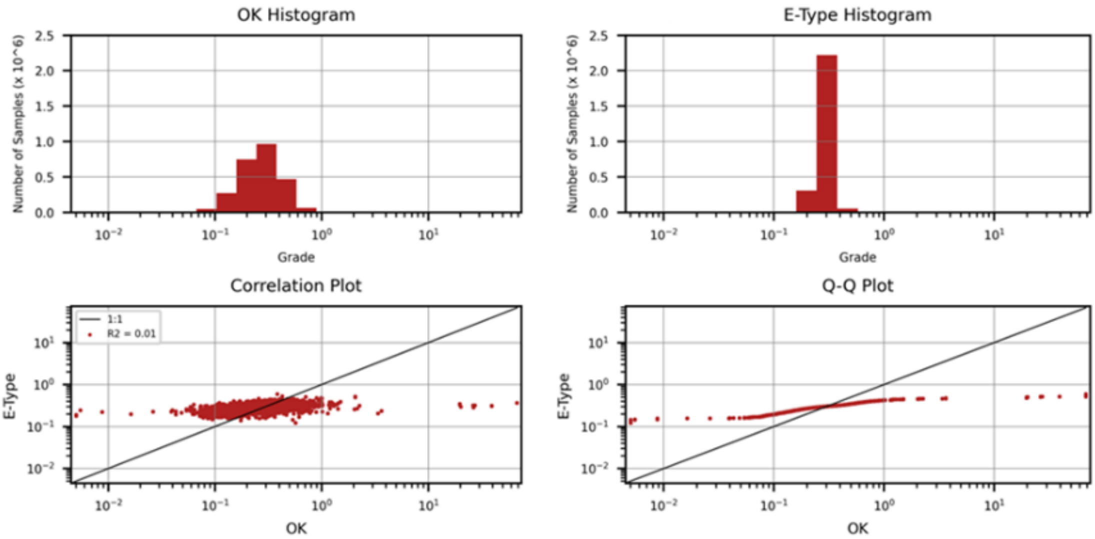


Figure 14-22: OK vs E-type (LG)

Source: GE21, 2025.

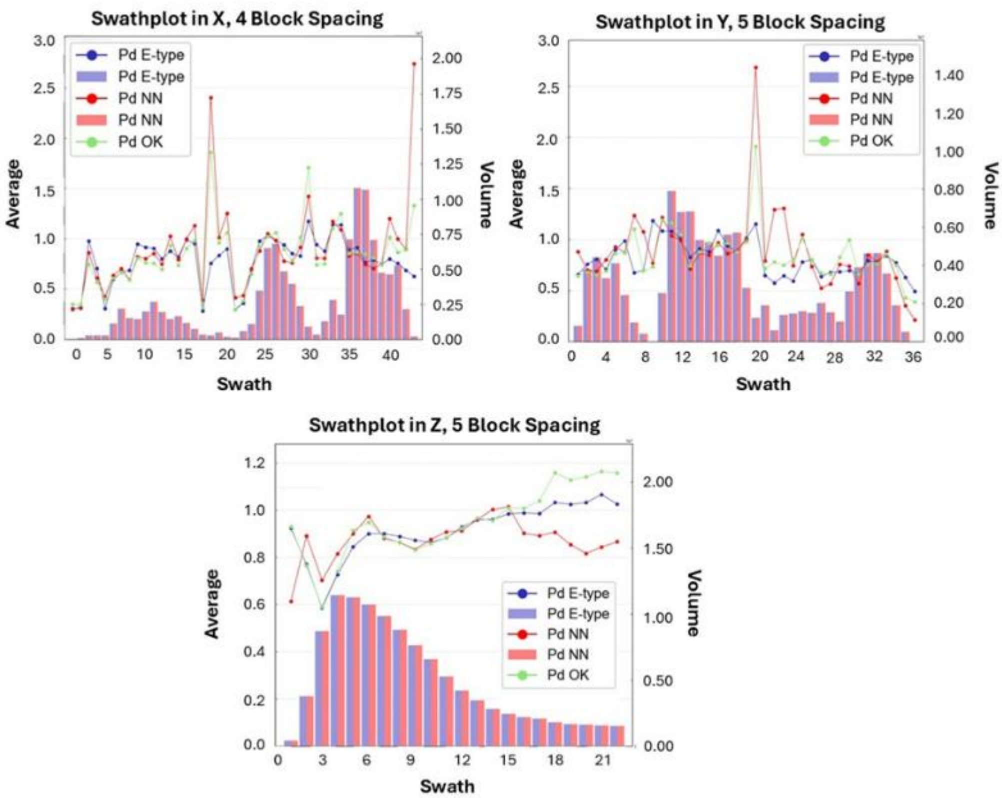


Figure 14-23: Swath plot E-type vs NN vs OK grades Pd (ppm)

Source: GE21, 2025.

14.6 Classification of Mineral Resources

The Mineral Resource was classified per CIM Standards (2014) and CIM Best Practices Guidelines (2019), utilizing geostatistical and classical methods, along with economically- and mining-appropriate parameters relevant to the deposit type.

The Mineral Resource definitions by CIM are transcribed below:

- A “Mineral Resource” is a concentration or occurrence of diamonds, a natural solid inorganic material or natural fossilized solid organic material, including base and precious metals, coal and industrial minerals in the earth’s crust or in the earth’s crust in such form and quantity and of such grade or quality that allows reasonable prospects of economic extraction. The location, quantity, level, geological characteristics, and continuity of a Mineral Resource are known, estimated, or interpreted from specific geological evidence and knowledge.
- An “Inferred Mineral Resource” is that part of a Mineral Resource for which the quantity and level or quality can be estimated on the basis of geological evidence and limited sampling and reasonably presumed but not verified geological and grade continuity. The estimation is based on limited information and sampling collected using appropriate techniques from locations such as outcrops, trenches, wells, and drill holes.
- An “Indicated Mineral Resource” is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and assessment of the deposit’s economic viability. The estimation is based on thorough and reliable exploration and testing information gathered using appropriate techniques from locations such as outcrops, trenches, wells, works, and drill holes spaced far enough apart for geological and level continuity to be reasonably assumed.
- A “Measured Mineral Resource” is that part of a Mineral Resource for which quantity, level or quality, densities, shape, and physical characteristics are so well established that they can be estimated with sufficient confidence to allow the appropriate application of technical and economic parameters, to support production planning and assessment of the deposit’s economic viability. The estimation is based on thorough and reliable exploration, sampling, and analysis of information gathered using appropriate techniques from locations such as outcrops, trenches, wells, works, and drill holes spaced far enough apart to confirm geological and level continuity.

The classification boundaries made by GE21 for the Measured, Indicated, and Inferred categories were established through an approach that considered a comprehensive set of factors. These factors included the sampling procedure analysis, the sample grid spacing, the survey methodology, and the quality of assay data. Additionally, drilling spacing and the progressive expansion of the search radius during grade estimation stages were also considered, as well as the average anisotropic distance of the samples and the continuity of pegmatite mineralization. This multi-faceted approach ensured the robustness and accuracy of the classification process.

To classify Mineral Resources, a study of spatial continuity for Pd Equivalent was conducted using variography followed by OK interpolation. This study established a continuity zone suitable for considering:

- The Measured Mineral Resource was classified according to a reference grid of approximately 45m x 45m, with a minimum number of 3 holes in the section along the strike and dip directions, surrounded by the pit shell.
- The Indicated Mineral Resource classification had as a reference a drilling grid of approximately 75m x 75m, extending both along the strike and dip directions, and requiring a minimum of two drill holes.
- Manual post-processing was undertaken to construct wireframes representing the volumes categorized as Measured and Indicated, while considering the blocks within the resource pit shell.
- The Inferred Mineral Resource classification is all remaining estimated blocks within the resource pit shell.

The total Mineral Resources all lie within the Mining Rights boundaries.

According to CIM Guidelines, the Mineral Resource classification should be supported by Reasonable Prospect for Eventual Economic Extraction (RPEEE), which GE21 usually performs through a mathematical model pit shell which limits the blocks classified as a mineral resource generated from an economic and geometric function.

GE21 performed a pit optimization study to classify the project's Mineral Resources to ensure the RPEEE was met. Parameters in the benefit function are presented in Table 14-15.

The Luanga Project's updated, pit-constrained MRE has an effective date of February 18, 2025. It comprises 36 Mt at 2.00 g/t Pd Eq for a total of 2.3 Moz Pd Eq in the Measured category, 122 Mt at 2.06 g/t Pd Eq for 8.0 Moz Pd Eq in the Indicated category, 158 Mt at 2.04 g/t Pd Eq for a total of 10.4 Moz Pd Eq in the Measured + Indicated categories, and 78 Mt at 2.01 g/t Pd Eq for a total of 5.0 Moz Pd Eq in the Inferred category (Table 14-16).

Figure 14-24 shows a perspective view of the MRE classification.

GE21 Consultoria Mineral

Table 14-15: Pit parameters generated by RPEEE

Optimization Parameters - RPEEE				
Item				Unit
Lithotype	Fresh & Weathered & High Talc			-
Slope Angle	Weathered		40	°
	Fresh / High Talc		50	°
Mining	Density		Block Model	
	Mining Recovery		100	%
	Mining Dilution		0	%
	MCAF		ANM Mineral Rights	
	Cut-off grade (Whittle)	Fresh	-	-
		Weathered	-	-
Processing	Metallurgic Recovery - Weathered	Pd	81.0%	Mill
		Pt	23.0%	Mill
		Rh	54.0%	Mill
		Au	90.0%	Mill
		Ni	0.0%	Mill
	Metallurgic Recovery - Fresh	Pd	77.0%	Mill
		Pt	81.0%	Mill
		Rh	51.0%	Mill
		Au	48.0%	Mill
		Ni	50.0%	Mill
	Metallurgic Recovery - High Talc	Pd	51.0%	Mill
		Pt	55.5%	Mill
		Rh	27.3%	Mill
		Au	27.0%	Mill
		Ni	0.0%	Mill
Costs	Weathered	Mining Cost	2.00	US$/t mined
		Processing Cost	7.50	US$/t ROM
		Grade Control	1.00	
		Logistics	0.50	
		Rehabilitation	1.00	
		G&A	1.50	
	Fresh / High Talc	Mining Cost	3.00	US$/t mined
		Processing Cost	9.00	US$/t ROM
		Grade Control	1.00	
		Logistics	0.50	
		Rehabilitation	1.00	
		G&A	1.50	
Selling	Price	Pd	1,380	US$/oz
		Pt	1,100	US$/oz
		Rh	6,200	US$/oz
		Au	1,500	US$/oz
		Ni	7.10	US$/lb
	Royalties	All	2.0	%

Source: GE21, 2025.

GE21 Consultoria Mineral

Table 14-16: MRE statement at a cut-off of 0.5g/t Pd Eq*

Resource	Classification	Domain	Mass	*Average Value*						*Material Content*					
			Mass	Pd eq	Pd	Pt	Au	Rh	Ni	Pd eq	Pd	Pt	Au	Rh	Ni
			Mt	ppm	g/t	g/t	g/t	g/t	%	koz	koz	koz	koz	koz	klb
Open Pit	Measured	Ox	4	1.51	0.90	0.88	0.05	0.12	0.00	197	117	115	7	15	—
		High Talc	—	—	—	—	—	—	—	—	—	—	—	—	—
		Fresh	32	2.06	0.97	0.67	0.04	0.08	0.11	2,144	1,009	694	46	88	77,621
		Total	36	2.00	0.96	0.69	0.04	0.09	0.10	2,340	1,126	809	53	104	77,621
	Indicated	Ox	6	1.51	0.97	0.73	0.04	0.11	0.00	314	200	151	9	23	0
		High Talc	2	1.83	1.12	0.54	0.11	0.08	0.13	146	89	43	9	6	6,952
		Fresh	113	2.09	0.99	0.59	0.05	0.09	0.14	7,599	3,583	2,133	193	318	344,092
		Total	122	2.06	0.99	0.59	0.05	0.09	0.13	8,058	3,872	2,326	210	348	351,044
	Measured + Indicated	Ox	10	1.51	0.94	0.79	0.04	0.11	0.00	510	317	266	15	38	—
		High Talc	2	1.83	1.12	0.54	0.11	0.08	0.13	146	89	43	9	6	6,952
		Fresh	145	2.08	0.98	0.60	0.05	0.09	0.13	9,743	4,592	2,827	239	407	421,713
		Total	158	2.04	0.98	0.62	0.05	0.09	0.12	10,399	4,998	3,135	262	451	428,665
	Inferred	Ox	3	1.57	0.88	1.04	0.05	0.13	—	130	73	86	4	11	—
		High Talc	0	1.76	1.08	0.53	0.10	0.07	0.14	5	3	2	0	0	292
		Fresh	75	2.02	0.97	0.58	0.05	0.08	0.13	4,878	2,344	1,389	123	191	214,690
		Total	78	2.01	0.97	0.59	0.05	0.08	0.13	5,013	2,421	1,476	128	202	214,981

Notes:

1. The MRE has been prepared by Porfirio Cabaleiro Rodriguez, Mining Engineer, BSc (Mine Eng), MAIG, director of GE21 Consultoria Mineral Ltda., an independent Qualified Persons (QP) under NI43-101. The effective date of the MRE is February 18, 2025.
2. Mineral resources are reported using the 2014 CIM Definition Standards and were estimated in accordance with the CIM 2019 Best Practices Guidelines, as required by National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).
3. Mineral resources that are not Mineral Reserves do not have demonstrated economic viability. There is no certainty that all Mineral Resources will be converted into Mineral Reserves.
4. Chemical elements are estimated using different estimation methodologies according to the Weathering Model. Ordinary Kriging was applied to the Oxidized domain, while the Turning Bands Simulation was applied to fresh rock.
5. This MRE includes Inferred Mineral Resources, which have had insufficient work to classify them as Indicated Mineral Resources. It is uncertain but reasonably expected that inferred Mineral Resources could be upgraded to indicated Mineral Resources with continued exploration.
    - The Mineral Resource Estimate is reported/confined within an economic pit shell generated by Dassault Geovia Whittle software, using the following assumptions (Generated from work completed for Bravo and historical test work):
        - Metallurgical recovery in sulphide material of 77% Pd, 81% Pt, 51% Rh, 48% Au, 50% Ni to a saleable Ni-PGM concentrate.
        - Metallurgical recovery in oxide material of 81% Pd, 23% Pt, 54% Rh, 90% Au to a saleable PGM ash residue (Ni not applicable).
        - Metallurgical recovery in high-talc sulphide material of 51% Pd, 55% Pt, 27% Rh, 27% Au, 0% Ni to a saleable Ni-PGM concentrate Independent Geotechnical Testwork – Overall pit slopes of 40 degrees in oxide and 50 degrees in Fresh Rock.
        - Densities are based on 27,170 drill hole cores and 112 in situ sample density measurements. The Mineral Resources are reported on a dry density basis.

- External downstream payability has not been included, as the base case MRE assumption considers internal downstream processing.

d. Payable royalties of 2%, (only considering CFEM, for reserves, a complete set of royalties must be considered)

6. Metal Pricing
   b. Metal price assumptions are based on 10-year trailing averages (2014-2023): Pd price of US$1,380/oz, Pt price of US$1,100/oz, Rh price of US$6,200/oz, Au price of US$1,500/oz, Ni price of US$7,10/lb.
   e. Palladium Equivalent (PdEq) Calculation
   f. The PdEq equation is: PdEq = Pd g/t + F1 + F2 + F3 + F4
   Where: $F1 = \frac{(Pt_p * Pt_R)}{(Pd_p * Pd_R)} Pt_t$ $F2 = \frac{(Rh_p * Rh_R)}{(Pd_p * Pd_R)} Rh_t$ $F3 = \frac{(Au_p * Au_R)}{(Pd_p * Pd_R)} Au_t$ $F4 = \frac{(Ni_p * Ni_R)}{(Pd_p * Pd_R)} Ni_t$
   $_P$ = Metal Price
   $_R$ = Metallurgical Recovery

7. Costs are taken from comparable projects in GE21's extensive database of mining operations in Brazil, which includes not only operating mines, but recent actual costs from what could potentially be similarly sized operating mines in the Carajás. Costs considered a throughput rate of ca. 10Mtpa:
   c. Mining costs: US$2.00/t oxide, US$3.00/t Fresh Rock. Processing costs: US$9.00/t fresh rock, US$7.50/t oxide. US$1.50/t processed, for General & Administration. US$1.00/t processed for grade control. US$0.50/t processed for rehabilitation.
   d. Several of these considerations (metallurgical recovery, metal price projections, for example) should be regarded as preliminary in nature, and therefore, PdEq calculations should be regarded as preliminary in nature.

8. The current MRE supersedes and replaces the Previous Estimate (2023), which should no longer be relied upon.
9. The QP is not aware of political, environmental, or other risks that could materially affect the potential development of the Mineral Resources other than those typical for mining projects at this stage of development and as identified in this report.
10. Totals may not sum due to rounding.

Source: GE21, 2025.

Table 14-17: MRE statement based on mineralization style

					Average Value						Material Content					
Resource	Classification	Weathering	Min Styles	Mass	Pd eq	Pd	Pt	Au	Rh	Ni	Pd eq	Pd	Pt	Au	Rh	Ni
				Mt	g/t	g/t	g/t	g/t	g/t	%	kOz	kOz	kOz	kOz	kOz	klb
Open Pit	Measured	Ox	CENTRAL_LSZ	0.2	0.80	0.59	0.38	0.03	0.03	—	6	4	3	0	0	—
			CENTRAL_MSZ	0.4	1.34	0.95	0.43	0.09	0.06	—	19	14	6	1	1	—
			CENTRAL_NR	0.1	0.70	0.41	0.18	0.14	0.03	—	2	1	0	0	0	—
			NORTH_LSZ	0.2	2.02	1.38	0.98	0.02	0.13	—	14	10	7	0	1	—
			NORTH_LSZ+CHR	1.4	1.50	0.88	1.47	0.02	0.09	—	65	38	64	1	4	—
			NORTH_MSZ	1.5	1.71	0.95	0.67	0.07	0.18	—	83	46	33	3	8	—
			NORTH_SZ	0.2	1.03	0.56	0.30	0.02	0.13	—	7	4	2	0	1	—
			Total	4.1	1.51	0.90	0.88	0.05	0.12	—	197	117	115	7	15	—
		Fresh	CENTRAL_LSZ	9.2	1.42	0.70	0.44	0.03	0.05	0.08	421	208	129	8	14	16,652.92
			CENTRAL_MSZ	6.1	3.06	1.59	0.68	0.08	0.10	0.24	597	310	133	16	20	31,905.83
			CENTRAL_NR	1.4	1.63	0.71	0.35	0.07	0.06	0.18	74	32	16	3	3	5,583.34
			NORTH_LSZ	1.1	2.12	0.98	0.73	0.05	0.09	0.10	75	35	26	2	3	2,455.10
			NORTH_LSZ+CHR	6.5	2.15	0.97	0.89	0.03	0.09	0.06	450	202	187	6	20	8,319.56

GE21 Consultoria Mineral

					Average Value						Material Content					
			NORTH_MSZ	6.9	2.05	0.86	0.78	0.04	0.12	0.07	452	190	172	9	26	10,545.82
			NORTH_SZ	1.3	1.83	0.77	0.76	0.03	0.07	0.08	75	32	31	1	3	2,158.09
			Total	32.4	2.06	0.97	0.67	0.04	0.08	0.11	2 144	1 009	694	46	88	77,620.66
		Total	CENTRAL_LSZ	9.4	1.41	0.70	0.44	0.03	0.05	0.08	427	213	132	9	15	16,652.92
			CENTRAL_MSZ	6.5	2.95	1.55	0.66	0.08	0.10	0.22	616	323	139	18	21	31,905.83
			CENTRAL_NR	1.5	1.58	0.69	0.34	0.08	0.06	0.17	76	33	16	4	3	5,583.34
			NORTH_LSZ	1.3	2.11	1.04	0.77	0.05	0.10	0.08	89	44	33	2	4	2,455.10
			NORTH_LSZ+CHR	7.9	2.04	0.95	0.99	0.03	0.09	0.05	515	241	251	7	24	8,319.56
			NORTH_MSZ	8.4	1.99	0.88	0.76	0.05	0.13	0.06	535	236	205	12	34	10,545.82
			NORTH_SZ	1.5	1.71	0.74	0.70	0.03	0.08	0.07	82	36	33	1	4	2,158.09
			Total	36.5	2.00	0.96	0.69	0.04	0.09	0.10	2 340	1 126	809	53	104	77,620.66
	Indicated	Ox	CENTRAL_LSZ	1.3	1.52	1.02	0.78	0.04	0.09	—	64	43	33	2	4	—
			CENTRAL_MSZ	1.4	1.37	0.94	0.52	0.05	0.09	—	64	44	24	2	4	—
			CENTRAL_NR	0.1	0.75	0.49	0.21	0.08	0.03	—	2	1	0	0	0	—
			NORTH_LSZ	0.5	4.31	2.31	1.61	0.03	0.53	—	70	37	26	1	9	—
			NORTH_LSZ+CHR	0.7	1.26	0.67	1.49	0.01	0.08	—	28	15	33	0	2	—
			NORTH_MSU	0.1	0.78	0.44	0.43	0.04	0.06	—	2	1	1	0	0	—
			NORTH_MSZ	0.7	1.15	0.70	0.61	0.05	0.08	—	26	16	14	1	2	—
			NORTH_SZ	0.2	0.74	0.47	0.25	0.03	0.06	—	5	3	2	0	0	—
			SW_LSZ	0.4	1.36	1.05	0.43	0.03	0.06	—	15	12	5	0	1	—
			SW_MSZ	1.1	1.11	0.82	0.37	0.06	0.05	—	38	28	12	2	2	—
			SW_NR	0.0	0.72	0.45	0.21	0.07	0.04	—	1	0	0	0	0	—
			Total	6.4	1.51	0.97	0.73	0.04	0.11	—	314	200	151	9	23	—
		High Talc	SW_LSZ	0.2	1.45	0.84	0.48	0.02	0.08	0.02	11	6	4	0	1	106.43
			SW_MSZ	2.2	1.87	1.14	0.55	0.12	0.08	0.14	134	82	39	8	5	6,767.92
			SW_NR	0.0	1.82	1.12	0.56	0.08	0.07	0.16	1	1	0	0	0	77.48
			Total	2.5	1.83	1.12	0.54	0.11	0.08	0.13	146	89	43	9	6	6,951.83
		Fresh	CENTRAL_LSZ	23.3	1.67	0.81	0.47	0.04	0.06	0.11	1,247	603	348	31	48	55,944.03
			CENTRAL_MSZ	33.3	2.64	1.32	0.67	0.07	0.10	0.18	2,823	1,414	713	72	112	129,117.13
			CENTRAL_NR	3.6	1.68	0.76	0.39	0.06	0.06	0.17	192	87	44	7	6	13,171.62
			NORTH_LSZ	4.0	1.73	0.68	0.59	0.04	0.11	0.10	225	88	77	5	14	8,717.12
			NORTH_LSZ+CHR	5.7	1.95	0.86	0.86	0.03	0.07	0.06	356	156	157	5	13	7,397.52
			NORTH_MSU	3.3	2.52	0.91	0.66	0.05	0.13	0.28	269	97	71	5	13	20,809.50
			NORTH_MSZ	13.9	2.32	0.90	0.72	0.04	0.13	0.17	1,035	399	323	20	60	52,036.89

GE21 Consultoria Mineral

					Average Value						Material Content					
			NORTH_MSZ+MSU	0.2	2.78	1.05	0.81	0.04	0.15	0.25	16	6	5	0	1	983.01
			NORTH_SZ	2.4	1.94	0.85	0.77	0.04	0.08	0.07	148	65	59	3	6	3,934.26
			SW_LSZ	4.1	0.84	0.44	0.26	0.02	0.03	0.03	111	58	35	3	4	3,117.92
			SW_MSZ	18.7	1.89	0.98	0.49	0.07	0.07	0.11	1,138	592	293	39	40	46,394.12
			SW_NR	0.6	2.02	0.93	0.47	0.08	0.07	0.19	38	17	9	1	1	2,469.17
			Total	113.0	2.09	0.99	0.59	0.05	0.09	0.14	7,599	3,583	2,133	193	318	344,092.29
		Total	CENTRAL_LSZ	24.6	1.66	0.82	0.48	0.04	0.07	0.10	1,311	646	381	32	52	55,944.03
			CENTRAL_MSZ	34.7	2.58	1.30	0.66	0.07	0.10	0.17	2,887	1,457	737	74	116	129,117.13
			CENTRAL_NR	3.6	1.66	0.76	0.39	0.06	0.05	0.16	194	88	45	7	6	13,171.62
			NORTH_LSZ	4.6	2.02	0.86	0.70	0.04	0.15	0.09	295	125	103	6	22	8,717.12
			NORTH_LSZ+CHR	6.4	1.88	0.84	0.93	0.03	0.07	0.05	384	171	190	6	15	7,397.52
			NORTH_MSU	3.4	2.49	0.90	0.66	0.05	0.12	0.28	271	98	72	5	14	20,809.50
			NORTH_MSZ	14.6	2.26	0.89	0.72	0.04	0.13	0.16	1,062	416	337	21	62	52,036.89
			NORTH_MSZ+MSU	0.2	2.78	1.05	0.81	0.04	0.15	0.25	16	6	5	0	1	983.01
			NORTH_SZ	2.6	1.85	0.82	0.73	0.04	0.08	0.07	153	68	60	4	7	3,934.26
			SW_LSZ	4.7	0.91	0.51	0.29	0.02	0.03	0.03	137	76	43	3	5	3,224.35
			SW_MSZ	22.0	1.85	0.99	0.49	0.07	0.07	0.11	1,310	702	345	50	47	53,162.05
			SW_NR	0.6	1.96	0.92	0.47	0.08	0.07	0.18	40	19	9	2	1	2,546.65
			Total	121.9	2.06	0.99	0.59	0.05	0.09	0.13	8,058	3,872	2,326	210	348	351,044.13
	Measured + Indicated	Ox	CENTRAL_LSZ	1.5	1.41	0.95	0.72	0.04	0.08	—	70	47	36	2	4	—
			CENTRAL_MSZ	1.9	1.36	0.94	0.50	0.06	0.08	—	83	57	30	3	5	—
			CENTRAL_NR	0.2	0.72	0.44	0.19	0.12	0.03	—	4	2	1	1	0	—
			NORTH_LSZ	0.7	3.62	2.03	1.42	0.03	0.41	—	84	47	33	1	9	—
			NORTH_LSZ+CHR	2.0	1.42	0.81	1.48	0.02	0.09	—	94	53	97	1	6	—
			NORTH_MSU	0.1	0.78	0.44	0.43	0.04	0.06	—	2	1	1	0	0	—
			NORTH_MSZ	2.2	1.53	0.87	0.65	0.07	0.14	—	109	62	47	5	10	—
			NORTH_SZ	0.4	0.89	0.52	0.28	0.02	0.09	—	12	7	4	0	1	—
			SW_LSZ	0.4	1.36	1.05	0.43	0.03	0.06	—	15	12	5	0	1	—
			SW_MSZ	1.1	1.11	0.82	0.37	0.06	0.05	—	38	28	12	2	2	—
			SW_NR	0.0	0.72	0.45	0.21	0.07	0.04	—	1	0	0	0	0	—
			Total	10.5	1.51	0.94	0.79	0.04	0.11	—	510	317	266	15	38	—
		High Talc	SW_LSZ	0.2	1.45	0.84	0.48	0.02	0.08	0.02	11	6	4	0	1	106.43
			SW_MSZ	2.2	1.87	1.14	0.55	0.12	0.08	0.14	134	82	39	8	5	6,767.92
			SW_NR	0.0	1.82	1.12	0.56	0.08	0.07	0.16	1	1	0	0	0	77.48

GE21 Consultoria Mineral

					Average Value						Material Content					
			Total	2.5	1.83	1.12	0.54	0.11	0.08	0.13	146	89	43	9	6	6,951.83
		Fresh	CENTRAL_LSZ	32.5	1.60	0.78	0.46	0.04	0.06	0.10	1,668	811	477	39	63	72,596.95
			CENTRAL_MSZ	39.4	2.70	1.36	0.67	0.07	0.10	0.19	3,420	1,724	845	88	131	161,022.96
			CENTRAL_NR	5.0	1.67	0.75	0.38	0.07	0.06	0.17	266	119	60	11	9	18,754.97
			NORTH_LSZ	5.2	1.82	0.74	0.62	0.04	0.10	0.10	301	123	103	7	17	11,172.21
			NORTH_LSZ+CHR	12.2	2.06	0.92	0.88	0.03	0.08	0.06	805	358	344	11	33	15,717.08
			NORTH_MSU	3.3	2.52	0.91	0.66	0.05	0.13	0.28	269	97	71	5	13	20,809.50
			NORTH_MSZ	20.7	2.23	0.89	0.74	0.04	0.13	0.14	1,487	590	495	29	85	62,582.71
			NORTH_MSZ+MSU	0.2	2.78	1.05	0.81	0.04	0.15	0.25	16	6	5	0	1	983.01
			NORTH_SZ	3.7	1.90	0.82	0.77	0.04	0.08	0.08	224	97	90	5	9	6,092.35
			SW_LSZ	4.1	0.84	0.44	0.26	0.02	0.03	0.03	111	58	35	3	4	3,117.92
			SW_MSZ	18.7	1.89	0.98	0.49	0.07	0.07	0.11	1,138	592	293	39	40	46,394.12
			SW_NR	0.6	2.02	0.93	0.47	0.08	0.07	0.19	38	17	9	1	1	2,469.17
			Total	145.4	2.08	0.98	0.60	0.05	0.09	0.13	9,743	4,592	2,827	239	407	421,712.96
		Total	CENTRAL_LSZ	34.0	1.59	0.79	0.47	0.04	0.06	0.10	1,737	858	513	41	67	72,596.95
			CENTRAL_MSZ	41.3	2.64	1.34	0.66	0.07	0.10	0.18	3,503	1,781	876	92	136	161,022.96
			CENTRAL_NR	5.1	1.64	0.74	0.37	0.07	0.05	0.17	269	121	61	11	9	18,754.97
			NORTH_LSZ	5.9	2.04	0.90	0.72	0.04	0.14	0.09	384	169	135	8	26	11,172.21
			NORTH_LSZ+CHR	14.2	1.97	0.90	0.96	0.03	0.08	0.05	899	412	441	13	39	15,717.08
			NORTH_MSU	3.4	2.49	0.90	0.66	0.05	0.12	0.28	271	98	72	5	14	20,809.50
			NORTH_MSZ	23.0	2.16	0.88	0.73	0.05	0.13	0.12	1,597	652	542	33	96	62,582.71
			NORTH_MSZ+MSU	0.2	2.78	1.05	0.81	0.04	0.15	0.25	16	6	5	0	1	983.01
			NORTH_SZ	4.1	1.80	0.79	0.72	0.04	0.08	0.07	236	104	94	5	11	6,092.35
			SW_LSZ	4.7	0.91	0.51	0.29	0.02	0.03	0.03	137	76	43	3	5	3,224.35
			SW_MSZ	22.0	1.85	0.99	0.49	0.07	0.07	0.11	1,310	702	345	50	47	53,162.05
			SW_NR	0.6	1.96	0.92	0.47	0.08	0.07	0.18	40	19	9	2	1	2,546.65
			Total	158.4	2.04	0.98	0.62	0.05	0.09	0.12	10,399	4,998	3,135	262	451	428,664.79
	Inferred	Ox	CENTRAL_LSZ	0.2	0.81	0.52	0.40	0.03	0.06	—	5	3	2	0	0	—
			CENTRAL_MSZ	0.0	0.88	0.60	0.31	0.03	0.06	—	0	0	0	0	0	—
			CENTRAL_NR	0.0	0.62	0.42	0.17	0.09	0.02	—	0	0	0	0	0	—
			NORTH_LSZ	0.7	2.38	1.25	0.96	0.06	0.28	—	51	27	20	1	6	—
			NORTH_LSZ+CHR	0.5	1.34	0.52	2.33	0.01	0.09	—	23	9	40	0	2	—
			NORTH_MSU	0.1	1.04	0.62	0.54	0.04	0.08	—	2	1	1	0	0	—
			NORTH_MSZ	0.3	1.05	0.56	0.79	0.02	0.10	—	10	5	8	0	1	—

GE21 Consultoria Mineral

					Average Value						Material Content					
			NORTH_SZ	0.1	0.86	0.46	0.62	0.02	0.08	—	2	1	1	0	0	—
			SW_LSZ	0.5	1.76	1.26	0.62	0.09	0.08	—	26	18	9	1	1	—
			SW_MSZ	0.3	1.24	0.88	0.44	0.09	0.05	—	11	8	4	1	0	—
			SW_NR	0.0	0.60	0.46	0.16	0.06	0.01	—	0	0	0	0	0	—
			Total	2.6	1.57	0.88	1.04	0.05	0.13	—	130	73	86	4	11	—
		High Talc	SW_LSZ	0.0	0.57	0.32	0.22	0.01	0.02	0.01	0	0	0	0	0	0.67
			SW_MSZ	0.1	1.81	1.11	0.54	0.11	0.07	0.14	5	3	2	0	0	291.09
			Total	0.1	1.76	1.08	0.53	0.10	0.07	0.14	5	3	2	0	0	291.76
		Fresh	CENTRAL_LSZ	13.5	1.61	0.78	0.45	0.04	0.06	0.11	697	336	196	18	26	32,097.18
			CENTRAL_MSZ	25.5	2.60	1.32	0.65	0.07	0.10	0.18	2,132	1,080	534	55	79	99,268.68
			CENTRAL_NR	5.1	1.79	0.85	0.43	0.06	0.06	0.16	291	138	70	10	10	17,389.35
			NORTH_LSZ	8.3	1.34	0.56	0.53	0.03	0.06	0.07	358	148	140	7	16	12,100.96
			NORTH_LSZ+CHR	5.9	2.07	0.92	0.86	0.03	0.09	0.07	390	174	162	6	16	8,435.83
			NORTH_MSU	0.8	2.25	0.84	0.64	0.04	0.13	0.20	57	21	16	1	3	3,487.55
			NORTH_MSZ	5.1	1.88	0.72	0.61	0.04	0.11	0.14	311	119	100	6	18	15,354.68
			NORTH_MSZ+MSU	0.1	2.43	0.93	0.71	0.04	0.14	0.20	5	2	1	0	0	254.20
			NORTH_ND	0.3	3.14	1.13	0.76	0.09	0.19	0.33	31	11	8	1	2	2,262.27
			NORTH_SZ	0.3	1.86	0.78	0.77	0.03	0.09	0.06	19	8	8	0	1	453.99
			SW_LSZ	4.9	1.66	0.88	0.44	0.04	0.06	0.09	261	138	69	7	9	10,020.39
			SW_MSZ	5.0	1.97	1.02	0.50	0.07	0.07	0.12	317	164	81	11	11	13,244.47
			SW_NR	0.2	1.26	0.66	0.33	0.05	0.04	0.07	9	5	2	0	0	320.05
			Total	74.9	2.02	0.97	0.58	0.05	0.08	0.13	4,878	2,344	1,389	123	191	214,689.59
		Total	CENTRAL_LSZ	13.6	1.60	0.77	0.45	0.04	0.06	0.11	701	339	199	18	26	32,097.18
			CENTRAL_MSZ	25.5	2.60	1.32	0.65	0.07	0.10	0.18	2,132	1,080	534	55	79	99,268.68
			CENTRAL_NR	5.1	1.78	0.85	0.43	0.06	0.06	0.16	291	138	70	11	10	17,389.35
			NORTH_LSZ	9.0	1.42	0.61	0.56	0.03	0.08	0.06	409	175	161	8	22	12,100.96
			NORTH_LSZ+CHR	6.4	2.01	0.89	0.98	0.03	0.09	0.06	413	183	202	7	18	8,435.83
			NORTH_MSU	0.8	2.17	0.82	0.64	0.04	0.13	0.19	59	22	17	1	3	3,487.55
			NORTH_MSZ	5.4	1.84	0.71	0.62	0.04	0.11	0.13	321	124	108	6	19	15,354.68
			NORTH_MSZ+MSU	0.1	2.43	0.93	0.71	0.04	0.14	0.20	5	2	1	0	0	254.20
			NORTH_ND	0.3	3.14	1.13	0.76	0.09	0.19	0.33	31	11	8	1	2	2,262.27
			NORTH_SZ	0.4	1.70	0.73	0.74	0.03	0.09	0.05	21	9	9	0	1	453.99
			SW_LSZ	5.4	1.66	0.91	0.45	0.05	0.06	0.08	287	157	78	8	10	10,021.06
			SW_MSZ	5.4	1.93	1.02	0.50	0.07	0.07	0.11	333	175	86	12	12	13,535.55

GE21 Consultoria Mineral

					Average Value						Material Content					
			SW_NR	0.2	1.21	0.64	0.32	0.05	0.04	0.06	9	5	2	0	0	320.05
			Total	77.6	2.01	0.97	0.59	0.05	0.08	0.13	5,013	2,421	1,476	128	202	214,981.35

Notes:

1. The MRE has been prepared Porfirio Cabaleiro Rodriguez, Mining Engineer, BSc (Mine Eng), MAIG, director of GE21 Consultoria Mineral Ltda., an independent Qualified Persons (QP) under NI43-101. The effective date of the MRE is February 18, 2025.
2. Mineral resources are reported using the 2014 CIM Definition Standards and were estimated in accordance with the CIM 2019 Best Practices Guidelines, as required by National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).
3. Mineral resources that are not Mineral Reserves do not have demonstrated economic viability. There is no certainty that all Mineral Resources will be converted into Mineral Reserves.
4. Chemical elements are estimated using different estimation methodologies according to the Weathering Model. Ordinary Kriging was applied to Oxidized domain while the Turning Bands Simulation was applied for fresh rock.
5. This MRE includes Inferred Mineral Resources which have had insufficient work to classify them as Indicated Mineral Resources. It is uncertain but reasonably expected that inferred Mineral Resources could be upgraded to indicated Mineral Resources with continued exploration.
6. The Mineral Resource Estimate is reported/confined within an economic pit shell generated by Dassault Geovia Whittle software, using the following assumptions (generated from work completed for Bravo and historical test work):
   - Metallurgical recovery in sulphide material of 77% Pd, 81% Pt, 51% Rh, 48% Au, 50% Ni to a saleable Ni-PGM concentrate.
   - Metallurgical recovery in oxide material of 81% Pd, 23% Pt, 54% Rh, 90% Au to a saleable PGM ash residue (Ni not applicable).
   - Metallurgical recovery in high-talc sulphide material of 51% Pd, 55% Pt, 27% Rh, 27% Au, 0% Ni to a saleable Ni-PGM concentrate.
   - Independent Geotechnical Testwork – Overall pit slopes of 40 degrees in oxide and 50 degrees in Fresh Rock.
   - Densities are based on 27,170 drill hole core and 112 in situ samples density measurements. The Mineral Resources are reported on a dry density basis. External downstream payability has not been included, as the base case MRE assumption considers internal downstream processing.

   a. Payable royalties of 2%, (only considering CFEM, for reserves, a complete set of royalties must be considered)

   - Metal Pricing
     - Metal price assumptions are based on 10-year trailing averages (2014-2023): Pd price of US$1,380/oz, Pt price of US$1,100/oz, Rh price of US$6,200/oz, Au price of US$1,500/oz, Ni price of US$7,10/lb.
     - Palladium Equivalent (PdEq) Calculation
     - The PdEq equation is: PdEq = Pd g/t + F1 + F2 + F3 + F4

   Where: $\boldsymbol{F1} = \frac{(Pt_p * Pt_R)}{(Pd_p * Pd_R)} \boldsymbol{Pt_t}$, $\boldsymbol{F2} = \frac{(Rh_p * Rh_R)}{(Pd_p * Pd_R)} \boldsymbol{Rh_t}$, $\boldsymbol{F3} = \frac{(Au_p * Au_R)}{(Pd_p * Pd_R)} \boldsymbol{Au_t}$, $\boldsymbol{F4} = \frac{(Ni_p * Ni_R)}{(Pd_p * Pd_R)} \boldsymbol{Ni_t}$

   $_P$ = Metal Price

   $_R$ = Metallurgical Recovery

   - Costs are taken from comparable projects in GE21's extensive database of mining operations in Brazil, which includes not only operating mines, but recent actual costs from what could potentially be similarly sized operating mines in the Carajás. Costs considered a throughput rate of ca. 10Mtpa:
     - Mining costs: US$2.00/t oxide, US$3.00/t Fresh Rock. Processing costs: US$9.00/t fresh rock, US$7.50/t oxide. US$1.50/t processed, for General & Administration. US$1.00/t processed for grade control. US$0.50/t processed for rehabilitation.
     - Several of these considerations (metallurgical recovery, metal price projections for example) should be regarded as preliminary in nature, and therefore PdEq calculations should be regarded as preliminary in nature.
7. The current MRE supersedes and replaces the Previous Estimate (2023), which should be no longer relied upon.
8. The QP is not aware of political, environmental, or other risks that could materially affect the potential development of the Mineral Resources other than those typical for mining projects at this stage of development and as identified in this report.
9. Totals may not sum due to rounding.

Source: GE21, 2025.

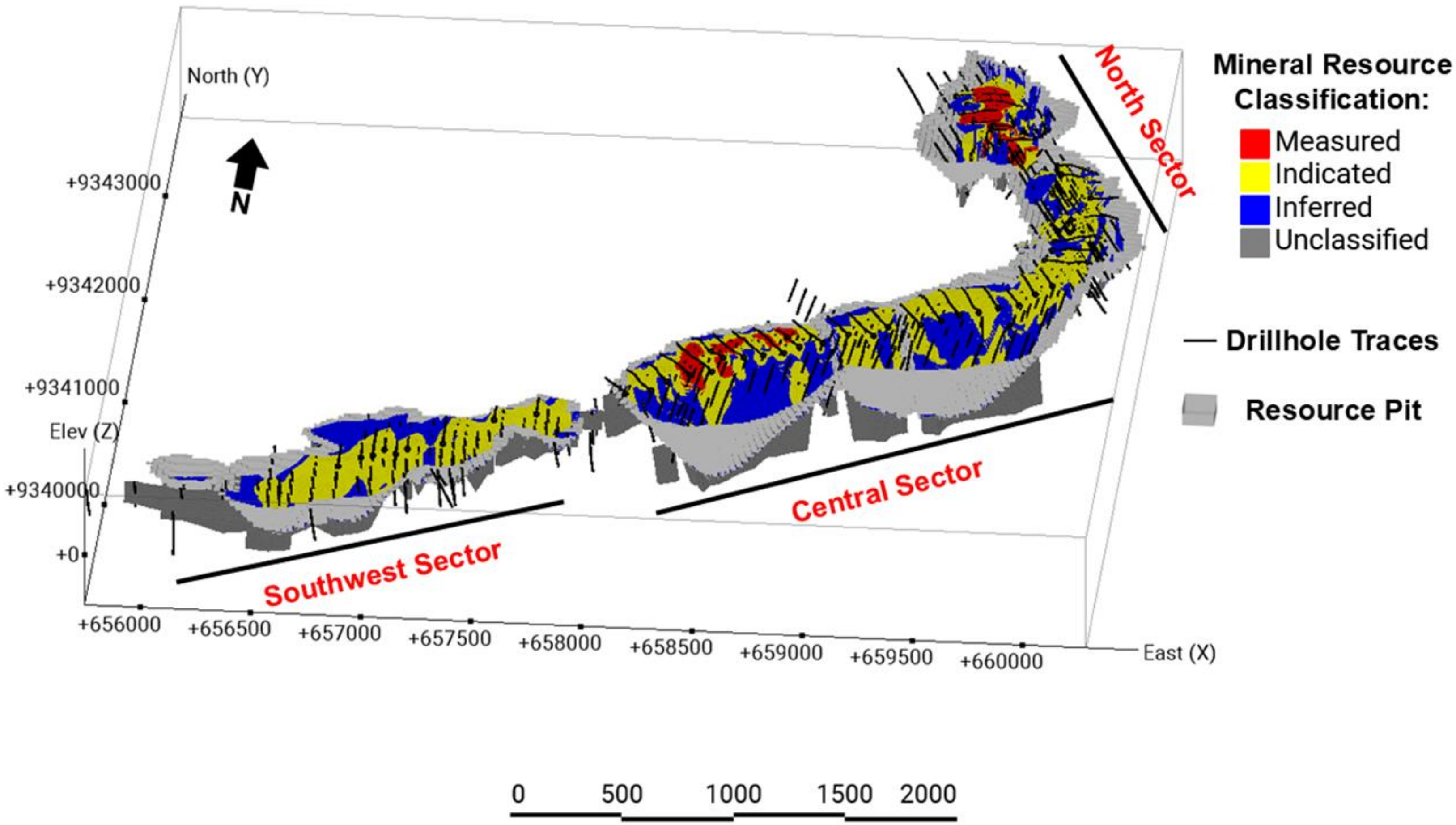


Figure 14-24: Mineral Resource classification 3D view

Source: GE21, 2024.

14.7 QP Opinion

Based on the validation methods employed, GE21 considers the results obtained for the estimate via Conditional Simulation acceptable and notes that no observable overall or local bias, as demonstrated by the NN and Swath Plot check analysis. GE21 also considers the quality of the data used for the estimate acceptable for classifying the Mineral Resources estimate.

If a further MRE upgrade is deemed desirable, GE21 recommends future work targets (see Section 26 - Recommendations for additional detail):

- Drilling at depth in areas where the constraining pit that encapsulates the reported Luanga MRE is limited due to the absence of drill data at depth.
- Further refinement of the geological and mineralogical models, which may result in unlocking a modest gain in MRE metal grades. An update of the mineralization geological model adopting an approach with implicit modelling methods and reducing domain internal dilution.
- Completion of the outstanding 2024 metallurgical test work, and 2025 metallurgical test work program.
- Mineral Resource estimation by the conditional simulation method defining the SMU to define the recoverable mineral resource.

15 MINERAL RESERVE ESTIMATES

The Project is currently classified as an advanced-stage exploration project. This PEA represents a preliminary technical and economic study of the potential viability of the Project and includes Inferred Mineral Resources. These resources are considered too speculative geologically to enable the application of economic considerations that would allow them to be classified as Mineral Reserves. There is no certainty that the economic forecasts presented in this PEA will be realized.

16 MINING METHODS

This section describes the conceptual mine plan used in this PEA level study, according CIM NI 43-101 Standards of Disclosure for Mineral Projects. Open pit mining was selected for mine production, with an approach that maximizes production while optimizing Project economics.

16.1 Introduction

The Luanga deposit was projected to be mined by conventional open pit mining methods for 17 years, at a plant feed rate of 5.1 Mt/y in the first year and 10 Mt/y from second year until end of life of mine. The in-pit measured+indicated Mineral Resources is 113 Mt grading 2.68 g/t Pd_Equ, and in-pit inferred Mineral Resources of 52 Mt grading 2.59 g/t Pd_Equ, based on long-term palladium, platinum, gold, rhodium and nickel selling prices of, respectively, US$ 1,271/oz, US$ 1,350/oz, US$ 3,096/oz, US$ 5,000/oz and US$ 9.07/lb.

Development of the LoM (life-of-mine) plan includes pit optimization, pit design, mine scheduling and the application of economic and metallurgical modifying factors to Indicated and Inferred Mineral Resources. The tonnages and grades reported in this report are inclusive of estimated ROM loss and mining dilution.

The Mineral Resources estimate, with effective date of February 18, 2025, was prepared by Porfirio Cabaleiro Rodriguez, BSc (Mine Eng), FAIG, a QP as defined under National Instrument 43-101 regulations.

Pit optimization was performed to establish the pit boundary for mining planning, applying reasonable modifying factors to the Mineral Resources estimate. A pit design was developed based upon operational and technical parameters, resulting in an estimated mine life of 23 years.

The final pit and the mine planning was based on pit optimization using GEOVIA Whittle software. The mining plan described in this report is based on Measured+Indicated and Inferred Mineral Resources constrained to the pit designed.

Mineable Resources are an estimate of the grade and tonnage in which the LoM plan was developed. Mineral Resource that are not Mineral Reserves do not have demonstrated economic viability.

Vegetation and topsoil will be removed and stockpiled for use in future site reclamation. The ROM (Run-of-Mine) and waste will be mined with 10 m high benches, focusing on reducing mining dilution and increasing ROM recovery.

Drilling and blasting operations are expected for production purposes. A staggered blast pattern forming equilateral triangles was selected for the blast design, as it provides optimal distribution of explosive energy and promotes efficient rock fragmentation. Blasting parameters were defined in accordance with this configuration and are summarized in Table 16-1.

The ROM and waste will be loaded into rigid frame off-road trucks and hauled to the processing plant. The waste will be disposed in a Waste Rock Stprage Facility (WRSF), and filtered tailings will be disposed within a Dry Stacking Facility (DSF).

The mineralized oxided material was not considered as ROM in this PEA study, as there is low evidence of its technical viability to the effective date of this report. This material will be stockpiled in a diferent area for future uses.

Drainage channels will be constructed along the mining area perimeters to collect rainwater and sediments, avoiding impacts on local terrain, vegetation and drainage. The channels will direct water to sumps located on the bottom of the pit, where it will be pumped for reuse.

Table 16-1: Hole Characteristics

Hole Characteristics	
Bench Height (m)	10
Hole Diameter (mm)	100
Subdrilling (m)	1
Total Hole Length m)	11
Explosive Characteristics	
ANFO Density (kg/m³)	850
Charge Density (kg/m)	7
Explosive Mass (kg)	57.52
Charge Length (m)	8.62
Blast Pattern Configuration	
Spacing	4.0
Burden	4.5

Source: GE21, 2025.

16.2 Pit Optimization

To define the boundary and production sequence for the subsequent design, a mining plan study was developed for pit optimization. This analysis uses geometric and economic criteria to determine the extent of the deposit that can be profitably mined and the corresponding cutoff grade.

16.2.1 Methodology

The pit optimization analysis was developed using the GEOVIA Whittle® software. The optimizer uses the Lerchs-Grossman algorithm to determine economic pit shells based on mining and processing cost inputs, revenue per block, and other parameters, as presented in Table 16-2. The costs were further detailed and developed throughout the PEA study; therefore, those presented in sections 21 and 22 may not fully correspond to the costs in this section.

GE21 Consultoria Mineral

Table 16-2: Pit optimization first pass parameters

Item			Unit	Value
Slope Angle	Saprolite		°	43
	Weathered (oxidized)		°	46
	Fresh Rock - poor quality		°	36
	Fresh Rock - medium quality		°	42
	Fresh Rock - good quality		°	55
Mining	Block Size (minimum)		X-Y-Z (m)	3.125 x 3.125 x 1.25
	Block Size (SMU)			12.5 x 12.5 x 5
	Density		g/cm³	Block Model
	Mining Recovery		%	95
	Mining Dilution		%	5
	Physical Constraints			ANM Mineral Rights Road (10m buffer) Electrical Transmission Line (100m buffer)
Processing	Metallurgic Recovery - Fresh	Pd	Mill	77%
		Pt	Mill	81%
		Rh	Mill	52%
		Au	Mill	50%
		Ni	Mill	62%
	Metallurgic Recovery - High Talc	Pd	Mill	51%
		Pt	Mill	55.5%
		Rh	Mill	27.3%
		Au	Mill	27%
		Ni	Mill	0%
Smelter Recoveries		Pd	%	99%
		Pt	%	99%
		Rh	%	95%
		Au	%	95%
		Ni	%	98%
Costs	Weathered	Mining Cost	US$/t mined	2.10
	Fresh / High Talc	Mining Cost	US$/t mined	Waste=2.9 e ROM= 3.2
		Processing Cost	US$/t ROM	7.64
		Grade Control		1.00
		Rehabilitation		1.00
		G&A		5.00
		Thickening and filtration dry stack		4.00
		Smelting and Refining	US$/t concentrate	251
		Smelter G&A		80
Selling	Price	Pd	US$/t.oz	1,271
		Pt	US$/t.oz	1,350
		Rh	US$/t.oz	5,000
		Au	US$/t.oz	3,096
		Ni	US$/lb	9.07
	Royalties	All	%	2.0

Source: GE21, 2025.

16.2.2 Economic Function

The economic function is the estimate of the value of a block, based on the costs, selling prices, and qualities of the products expected to be produced. The mineralized block receives the difference between the sum of all revenue generated from the sale of products and the sum of all mining, processing, and selling expenses.

The economic function is represented by the following equation:

$$EBV = (To \times g \times re \times P - To \times PC) - T \times MC$$

Where:

- EBV = economic block value,
- To = amount of above cut-off mineralized material in the block (tonne),
- g = grade (% or ppm),
- re = recovery, the proportion of product recovered by processing the above cut-off mineralized material (%),
- P = the price obtainable per unit of product sold ($\$ \times tonne^{-1}$),
- MC = the cost of mining a tonne of waste ($\$ \times tonne^{-1}$),
- PC = cost per tonne of mining the material as as above cut-off mineralized material and processing it ($\$ \times tonne^{-1}$),
- T = total amount of above cut-off mineralized material and waste in the block (To + Tw) (tonne), and
- Tw = amount of waste in the block (tonne).

16.2.3 Optimization Results

A serie of 18 pit shells was generated by varying the revenue factor from 30% to 200%. The pit associated with a revenue factor of 100% (highlighted in Table 16-3 and Figure 16-1) was selected to guide the final pit design. Figure 16-2 shows the selected pit shell boundaries and Mineral Resources distribution.

Table 16-3: Pit optimization results from GEOVIA Whittle®

Revenue Factor	ROM	Waste	SR	Total Moved	Pd [2]	Pt [2]	Au [2]	Rh [2]	Ni [2]	NPV [1]
	(Mt)		(t/t)	(Mt)	(g/t)				(%)	(MUS$)
30%	2	5	2.78	7	1.91	1.05	0.10	0.17	0.20	193
40%	38	93	2.48	131	1.35	0.72	0.06	0.11	0.19	2,481
50%	78	242	3.10	320	1.23	0.73	0.06	0.11	0.16	4,425
60%	110	432	3.95	542	1.20	0.71	0.06	0.11	0.15	5,614
70%	126	563	4.48	689	1.18	0.70	0.06	0.10	0.15	6,030
80%	141	708	5.04	849	1.16	0.69	0.06	0.10	0.15	6,290
90%	157	842	5.36	999	1.13	0.66	0.06	0.10	0.15	6,411
100%	**171**	**906**	**5.28**	**1,077**	**1.07**	**0.64**	**0.06**	**0.09**	**0.14**	**6,430**
110%	181	936	5.16	1,117	1.04	0.62	0.05	0.09	0.14	6,422
120%	191	948	4.96	1,139	1.00	0.60	0.05	0.09	0.14	6,406
130%	199	954	4.79	1,153	0.97	0.58	0.05	0.08	0.13	6,390
140%	207	969	4.69	1,175	0.95	0.57	0.05	0.08	0.13	6,354
150%	209	977	4.67	1,186	0.94	0.57	0.05	0.08	0.13	6,336
160%	210	985	4.68	1,195	0.94	0.57	0.05	0.08	0.13	6,320
170%	211	991	4.69	1,202	0.94	0.56	0.05	0.08	0.13	6,308
180%	212	995	4.70	1,206	0.94	0.56	0.05	0.08	0.13	6,299
190%	212	999	4.71	1,211	0.94	0.56	0.05	0.08	0.13	6,291
200%	212	1,002	4.72	1,214	0.94	0.56	0.05	0.08	0.13	6,284

Notes:
1. NPV figures on a comparative basis. No economic assessment was developed.
2. Grade results include 5% dilution factor.

Source: GE21, 2025.

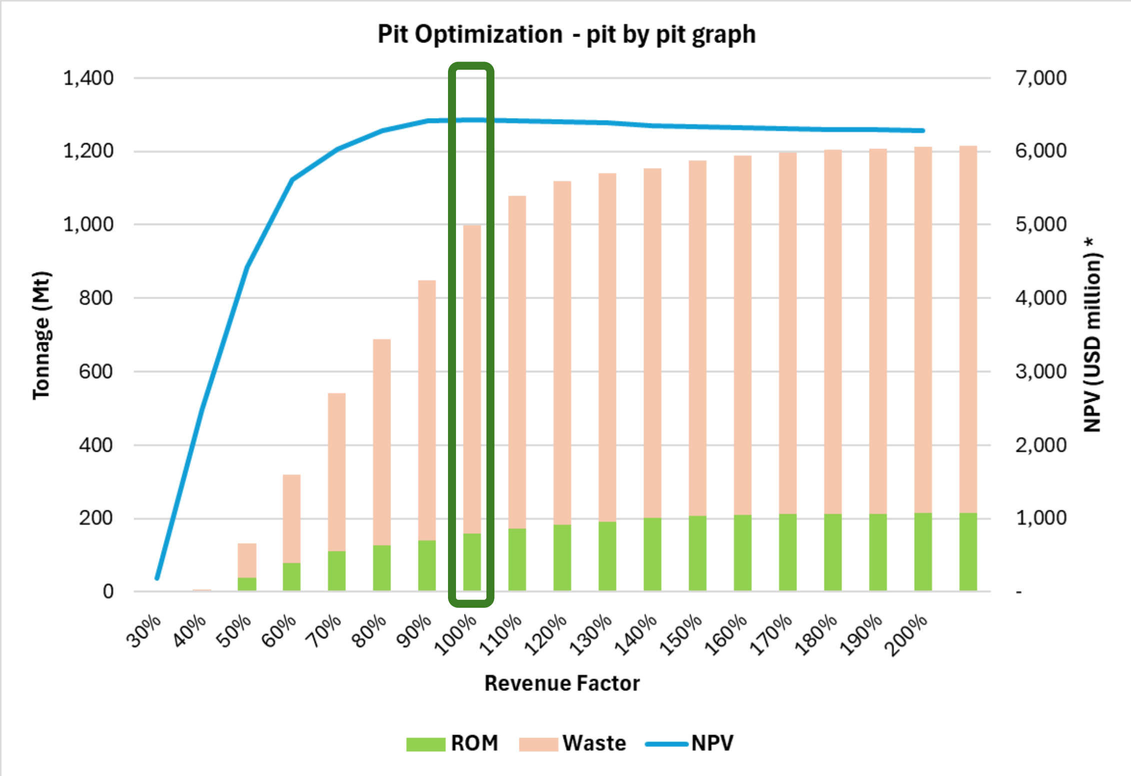


Figure 16-1: Pit-by-pit chart

Notes:
1. NPV figures on a comparative basis. No economic assessment was developed.

Source: GE21, 2025.

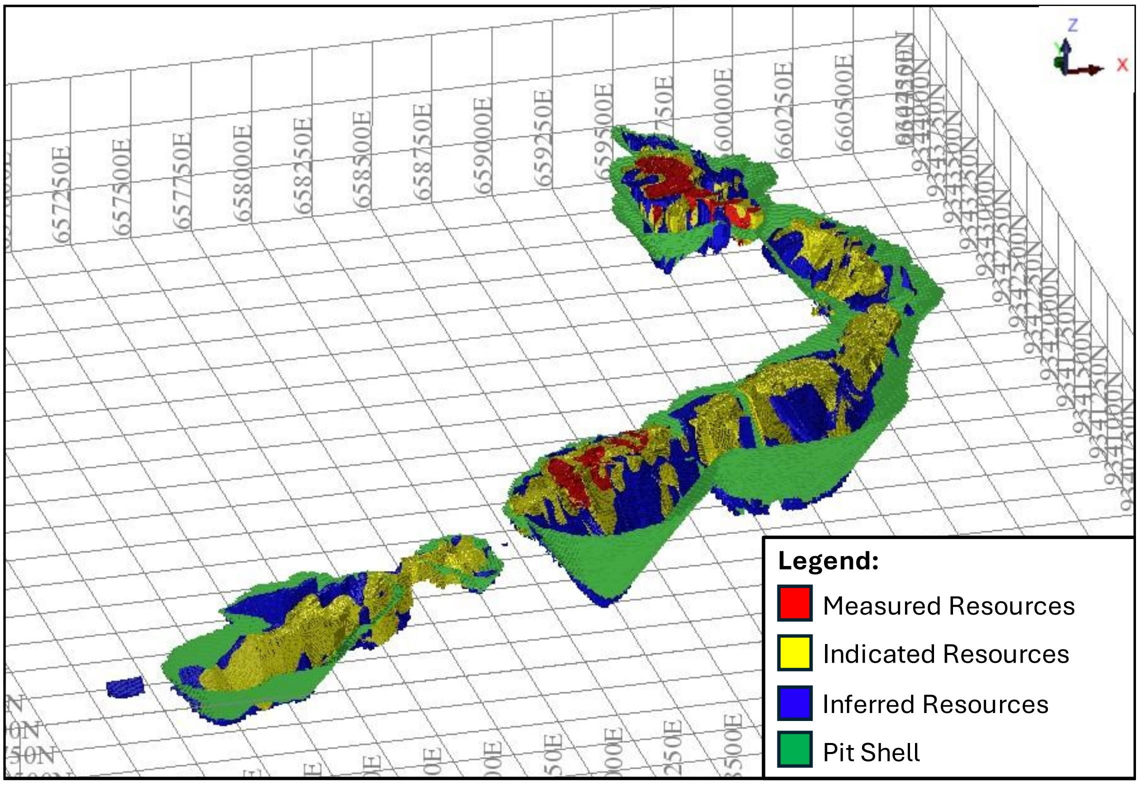


Figure 16-2: Selected pit shell boundaries

Source: GE21, 2025.

A cutoff grade was estimated based on the First Pass Parameters presented in Table 16-2, and an estimation of Palladium Equivalent (PdEq) grade. The PdEq equation used is:

$$\mathbf{PdEq = Pd\ (g/t) + F1 + F2 + F3 + F4}$$

Where:

$$F1 = \frac{(Pt_p * Pt_R)}{(Pd_p * Pd_R)} Pt_t \quad F2 = \frac{(Rh_p * Rh_R)}{(Pd_p * Pd_R)} Rh_t \quad F3 = \frac{(Au_p * Au_R)}{(Pd_p * Pd_R)} Au_t \quad F4 = \frac{(Ni_p * Ni_R)}{(Pd_p * Pd_R)} Ni_t$$

And:

$_P$ = Metal Price
$_R$ = Metallurgical Recovery

The estimated cutoff grade for the Mine Planning development is 0.87 g/t Pd_Equ.

The selected pit shell resulted in 172.3 Mt of ROM at 2.80 g/t Pd_Equ at the estimated cutoff. The material below the cutoff was designated as Opportunity Resources in this PEA report, as it presents economic potential depending on market conditions.

The WRSF was designed considering the mined volumes of both waste and Opportunity Resources, which should be disposed separately within the correspondent dumping area.

Table 16-4 presents the material within the pit shell, applying the cutoff of 0.87 g/t Pd_Equ. These are *in situ* figures: mining dilution and recovery factors were not applied.

Table 16-4: Pit shell results using a cutoff of 0.87 g/t PdEq

Resource Classification	Mass	Pd	Pt	Au	Rh	Ni	Pd Eq
	(Mt)	(g/t)				(%)	(g/t)
Measured	25	1.15	0.76	0.05	0.10	0.13	2.85
Indicated	91	1.12	0.64	0.06	0.10	0.16	2.81
Measured + Indicated	**116**	**1.12**	**0.67**	**0.06**	**0.10**	**0.15**	**2.82**
Inferred	**56**	**1.13**	**0.65**	**0.06**	**0.09**	**0.15**	**2.78**
Low Grade	25						
Waste	884						
Strip Ratio	5.3						
Total Moved	1,081						

* Cutoff grade of 0.87 g/t Pd_Eq applied

Source: GE21, 2025.

16.3 Final Pit Design

The next step in mining plan development is to design an operational pit as the basis for the production plan. The pit was designed based on the selected pit shell and includes bench-by-bench toes and crests and ramps to access the pit bottoms, resulting in a representation of the surface where the selected equipment will be able to operate efficiently and safely.

16.3.1 Methodology

The layout of the toes and crests of benches, safety berms, minimum operational areas, and access ramps for the mining operation was established, respecting the defined geometric and geotechnical parameters. The two main objectives were to minimize the loss of ROM material, assuming some increase in waste mining where it was necessary to reach ROM areas at lower benches, and define access and ramps to optimize the average haulage distances.

16.3.2 Geometric and Geotechnical Parameters, Groundwater

The geometric and geotechnical parameters were defined based on the specific features of the mined materials. Table 16-5 presents the parameters used for pit designs.

Table 16-5: Pit design geometric and geotechnical parameters

Parameter	Saprolite	Oxidized	Fresh Rock - poor quality	Fresh Rock - medium quality	Fresh Rock - good quality
Bench Slope (°)	63	68	51	61	81
Overall Slope (°)	43	46	36	42	55
Bench Height (m)	10				
Berm Width (m)	6				
Ramp Width (m)	25				
Ramp Gradient (%)	10%				
Minimum Bottom Width (m)	45				

Source: Luis Navarro, 2023.

Insufficient information is available to determine whether groundwater would affect pit designs but, based on information available from exploration drilling and nearby operating properties, it is not expected to be a significant issue.

16.3.3 Final PEA Pit Design Results

Figure 16-3 presents the final PEA pit design, and Table 16-6 lists the total material to be mined. An operational mining recovery of 95% and operational mining dilution of 5% were applied based on the characteristics of the pit and material and on projects with similar operational features.

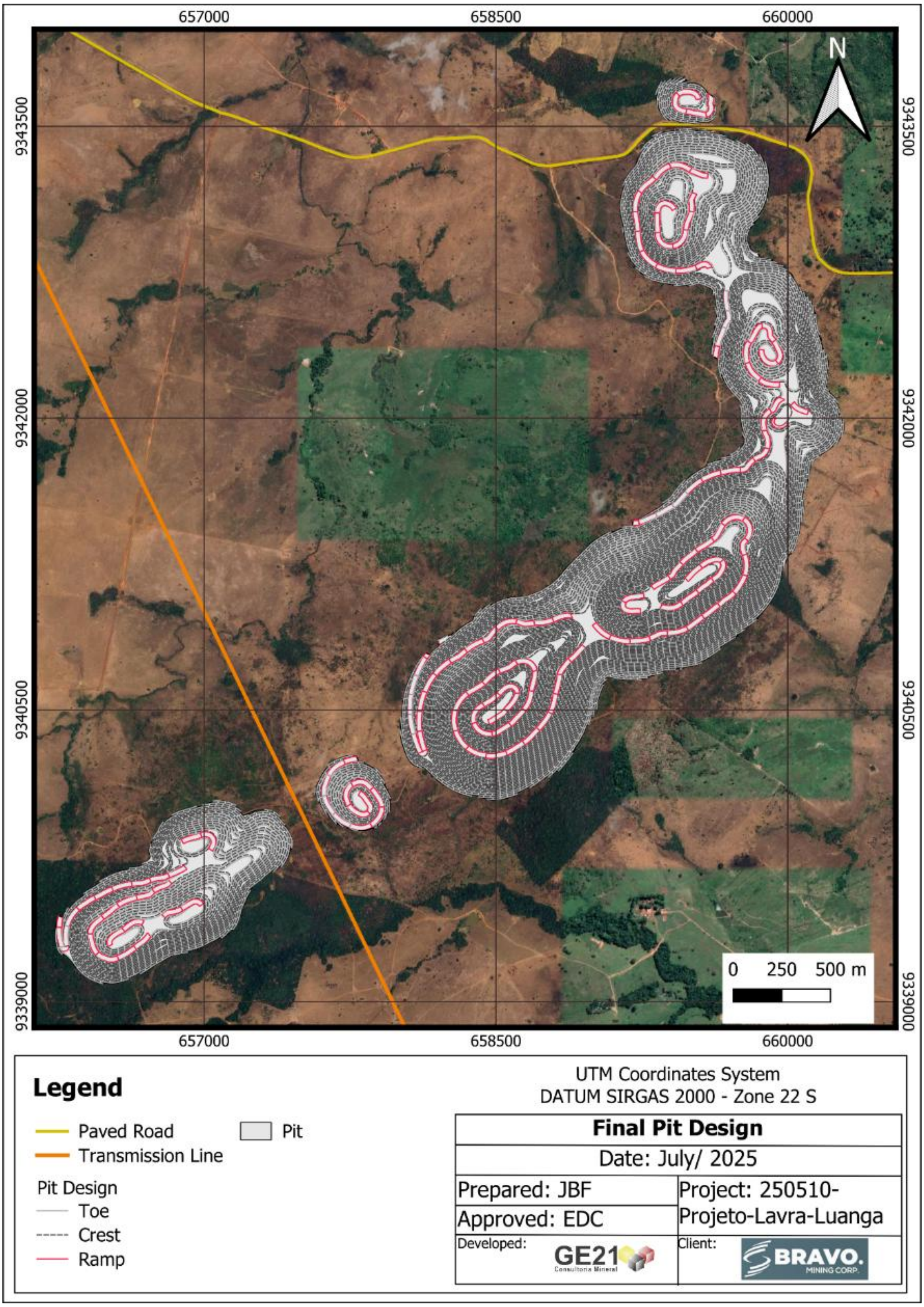


Figure 16-3: Final pit design

Source: GE21, 2025.

GE21 Consultoria Mineral

Table 16-6: PEA final pit results

Resource Classification	Type	Mass	Pd	Pt	Au	Rh	Ni	Pd Eq
		Mt	g/t				%	g/t
Measured	Fresh	24	1.09	0.72	0.05	0.10	0.12	2.72
	High Talc	-	-	-	-	-	-	-
	Subtotal	**24**	**1.09**	**0.72**	**0.05**	**0.10**	**0.12**	**2.72**
Indicated	Fresh	87	1.06	0.61	0.06	0.10	0.15	2.68
	High Talc	1	1.15	0.54	0.12	0.08	0.14	2.08
	Subtotal	**89**	**1.06**	**0.61**	**0.06**	**0.10**	**0.15**	**2.67**
Measured + Indicated	Fresh	111	1.07	0.64	0.05	0.10	0.14	2.69
	High Talc	1	1.15	0.54	0.12	0.08	0.14	2.08
	Total	**113**	**1.07**	**0.63**	**0.06**	**0.10**	**0.14**	**2.68**
Inferred	Fresh	52	1.05	0.61	0.05	0.09	0.14	2.59
	High Talc	0	1.10	0.53	0.11	0.07	0.14	2.00
	Subtotal	**52**	**1.05**	**0.61**	**0.05**	**0.09**	**0.14**	**2.59**
Low Grade		30						
Waste		1,124						
Strip Ratio		7.0						
Total Moved		**1,319**						

Notes:
1. Estimated cutoff grade: 0.87 g/t Pd_Eq
2. Mining recovery factor: 95%
3. Mining dilution factor: 5%

Source: GE21, 2025.

16.4 Mine Scheduling

Mine scheduling development focused on operational features of the mining operations and optimization of concentrate production. The production rate was defined as 10.0 Mt/year of ROM delivered to the processing plant.

The Mining Plan includes a pre-stripping period (year 0) with the development of 10 Mt of waste, and a ramp-up period of 1 year at 50% of the full production capacity.

A minimum of three active mining faces were considered to optimize the number of trucks accessing the same areas of the pit and corresponding ramps and roads, thereby increasing productivity. At the same time, higher grade and lower strip ratio zones were prioritized at the beginning of Life-of-Mine, to optimize the project's economics during the payback period.

The detailed mining sequence was developed using GEOVIA Minesched® software, with the following periods:

- Years 0 to 5: yearly basis
- Years 6 to 15: 5-year basis
- Years 16 and 17: last period

Figure 16-4 and Table 16-7 present the Mine Scheduling results.

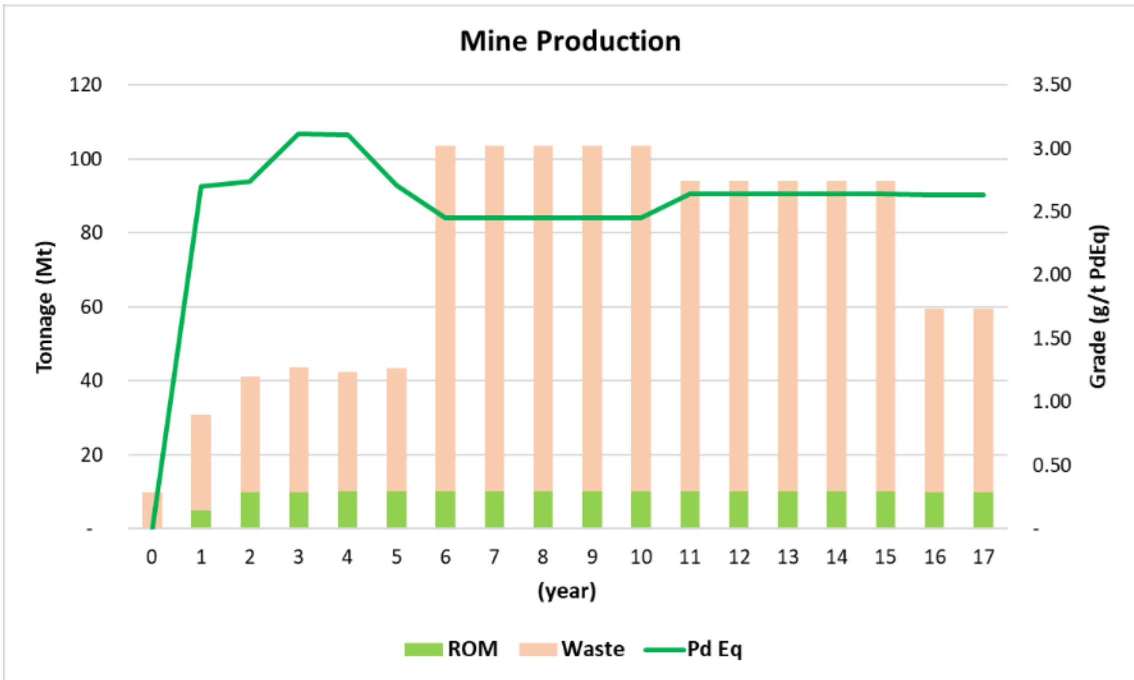


Figure 16-4: Mine scheduling results - graph

Source: GE21, 2025.

Table 16-7: Mine scheduling results - table

Year	ROM						PdEq	Waste		Total Moved	Strip Ratio
	Tonnage	Pd	Pt	Rh	Au	Ni	PdEq	weathered	fresh		
	(Mt)	(g/t)				(%)	(g/t)	Mt			(t/t)
0	-	-	-	-	-	-	-	5.0	5.0	10.0	-
1	5.1	1.09	0.63	0.10	0.07	0.14	2.70	14.9	10.8	30.7	5.1
2	10.0	1.10	0.63	0.09	0.07	0.15	2.74	14.1	17.0	41.1	3.1
3	10.0	1.28	0.63	0.09	0.07	0.20	3.12	9.5	24.3	43.7	3.4
4	10.0	1.32	0.64	0.10	0.06	0.18	3.10	11.1	21.3	42.4	3.2
5	10.1	1.02	0.69	0.11	0.05	0.14	2.71	2.4	31.0	43.5	3.3
6	10.0	1.00	0.60	0.09	0.05	0.13	2.46	8.3	85.3	103.6	9.3
7	10.0	1.00	0.60	0.09	0.05	0.13	2.46	8.3	85.3	103.6	9.3
8	10.0	1.00	0.60	0.09	0.05	0.13	2.46	8.3	85.3	103.6	9.3
9	10.0	1.00	0.60	0.09	0.05	0.13	2.46	8.3	85.3	103.6	9.3
10	10.0	1.00	0.60	0.09	0.05	0.13	2.46	8.3	85.3	103.6	9.3
11	10.0	1.03	0.65	0.10	0.05	0.14	2.64	0.0	84.0	94.0	8.4
12	10.0	1.03	0.65	0.10	0.05	0.14	2.64	0.0	84.0	94.0	8.4
13	10.0	1.03	0.65	0.10	0.05	0.14	2.64	0.0	84.0	94.0	8.4
14	10.0	1.03	0.65	0.10	0.05	0.14	2.64	0.0	84.0	94.0	8.4
15	10.0	1.03	0.65	0.10	0.05	0.14	2.64	0.0	84.0	94.0	8.4
16	9.9	1.09	0.59	0.09	0.05	0.14	2.64	0.0	49.6	59.5	5.0
17	9.9	1.09	0.59	0.09	0.05	0.14	2.64	0.0	49.6	59.5	5.0
Total	**165.3**	**1.06**	**0.63**	**0.09**	**0.06**	**0.14**	**2.65**	**98.2**	**1,055.0**	**1,318.6**	**7.0**

Notes:
1. Estimated cutoff grade: 0.87 g/t Pd_Eq
2. Mining recovery factor: 95%
3. Mining dilution factor: 5%

Source: GE21, 2025.

Based on the detailed mining sequence, the end-of-period pit designs were developed for years 1, 2, 3, 5, 10, 15, and the previously presented final pit (Figure 16-5 to Figure 16-11).

GE21 Consultoria Mineral

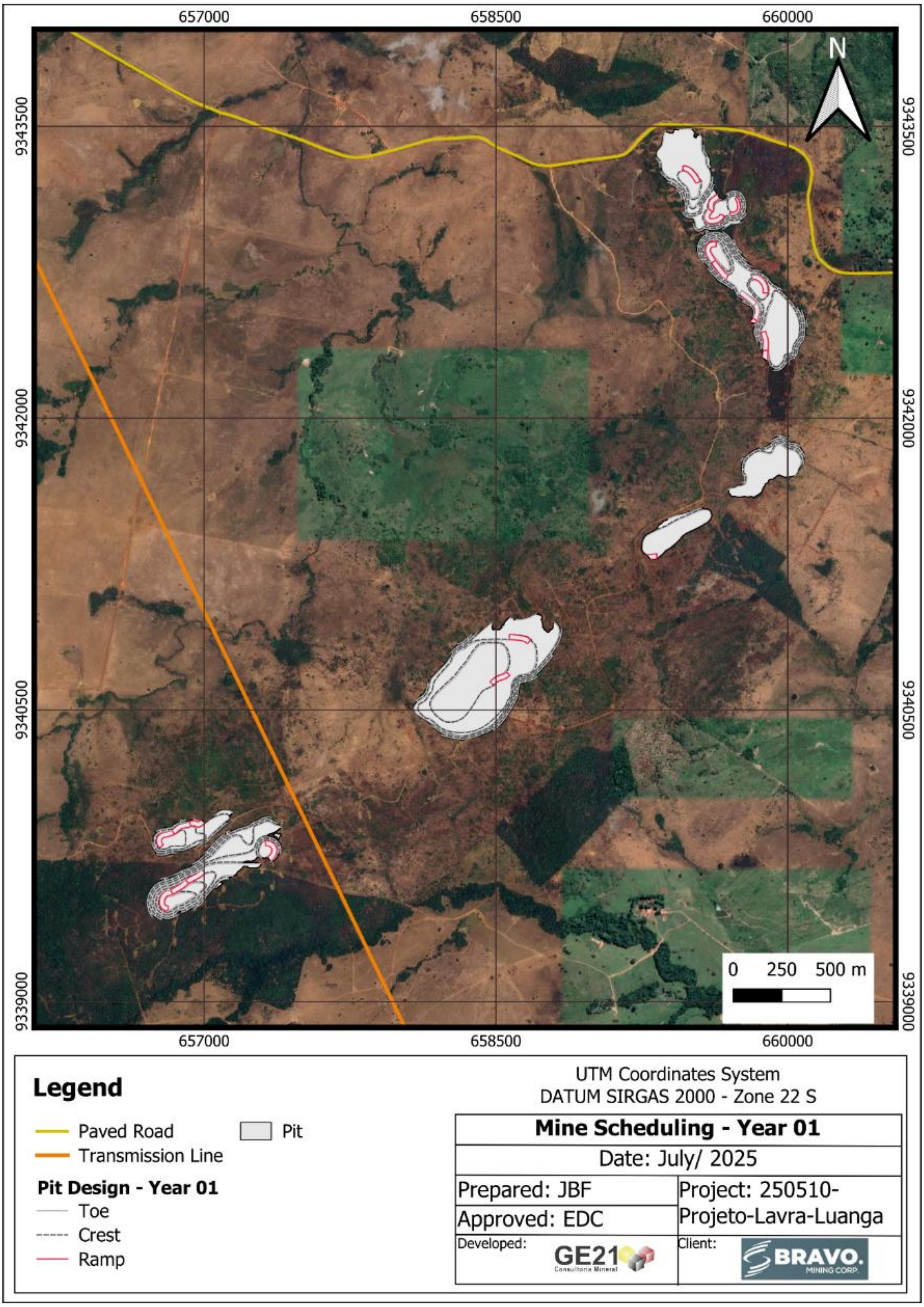


Figure 16-5: Mining schedule design – Year 1

Source: GE21, 2025.

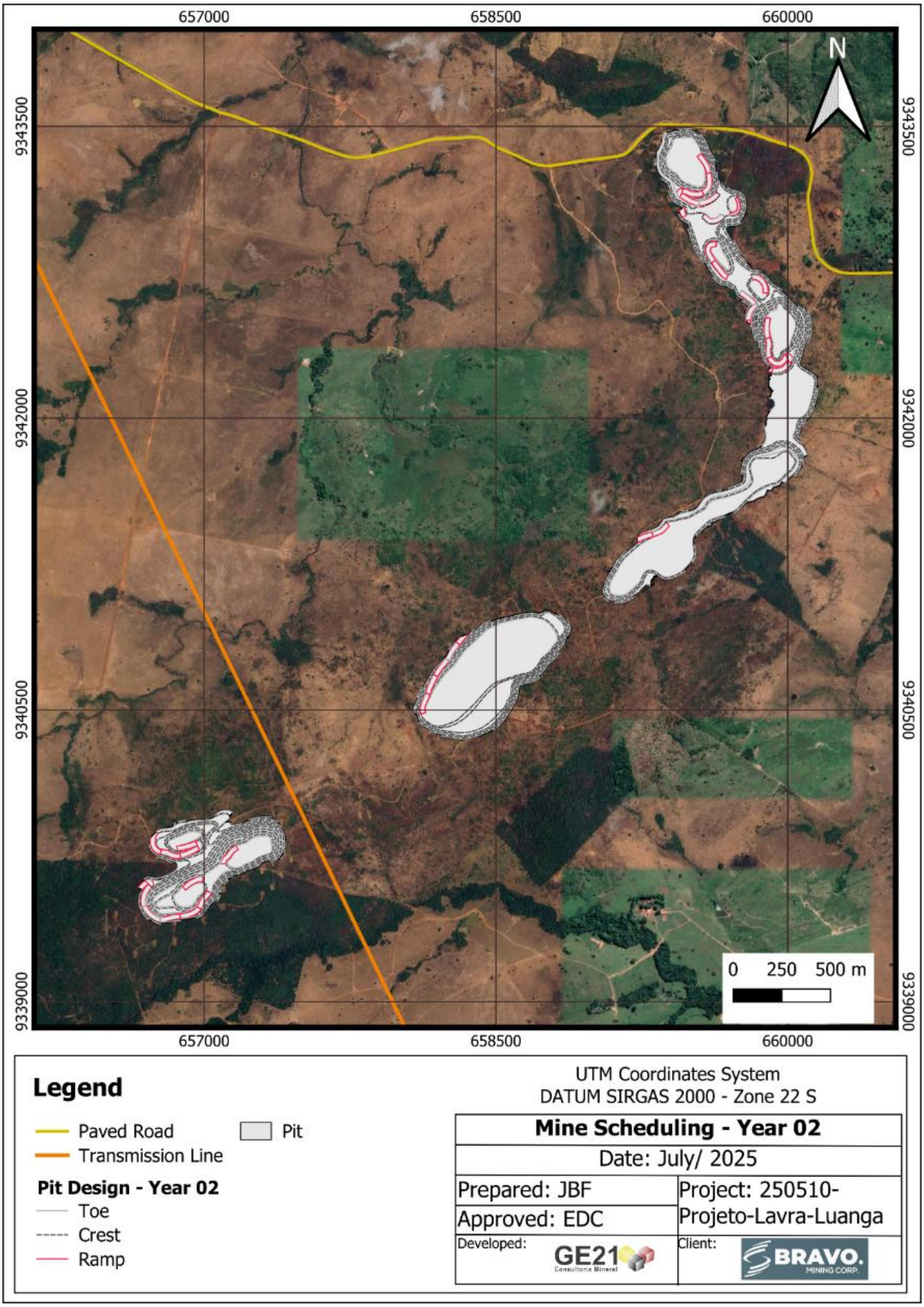


Figure 16-6: Mining schedule design – Year 2

Source: GE21, 2025.

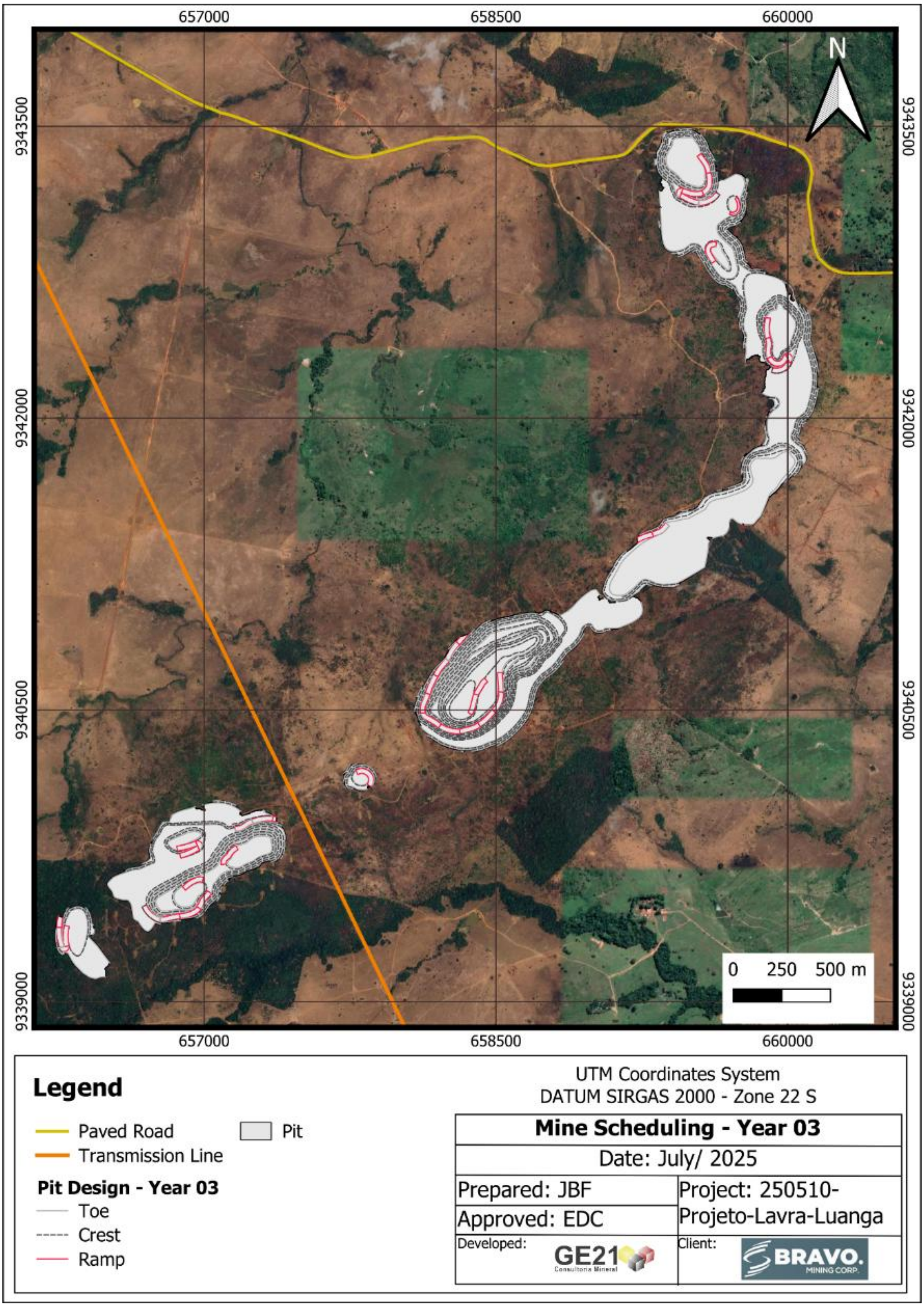


Figure 16-7: Mining schedule design – Year 3

Source: GE21, 2025.

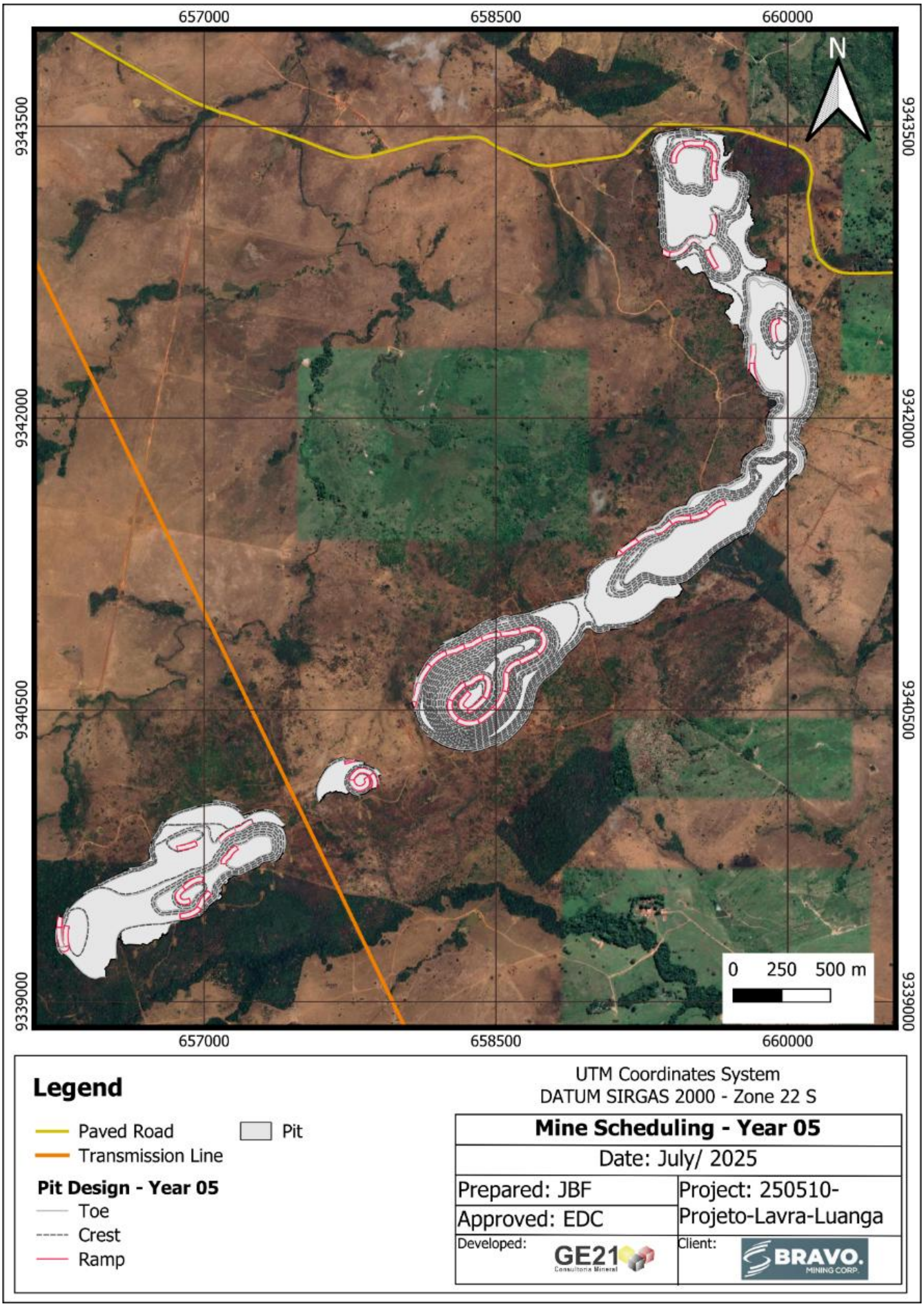


Figure 16-8: Mining schedule design – Year 5

Source: GE21, 2025.

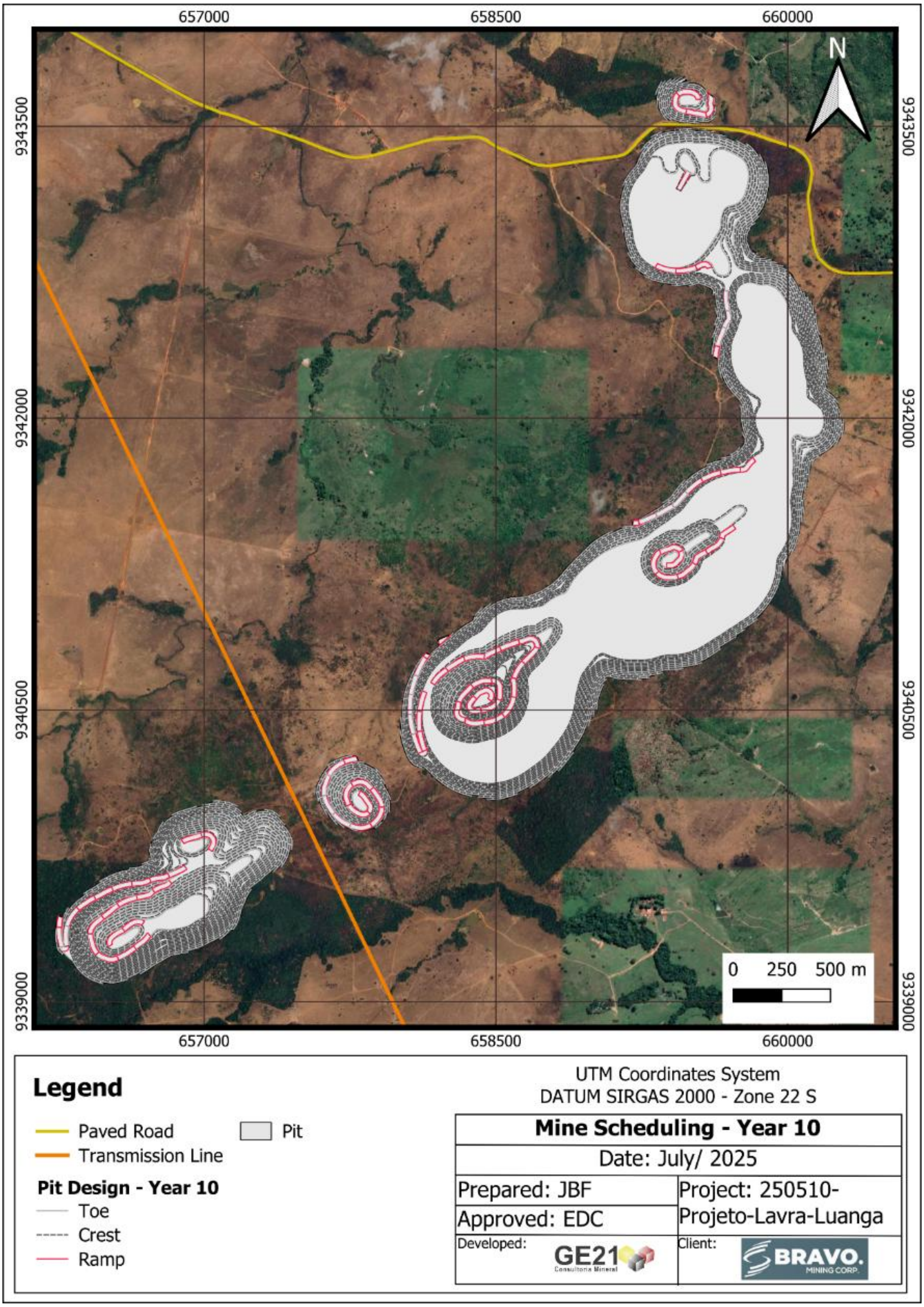


Figure 16-9: Mining schedule design – Year 10

Source: GE21, 2025.

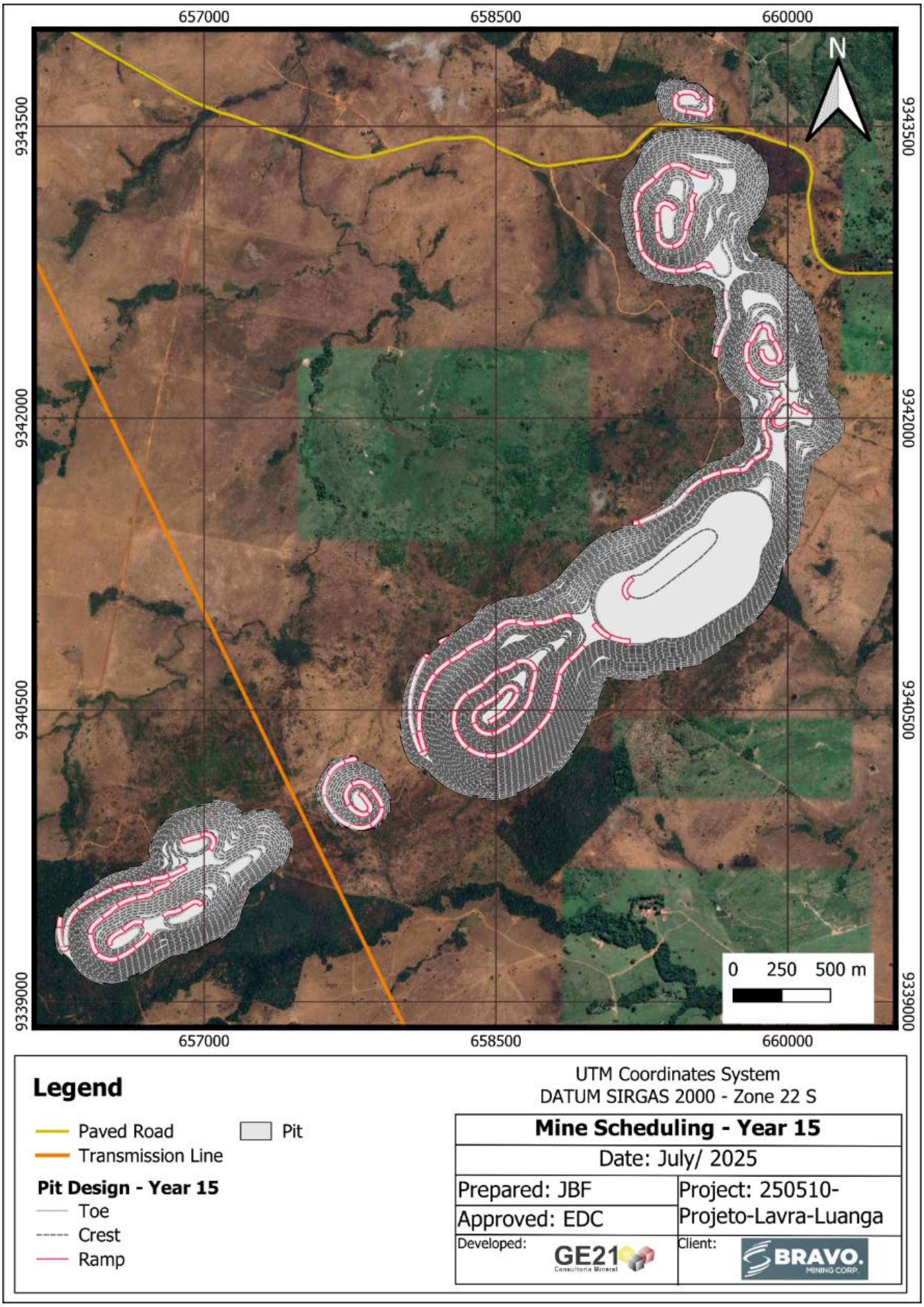


Figure 16-10: Mining schedule design – Year 15

Source: GE21, 2025.

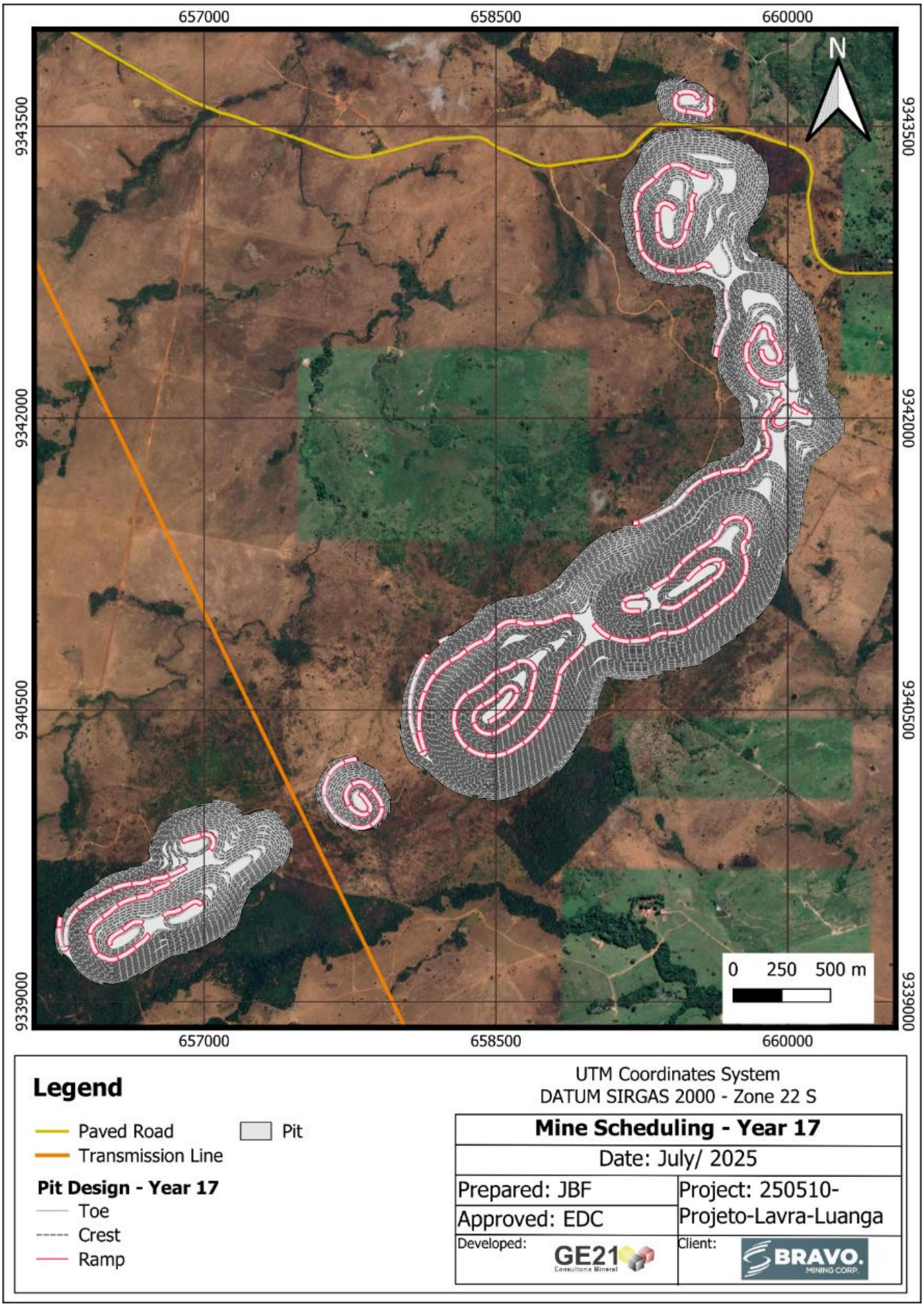


Figure 16-11: Mining schedule design – Year 17 (final pit)

Source: GE21, 2025.

16.5 Waste and Tailings Storage Facility

The waste material from the pit will be disposed off in a WRSF. The disposed volume was estimated considering a 5% compaction factor and 20% swelling factor.

A DSF will store filtered tailings from the processing plant. The disposed volume was estimated considering a 5% compaction factor and 15% swelling factor.

The conceptual designs were developed using assumed parameters based on the characteristics of the materials to be disposed of and locational features, as presented in Table 16-8. These parameters must be confirmed after comprehensive geotechnical studies.

Table 16-8: WRSF and DSF design parameters

Parameter	Waste	Tailings	Unit
Overall Slope	21	18	°
Bench Slope	29	26	°
Bench Height	10	10	m
Minimum Berm Width	7.5	10	m
Ramp Gradient	10		%
Ramp Width	25		m
Compaction factor	5		
Swelling factor	20	15	%

Source: GE21, 2025.

The conceptual WRSF sequence was developed with the following periods:

- Year 1
- Year 5
- Year 17 – end of LoM

The conceptual DSF sequence was developed considering that filtered stacking will start in year 3. The corresponding design were prepared with the following periods:

- Year 3
- Year 7
- Year 17 – end of LoM

Figure 16-12 to Figure 16-17 present the corresponding designs.

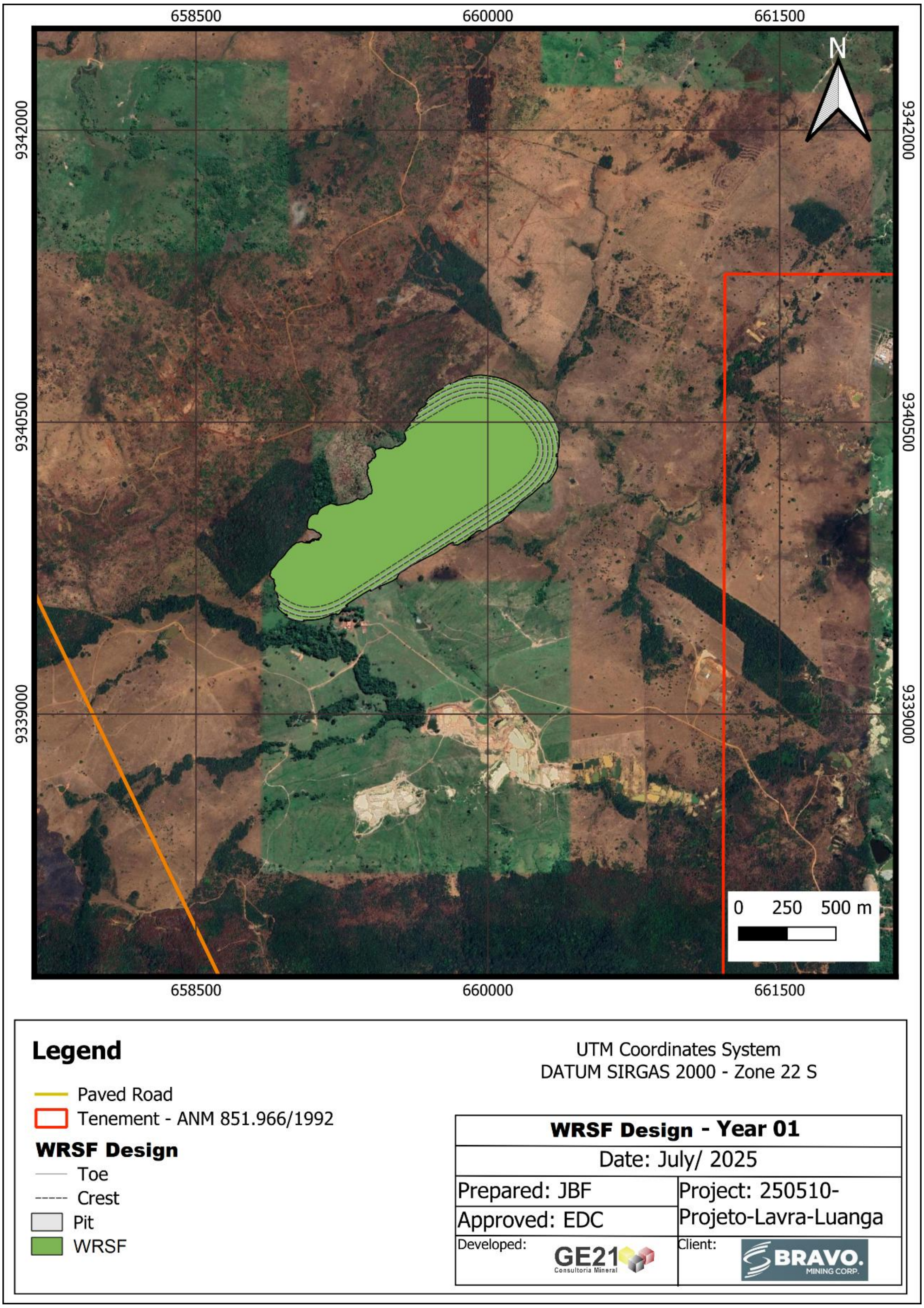


Figure 16-12: WRSF sequence – Year 1

Source: GE21, 2025.

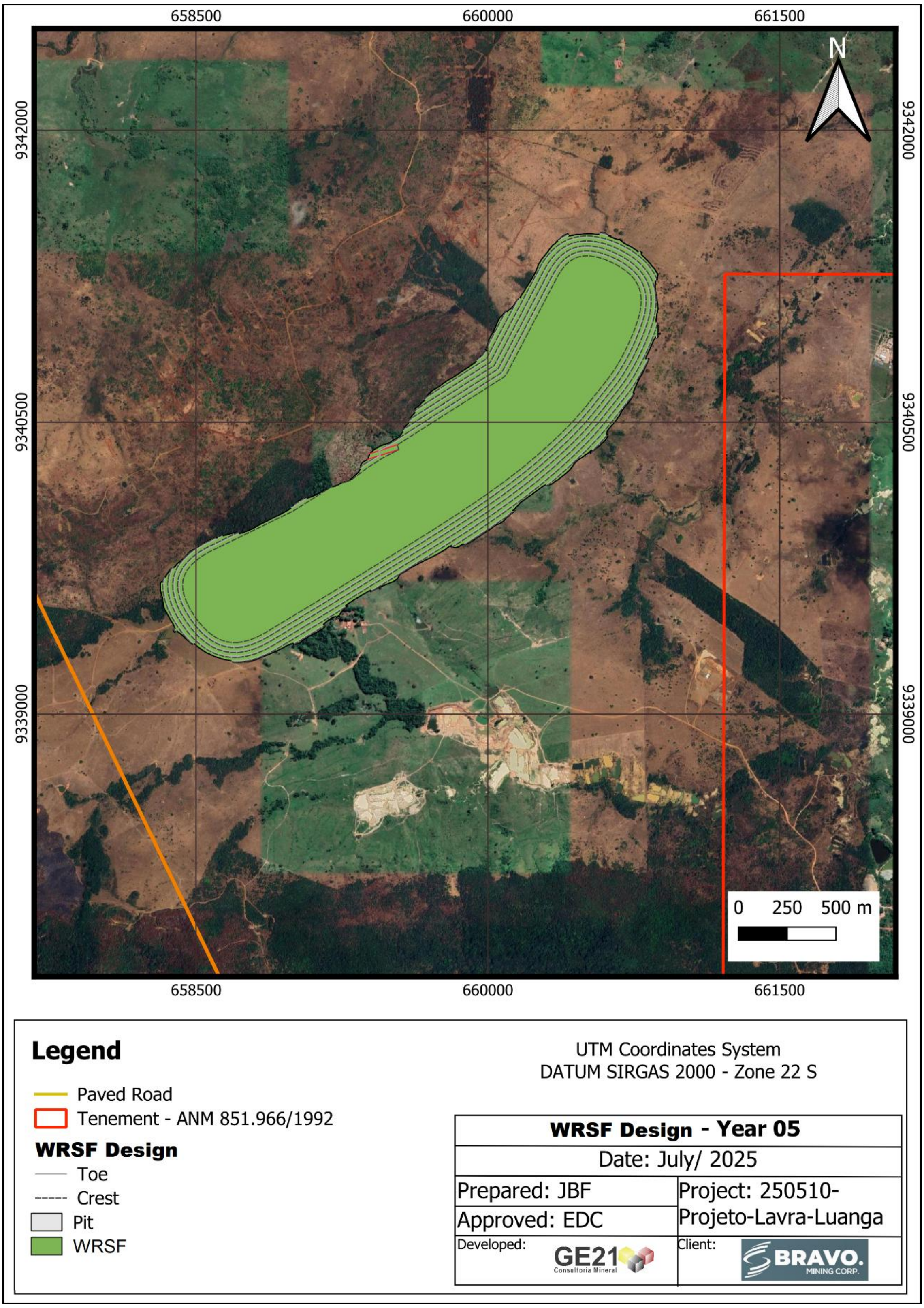


Figure 16-13: WRSF sequence – Year 5

Source: GE21, 2025.

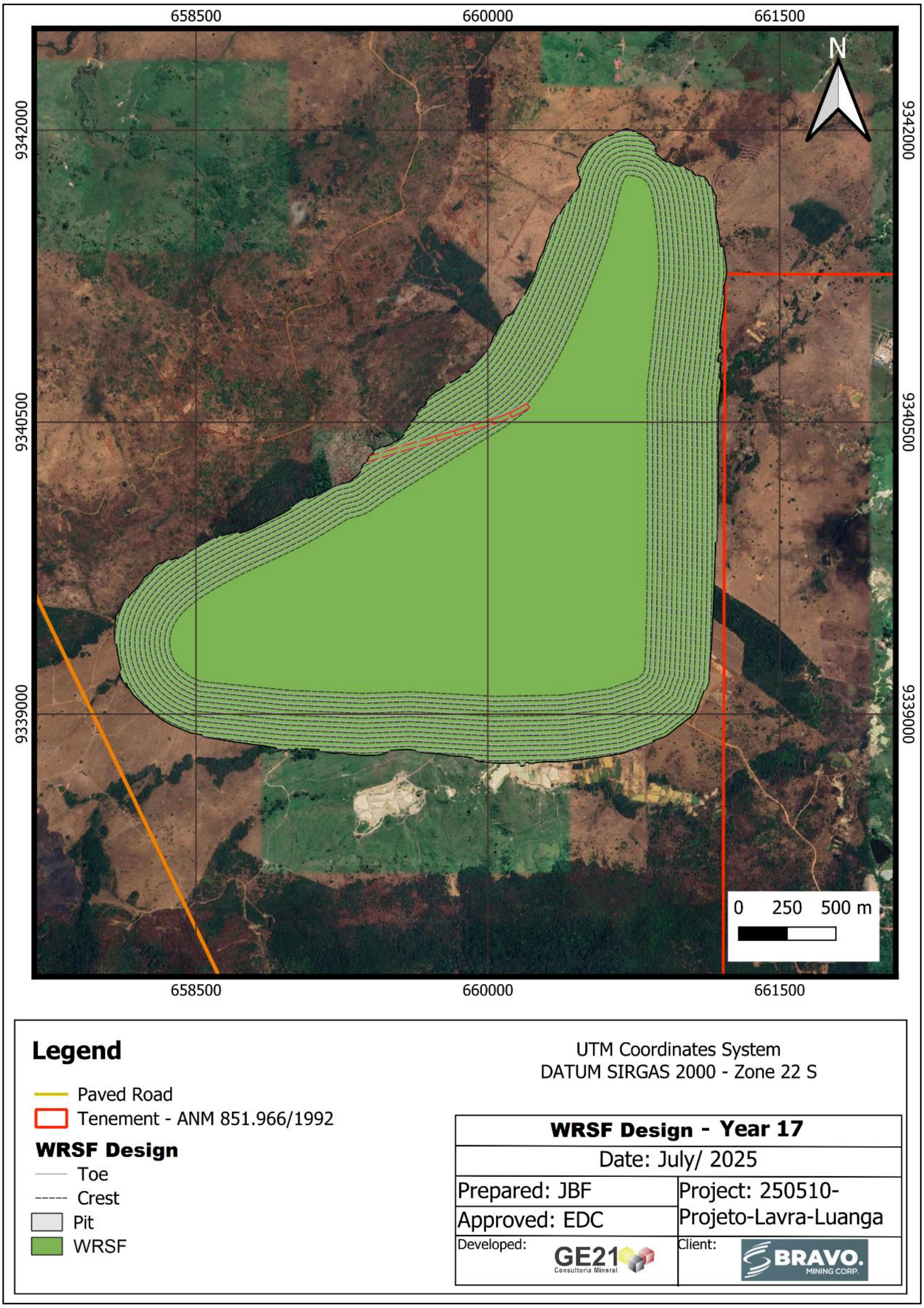


Figure 16-14: WRSF sequence – Year 17 (end of LoM)

Source: GE21, 2025.

GE21 Consultoria Mineral

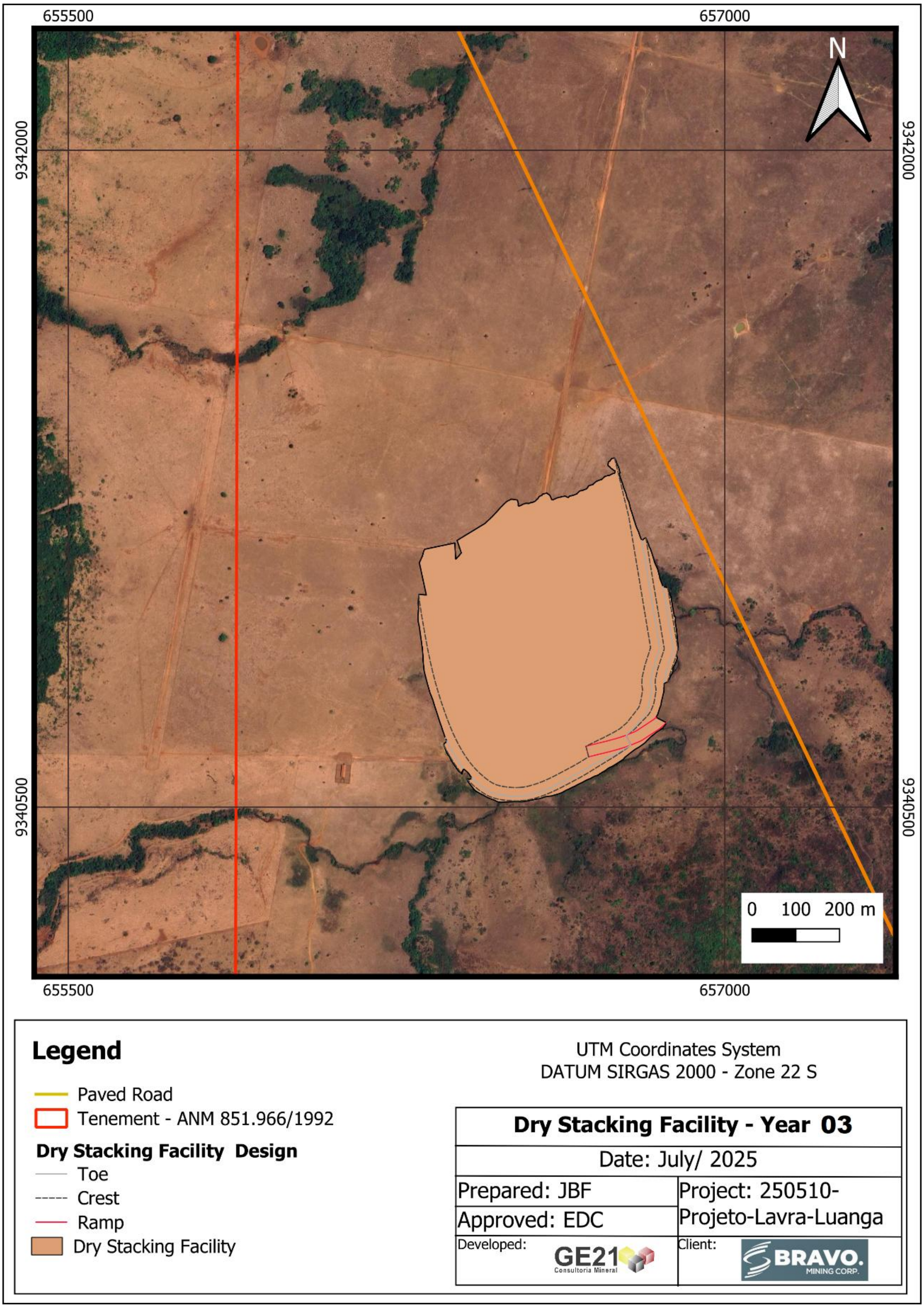


Figure 16-15: Dry Stacking Facility sequence – Year 3

Source: GE21, 2025.

GE21 Consultoria Mineral

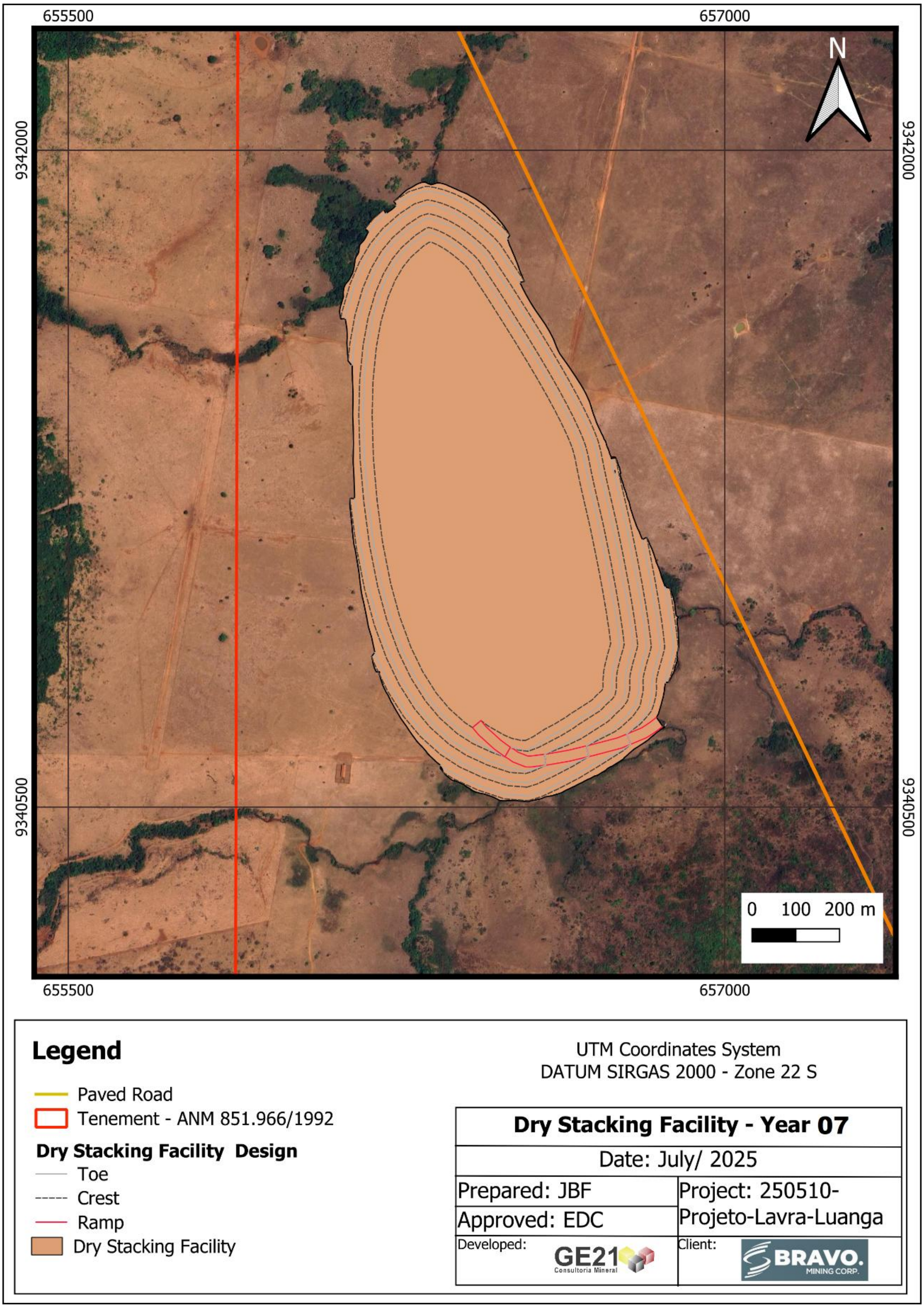


Figure 16-16: Dry Stacking Facility sequence – Year 7

Source: GE21, 2025.

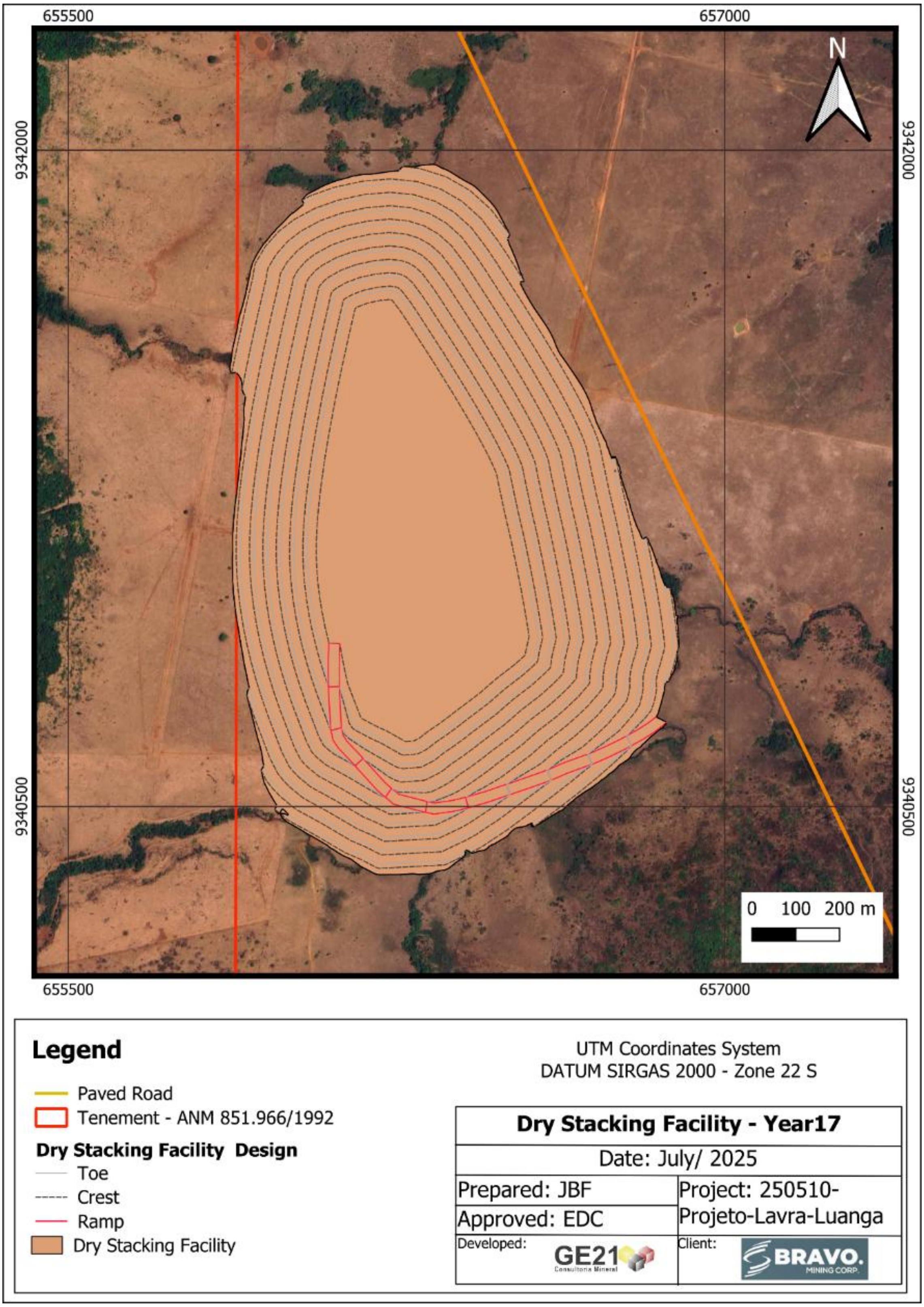


Figure 16-17: Dry Stacking Facility sequence – Year 17 (end of LoM)

Source: GE21, 2025.

16.6 Mining Equipment

At the Bravo Mine, the mining operations will be carried out by a third-party contractor. The mining equipment was sized according to the volumes of material to be removed each year. The ROM will be drilled, blasted, loaded and transported by trucks. The mining plan sequence was scheduled to feed the plant with highest ROM at the beginning and the lowest strip ratio.

The main mining activities will be:

- Digging or rock blasting of ROM and waste.
- Excavation, loading and transport of ROM and waste.
- Disposal of ROM in the primary crusher and/or stockpile balance and waste in the WRSF.
- Construction and maintenance of all internal accesses to the pit(s) and the WRSF.
- Maintenance of the accesses, drainages, and signaling of all access roads used in the operation.
- Implementation and maintenance of the mine's surface drainage systems at access points to the mining operation, waste deposit, stockpiles and other areas linked to mining operations.
- Mine infrastructure services, such as construction and maintenance of accesses to the mining areas, crusher, WRSF, workshops and offices, mine drainage and pumping services, access signaling, mine dewatering, etc.
- Build and maintain the operation’s support facilities (offices, workshops, cafeteria, living quarters, warehouses, changing rooms, ablution facilities, septic tanks, environmental, health and safety emergency, explosive magazine, electrical and hydraulic installations and others), in strict accordance with the Brazilian environmental standards and labour laws.

Besides the mining operation will be outsourced it was carried out an exercise for mining equipment sizing was based on the required production rate and specific features of the Project. Equipment models with demonstrated efficiency in similar operations in Brazil were selected.

16.6.1 Loading

The excavation and loading of ROM and waste will be performed using hydraulic excavators with bucket capacities ranging from 15 m³ to 38 m³, depending on the equipment class and operational requirements. The larger excavators, with buckets of 29 m³, will be allocated to primary production loading, while medium-sized excavators with 15–22 m³ buckets will support development and rehandling activities. All units are matched with haul trucks of up to 242 t payload to ensure efficient loading cycles.

16.6.2 Hauling

The transport of ROM, waste, and rehandled material will be performed using a fleet of off road trucks, with payload capacities of up to 242 t. The haulage cycle considers average distances projected for LoM. ROM material will be dumped on the ROM pad at the processing plant, while waste will be sent to designated disposal areas.

16.6.3 Fleet Sizing

Fleet sizing was determined based on the production schedule and operational parameters. The mine plans to move a maximum of 103.6 Mt/year, including 10 Mt/year of ROM, 85.3 Mt/year of waste, and 10 Mt/year of rehandled material. Equipment availability, utilization rates (e.g., UF of 85%, DF of 80%), and average travel speeds (20 km/h loaded, 25 km/h empty) were used in the calculations. Maximum equipment estimated includes 6 loading units, 44 haul trucks and 9 blast drill rigs of 6 inches.

Table 16-9: Equipment capacities and productivity factors

Parameter	Excavator	Truck
Bucket / Payload (m^3)	29	96.8
Bucket / Payload (t)	72.5	242
Fill Factor (%)	100	100
Availability (%)	80	80
Utilization (%)	85	85
Efficiency Factor (%)	85	90
Cycles per Hour	30	-

Source: GE21, 2025.

Table 16-10: Average haulage distances

Category		Average Haulage Distance
ROM	(km)	2.8
Waste		5.3
Tailings		4.0

Source: GE21, 2025.

GE21 Consultoria Mineral

17 RECOVERY METHODS

Disclaimer: In this chapter the term “ore” is used in a generic and non-reserve context to describe mineralized material that is considered economically processed in the Plant. The use of the term “ore” does not imply Mineral Reserves as defined under NI 43-101 and CIM Definition Standards, once Mineral Reserves require the completion of at least a Pre-Feasibility Study demonstrating economic viability, including consideration of all Modifying Factors.

17.1 General

The process plant will be designed based on the metallurgical test results up to now. The route was developed to concentrate fresh material due the reduced resources of oxidized material. This material will be stored and treated at the end of the mine life, according strategy to be defined.

The plant feed will be planned to process inittially 5 Mtpy, ramping up to 10 Mtpy from Year 2. Table 17-1 summarizes the preliminary design criteria and estimated sizing of the major equipment for the ore processing facility. These equipment specifications are based on benchmarking against similar operations and are subject to further refinement.

To support detailed engineering and definitive equipment sizing, particularly for the grinding, thickening, and filtration units, additional mineral characterization testwork will be conducted by Bravo.

Table 17-1: Operational parameters – crushing & grinding circuits

Operational Criteria			
General	Ore Feed Years 1 – 2	Mtpy	5
	Ore Feed from Year 3	Mtpy	10
	Average Feed Grade 4E	g/t	1.4
	Specific Gravity	t/m³	2.5
	Feed Moisture	%	5.0
	Density	t/m³	2.0
Crushing	Utilization	%	70
	Operating Hours	h	16.8
	Feed Rate	t/h	1631
Grinding	Utilization	%	92
	Operating Hours	h	22.1
	Feed Rate	t/h	1365
	BWi (circuit of fines)	kWh/t	13
	F80	mm	0.212
	P80	mm	0.028
Thickener	Settling Rate	t/(m²/h)	2
	Residence Time (Pilot Plant)	min	36
	Utilization	%	92
Flotation	Operating Hours	h	22.1
	Feed Rate	t/h	1365

Source: GE21, 2025.

Table 17-2 presents the preliminary characteristics of the main equipment..

Table 17-2: Main equipment characteristics.

Main Equipment Characteristics		
Equipment	Quantity	Characteristics
Primary Crusher	1	MMD 1300, 2 axis, 800 kW
SAG Mill	1	36' x 17', 12,8 MW
Pebble's Crusher	2	Conic, 500HP
Coarse Circuit Ball Mill	1	24' x 40', 13.5 MW
Fines Circuit Ball Mill	2	24' x 40', 13.5 MW
Hydrocyclones	2 x 68	15"
Thickener	1	30 m diam.
Flotation	To be defined	

Source: GE21, 2025.

Figure **17-1** shows the simplified flowsheet for the processing plant.

GE21 Consultoria Mineral

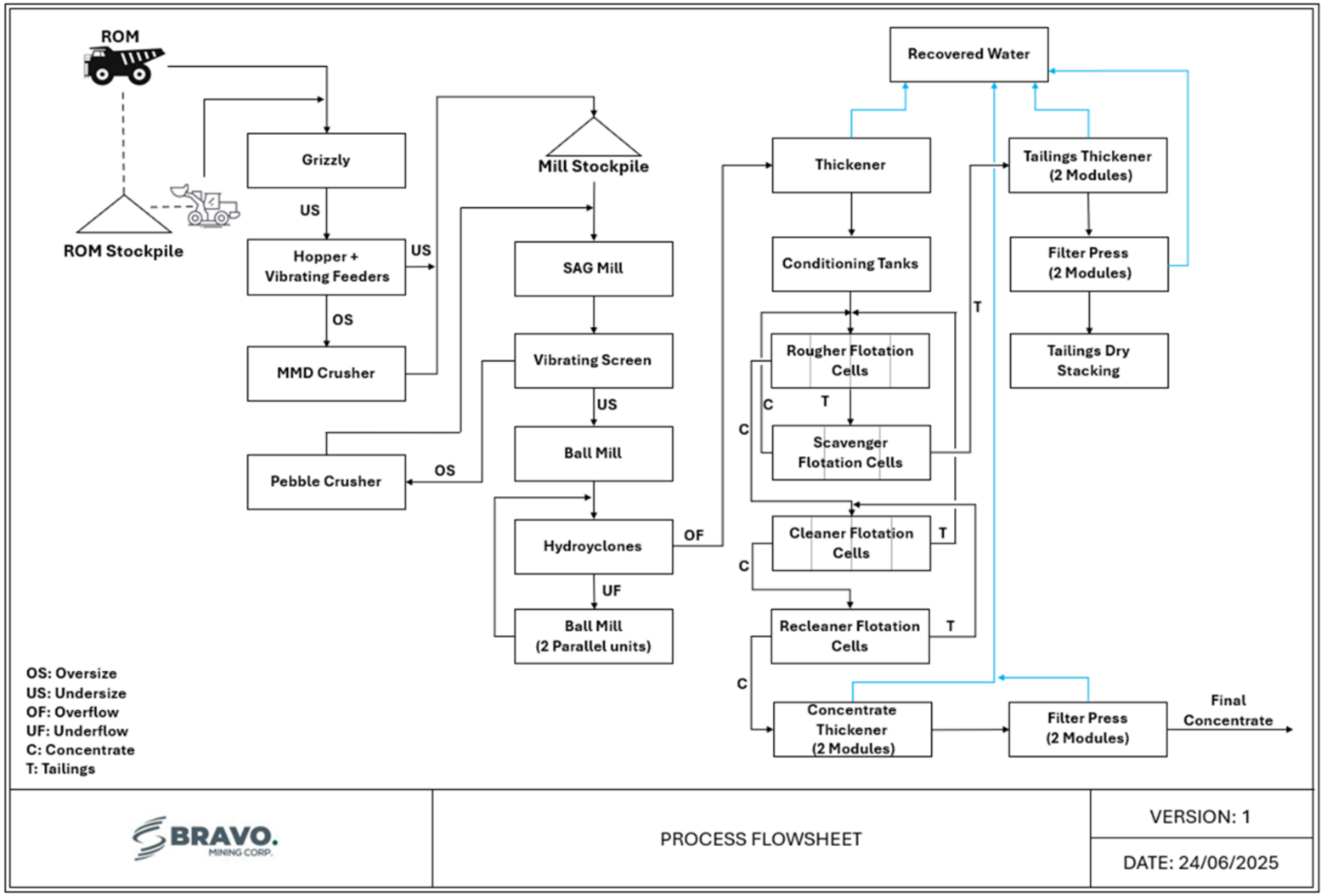


Figure 17-1: **Processing Plant Flowsheet**

Source: GE21, 2025.

17.2 Processing Plant

The ore extracted (ROM) is discharged in a primary crushing yard and either discharged directly into the crushing feed hopper through a fixed-type grizzly or stockpiled in the ROM stockpile around the hopper. In this case, the ore is reclaimed using a front-end loader and fedto a hopper. The ore is reclaimed from the hopper by an apron feeder and feeds a primary crusher, comprised of a 2 axis MMD crusher.

The crushed material is conveyed by a Long Distance Belt Conveyor (LDBC) and stockpiled in a conic pile, for being reclaimed by belt feeders to feed to a three-stage comminution circuit, comprising a semi-autogenous primary grinding (SAG 36') mill, a secondary open circuit ball mill and two tertiary closed circuit ball mills.

The SAG discharge feeds a classification screen, where the retained, the critical size, is conveyed to a conic pebble crusher, model HP500 or similar, which product returns to the SAG.

The passing material in the screen feeds an open-circuit 24" x 40" ball mill as a coarse grinding – 212 μm step. The mill discharge feeds a 15'' hydrocyclone battery in order to classify a 28 μm overflow. The underflow feeds 2 parallel 24" diameter ball mill units, in closed-circuit with the hydrocyclones battery.

The 28 μm material is very diluted due to the classification stage and passes through a high capacity thickener in order to raise the solids percentage for the upcoming processing stages, and also to recover water that is recycled back into the third grinding stage and water process tanks.

The underflow of the thickener slurry is eventually pumped to the conditioning tank , where additional water is added to adjust the slurry density to the flotation requirements of approximately 35% solids by weight.

Flotation is made up of four stages: rougher, scavenger, cleaner and recleaner.

Rougher flotation is comprised of 4 flotation cells. The concentrate generated from this stage feeds the cleaner stage, while rougher tailings feeds the scavenger cells.

The scavenger stage is comprised by 3 flotation cells, where concentrate generated is recycled to the rougher stage while scavenger tailings proceeds to the tailings thickening and filtering and dry stacking.

3 flotation cells comprises the cleaner stage, where the concentrate generated feeds the recleaner stage, while cleaner tailings are recycled to the rougher stage.

The recleaner stage is comprised of a single column cell, where tailings are recycled to the cleaner stage and the concentrate is pumped to a solid/liquid separation system, including thickener, concentrate storage in agitated tanks, and filtration in a pressure-plates filter.

It is expected to generate a total of 184 metric dry tonnes of concetrate per year withexpected grades for economic metals shown in Table 17-3:

Table 17-3: Concentrate metal grades

Element (unit)	Grade
Pd (ppm)	44.36
Pt (ppm)	27.73
Rh (ppm)	2.54
Au (ppm)	1.63
Ni (%)	4.72

Source: GE21, 2025.

17.3 Dry Stacking

The tailings from the ore processing will be filtered in pressure plates and disposed of as dry stacked material in a dedicated storage facility.

Flotation tailings will be thickened in a high-capacity thickener. The overflow returns to the process water tank to be recycled into the process. The undeflow is discharged into an agitated tank and pumped by a variable-frequency-drive (VFD) slurry pumps to a series of filter presses.

The filtering plant will be located neart he storage pile, and the filtered tailings (“filter cake”) are discharged onto belt conveyors, forming conical piles. A front-end loader loads the material to haul trucks and transported to the dry stacking storage area.

The DSF design parameters are presented in Section 16 of this report. The stucture will have a storage capacity of 65.7 Mm^3 and will occupy an area of 122. Tailings will be stacked with a degree of compression of 95% of the Standard Proctor Compaction at the edges of the stack, and 85% of the normal proctor in the center of the stack. The layers must be laid and compacted parallel to the face of the pile slope, maintaining a slope of 2% or indicated in the project for permanent or temporary drainage and thus allowing the runoff of rainwater. The thickness of the layers after compaction should be around 20 cm, with systematic topographic control.

The conceptual DSF design is presented in Section 18 of this report.

The filtered tailings, with an expected moisture content of 12%, will be deposited and compacted according to the following procedure:

- Truck loading, transport, dumping, spreading, and compaction using roller compactors.
- Moisture adjustment as required to meet compaction specifications.
- Quality control to ensure compaction and drainage levels.
- Topographic control of advancement and surface drainage.
- General site services as required for safe and compliant operation.

Compaction will be controlled primarily by the number of roller passes, validated through preliminary test pads. Additional quality control will include:

- Proctor compaction testing.

- Field density testing using sand cone or equivalent.
- In-situ moisture determination.
- Cross slope of verges to be considered: 3%.

17.4 Water Supply

17.4.1 Fresh water

The project includes the construction of a small dam on a watercourse located within the project area. The proposed structure will have a maximum height of 15 metres, forming a reservoir with a normal resevoir elevation of 183.5 metres above sea level and a total storage capacity of approximately 2.7 million cubic metres of water.

Hydrological studies indicate that this volume is sufficient to meet the operational water demand of the project under normal climatic conditions. Under scenarios of prolonged or extreme drought, the risk of supply interruption is considered moderate. To mitigate this risk, a supplementary water captation system is planned.

A contingency measure involves the captation of water from the Sereno River, located north of the plant site and outside the project boundary. The proposed system will divert up to 136 cubic metres per hour (m^3/h) to the main reservoir via a 4,880-meter-long pipeline.

The contributing catchment area for the watercourse within the plant site is estimated at 26.3 km^2, while the catchment area at the external intake point on the Sereno River is approximately 88.1 km^2.

17.4.2 Process water

Process water requirements, primarily for use in grinding and flotation, will be met through the reuse of water recovered from the overflow streams of both the concentrate and tailings thickeners.

The reclaimed water will be collected in a dedicated storage tank and subsequently distributed to various industrial demands across the plant, including processing circuits and area cleaning operations. This closed-loop system is designed to reduce raw water intake and optimize overall water efficiency within the operation.

17.5 Power Supply

The supply of electricity for the Project, with an estimated final demand of 80 MW, will be carried out through a transmission line at a voltage of 230 kV. The transmission line will be 35 km long from the Carajás Substation and will be built by Equatorial Energia to supply the main substation of the project.

The planned investment is US$ 17.3 million and the execution schedule foresees a period of 36 months to complete the work.

The investment estimate foresees the acquisition of equipment, materials and services, in addition to negotiation with surface owners along the right-of-way of the 230 kV Transmission Line.

Figure 17-2 below illustrates the preliminary design of the main electrical substation, where the input voltage of 230 kV will be lowered to 13.8 kV and then distributed to the areas. The main electrical substation will occupy an area of 6,500 m^2.

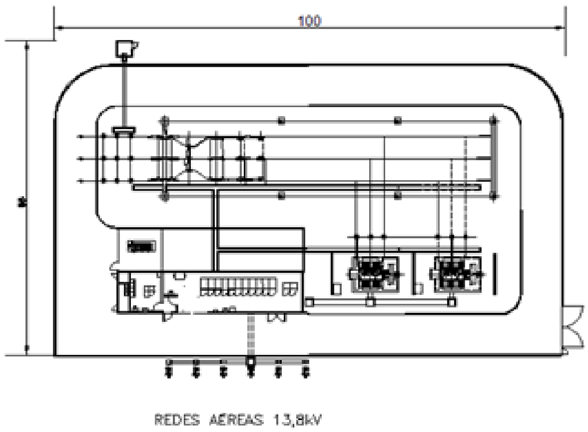


Figure 17-2: Main Substation

Source: GE21, 2025.

The main substation will be located near to the processing plant, where the project's largest consumption units are located. This substation houses the transformers and other switching, control and protection equipment.

From the panel with 15 kV circuit breakers, installed in the Electrical Room of the Main Substation, the following are derived:

- 13.8 kV overhead networks, simple radial system, for load centers outside the industrial plant;
- Substation of the auxiliary buildings, explosive stores and water collection;
- Cable beds or duct network (electrical envelopes) with 13.8 kV circuits for loads within the Industrial Plant:
  - 13.8 kV - Crushing and metallurgy substations, 8 MW.
  - 460 V - Grinding and thickening auxiliaries, leaching, CIL, AVR, Detox, reagents, raw and sealing water, waste pumping and coal regeneration furnace.

18 PROJECT INFRASTRUCTURE

The Project occupies an area of approximately 1,600 hectares, comprising at least the following infrastructure and on-site facilities:

- Open Pit
- Site access roads
- Haul roads
- Waste Rock Storage Facility (WRSF)
- Dry stacking facility (DSF)
- Settling ponds
- Maintenance facilities (truck shop, plant workshop and warehouse)
- Process water treatment facility
- Tailings Dam (first 2 years of operation)
- Water Dam
- Administrative buildings (Admin & Finance, Management, Engineering and Geology Offices, Support Services, Parking lot, Cafeteria, Locker Room and Restrooms (Male/Female), Security, Medical Post, HSEC (Health, Safety, Environment & Communities)
- Access gate
- Laboratory
- Explosives magazine
- Main electrical substation
- 230 kV transmission line
- Water catchment system
- Communication system
- Control Room

18.1 Roads

Project is well served by paved roads. Access to the area can be made by road or air to the city of Marabá or Parauapebas, an approximate distance of 530 km.

Leaving Marabá, land access is via the BR–155 highway, traveling 100 km to the junction with PA–275, in the city of Eldorado dos Carajás. From this junction, head west until km 16 of the PA–275 highway. From km 16, heading N-NW, travel 18 km on the paved municipal road, accessing the town of Serra Pelada, to the Luanga Deposit. This route totals 134 km of paved roads.

Leaving from Curionópolis, access is via the PA–275 highway, heading east until km 16, 14 km after the city of Curionópolis. From this point, it is 18 km along a paved municipal road to the deposit, making a total journey of 32 km.

In Marabá and Parauapebas there are airports with commercial flights to Belém, Brasília and Belo Horizonte.

The access options are presented in Figure 18-1.

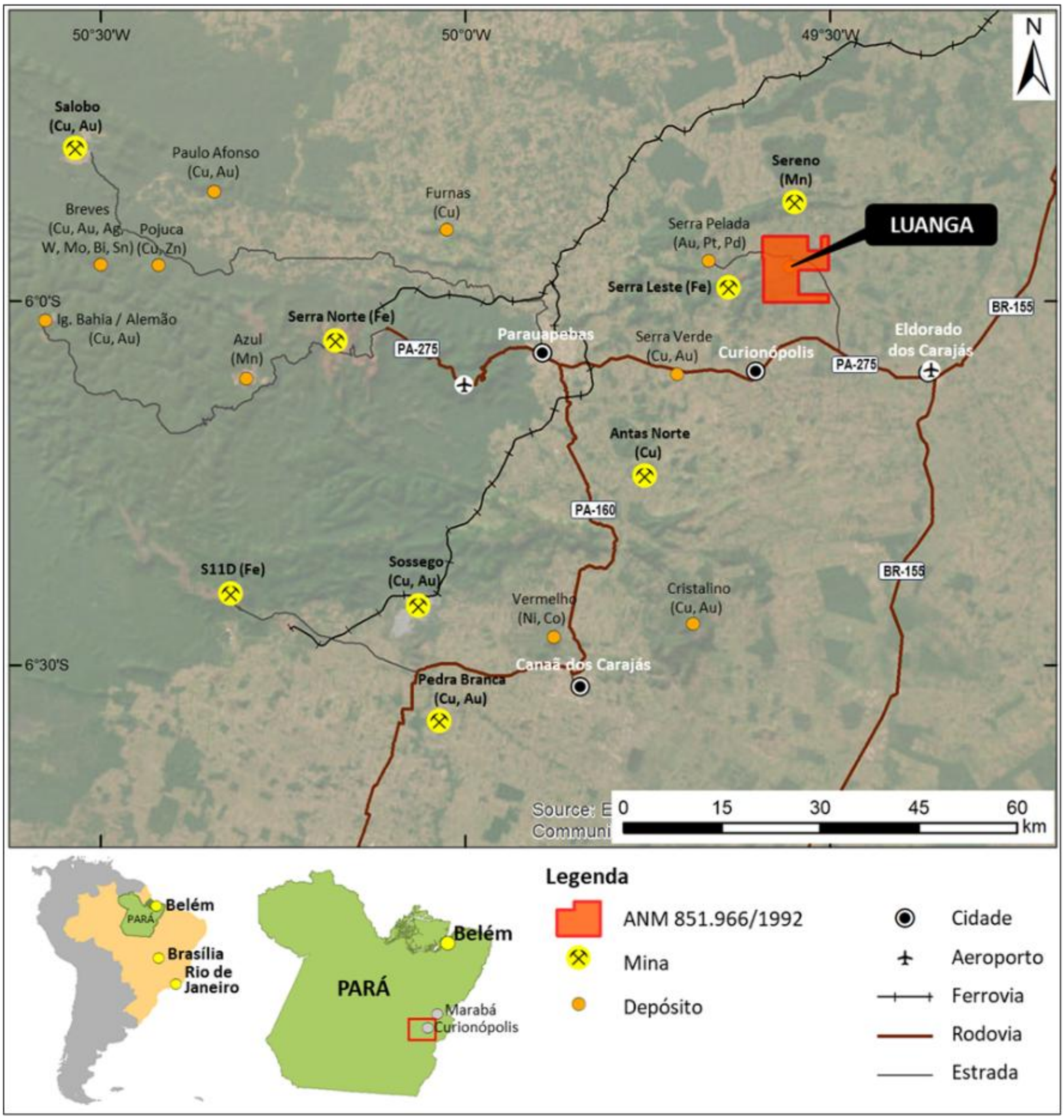


Figure 18-1: Access options

Source: Bravo, 2023.

18.2 Power Supply

Bravo team conducted a power supply alternatives study with support from a specialized consultancy. The selected alternative for the Project's power supply consists of a direct connection to the basic electric grid through the Carajás Substation at 230 kV. This solution includes the construction of a dedicated connection bay at the Carajás Substation and the installation of a 35 km long 230 kV transmission line linking the substation to a new on-site substation to be built for the Project. The new substation will be equipped with a 100 MVA transformer (230 kV / 13.8 kV) to meet the internal power demand of the operation.

This option was considered technically feasible, offering a more robust connection solution that is well-suited to the Project's projected power demand and operational characteristics. Figure 18-2 presents a schematic design of the selected option.

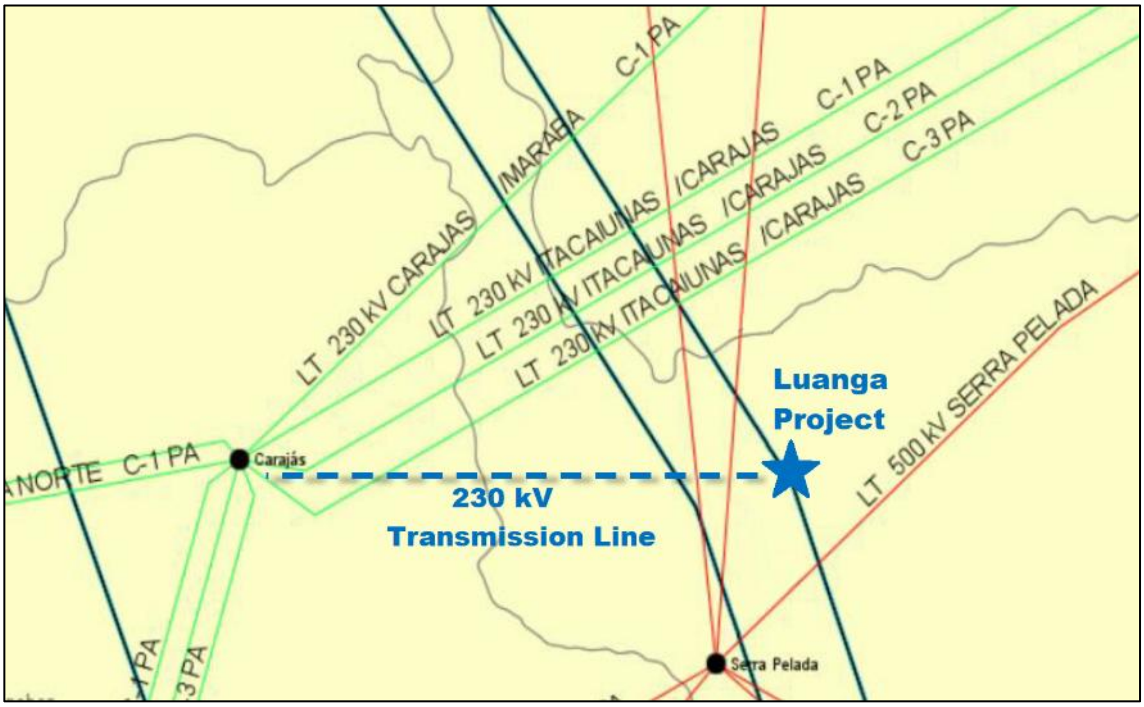


Figure 18-2: Power supply – schematic sketch

Source: TSE Energy Consulting/Replace Consultoria, 2023. Adapted by GE21.

The main substation in the Project area will comprise an external concrete pad for high-voltage equipment and a control area made up of electrical panel room and cable room. The substation infrastructure includes:

- Grounding
- Protection against lightning strikes
- Protection of equipment from a ground fault current
- Reliable voltage
- Emergency power generators
- Supervision, control, and protection room
- Fire detection, alarms, and firefighting with a clean fire suppression agent
- Access control
- Air conditioning and pressurization
- Safety signs and escape routes

18.3 Mine Drainage and Dewatering

The open pits perimeters and its upper benches will have drainage channels to direct water to the collection system. The run-off and underground water will be retained by sumps, and when possible will be pumped out from pit. The main pumping system will be installed at pit bottom to work during high-rainfall or high water percolation periods. When possivle, the water will be diverted to sumps and pumped for reuse. The water excess volume will be directed to the natural drainage system, after being in compliance with environmental parameters.

The detailed drainage and dewatering projects will be developed in the next phases, properly sized based on hydrogeological and hydrological studies. Fines sedimentation sumps will be constructed within the mine area to fine materials generated during mining activities, particularly from drainage and run-off. It will be strategically located to follow the natural topography, reducing earthworks and environmental disturbance. The sumps will contribute to erosion control, water management, and regulatory compliance, with its integrity monitored through standard controls.

18.4 Waste and Dry Stacking Facility

The WRSF and DSF were dimensioned to receive, respectively, waste material from the mining areas, and dry tailings from the processing plant, throughout the Life-of-Mine. A contingency factor of 5% was considered in volume requirement estimates to account for uncertainties surrounding material features and operational issues.

Drainage channels will be constructed along those facilities perimeters to collect rainwater and carried sediments, reducing their impact on the local vegetation and terrain. Those channels will direct the water to settling ponds located at the bottom of the slope. When possible water will be pumped for reuse.

The geotechnical and design parameters are presented in Section 16 of this report. The WRSF and DSF design are presented in Figure 18-3 and Figure 18-4.

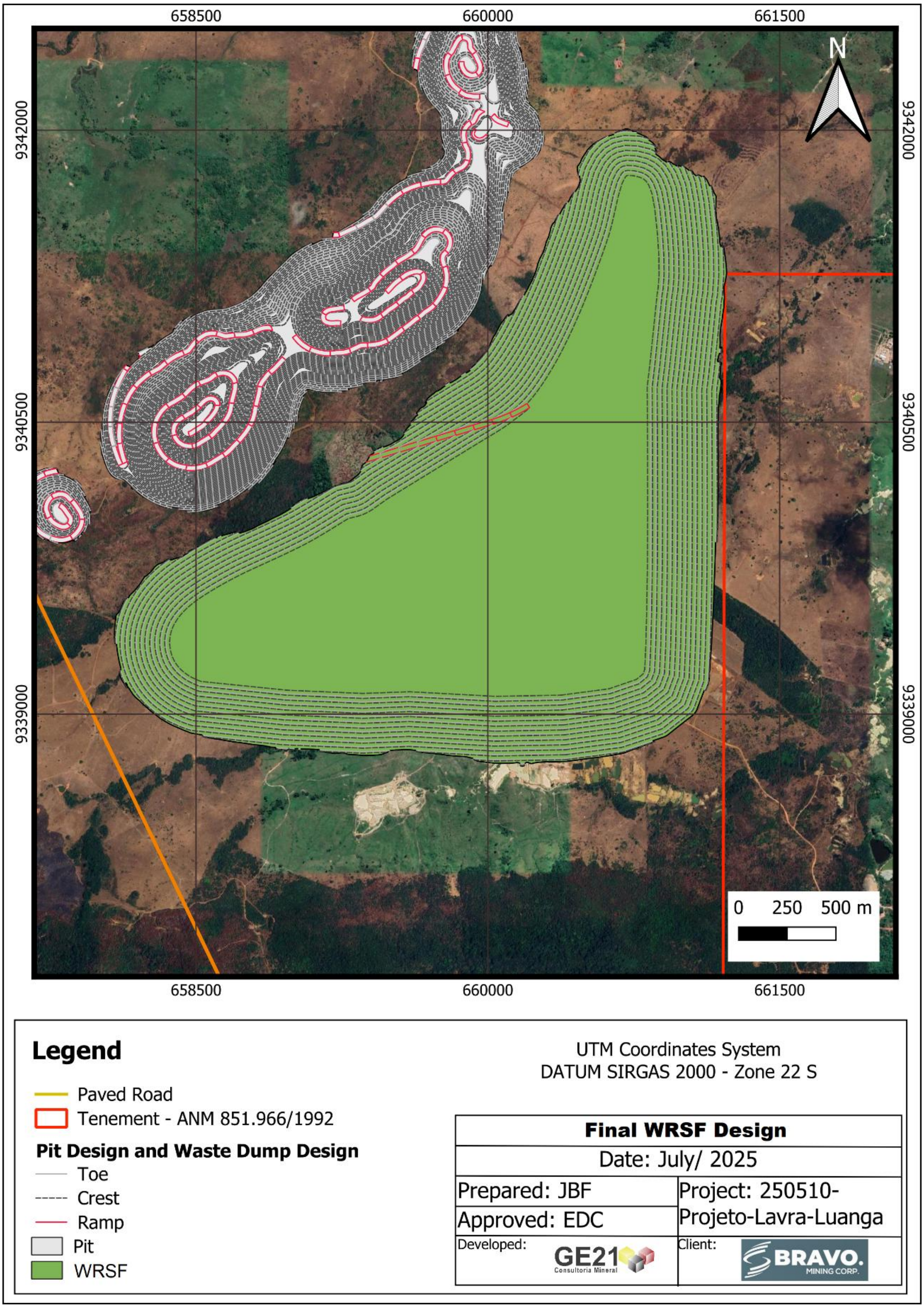


Figure 18-3: WRSF design

Source: GE21, 2025.

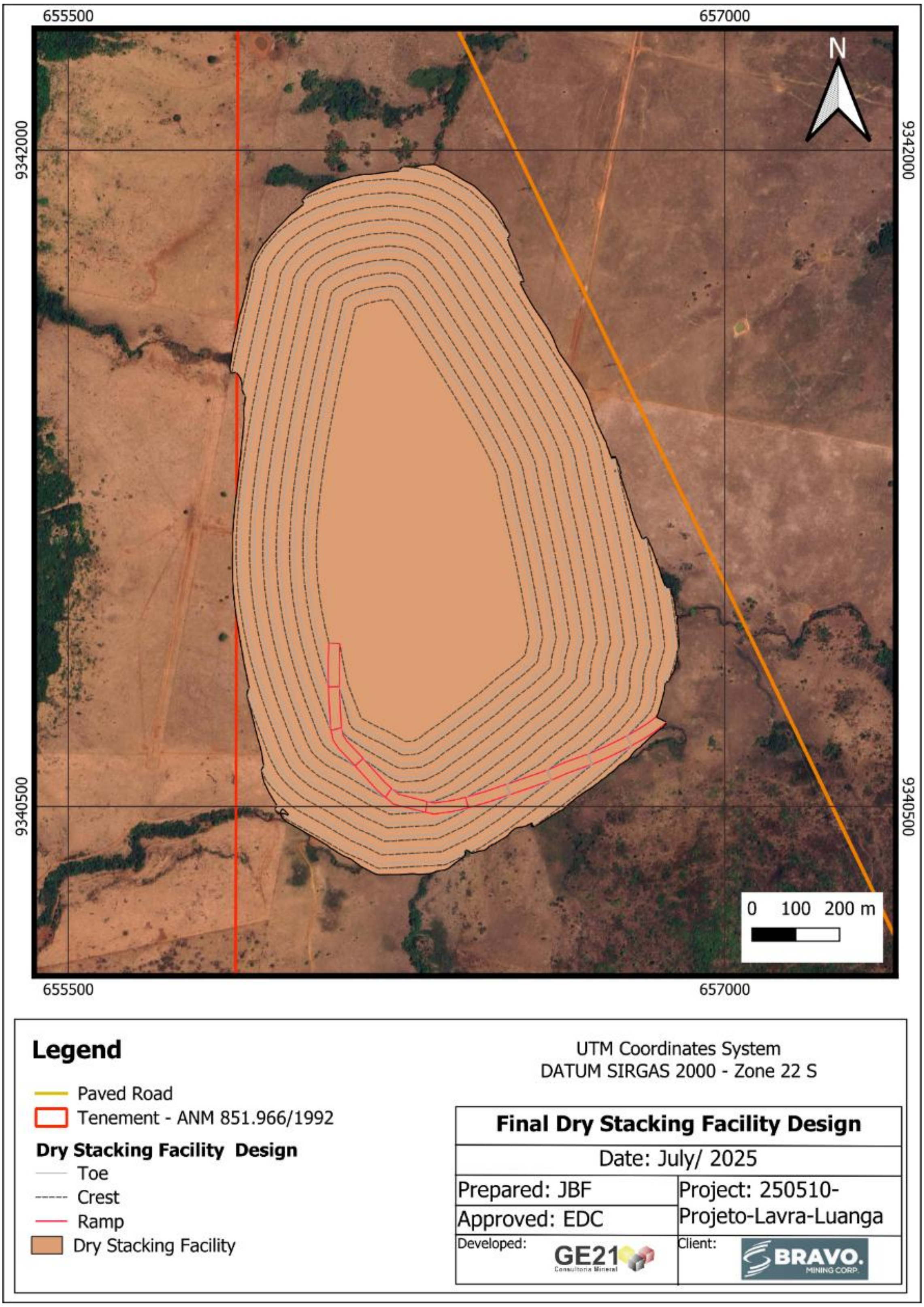


Figure 18-4: Dry Stacking Facility design

Source: GE21, 2025.

18.5 Tailings Dam

During the first 2 years of operation, the tailings produced in the processing plant will be disposed in a tailings dam facility close to the plant area. A total of 5.0 million m^3 will be disposed in the selected area, and the structure project and design will be detailed on future studies.

18.6 Raw and Potable Water

The water in the plant will be used on a closed circuit. Water loss will occur through product and tailings moisture and minor evaporation. This reuse will minimize the need for new make-up water.

Raw water for human and general use will be pumped from water wells. To achieve the best environmental practices, the water supply sources will be investigated during the next phase of the Project. Further, potable water will be pumped from wells drilled through water table and will be distributed to the processing plant and site buildings.

Among the water supply alternatives studied for the processing plant, the most appropriate and rational involves the construction of a watercourse dam within the project area, with a maximum height of 15 meters, creating a reservoir with a mass elevation at elevation 183.5 RL with capacity of 2.7 million m^3 of water.

This volume will be sufficient to meet the project's demands, with moderate risk in cases of extreme rainfall deficit. As a mitigating measure, it was planned to capture 136 m^3/h via transposition of water from the Sereno River, at a point in the region north of the plant (outside the project area).

The distance from the pipeline that will bring water collected from the Sereno River to the reservoir will be 4,880 meters. The area of the contribution basin of the water body that runs through the plant is 26.3 km^2 and the area of the contribution basin of the point where external water collection will occur is 88.1 km^2.

Figure 18-5 below shows the contribution basins and the route of the pipeline for water collection.

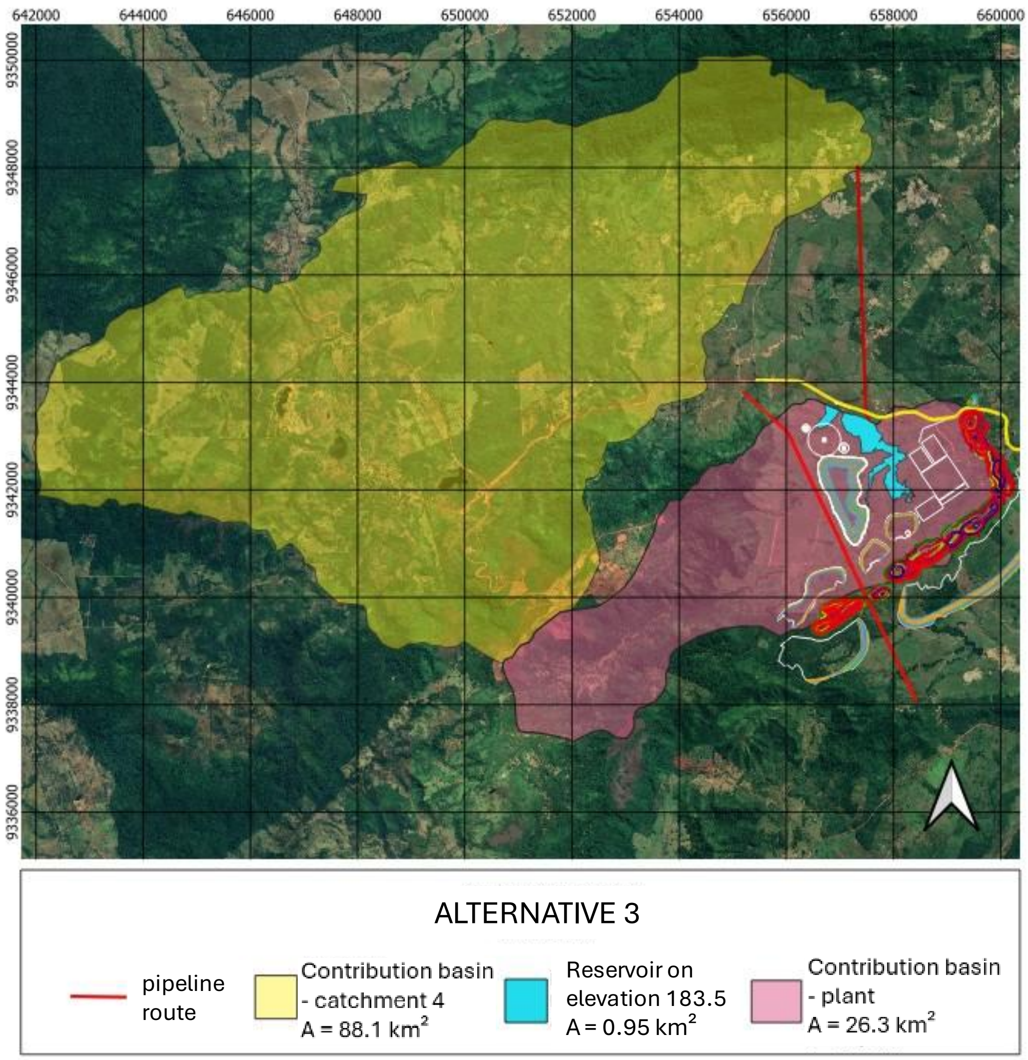


Figure 18-5: Contribution basins and adductor

Source: Geohydrotech, 2025.

18.7 Personnel Areas

An area close to the main gate and to the Project facilities will be designated for the mining operation buildings and infrastructure. It will include a warehouse to serve as a centralized storage hub for equipment, spare parts, and consumables, ensuring efficient operations and rapid access and inventory management. A shop building will minimize downtime and optimize equipment performance by housing all the tools and equipment needed to maintain and repair the mining fleet and ancillary equipment. Further, it will be equipped with residue collection and separation.

Administration offices will be located close to the processing plant area, sharing a building with meeting rooms, workstations, washrooms, and warehouse.

A canteen, kitchen area, cafeteria, and toilets will be centrally located and easily accessible by all restaurant, house keeping, administrative, maintenance, mining, and processing employees.

Figure 18-6 shows the Project's Master Plan.

GE21 Consultoria Mineral

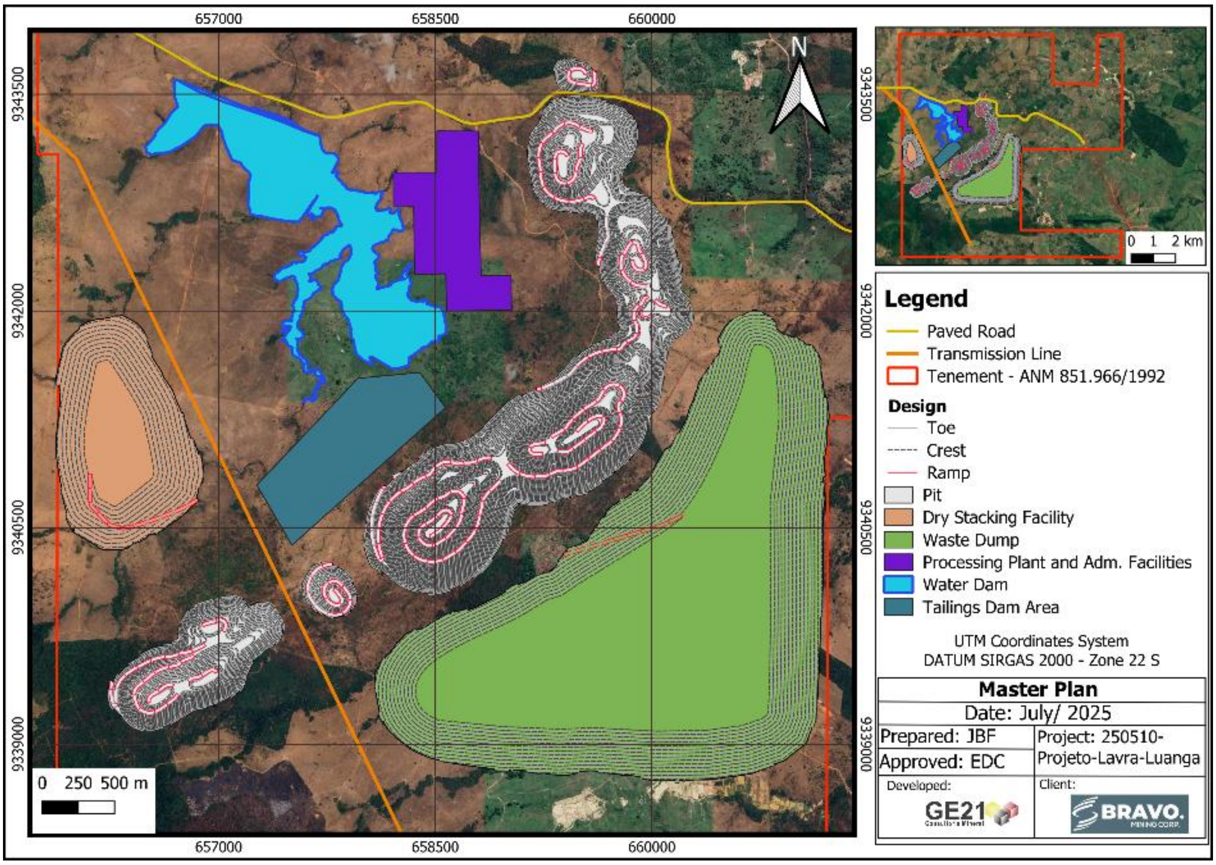


Figure 18-6: Master Plan

Source: GE21, 2025.

18.8 Communications System

Communication and control hardware will be distributed, and the network expanded as development progresses. Radios will be hard-wired into mobile equipment. Handheld radios will be strategically located at base stations in key locations such as the main gate, processing plant, workshops, and administration offices. A redundant integrated cabling system will be supported by hybrid network architecture (i.e., optical fiber for the backbone and copper twisted pair cable for the local network). Single-mode fiber optic cable will be used for low-loss and high-bandwidth optical systems. This type of cable is capable of continuous operation with good performance in an environment with high levels of suspended dust, moisture, and noise.

GE21 Consultoria Mineral

19 MARKET STUDIES AND CONTRACTS

19.1 Overview

The proposed Luanga PGM operation is anticipated to enter the market at a time of continuing structural deficits across several key precious and base metals, notably platinum, palladium, rhodium, gold and nickel. Market fundamentals indicate sustained demand for PGMs driven primarily by the automotive and emerging hydrogen energy sectors, while constrained global mine supply—exacerbated by geopolitical uncertainty and declining South African production—continues to support higher long-term pricing.

The Project will produce a flotation concentrate containing PGMs (Pt, Pd, Rh), Au, and Ni. Given Brazil's strategic position and access to Atlantic shipping routes, concentrate sales are expected to target Southern African or Chinese metallurgical processors. The Company has also had informal discussions with international buyers in Europe to understand long-term contracts or/and tolling arrangements. Vertical integration will produce products for the refinery or end-metal user market with options to deliver into the Southern African, Chinese, European or North American market.

Market investigations have indicated that capacity in both the concentrate and refinery feed stack markets is available to due decreasing concentrate volumes production within Southern Africa, substantial capacity unlocking to due recent debottlenecking by major smelters and additional refinery capacity coming online in North America during 2025. This aspect is presented in Figure 19-1.

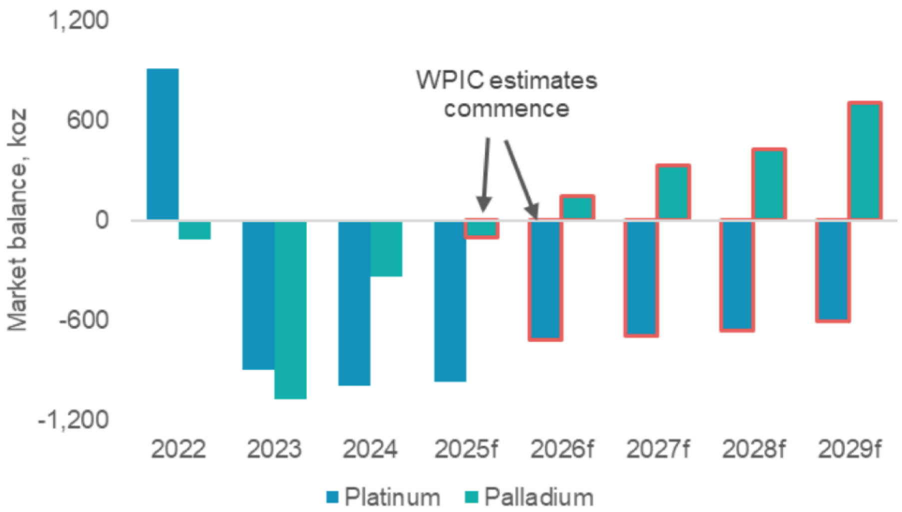


Figure 19-1: Platinum and Palladium market balance

Source: Metals Focus 2022 to 2023 (palladium) and 2022 to 2025f (platinum), Company guidance, WPIC Research.

Platinum is forecast to remain in deficit at least until 2029 while palladium's market positive market balance beyond 2026 is assumed based on return of recycling volumes. In the absence of these returning recycling volumes, the palladium market is anticipated to remain in a modest supply deficit like platinum.

19.2 Concentrate Marketability

Global smelting and refining infrastructure have increasingly adapted to intermediate PGM products, with contracts now common for flotation concentrates, converter mattes and high-grade residues. PGM concentrates are primarily refined in South Africa and Europe, with limited but expanding capacity in Asia and North America. Brazil lacks existing PGM smelting facilities and is contemplated to be addressed by the PEA's vertically integrated scenario.

Contracts for concentrate sales or tolling typically incorporate key commercial terms such as metal accountabilities and payable percentages, treatment and refining charges (TC/RCs), penalties for deleterious elements (e.g., Cr_2O_3, As, Sb, Se), delivery logistics and pipeline duration, advance payment mechanisms, force majeure, dispute resolution and renewal clauses.

The sulphur and iron content of the Luanga concentrate will be within the smelter acceptance limits. The anticipated concentrates are low in chromium dioxide since the chromite concentrations in the mineralised ROM is low to absent. MgO concentrate content is in line with other Southern African concentrate producers and pose no limits on treatability through standard 6-in-line electric arc furnaces.

19.3 Platinum Group Metal Market Outlook

19.3.1 Platinum (Pt)

Global platinum demand remains bifurcated between industrial (automotive, chemical, glass) and investment (jewellery, bullion). While automotive demand has softened due to a shift from diesel to gasoline vehicles, Pt is poised to benefit from fuel cell electric vehicle (FCEV) growth, where Pt-based catalysts dominate. Average Pt loadings per FCEV range from 10–20 g, nearly double traditional diesel vehicle requirements.

Jewellery demand, historically concentrated in China and India, has seen substantial growth in recent months, with high gold prices driving demand to platinum while targeted promotional campaigns maintain interest. Recycling of old jewellery stock, particularly in Asia, continues to modulate short-term market balances.

Supply remains concentrated in South Africa and Russia. South African output faces structural constraints—declining grades, deepening shafts and labour unrest—resulting in a tight long-term outlook. New primary supply, including the Luanga Project, is essential to offset declining mined production.

19.3.2 Palladium (Pd)

Palladium is a critical component of three-way catalytic converters in gasoline engines. Its demand surged to a record 9.7 Moz in 2019, driven by tightening emission standards in China (China VI) and Europe (Euro 6). Although increased substitution by Pt is underway to manage costs, Pd remains indispensable in emission control.

Primary supply is dominated by Russia (Norilsk) and South Africa. Geopolitical factors and aging assets pose risks to continued output. Long-term demand remains robust, bolstered by hybrid vehicle growth, which maintains significant internal combustion engine content.

19.3.3 Rhodium (Rh)

Rhodium is the most volatile and illiquid of the PGMs, with a small market and limited number of suppliers. It plays a vital role in removing NOx emissions in catalytic converters, particularly under Real Driving Emissions (RDE) regimes. Automotive demand continues to outstrip supply, with prices historically peaking above US\$25,000/oz.

Given the extremely small size of the Rh market, even marginal changes in supply or demand can materially impact pricing. The Luanga Project's Rh content, though minor in absolute terms, is commercially significant due to the metal's unit value.

19.4 Gold (Au) Market

Gold remains a globally liquid and fungible commodity, serving dual roles as a monetary metal and an industrial component. Its pricing is primarily influenced by macroeconomic factors—interest rates, currency strength, inflation expectations—and geopolitical tensions.

Gold constitutes less than 3% of potential project revenue.

19.5 Nickel (Ni) Market

Nickel demand is bifurcated between stainless steel (≈70%) and emerging battery applications for electric vehicles (EVs). The latter has reshaped the nickel supply chain, with Class 1 nickel (sulphate or briquette form) commanding premiums for battery-grade specifications.

Long-term nickel prices are forecast to remain strong on the back of EV adoption. The Luanga Project's nickel output will be a key value contributor and may be marketed as part of an alloy or intermediate concentrate product. Payability for Ni in concentrates typically ranges from 75–90% depending on form and destination.

19.6 Supply Constraints and Strategic Positioning

South African production continues to dominate global PGM supply; however, structural issues—ageing infrastructure, socio-political instability, and cost inflation—constrain expansion. Russian production, particularly Pd and Ni, faces significant geopolitical risk. The result is a tightening market for key PGMs.

- The Luanga Project is well-positioned to benefit from:
- Proximity to Atlantic ports and international shipping routes
- Diversification of global PGM supply chains
- Increasing global demand for cleaner transportation technologies

GE21 Consultoria Mineral

- Brazil's established mining jurisdiction and regulatory framework

19.7 Offtake and Marketing Strategy

Informal preliminary concentrate offtake discussions are underway with international PGM refiners and base metal processors. The Project will pursue a multi-pronged offtake approach that may include:

- Treatment arrangements with established refiners
- Long-term purchase contracts with metal accountabilities
- Strategic partnerships for local smelting and downstream integration

Commercial terms will be structured to optimise revenue from all payable metals—Pt, Pd, Rh, Au, Ni and Cu—while mitigating payment delays through advance payment agreements and hedging options where feasible.

The base case considers established payabilities for metals in concentrate, net of TC/RCs. Logistics costs are fully costed and accounted for in the economic model. The resulting payabilities for metals in concentrate are presented in Table 19-1.

Table 19-1 –Payabilities for metals in concentrate

Metal	Payability (%)
Pd	85
Pt	85
Rh	84
Au	84
Ni	72

Source: Bravo, 2025.

19.8 Pricing and Economics

The financial model underpinning this technical report assumes the long-term, real metal prices presented in Table 19-2, informed by consensus forecasts and peer feasibility studies.

Table 19-2 –Metal price deck

Commodity	Luanga 2025 PEA Price Deck	Source	Spot price at time of determination	Unit
Palladium	1,271	Investec LT Real June 2025	1,172	US$/oz
Platinum	1,500	Investec LT Real June 2025	1,444	US$/oz
Rhodium	6,000	GE21	5,540	US$/oz
Gold	3,251	Consensus Economics LT Real June 2025	3,336	US$/oz
Nickel	17,637	Investec LT Real June 2025	15,100	US$/t

Source: various, as presented in column #3.

The above pricing assumptions equate to an assumed basket price of $1,555/oz. The basket price based on commodity spot price at the time of finalisation was $1,463/oz constituting a 6% difference between assumed pricing and spot price.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACTS

20.1 Introduction

This section presents studies conducted to date to develop a baseline for the Project and also to support the Environmental Impact Assessment and its respective Environmental Impact Report (EIA/RIMA), which are required for the environmental licensing of the Luanga Project.

20.2 Project location and environmental licensing

With the objective of initiating the licensing of the Project, Bravo Mineração filed, in March 2022, a request for a Term of Reference with SEMAS-PA (Pará State Secretariat for Environment and Sustainability) for the preparation of the EIA/RIMA. In response, SEMAS issued the Term of Reference that guided the studies, and the company Brandt Meio Ambiente was contracted for its elaboration. The preparation of this EIA constitutes compliance with a fundamental legal requirement, determined by Brazilian environmental legislation, to instruct the licensing process and obtain the Preliminary License (LP) from the SEMAS.

The field surveys and secondary data collection required by the TR were carried out between 2022 and 2024, covering the two climatic seasons (rainy and dry) characteristic of the Amazon region. Among the field studies conducted are surveys of fauna, flora, hydrology, and hydrogeology, sampling of surface and groundwater, sediments, diagnosis of speleological and archaeological heritage, and the socioeconomic diagnosis of the region. Additionally, analyses of environmental liabilities and risk studies were performed. The EIA/RIMA was filed with SEMAS-PA in June 2024 for technical analysis.

20.2.1 Regulatory Framework

Mining activities require preliminary Environmental Permitting. These activities are subject to a hierarchical set of environmental laws and regulations at the federal, state, and municipal levels in Brazil. Compliance with this legal framework is mandatory and serves as the basis for the entire permitting process and for the future operations of the project.

The 1988 Federal Constitution is the cornerstone of all Brazilian environmental legislation. Article 225 establishes the right to an ecologically balanced environment and imposes on the government and the community the duty to defend and preserve it. Crucially for mining, § 2º of the same article determines that "those who exploit mineral resources are obliged to recover the degraded environment, according to the law".

At the federal level, the main regulation is Law No. 6.938/1981, which establishes the National Environmental Policy (PNMA). This law defines the principles and instruments of environmental management in the country. Resolutions of the National Environmental Council (CONAMA) detail the technical aspects of the legislation. The most relevant for the project are

GE21 Consultoria Mineral

CONAMA Resolution No. 001/1986, which defines the guidelines for the EIA/RIMA, and CONAMA Resolution No. 237/1997, which regulates the environmental permitting procedures. Annex I of CONAMA Resolution No. 237/97 lists the activities and projects that use environmental resources, which are effectively or potentially polluting, and are subject to Environmental Permitting.

Federal law and the CONAMA resolutions structure the environmental permitting in a three-phase model, consisting of:

- **Preliminary License (LP):** Granted in the preliminary planning phase, attesting to the project's environmental feasibility.
- **Installation License (LI):** Authorizes the installation of the project.
- **Operating License (LO):** Authorizes the start of the mine's and processing plant's activities.

Finally, it should be noted that Complementary Law No. 140/2011 defines the responsibilities of each federative entity for environmental permitting. The responsibility lies with the environmental agency of the State of Pará. Thus, the project is subject to Pará State Law No. 5,887/1995 and to the other regulations and instructions of SEMAS.

20.2.2 Permitting and Licensing Status

The mining process with ANM is No. 851.966/1992. The ownership of the mining rights was transferred to Bravo Mineração Ltda. in November 2021. The Mining License Application was submitted in 2014 by the previous owner (VALE S.A.), and the process is currently active and in the Mining License Application phase.

As of the effective date of this report, the Project is in the LP application phase, with the EIA/RIMA having been filed with SEMAS-PA in June 2024. The issuance of the LP by SEMAS will indicate the approval of the project's design and location, attesting to its environmental viability and establishing the conditions to be met in the subsequent licensing phases.

In addition to the main environmental license, the project will require other authorizations, such as a Water Use Grant and a Vegetation Suppression Authorization (ASV).

20.2.3 Water Resources

The local hydrographic basin is located within the Amazon River basin, with the Tocantins River being one of its main tributaries. The main watercourse in the project area is the Itacaiúnas River, of which the Vermelho River sub-basin is a part. Locally, the area is contained within the Luanga Creek basin, which drains the area and flows into the Sereno River, a tributary of the Vermelho River. The local hydrography is composed of a low-order drainage network. The main channels are perennial; however, some springs dry up completely in the dry season, and the volumes and flows of the creeks reduce significantly when compared to the peaks of the rainy season.

The local hydrography is presented in Figure 20-1.

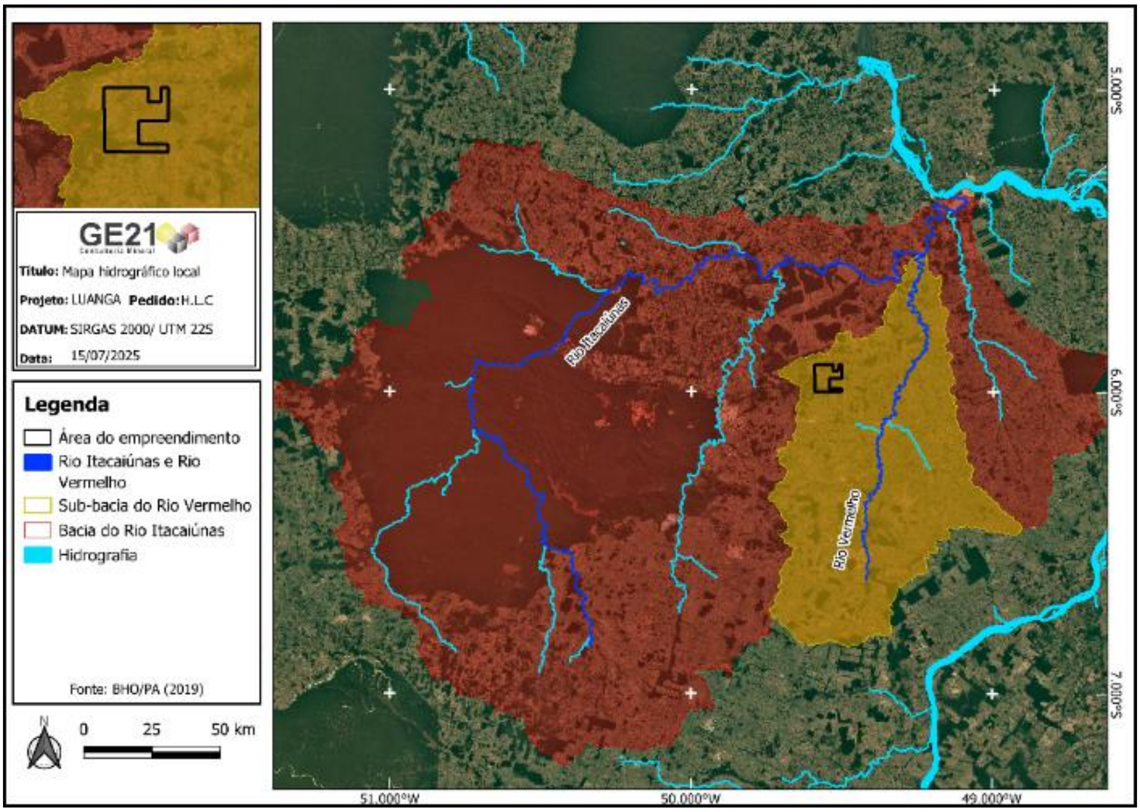


Figure 20-1: Local hydrographic map

Source: GE21, 2025.

To assess the possible impacts of the project on the basin, a detailed water balance was carried out. The raw water demand for the operation varies annually, reflecting the different phases of the project. According to the Economic Exploitation Plan (PAE), the demand starts at 126 m³/h in the first year and reaches a peak of 534 m³/h from the fourth year onwards, with the start of the rock ore (sulfide) plant's operation and its simultaneous operation with the first one.

Table 20-1 presents the estimated annual demands for new water.

Table 20-1 - Summary of annual demands for new (raw) water

Year	Demands for new water (m^3/h)
01	126.0
02	126.0
03	330.0
04	534.0
05	534.0
06	471.0
07	408.0
08	408.0
09	408.0
10	408.0
11	408.0
12	408.0

Source: Bravo, 2023.

20.2.4 Water Balance and Supply Solution

The water balance analysis, presented in the supporting studies, shows that the direct abstraction of water from Luanga Creek is not feasible. The Q95 reference flow (the flow that is present in the river for at least 95% of the time) is equal to zero for most of the year (from May to November), which confirms the insufficiency of the watercourse to meet the project's demand during the dry season. As there is a surplus of water during the rainy season, the project includes a central containment and flow regulation structure, which is the main reservoir. A comprehensive drainage system will be implemented in both the pits and the waste and tailings piles, which will direct rainwater to containment basins and, finally, to the main reservoir. This will prevent direct discharge into the watercourses without proper management.

The reservoir will play a fundamental role in the project's water management, serving as the main source of water for maintaining the operation during the dry season, and absorbing the excess flow from the drainage basin during the rainy season. Another important function is to act as a water body for the controlled dilution of treated effluents, should monitoring indicate the need. Thus, the treatment of effluents, when necessary, will be carried out before their discharge into the reservoir, which in turn will have a spillway designed to release excess water in a controlled manner, always maintaining the ecological flow of the downstream watercourse.

Regarding the volume of rainwater for reuse in the process, the project was designed to maximize recirculation, using the water recovered from the thickeners and the reservoir itself to supply the demands of the processing plants

20.2.5 Effluent Treatment

The activities of the Project will generate effluents of different natures, which will receive specific treatment before any discharge or reuse, in compliance with environmental legislation, especially CONAMA Resolution No. 430/2011. The main effluents are from the hydrometallurgical process (which contain cyanide), sanitary effluents, and industrial effluents (oils and greases).

Sanitary effluent, generated in the offices, locker rooms, and cafeterias, will be collected by a network of pipes and treated in compact Sewage Treatment Plants (STPs). For industrial effluents, such as those from maintenance workshops and equipment washing yards that may contain oils and greases, a treatment system composed of oil-water separators (OWS) will be implemented to ensure the removal of these contaminants before final disposal.

Continuous monitoring of the quality of the treated effluents and the receiving water bodies will be one of the project's conditions, to ensure the effectiveness of the treatment systems and the protection of the region's water resources.

20.2.6 Hydrogeology

Studies were conducted to develop a conceptual hydrogeological model, with the objective of evaluating the possible influences of the pit opening and its effects on the groundwater flows in the region.

The hydrogeological characterization of the area was based on field data, obtained from the installation of monitoring wells, and on geological and structural information. The local aquifer system is characterized as mixed—porous and fissured—with differentiated water circulation in two subsystems that have direct hydraulic connection: a shallow aquifer flow, associated with the more superficial and weathered geological units, and another deep flow, of a fissured nature, with strong control by rock structures.

In general, the circulation of groundwater at depth is quite restricted due to the low porosity and low fracturing of the rock mass. The shallow aquifer system is unconfined, and its water table follows the topography of the terrain, being recharged mainly by rainfall infiltration. This system is responsible for feeding the watercourses and springs in the region.

Simulations based on the conceptual model indicate that the opening of the pit will promote a lowering of the water table in its immediate vicinity. However, due to the low interaction between the deep fracture system and the shallow aquifer, no significant changes are expected in the general hydrodynamics of the area or in the flows that sustain the adjacent springs and streams.

20.2.7 Vegetation

The Project is located in the Amazon Biome, with the Open Ombrophilous Forest being the representative vegetation of the region. Over the years, a large part of the forest formations was replaced by pastures and agriculture. The diagnosis of the vegetation cover carried out in the direct influence area identified pasture as the predominant land use, followed by fragments of secondary vegetation in different stages of regeneration, demonstrating that most of the area where the project will be developed is already anthropized.

In the existing forest fragments, a detailed inventory was carried out that recorded a rich floristic diversity, with hundreds of species distributed in dozens of botanical families, the most representative being Fabaceae, Sapotaceae, and Burseraceae. Isolated arboreal individuals were also identified amidst the pasture, including specimens of legally protected and endangered species, such as Bertholletia excelsa (Brazil nut tree).

20.2.8 Wildlife

Field surveys were carried out in the two climatic seasons (rainy and dry) for the identification and evaluation of the terrestrial and aquatic fauna that occurs in the surroundings of the project.

Regarding medium and large-sized mammals, several species were recorded in the forest fragments of the study area, some considered common and widely dispersed, such as the squirrel monkey (Saimiri sciureus), the agouti (Dasyprocta leporina), and the crab-eating fox (Cerdocyon thous). Others are rarer, including top predators such as the jaguar (Panthera onca) and the ocelot (Leopardus pardalis). Among the endangered species, traces of the giant armadillo

(Priodontes maximus), the white-lipped peccary (Tayassu pecari), and the tapir (Tapirus terrestris) were recorded.

Regarding birds, hundreds of species were recorded, highlighting the presence of species endemic to the Amazon, such as the White-bellied antbird (Pyriglena leuconota), and migratory species. Some of the recorded species are game species (used for hunting), such as the Spix's guan (Penelope pileata) and the aracari (Pteroglossus bitorquatus).

During the field campaigns, dozens of species of reptiles and amphibians were recorded. No endangered species were identified in the region; however, the boa constrictor (Boa constrictor) is listed in the appendices of CITES (Convention on International Trade in Endangered Species of Wild Fauna and Flora) as a species that, although not endangered, could become so if trade is not controlled.

Regarding fish, dozens of species were recorded in the water bodies of the project area. Despite the occurrence of several species restricted to the Amazon region, none are considered endemic to the specific area of the project or are in any category of threat of extinction.

20.2.9 Socioeconomic Aspects

The area is located in the rural area of the municipality of Curionópolis, surrounded by rural properties. For the diagnosis and impact assessment of the EIA/RIMA, the municipalities of Curionópolis, Parauapebas, and Eldorado dos Carajás were considered as the area of influence, with Curionópolis, where the project will be installed, being the direct area of influence.

The population growth rate in Curionópolis is considered high, largely driven by mining activity in the region. The demographic density of the municipality is approximately 10.9 inhabitants/km², and the urbanization rate is high, exceeding 80%, with most of the population concentrated in the municipal seat. In the area of influence of the project, the social groups are distributed among rural producers, artisanal miners (with emphasis on the proximity to the Serra Pelada district), and residents of urban areas. In the area directly affected by the Project, the groups of rural producers predominate.

Regarding educational services, the municipality of Curionópolis has a network of early childhood, elementary, and high school establishments. Although the Basic Education Development Index (IDEB) has shown improvement, data from the IBGE indicate a literacy rate for the population over 15 years old that still requires attention.

Regarding the health structure, Curionópolis has a municipal hospital and Basic Health Units (UBS) that make up the care network. However, the health infrastructure, including the number of beds and professionals, is considered limited to keep up with the population growth of the region, reflecting a common reality in many municipalities in the country. The sanitary conditions for endemic diseases, such as malaria and dengue, are a point of attention in the region. On the other hand, a positive fact is the downward trend in the infant mortality rate in the municipality.

Basic sanitation and energy supply present different scenarios. The water supply and garbage collection serve a significant portion of the urban area. However, the sewage network coverage is still incipient. The electricity supply, in turn, is provided by a private utility and serves most of the municipality.

The economic activities of Curionópolis and the region are strongly based on mining and agriculture, with an emphasis on cattle ranching. In 2019, the employed population in the municipality had an average monthly salary of 2.3 minimum wages. The Project will be inserted into this context, with the expectation of strengthening the local economy through the generation of direct and indirect jobs and an increase in tax revenue.

20.2.10 Legally Protected Areas and Traditional Population

The Project area is not located within Conservation Units (Parks, Environmental Protection Areas, Ecological Stations, etc.) nor in their buffer zones. Likewise, no Indigenous Lands, Quilombola Communities, or settlement projects were identified overlapping its direct area of influence as part of the Project area.

20.2.11 Land Use and Occupation

The area of influence of the Project, in the municipality of Curionópolis, is predominantly occupied by pastures, a reflection of the extensive cattle ranching activity in the region. The diagnosis of vegetation cover and land use and occupation, carried out for the project, identified pastures as the main formation, followed by fragments of Open Ombrophilous Forest in different stages of regeneration and, to a lesser extent, by areas of subsistence agriculture. The area is, therefore, predominantly composed of already anthropized areas, with an emphasis on cattle ranching. The history of mining and artisanal mining activities has also left its mark on the local landscape.

The land use and cover situation is presented in Figure 20-2.

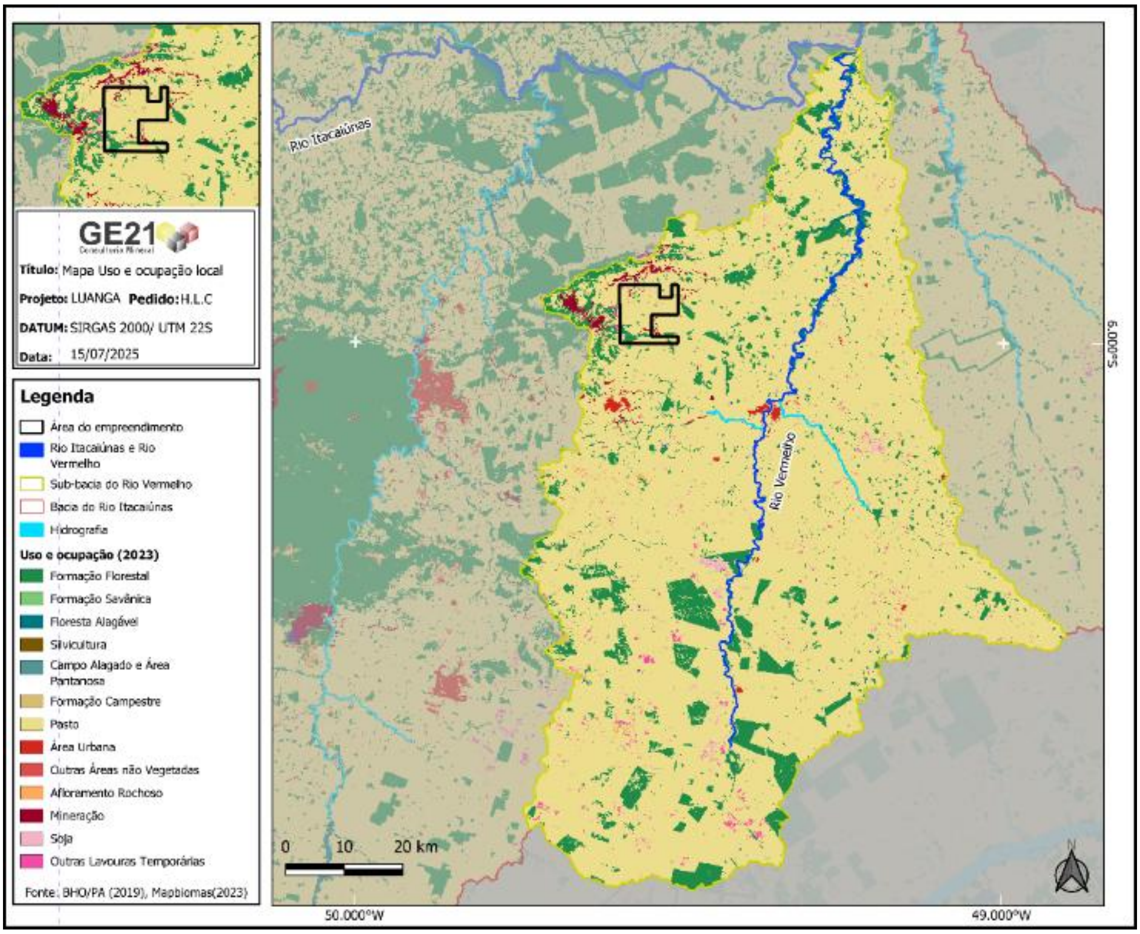


Figure 20-2: Land use and cover map of the Direct Influence Area

Source: GE21, 2025.

20.3 Main Environmental and Social Impacts

The impacts listed below deserve to be highlighted, between positive and negative:

- Stimulation of the local and regional economy.
- Tax collection (CFEM, ISS, etc.).
- Job and income generation.
- Interference in the processes of superficial soil dynamics.
- Suppression and fragmentation of native vegetation.
- Disturbance of wildlife.
- Loss of habitats and alteration in ecological processes; and
- Alteration in the local water regime.

The positive impacts identified for the Project, in both the implementation and operation phases, are those related to the socioeconomic environment, such as the dynamization of the local and regional economy, tax collection, and the generation of employment and income, bringing benefits not only to the municipality of Curionópolis but also to Parauapebas and Eldorado dos Carajás.

Among the negative impacts, those related to the suppression of native vegetation and interventions on water resources stand out.

As previously described, the area and region where the Project will be developed is inserted in a matrix that is already mostly anthropized and has undergone an intense process of replacing the vegetation cover for the formation of pastures. For the implementation of the project, the suppression of vegetation in the remaining fragments of Open Ombrophilous Forest is estimated. Such environments, whether forested or not, constitute important areas for foraging, shelter, and reproduction for several species of animals.

The main direct intervention on water resources will be the construction of the dam on Luanga Creek for the formation of the water reservoir. This action will result in a negative impact on the downstream flow regime and the flooding of the reservoir area. However, the reservoir project provides for the maintenance of an ecological flow for the watercourse, and its main role is to regulate the flow, ensuring the project's supply without compromising the water availability of the basin during the dry season. Additionally, the hydrogeological studies indicate that the activities of lowering the water table, necessary for the opening of the pit, should not imply significant changes in the water availability of the springs and water points in the direct area of influence, since these are mainly sustained by direct surface runoff and the shallower porous aquifer system.

20.4 Mitigating, Measures and Plans - Environmental and Social Programs

The management of impacts will be carried out through a robust environmental control system, based on Bravo's ESG Policy. This policy establishes the commitment to continuous improvement and environmental protection, based on principles such as the assessment of all potential impacts, the minimization of waste generation, the progressive rehabilitation of areas, dialogue with stakeholders, and compliance with the best industry practices. To operationalize this policy and to manage, mitigate, and compensate for the identified impacts, a comprehensive set of environmental and social programs has been proposed, which will be detailed below.

- Programs for the Physical Environment
  - Environmental Management Program for Construction
  - Drainage, Erosive Processes, and Sediment Control Program
  - Water Resources Management and Effluent Monitoring Program
  - Solid Waste Management Program (SWMP)
  - Water Quality Monitoring Program (Surface and Groundwater)
  - Atmospheric Emissions Control and Air Quality Monitoring Program
  - Noise and Vibration Management and Monitoring Program
- Programs for the Biotic Environment
  - Wildlife and Aquatic Ecosystems Monitoring Program
  - Fauna Scare and Rescue Program
  - Roadkill Monitoring Program

- Degraded Area Recovery Plan (DARP)
- Germplasm Rescue and Salvage and Forest Replacement Program
- Environmental Compensation Program

- Programs for the Socioeconomic Environment
  - Social Communication and Socio-institutional Relations Program
  - Local Labor Training and Prioritization Program
  - Economic Diversification Support Plan (EDSP)
  - Environmental Education Program

20.5 Mine Closure Plan

The conceptual mine closure plan, detailed in the PAE, aims for the environmental rehabilitation of the area and the physical and chemical stability of the remaining structures. The plan meets the requirements of Brazilian legislation, such as ANM Resolution No. 68/2021, and follows international best practice guidelines (ICMM).

The closure actions include the decommissioning of all industrial and administrative facilities through selective demolition, maximizing material recycling and the correct disposal of waste. The rehabilitation of geotechnical structures includes the reshaping and revegetation of the waste and dry-stacked tailings piles to ensure their long-term stability. The open pits are expected to form lakes, for which the filling and water quality will be monitored.

The total estimated cost for the execution of the closure plan is R$ 103.7 million (or US$ 17.9 million), to be disbursed over a 10-year period following the cessation of operations. A post-closure monitoring program will be implemented to assess the effectiveness of the rehabilitation measures and to verify the stability of the area before its return to the surface rights holders.

21 CAPITAL AND OPERATING COSTS

21.1 Basis of Estimate

The capital cost estimate was based on GE21 database, usual indexes from mineral industry (Cost Mine Magazine) and public reports from similar operations in the region. Due to the methodology used to develop the capital estimate and the conceptual level of engineering definition, the estimate has an accuracy of -30% +50%, which is in accordance with the Association for the Advancement of Cost Engineering International (AACE International) guidelines for a PEA study.

Data input for the estimates has been obtained from conceptual project developed internally, including mining schedule, process plant conceptual route.

The costs reference prior to 2025 were update by exchange rates and inflation in the period.

21.2 Capital Cost

The total initial capital cost summary is presented in Table 21-1. For the purposes of this report, two scenarios were evaluated: a base case, in which the flotation concentrated is sold as-is for external refining; and an alternative case, in which the concentrate is processed at an in-house refinery, and the final product—a Pt-Pd-Rh-Au-Ni alloy—is sold.

The total CAPEX (Capital Expenditures) for the base-case is estimated in US$ 592.9, comprising US$ 495.8 refers initial investment and US$ 97.1 in sustaining capital.

Table 21-1: Capital Costs Estimates

CAPEX Summary (MUSD)			
AREA	TOTAL	INITIAL	SUSTAINING
Mine	59.4	36.7	22.7
Plant & Dry Stack	401.9	319.2	74.4
Infrastructure	19.4	19.4	-
Transmission Line and Substation	17.3	17.3	-
Indirect	94.8	94.8	-
Total CAPEX (Base Case)	**592.9**	**495.8**	**97.1**
Mine Closure	17.9		

Source: GE21, 2025.

21.2.1 Direct Costs - Mining

For the purpose of this report, mining operations will be outsourced under Bravo's supervision.

Major components of capital costs include mine preparation and infrastructure works, such as pre stripping, topsoil removal and vegetal suppression, WRSF area preparation, will be contractors' expenses (including mobilization and site construction and other costs associated with mine start-up activities). A contingency of 20% is included in all CAPEX items. Pre Stripping costs exclude contingencies and are priced at the same contractor rates used for the waste mining costs, benchmarked to current contractor rates in the Carajás region. Table 21-2 shows the initial and sustaining capital estimated.

Table 21-2: Mine CAPEX Estimate

Mine CAPEX Estimate (MUSD)				
Description		Total	Initial	Sustaining
Mine Preparation		**31.2**	**27.0**	**4.2**
	Pre Stripping	25.0	25.0	0,0
	Dewatering Pumping System	0.7	0.3	0,4
	Deforesting	2.1	0.9	1,2
	Grubbing and Topsoil Removal	3.3	0.8	2,5
Technical Studies		**2.3**	**2.3**	**0.0**
	Hydrogeological Study	0.7	0.7	0,0
	Geotechnical Study	1.5	1.5	0,0
	Acid Drainage Potential Study	0.1	0.1	0,0
Contractor's Mobilization		**14.8**	**1.1**	**13.7**
WRSF Base Preparation		**9.6**	**4.8**	**4.8**
	Deforesting	1.2	0.6	0,6
	Grubbing and Topsoil Removal	6.6	3.3	3,3
	Underdrain	1.0	0.5	0,5
	Peripherical Drain Channel	0.8	0.4	0,4
Mine Accesses		**1.5**	**1.5**	**0.0**
TOTAL		**59.4**	**36.7**	**22.7**

Source: GE21, 2025.

21.2.2 Direct Costs – Processing Plant

The process equipment requirements were defined based on a conceptual flowsheet, developed from the test work conducted throughout the project's development. Mechanical equipment, buildings, and supporting facilities were estimated using cost indexes published in specialized sources (such as *Mining Cost*), benchmarking data from similar projects, and the GE21 internal database. Table 21-3 shows the estimated plant and infrastructure capital by area.

Table 21-3: Plant CAPEX and Infrastructure Estimate

Plant CAPEX Estimate (MUSD)				
DIRECT COST				
AREA	**DESCRIPTION**	**TOTAL**	**INITIAL**	**SUSTAINING**
Plant		**283.2**	**283.2**	
	Crushing	66.0	66.0	
	Grinding	96.0	96.0	
	Flotation	51.6	51.6	
	Concentrate Dewatering	14.4	14.4	
	Reagents	9.6	9.6	
	Utilities	24.0	24.0	
	Facilities	21.6	21.6	
Infrastructure		**38.4**	**36.0**	**2.4**
	Earthmoving & Internal Roads	19.2	19.2	
	Water Storage Dam	6.0	6.0	
	Support Buildings	3.6	3.6	
	Tails Storage Facility	9.6	7.2	2.4
Dry Stacking		**80.3**	**8.3**	**72.0**
	Preparation	33.5	8.3	25.2

Plant CAPEX Estimate (MUSD)				
	Plant	46.8		46.8
230 kV Transmission Line and 100 MVA SE		**17.3**	**17.3**	
TOTAL		**419.3**	**344.9**	**74.4**

Source: GE21, 2025.

21.2.3 Infrastructure – Installation

For site construction, Table 21-4 shows the estimated capital required.

Table 21-4: Site construction estimate

Installation CAPEX Estimate (MUSD)	
ITEM	**TOTAL**
Installation	**19.4**
Ancillary Facilities	17.4
Construction Site	2.0

Source: GE21, 2025.

21.2.4 Indirect Costs

Indirect costs include that not directly related to direct construction but are necessary for Project completion. Table 21-5 shows the estimated capital for indirect costs. Contingency of 20% is included in the costs.

Table 21-5: Indirect cost estimate

Indirect Cost Estimate (US$)	
Description	MUSD
Freight, Technical Support	20.4
Spare Parts & First Fill	36.0
EPCM	24.0
Owner's Cost	8.4
Insurance and Taxes	6.0
Total Indirect Costs	94.8

Source: GE21, 2025.

21.2.5 Closure Costs

Closure and reclamation costs were estimated based on the projected areas to be rehabilitated, as defined in the conceptual general arrangement. A summary of the cost estimate is presented in Table 21-6. Smelter closure was not included.

GE21 Consultoria Mineral

Table 21-6: Closure costs estimate.

Mine Closure Estimated CAPEX		
AREA	**ITEM**	**MUSD**
Industrial & Administrative Facilities		**4.5**
	Demolition & Cleanup	2.7
	Reclaiming	1.8
Mine		**1.3**
	Pit Reclaiming	0.9
	Drainage	0.4
WRSF, DSF & Sumps		**4.2**
	Reclaiming	2.6
	Drainage	1.6
Tails Dam Decharacterization		**1.4**
	Preparation	0.3
	Decharacterization	1.1
Auxiliary Structures		**2.7**
	Reclaiming	1.9
	Drainage	0.8
Fencing		**1.5**
Social		**1.5**
Environmental Monitoring		**0.7**
TOTAL		**17.9**

Source: GE21, 2025.

21.3 Operational Costs – OPEX

Operating costs (OPEX) include the ongoing cost of operations related to mining, processing, tailings disposal and general administrative (G&A) costs. It covers all direct costs necessary for the operation of the enterprise.

21.3.1 Basis of estimate

Costs were estimated in Brazilian currency and converted to US$ using an exchange rate of R$ 5.80 : US$ 1.00.

Table 21-7 summarizes the operational costs estimated for the Project.

Table 21-7: Operational costs summary.

Description	Unit	OPEX
Mine	US$/t processed	22.80
Process	US$/t processed	12.12
Freight	US$/t processed	0.94
G&A	US$/t processed	5.00
Total	**US$/t processed**	**40.86**

Source: GE21, 2025.

21.3.2 Mining OPEX

Costs were originally estimated in Brazilian Reais (BRL) and subsequently converted to U.S. dollars (US$) using an exchange rate of R$ 5.80 to US$ 1.00.

For the purposes of this PEA, mining operations are assumed to be outsourced under a fixed unit cost contract, covering both mineralized material and waste, friable and fresh rock material, and based on an average haul distance. Mine management costs were included in this estimation.

Estimated mining costs reflect typical rates currently practiced in the northern region of Brazil, where the project is located, and were validated using data from the GE21 internal database.

Table 21-8 summarises the mining costs assumed to the outsourced operation.

Table 21-8: Mine OPEX

MINING OPEX	
Material	Cost (US$/t mined)
Mineralized Material - ROM	3.20
Weathered Waste	2.10
Fresh Waste	2.90

Source: GE21, 2025.

21.3.3 Processing OPEX

Plant & processing operating costs were estimated based on similar operations around the site is located, in the Carajás Mineral Province. Table 21-9 shows the cost considered for OPEX estimate.

Table 21-9: Plant Cost - OPEX

Plant Operational Cost Estimate		
Item	Description	US$/t
Materials	Crusher Liner	0.16
	Mill Liners	0.25
	Grinding Media	0.75
	Lubricants	0.10
	Maintenance Parts	1.10
	Subtotal	**2.36**
Power	Power	0.73
	Subtotal	**0.73**
Reagents	CMC	2.98
	PAX	0.55
	MIBC	0.38
	W31	0.27
	Subtotal	**4.18**
Labour	Operation	0.75
	Maintenance	0.28
	Subtotal	**0.75**
Dry Stacking	Thickening & Filtering	3.20
	Tails Handling	0.54
	Subtotal	**3.74**
Mobile Equipment	Services	0.08
	Subtotal	**0.08**
Total		**12.12**

Source: GE21, 2025.

21.3.4 Freight and Logistics Costs

For purpose of this PEA, a fixed cost of US$ 50.86/t of concentrate was assumed, based on recent references of other projects in the region.

21.3.5 General & Administrative Costs – G&A

For purpose of this PEA, a fixed cost of US$ 5.00/t ROM was assumed, based on recent references of other projects in the region and GE21's database.

22 ECONOMIC ANALYSIS

The economic analysis for the Luanga Project is based on Mineral Resources data, including the annual mining schedule previously presented in this report. The outcome of the economic analysis is subject to known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from them. Further, the assumed feed to the process plant in the PEA includes Inferred Mineral Resources, which have had insufficient work to classify them as Indicated Mineral Resources. It is uncertain but reasonably expected that Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration. Further, Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. There is no certainty that all Mineral Resources will be converted into Mineral Reserves and there is no certainty that the results of this PEA will be realized.

The information on which this analysis is based is listed below:

- Mineral Resources Estimates
- Assumed fixed exchange rate of R$ 5.80/US$
- Proposed mine production plan
- Projected mining and processing recovery rates
- Fixed installed processing plant capacity
- Assumptions on closure costs
- Assumptions on environmental, licensing, and social risks
- Changes in production costs relative to the assumptions

This analysis does not rely on:

- Unrecognized environmental risks
- Unanticipated recovery expenses
- Unexpected variations in the quantity of mineralized material, grade, or recovery rates
- Different geotechnical and/or hydrogeological considerations during operations
- Unexpected variations in the quantity of mineralized material, grade, metallurgical recovery efficiency, and plant recovery efficiency
- Assumptions about geotechnical or hydrogeological conditions during operations
- Accidents, labor disputes, and other mining industry risks
- Changes in tax rates
- Assumptions of commercial discounts not foreseen in the financial analysis

22.1 Methodology

An economic model was developed to estimate the post-tax annual cash flow and sensitivity analysis of the project based on an assumed discount rate of 8%. Capital and operating

GE21 Consultoria Mineral

cost estimates are described in Section 21 of this Report. The economic analysis was performed without inflation.

The economic analysis was conducted with the following assumptions:

- Year 0 corresponds to the beginning of the pre-production phase
- Price inflation and escalation factors are ignored (constant dollar basis)
- Results are based on 100% equity capital
- Project revenue is derived from the sale of a basket of metals including PGM (Pd, Pt, Rh), Gold (Au), and Nickel (Ni)
- All production is sold at the year of production

22.2 Discount Rate

The financial analyses were conducted based on discounted cash flow using 8% discount rate. The discount rate directly impacts the Net Present Value (NPV).

22.3 Exchange Rate Forecast

The exchange rate was defined based on parameters adopted in international projects, not using values projected by any financial institution. The exchange rate used was R$ 5.80/US$.

22.4 Taxes and Duties

Payable taxes were estimated by applying existing tax laws to revenues associated with production. A detailed description is provided in the following subsections.

22.4.1 Financial Compensation for the Exploration of Mineral Resources

The CFEM is a federal royalty fee paid to the Brazilian government for the extraction and economic exploitation of Brazilian mineral resources. The tax rate varies between 1% and 3%, depending on the type of mineral product, and is applied to net revenue.

For the mineral products, the applicable tax rates are 1.5% for gold and 2.0% for the other metals (PGM and Nickel).

22.4.2 Income Tax and Social Contribution

Income Tax (Corporate Income Tax) applies to the profits obtained by companies and other legal entities. It is calculated based on the accounting result determined by the legal entity at the end of a reporting period, such as a quarter or fiscal year. In Brazil, the income tax rate for companies taxed under the Real Profit regime is 15% of taxable income, with the possibility of an additional 10% on the portion of profit exceeding the calculation base of BRL 20,000/month or BRL 240,000/year.

Companies located in the Amazon region may benefit from certain tax incentives. SUDAM is an administratively and financially independent federal government agency that oversees development in the Amazon region. The region includes the state of Pará in which the Project is located. Under the concession program, companies can receive either partial or complete tax exemption on income taxes for Brazilian companies.

The tax exemption applies only to income from facilities operating in the designated region and consists of a reduction of 75% off the regular corporate income tax (25%). For the purposes of the PEA, the financial model factors in a reduction of the corporate income tax rate plus social contribution of 34% (25% + 9%) to the 15.25% (25% x 25% + 9%) rate available under the SUDAM regime for the Project. The concession is available for an initial period of 10 years of operation.

The PEA assumes that the Luanga Project would be eligible for SUDAM tax exemption, but this can only be confirmed once an application has been submitted and approved.

22.4.3 Social Contribution

The Social Contribution is aimed at financing social security issues, which include health, social security, and social assistance. The tax rate is 9% and is also applied to earnings before income taxes.

22.4.4 Other Fees

The financial model includes a fee for control, monitoring, and supervision of research, mining, exploration, and exploitation of Mineral Resources, the Mineral Resources Inspection Fee (TFRM - Taxa de Fiscalização de Recursos Minerais), payable in the State of Pará. This tax is charged on PGM, Nickel, Copper production, at a rate of BRL 14.40 per ton of mill feed extracted. This tax is charged on Gold production, at a rate of BRL 9.60 per gram of production.

22.5 Royalty Rights

In addition to governmental tax obligations, the project is subject to the payment of contractual royalties to third parties, calculated on revenue. The applicable royalties are 2.0% to BNDES, 1.0% to Vale, and royalties to the surface rights holders of 0.75% for gold and 1.0% for other metals.

22.6 Working Capital

A high-level estimate of working capital is incorporated into the cash flow based on accounts receivable (30 days), inventories (30 days), accounts payable (30 days), and taxes and deductions (30 days). These terms serve as a flexible and adjustable starting point that can be revised as the project progresses, and more information becomes available.

22.7 Depreciation

Depreciation was estimated in a simplified manner, depreciating the investments annually over the life-of-mine at a rate of 10% per year. The 10% rate is an assumption based on similar projects, adopted as a flexible and adjustable starting point that can be revised as the project progresses, and more information becomes available.

22.8 Results

A simplified discounted cash flow was developed to evaluate the project based on economic-financial parameters, mining schedule results, initial and sustaining capital costs, and operating costs estimates.

22.9 Discounted Cash Flow

The discounted cash flow is presented in Table 22-1.

GE21 Consultoria Mineral

Table 22-1 – Discounted Cash Flow, Luanga Project PEA

Cash Flow	Unit	Total		1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
Pd Revenue	MUSD	4.685,33	-	147,92	288,81	342,60	352,63	276,32	264,89	264,89	264,89	264,89	264,89	275,11	275,11	275,11	275,11	275,11	288,53	288,53	-
Pt Revenue	MUSD	3.428,64	-	106,19	205,99	210,80	213,82	231,15	198,85	198,85	198,85	198,85	198,85	215,10	215,10	215,10	215,10	215,10	195,47	195,47	-
Rh Revenue	MUSD	1.282,18	-	43,25	76,84	79,62	83,16	92,25	71,73	71,73	71,73	71,73	71,73	80,68	80,68	80,68	80,68	80,68	72,51	72,51	-
Au Revenue	MUSD	396,51	-	14,61	31,02	29,81	25,60	21,85	21,64	21,64	21,64	21,64	21,64	23,64	23,64	23,64	23,64	23,64	23,61	23,61	-
Ni Revenue	MUSD	1.845,57	-	54,09	115,75	153,82	143,68	113,00	96,99	96,99	96,99	96,99	96,99	111,16	111,16	111,16	111,16	111,16	112,25	112,25	-
Sulph. Acid	MUSD	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Gross Revenue	MUSD	$ 11.638	-	366,05	718,42	816,65	818,90	734,57	654,10	654,10	654,10	654,10	654,10	705,68	705,68	705,68	705,68	705,68	692,38	692,38	-
Royalties	MUSD	$ 695	-	21,85	42,87	48,78	48,94	43,91	39,08	39,08	39,08	39,08	39,08	42,16	42,16	42,16	42,16	42,16	41,37	41,37	-
Net Revenue	MUSD	$ 10.943	-	344,20	675,55	767,88	769,95	690,66	615,01	615,01	615,01	615,01	615,01	663,52	663,52	663,52	663,52	663,52	651,01	651,01	-
Total opex	MUSD	$ 6.755	-	170,24	291,51	302,85	297,99	310,19	478,17	478,17	478,17	478,17	478,17	456,59	456,59	456,59	456,59	456,59	354,37	354,37	-
Opex Mine	MUSD	$ 3.770	-	78,65	110,88	122,26	117,03	127,32	296,95	296,95	296,95	296,95	296,95	275,60	275,60	275,60	275,60	275,60	175,48	175,48	-
Opex Plant	MUSD	$ 2.004	-	61,47	121,25	121,22	121,46	122,75	121,64	121,64	121,64	121,64	121,64	121,49	121,49	121,49	121,49	121,49	120,08	120,08	-
Smelting/ Refining	MUSD	$ -	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Freight	MUSD	$ 155	-	4,75	9,36	9,36	9,38	9,48	9,39	9,39	9,39	9,39	9,39	9,38	9,38	9,38	9,38	9,38	9,27	9,27	-
SG&A	MUSD	$ 827	-	25,36	50,02	50,01	50,11	50,64	50,18	50,18	50,18	50,18	50,18	50,12	50,12	50,12	50,12	50,12	49,54	49,54	-
Profit	MUSD	$ 4.188	-	173,96	384,03	465,03	471,97	380,47	136,84	136,84	136,84	136,84	136,84	206,93	206,93	206,93	206,93	206,93	296,64	296,64	-
TFRM	MUSD	$ 411	-	12,59	24,84	24,83	24,88	25,15	24,92	24,92	24,92	24,92	24,92	24,89	24,89	24,89	24,89	24,89	24,60	24,60	-
Depreciation	MUSD	$ 589	-	49,58	49,58	51,06	57,20	57,84	57,85	58,21	58,88	58,89	58,90	9,33	9,33	7,85	1,70	1,07	1,05	0,69	
EBIT (US$)	MUSD	$ 3.188	-	111,79	309,62	389,14	389,88	297,49	54,07	53,71	53,05	53,04	53,02	172,71	172,71	174,19	180,34	180,97	270,99	271,35	-
Income tax	MUSD	$ 634	-	9,50	26,32	33,08	33,14	25,29	4,60	4,57	4,51	4,51	4,51	58,72	58,72	59,22	61,32	61,53	92,14	92,26	-
Operational profit(US$)	MUSD	$ 2.554	-	102,29	283,30	356,06	356,74	272,21	49,48	49,15	48,54	48,53	48,51	113,99	113,99	114,97	119,02	119,44	178,85	179,09	-
(=) EBIT	MUSD	$ 3.188	-	111,79	309,62	389,14	389,88	297,49	54,07	53,71	53,05	53,04	53,02	172,71	172,71	174,19	180,34	180,97	270,99	271,35	-
Depreciation	MUSD	$ 589	-	49,58	49,58	51,06	57,20	57,84	57,85	58,21	58,88	58,89	58,90	9,33	9,33	7,85	1,70	1,07	1,05	0,69	
(=) EBITDA	MUSD	$ 3.777		161,37	359,20	440,20	447,09	355,33	111,93	111,93	111,93	111,93	111,93	182,04	182,04	182,04	182,04	182,04	272,04	272,04	-
Ebtida Margin	MUSD			47%	53%	57%	58%	51%	18%	18%	18%	18%	18%	27%	27%	27%	27%	27%	42%	42%	0%
(-) Capex	MUSD	$ 496	495,78	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
(-) Sustaining capital	MUSD	$ 97	-	14,77	61,50	6,31	0,15	3,63	6,62	0,15	0,15	-	-	-	-	-	-	-	-	3,83	-
(-) ARO	MUSD	$ 18	-	-	-	-	-	-	-	-	1,79	1,79	1,79	1,79	1,79	1,79	1,79	1,79	1,79	1,79	-
(+) Residual Value	MUSD	$ -																			
(+-) Working Capital	MUSD	$ -	-	39,26	29,87	9,43	- 0,25	- 5,48	9,13	0,00	0,00	14,61	0,00	- 3,77	-	- 0,04	- 0,17	- 0,02	- 20,27	- 0,01	- 72,31
(-) Income Tax	MUSD	$ 634	-	9,50	26,32	33,08	33,14	25,29	4,60	4,57	4,51	4,51	4,51	58,72	58,72	59,22	61,32	61,53	92,14	92,26	-
(=) After-tax Cash Flow to the Firm (FCFF)	MUSD	$ 2.647	- 495,78	112,61	303,00	397,68	414,19	335,52	98,20	107,36	107,41	92,81	107,42	127,09	123,32	122,86	120,90	120,53	200,17	179,79	72,31

Source: GE21, 2025.

The results are presented in Table 22-2.

Table 22-2 - Results

Discount Rate	8%
NPV (US$ X 1000)	$ 1,249
IRR (%)	49%
Discounted Payback After-Tax (years)	2.42

Source: GE21, 2025.

22.10 Sensitivity Analysis

Based on the result, a sensitivity analysis (Figure 22-1) was conducted to observe the Project's value and profitability behaviour in response to:

- Price variation
- CAPEX variation
- OPEX variation

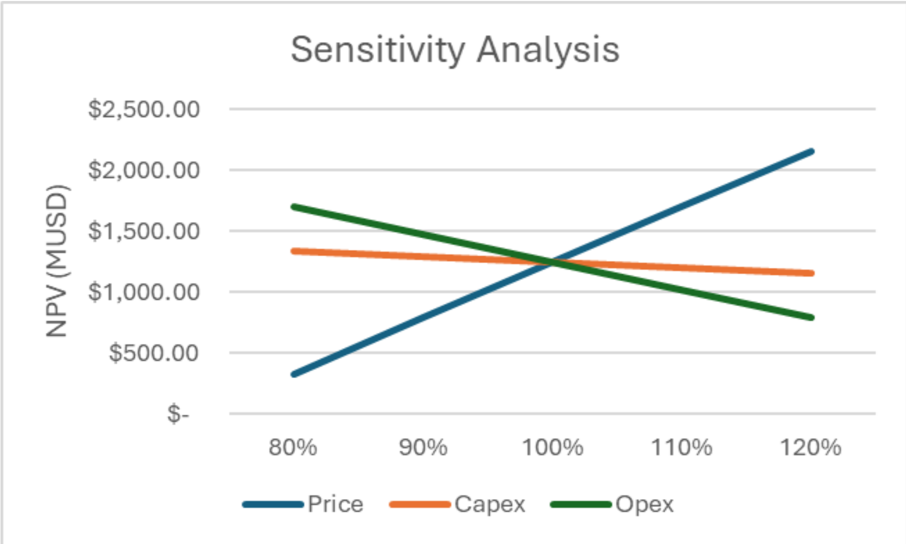


Figure 22-1 - Sensitivity Analysis

Source: GE21, 2025.

Table 22-3 below presents the sensitivity analysis indicating the variation of the NPV as a function of the discount rate and the reference price of Palladium Equivalent.

Table 22-3 - Sensitivity Analysis for Price vs Discount Rate

		Price (US$/oz t)				
		B.E.P.*	**-10%**	**Base Case**	**+10%**	**+20%**
	$ 1,248.54	**$ 839.08**	**$ 1,143.90**	**$ 1,271.00**	**$ 1,398.10**	**$ 1,525.20**
Discount Rate	**5%**	$ -391.31	$ 1,056.36	$ 1,627.51	$ 2,196.45	$ 2,765.39
	8%	$ -347.66	$ 795.41	$ 1,248.54	$ 1,699.95	$ 2,151.36
	10%	$ -325.96	$ 662.23	$ 1,055.17	$ 1,446.66	$ 1,838.15
	20%	$ -270.32	$ 265.44	$ 481.40	$ 696.68	$ 911.97
	49%	$ -245.31	$ -72.40	$ -1.26	$ 69.77	$ 140.80

*B.E.P.: Break Even Point at 8% ($/oz)

Source: GE21, 2025.

23 ADJACENT PROPERTIES

Within 10 km of the Project, there are two main mineral deposits of relevance to Luanga: the Serra Pelada Au + PGM deposit and the Serra Leste iron ore deposit (Figure 23-1), neither of which have similar geology to Luanga, and are unrelated to mafic-ultramafic intrusions. In addition, there are several minor gold occurrences, mostly operated by artisanal miners, in the area. These projects are located to the west of the Luanga Project.

The Serra Pelada Au + PGM deposit occurs 8 km west of Luanga in a tenement with a Mining License held by Serra Pelada Companhia de Desenvolvimento Mineral. During the 1980's, there were tens of thousands of illegal miners active in the Serra Pelada open pit, the largest gold mine in Brazil in its day. The pit reached 400 m in length by 300 m wide, to a depth over 120 m below surface, all dug by hand. History records that 1.04 Moz Au was extracted (Source: Meireles & Silva, 1988).

The Serra Leste high-grade hematite open pit iron ore mine occurs approximately 8.5 km southwest of the Project, in a tenement held by Vale. Serra Leste includes active open pit mining and a beneficiation process comprising screening, hydrocycloning, crushing and filtration (Source: Vale public records).

There is no open ground for new exploration claims surrounding the Luanga License, and Vale is the major holder of exploration claims in the region.

The QP has not been able to verify the information on the adjacent properties and observes that the information in Section 23 is not indicative of the mineralization on the property that is the subject of this Technical Report.

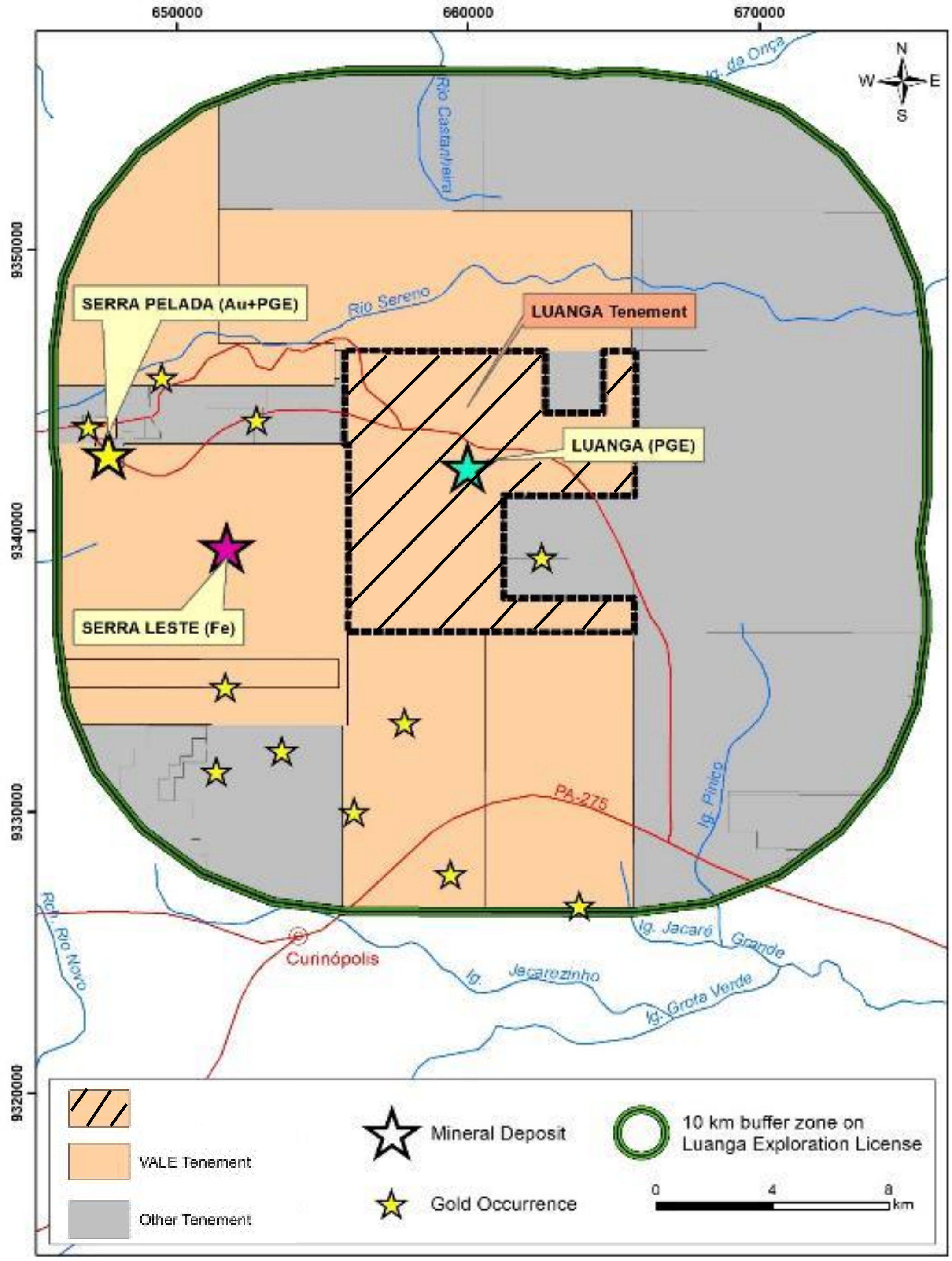


Figure 23-1: Mineral deposits adjacent to Luanga Project

Source: Bravo, 2023.

GE21 Consultoria Mineral

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Alternate Case - Vertical Integration

For effect of this PEA, a second scenario was developed, considering the on-site smelting of the flotation concentrate.

24.1.1 Introduction

The smelter is anticipated to treat up to 200,000 tonnes wet basis of PGM-rich, nickel sulphide concentrate, up to 800.000 ounces of PGM, 16.000 of nickel, and 120.000 tonnes of sulphuric acid annually. The metallurgical process comprises a single line fluidized bed roaster and direct-current (“DC”) electric arc furnace producing a PGM-enriched, ferronickel alloy for further precious and base metal refining. Extracted sulphur will be processed to sulphuric acid in a sulphuric acid plant.

Bravo conducted an internal review of potential processing technologies to which the Luanga concentrate will be amenable for treatment. Factors considered during review included technological appropriateness (maximising recovery, technical amenability), technology risk (execution and operational technical risks, intrinsic design factors), environmental management factors (waste and gases management) and financial considerations (CAPEX and OPEX).

Technologies that were included in this review included Flash Smelting, Ausmelt, AC Electric Arc Furnace Smelting, DC Reductive Electric Arc Furnace Smelting and hydrometallurgical process options.

DC Reductive EAF smelting was found to be the most appropriate and further pursued with preliminary bench-scale amenability tests, conducted in partnership with Arxo Metals in South Africa.

In the selected process, sulphide concentrates are roasted for sulphur removal followed by reductive smelting in a DC EAF. A PGM-rich FeNi alloy is then converted and atomized for further refining.

24.1.2 CAPEX Estimates

The process plant, infrastructure and utilities engineering has been developed to support a CAPEX estimate according to the AACE class 5 standard. This means that CAPEX estimate has a combined accuracy of between -20% and +50%. Total estimated is US$ 181.9 million.

Table 24-1 shows a summary of the estimates.

GE21 Consultoria Mineral

Table 24-1: Summary of smelter CAPEX estimate.

TOTAL SMELTER CAPEX ESTIMATE	
Item	**Total (US$)**
Major Equipment	83.5
Auxiliary Equipment	25.0
Shipping & Logistics	5.5
Engineering, Construction & Installation	44.0
Contingency (15%)	23.9
Total CAPEX	**181.9**

Source: Bravo, 2025.

In the Table 24-2 is presented a list of the major equipment and their characteristics, totalizing US$ 83.5 million.

Table 24-2: Major equipment estimates

MAJOR EQUIPMENT COSTS				
Equipment	**Unit Cost (MUSD)**	**Quantity**	**Total (MUSD)**	**Notes**
Rotary Dryer	1.2	1	1.2	Indirect heating, 30 t/h capacity
Fluidized Bed Roaster	8.5	1	8.5	Bubbling bed, 25 t/h, SO_2 capture
DC Electric Arc Furnace (EAF)	25.0	1	25.0	25 MW, slag/alloy separation
Peirce-Smith Converter	3.5	1	3.5	Fe removal, oxygen-blown
High-Pressure Water Atomizer	4.2	1	4.2	2.5 t/h powder production
Sulfuric Acid Plant (DCDA)	18.0	1	18.0	12 t/h H_2SO_4, double contact
Leaching & Solvent Extraction	6.8	1	6.8	Ni/Cu/PGM circuits
PGM Refinery	12.0	1	12.0	Aqua regia, precipitation
Off-Gas Scrubbers	2.5	1	2.5	HCl/As/F removal
Slag Handling System	1.8	1	1.8	Granulation, disposal
TOTAL			**83.5**	

Source: Bravo, 2025.

Table 24-3 shows the estimates for the auxiliary equipment and utilities.

Table 24-3: Auxiliary equipment CAPEX estimate.

AUXILIARY EQUIPMENT & UTILITIES		
Item	**Total (MUSD)**	**Notes**
Pumps, Compressors, Conveyors	5.2	Material handling
Cooling Towers & Chillers	3.8	Process cooling
Electrical Substation	6.0	35 MW capacity
Instrumentation & Control	4.5	Automation (DCS/PLC)
Dust Collection (Baghouses)	2.3	ESPs + filters
Water Treatment Plant	3.2	500 m^3/h capacity
TOTAL	**25.0**	

Source: Bravo, 2025.

Table 24-4 shows the estimated cost for logistics, considering maritime freight from Barcarena Port to China.

GE21 Consultoria Mineral

Table 24-4: Shipping & logistics CAPEX estimate.

SHIPPING & LOGISTICS		
Item	**Total (MUSD)**	**Notes**
Ocean Freight (40-ft Containers)	2.5	~200 containers
Heavy-Lift Charges	1.8	Roaster/EAF transport
Insurance	1.2	1.5% of equipment value
TOTAL	**5.5**	

Source: Bravo, 2025.

24.1.3 OPEX Estimates

The estimated costs for smelter operation are summarized in Table 24-5. The feed cost was not considered in the total OPEX.

Table 24-5: Smelter OPEX summary

SMELTER OPERATING COST			
Category	**Annual Cost ($M)**	**Unit Cost (US$/t)***	**% of Total**
Feed (Variable)	1,023.6		91.6%
Variable Costs	37.2	186.00	3.3%
Fixed Costs	17.7	88.50	1.6%
Byproduct Credits	(2.5)		-0.2%
Total OPEX	1,076.0	274.50	100%

*** Cost per tonne of concentrate.**

Source: Bravo, 2025.

Table 24-6 shows the detailed OPEX costs.

Table 24-6: Smelter OPEX

VARIABLE OPERATING COSTS				
Item	**Consumption**	**Unit Cost**	**Annual Cost ($M)**	**Specific Cost (US$/t)***
Energy & Water			**22.5**	**112.5**
Power	350 GWh/year	$0.02/kWh	7.0	35.0
Natural Gas	1.63M MMBtu/year	$6/MMBtu	9.8	49.0
Water	4.38M m³/year	$0.50/m³	2.2	11.0
Oxygen (Converting)	70.000 t/year	$50/t	3.5	17.5
Reagents & Consumables			**9.2**	**46.0**
Lime (CaO)	84.000 t/year	$50/t	4.2	21.0
Sulfuric Acid	28.000 t/year	$80/t	2.2	11.0
Coal (Reductant)	14.760 t/year	$120/t	1.8	9.0
Solvents (SX)	500 t/year	$2,000/t	1.0	5.0
TOTAL			**31.7**	**158.5**

*** Cost per tonne of concentrate.**

FIXED COST					
		Headcount	**Salary ($/year)**	**Annual Cost ($M)**	**Specific Cost (US$/t)***
Labor		**120**		**5,3**	**26,5**
	Operations	60	40000	2,4	12,0
	Maintenance	30	45000	1,4	6,8
	Management	20	60000	1,2	6,0
	Lab/QA	10	35000	0,4	1,8
Maintenance & Overheads				**12,2**	**61,0**
	Preventive Maintenance	2% of equipment CAPEX		3,6	18,0
	Spare Parts	1.3% of CAPEX		2,4	12,0
	Insurance	1.5% of CAPEX		2,7	13,5
	G&A	Office/IT costs		3,5	17,5
Total Fixed Costs				**17.7**	**87,5**

*** Cost per tonne of concentrate.**

Source: Bravo, 2025.

24.1.4 Economic Analysis

The economic analysis for the verticalization case was developed using the same assumptions as the base case scenario.

The discounted cash flow is presented in Table 24-7.

GE21 Consultoria Mineral

Table 24-7 – Discounted Cash Flow – Luanga Project, Alternate Case

Cash Flow	Unit	Total		1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
Pd Revenue	MUSD	5.475,90	-	172,32	341,50	399,03	410,71	321,84	311,26	311,26	311,26	311,26	311,26	320,42	320,42	320,42	320,42	320,42	336,05	336,05	-
Pt Revenue	MUSD	4.003,60	-	123,69	242,63	245,52	249,04	269,22	233,11	233,11	233,11	233,11	233,11	250,53	250,53	250,53	250,53	250,53	227,67	227,67	-
Rh Revenue	MUSD	1.455,41	-	48,92	88,41	90,05	94,05	104,33	81,88	81,88	81,88	81,88	81,88	91,24	91,24	91,24	91,24	91,24	82,01	82,01	-
Au Revenue	MUSD	452,62	-	16,53	36,53	33,72	28,95	24,71	25,02	25,02	25,02	25,02	25,02	26,73	26,73	26,73	26,73	26,73	26,70	26,70	-
Ni Revenue	MUSD	2.535,60	-	73,66	163,45	209,36	195,56	153,81	135,53	135,53	135,53	135,53	135,53	151,30	151,30	151,30	151,30	151,30	152,79	152,79	-
Sulph. Acid	MUSD	316,80	-	9,60	19,20	19,20	19,20	19,20	19,20	19,20	19,20	19,20	19,20	19,20	19,20	19,20	19,20	19,20	19,20	19,20	-
Gross Revenue	MUSD	$ 14.240	-	444,72	891,72	996,88	997,52	893,11	806,01	806,01	806,01	806,01	806,01	859,42	859,42	859,42	859,42	859,42	844,42	844,42	-
Royalties	MUSD	$ 832	-	25,98	52,08	58,41	58,48	52,25	47,02	47,02	47,02	47,02	47,02	50,21	50,21	50,21	50,21	50,21	49,31	49,31	-
Net Revenue	MUSD	$ 13.408	-	418,74	839,64	938,47	939,04	840,86	758,99	758,99	758,99	758,99	758,99	809,21	809,21	809,21	809,21	809,21	795,11	795,11	-
Total opex	MUSD	$ 7.364	-	188,91	328,35	339,68	334,89	347,49	515,13	515,13	515,13	515,13	515,13	493,50	493,50	493,50	493,50	493,50	390,86	390,86	-
Opex Mine	MUSD	$ 3.770	-	78,65	110,88	122,26	117,03	127,32	296,95	296,95	296,95	296,95	296,95	275,60	275,60	275,60	275,60	275,60	175,48	175,48	-
Opex Plant	MUSD	$ 2.004	-	61,47	121,25	121,22	121,46	122,75	121,64	121,64	121,64	121,64	121,64	121,49	121,49	121,49	121,49	121,49	120,08	120,08	-
Smelting/ Refining	MUSD	$ 764	-	23,43	46,20	46,19	46,28	46,78	46,35	46,35	46,35	46,35	46,35	46,29	46,29	46,29	46,29	46,29	45,76	45,76	-
Freight	MUSD	$ -	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
SG&A	MUSD	$ 827	-	25,36	50,02	50,01	50,11	50,64	50,18	50,18	50,18	50,18	50,18	50,12	50,12	50,12	50,12	50,12	49,54	49,54	-
Profit	MUSD	$ 6.044	-	229,83	511,29	598,79	604,15	493,37	243,86	243,86	243,86	243,86	243,86	315,70	315,70	315,70	315,70	315,70	404,25	404,25	-
TFRM	MUSD	$ 411	-	12,59	24,84	24,83	24,88	25,15	24,92	24,92	24,92	24,92	24,92	24,89	24,89	24,89	24,89	24,89	24,60	24,60	-
Depreciation	MUSD	$ 771	-	67,76	67,76	69,24	75,39	76,02	76,04	76,40	77,06	77,07	77,09	9,33	9,33	7,85	1,70	1,07	1,05	0,69	
EBIT (US$)	MUSD	$ 4.862	-	149,47	418,69	504,72	503,88	392,20	142,90	142,54	141,88	141,87	141,85	281,49	281,49	282,97	289,12	289,75	378,60	378,96	
Income tax	MUSD	$ 970	-	12,70	35,59	42,90	42,83	33,34	12,15	12,12	12,06	12,06	12,06	95,71	95,71	96,21	98,30	98,51	128,72	128,85	
Operational profit(US$)	MUSD	$ 3.893	-	136,77	383,10	461,81	461,05	358,87	130,76	130,43	129,82	129,81	129,79	185,78	185,78	186,76	190,82	191,23	249,88	250,11	
(=) EBIT	MUSD	$ 4.862	-	149,47	418,69	504,72	503,88	392,20	142,90	142,54	141,88	141,87	141,85	281,49	281,49	282,97	289,12	289,75	378,60	378,96	
Depreciation	MUSD	$ 771	-	67,76	67,76	69,24	75,39	76,02	76,04	76,40	77,06	77,07	77,09	9,33	9,33	7,85	1,70	1,07	1,05	0,69	
(=) EBITDA	MUSD	$ 5.633		217,23	486,46	573,96	579,27	468,22	218,94	218,94	218,94	218,94	218,94	290,82	290,82	290,82	290,82	290,82	379,65	379,65	-
Ebtida Margin	MUSD			52%	58%	61%	62%	56%	29%	29%	29%	29%	29%	36%	36%	36%	36%	36%	48%	48%	0%
(-) Capex	MUSD	$ 678	677,63	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
(-) Sustaining capital	MUSD	$ 97	-	14,77	61,50	6,31	0,15	3,63	6,62	0,15	0,15	-	-	-	-	-	-	-	-	3,83	-
(-) ARO	MUSD	$ 18	-	-	-	-	-	-	-	-	1,79	1,79	1,79	1,79	1,79	1,79	1,79	1,79	1,79	1,79	-
(+) Residual Value	MUSD	$ -																			
(+-) Working Capital	MUSD	$ -	-	45,95	37,64	13,81	- 0,36	- 6,97	8,57	0,00	0,00	10,75	0,00	- 6,07	-	- 0,04	- 0,17	- 0,02	- 20,38	- 0,01	- 82,70
(-) Income Tax	MUSD	$ 970	-	12,70	35,59	42,90	42,83	33,34	12,15	12,12	12,06	12,06	12,06	95,71	95,71	96,21	98,30	98,51	128,72	128,85	-
(=) After-tax Cash Flow to the Firm (FCFF)	MUSD	$ 3.986	- 677,63	158,58	413,23	517,24	536,80	441,86	198,22	206,82	206,88	196,14	206,88	201,18	195,11	194,65	192,69	192,32	271,31	250,82	82,70

Source: GE21, 2025.

The results are presented in Table 24-8.

Table 24-8 - Results

Discount Rate	8%
NPV (US$ X 1000)	$ 1,861
IRR (%)	49%
Discounted Payback After-Tax (years)	2.43

Source: GE21, 2025.

Based on the result, a sensitivity analysis (Figure 24-1) was conducted to observe the Project's value and profitability behaviour in response to:

- Price variation
- CAPEX variation
- OPEX variation

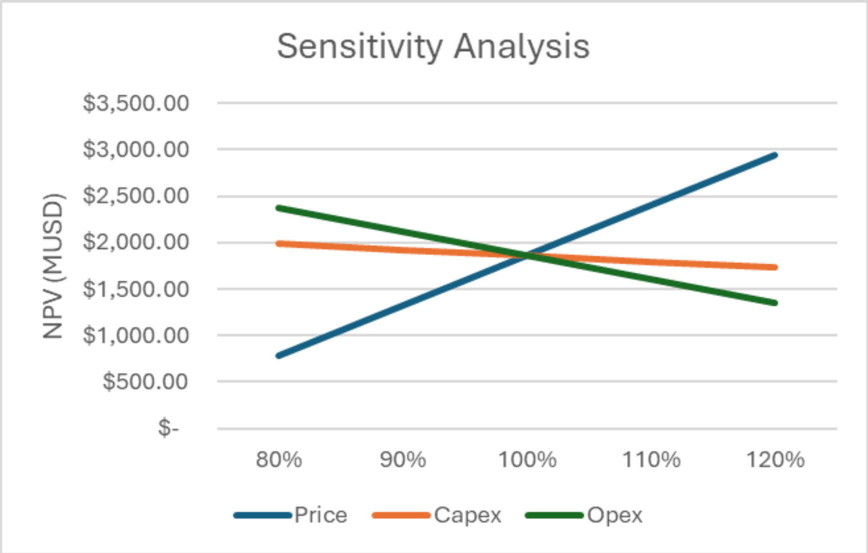


Figure 24-1 - Sensitivity Analysis

Source: GE21, 2025.

Table 24-9 below presents the sensitivity analysis indicating the variation of the Net Present Value (NPV) as a function of the discount rate and the reference price of Palladium Equivalent.

Table 24-9 - Sensitivity Analysis for Price vs Discount Rate

		Price (US$/oz t)				
		B.E.P.*	**-10%**	**Alternate Case**	**+10%**	**+20%**
	$ 1,861.03	**$ 839.08**	**$ 1,143.90**	**$ 1,271.00**	**$ 1,398.10**	**$ 1,525.20**
Discount Rate	**5%**	$ 91.27	$ 1,755.37	$ 2,436.79	$ 3,118.22	$ 3,799.64
	8%	$ -0.00	$ 1,320.22	$ 1,861.03	$ 2,401.85	$ 2,942.67
	10%	$ -46.09	$ 1,098.64	$ 1,567.75	$ 2,036.85	$ 2,505.95
	20%	$ -182.46	$ 445.74	$ 703.86	$ 961.98	$ 1,220.10
	49%	$ -291.04	$ -85.21	$ 0.00	$ 85.21	$ 170.42

*B.E.P.: Break Even Point at 8% ($/oz)

Source: GE21, 2025.

25 INTERPRETATION AND CONCLUSIONS

The QPs note the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for this PEA Report.

25.1 Mineral Exploration and Geology

In general terms, the geological descriptions, sampling procedures and density tests that were evaluated and found to be of acceptable quality and in accordance with industry best practices. It was noted that the data collection process was executed with the aim of maintaining data security. Data was stored in a standardized database, which was found to be secure and auditable. GE21 supervised the process through which density was determined and concluded that it was in conformity with industry best practices.

25.2 QA/QC

GE21 performed the evaluation of the QA/QC data, which includes Blanks, CRMs, Field Duplicates, Check Assays, and Umpire Check Assays.

The AMIS standards data used by Bravo were revised for QA/QC programs. The recommendations state that new Reference Grades and Standard Deviations should be obtained from the data acquired during the Project (after outlier treatment). After this procedure, new reference values were generated. No significant biases were found after the correction of Reference Values. Control Charts using the new calculated reference value were done.

Although the Vale database was historical in nature, the validation and correlation procedures applied, and the results obtained, enable the QP of this report to consider this database to be valid for Mineral Resource estimation work.

QA/QC procedures, sampling methodology, and analytical methods applied by Bravo are within the industry's best practices standard. The QP responsible for this report, considering the data presented in Section 11, is of the opinion that the Luanga Project's Database is suited for a Mineral Resource Estimation work.

25.3 Geological Model

The procedure that was adopted to produce the 3D geological models (wireframes), consisting of generating triangulations between interpreted geological cross sections, was executed properly and aligns with the opinions of GE21 staff.

25.4 Grade Estimation

The heterogeneity of the geological model lead GE21 to select the Turning Bands Simulation method to estimate the grades for Luanga Project.

The variograms that were used in the estimation method are satisfactory and consistent with respect to the grade estimation that was calculated via Simulation (E-Type), making use of search anisotropy determined in the variography study. A valid conditional simulation in geostatistics ensured that simulated values honor both spatial continuity and the data distribution

To classify Mineral Resources, a study of spatial continuity for Pd Equivalent was conducted using variography followed by ordinary kriging interpolation. This study established a continuity zone suitable for considering:

- The Measured Mineral Resource was classified according to a reference grid of approximately 45mx45m, with a minimum number of 3 holes in section along the strike and dip directions, surrounded by the pit shell.
- The Indicated Mineral Resource classification had as a reference a drilling grid of approximately 75m x 75m, extending both along the strike and dip directions, and requiring a minimum of two drill holes.
- Manual post-processing was undertaken to construct wireframes representing the volumes categorized as Measured and Indicated, while considering the blocks within the resource pit shell.
- The Inferred Mineral Resource classification is all remaining estimated blocks within the resource pit shell.

GE21 considers the Mineral Resource classification model, and the analysis of criteria for the classification of those Mineral Resources, to be satisfactory although some items could be improved.

25.5 Reasonable Prospects for Eventual Economic Extraction (RPEEE)

GE21 has not identified any mining, metallurgical, infrastructure, permitting, legal, political, environmental, technical, or other relevant factors that could materially affect the potential development of the Mineral Resources, other than those typical for a mineral deposit at this stage of development and as identified in this report.

GE21 performed a pit optimization study to classify the project's Mineral Resources to ensure the requirement for RPEEE was met.

25.6 Mineral Resources

Luanga Project's maiden, pit-constrained MRE has an effective date of February 18, 2025. In summary, it comprises

- 36 Mt at 2.00 g/t Pd Eq for a total of 2.3 Moz Pd Eq in the Measured category,
- 122 Mt at 2.06 g/t Pd Eq for 8.0 Moz Pd Eq in the Indicated category,
    - 158 Mt at 2.04 g/t Pd Eq for a total of 10.4 Moz Pd Eq in the Measured + Indicated categories, and
- 78 Mt at 2.01 g/t Pd Eq for a total of 5.0 Moz Pd Eq in the Inferred category.

Mineral Resource estimate includes Inferred Mineral Resources, which have had insufficient work to classify them as Indicated Mineral Resources. It is uncertain but reasonably expected that inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration. Further, Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

25.7 Mine Plan

The mine plan was developed based on the Mineral Resource estimate and an open pit optimization process. The final pit design resulted in 113 Mt at 2.68 g/t Pd_Eq of Measured + Indicated Resources, and 52 Mt at 2.59 g/t Pd_Eq of Inferred Resources, using a 0.87 g/t Pd_Eq cut-off, 95% mining recovery and 5% mining dilution factors. Waste within the pit design sum 1,124 Mt, and Low Grade material of 30 Mt, resulting in 7.0 strip ratio.

The mining schedule was developed with a focus on operational features of the mining operations and optimizing concentrate production. The production rate was defined as 10.0 Mt/year of run-of-mine delivered to the processing plant, including a ramp-up period of 1 year at 50% the full production rate. The mine life is estimated at approximately 17 years.

A minimum of three active mining faces was considered to optimize the number of trucks accessing the same areas of the pit and corresponding ramps and roads, thus avoiding a decrease in productivity.

Vegetation and topsoil will be removed and stockpiled for use in future site reclamation. The ROM and waste will be mined with 10 m high benches, focusing on reducing mining dilution and increasing ROM recovery.

Drilling and blasting operations are expected for production purposes. A staggered blast pattern forming equilateral triangles was selected for the blast design, as it provides optimal distribution of explosive energy and promotes efficient rock fragmentation.

The ROM and waste will be loaded into rigid frame off-road trucks and hauled to the processing plant. The waste will be disposed in a Waste Rock Storage Facility, and filtered tailings will be disposed within a Dry Stacking Facility.

25.8 Mineral Processing

The recovery method proposed for the Project is based on the results of initial metallurgical testing carried out since and benchmarking against similar operations.

For the purposes of this Report, the selected metallurgical route is based on the most recent locked cycle tests (LCT) campaign. Although previous processing routes and historical tests results have been referenced, the selected route was developed specifically for the concentration of hard rock, given the limited amount of oxidized material. Data from the previous testworks should be taken into account during the development of the feasibility study.

The processing plant is planned to operate initially at 5 Mtpy, ramping up to 10 Mtpy by the second year of operation. The proposed plant includes a three-stage grinding circuit followed by a flotation process composed of rougher, scavenger, cleaner, and recleaner stages. Concentrate is expected to be produced at a rate of 184,000 dry tonnes per year.

Tailings will be filtered and dry stacked. The dry stacking facility and water management system are conceptually defined, with design parameters aligned with industry best practices for environmental performance and operational reliability. The water balance is based on a closed-loop system that maximizes reuse and minimizes raw water intake, supported by a reservoir within the project boundary and a contingency intake from the nearby Sereno River.

Power supply will be secured through a dedicated 230 kV transmission line from the Carajás Substation, with a planned capacity of 80 MW. While the proposed recovery method is robust and consistent with the characteristics of the mineralization, further metallurgical testwork is planned to support detailed engineering and equipment selection, particularly in grinding, thickening, and filtration stages.

25.9 Infrastructure

The main infrastructure for the Project includes the processing plant, Waste Rock Storage Facility, and Dry Stacking Facility, tailings dam (for the first 2 years of operation), and water dam, covering an area of approximately 1,600 hectares.

Supporting infrastructure comprises site access roads, settling ponds, maintenance facilities (truck shop, plant workshop and warehouse), process water treatment facility, administrative buildings (admin & finance, management, engineering and geology offices, support services, parking lot, restrooms, security, medical post, HSEC (Health, Safety, Environment & Communities), access gate, laboratory, explosives magazine, main electrical substation, 230 kV transmission line, water catchment system, communication system, and control room.

25.10 Environmental, Permitting, and Social Considerations

The Project is advancing through the environmental licensing process in accordance with Brazilian legislation. Following the issuance of the Terms of Reference by SEMAS-PA, the EIA/RIMA studies were completed between 2022 and 2024 and formally submitted in June 2024 to support the application for the Preliminary License (LP) which was granted on to Bravo by the Pará State Environmental Agency in February 2025. These studies covered both climatic seasons and addressed key aspects such as fauna, flora, hydrology, hydrogeology, and socioeconomic conditions. Additional permitting requirements, such as the Water Use Grant and Vegetation Suppression Authorization, will be addressed in subsequent phases.

The area of influence is predominantly anthropized, with pasture and rural uses, and does not overlap with protected areas or traditional communities. Socioeconomic conditions in Curionópolis reflect a dependence on mining and cattle ranching, with limited health infrastructure and partial basic sanitation coverage. The Project is expected to generate positive socioeconomic impacts, including job creation, local economic growth, and tax revenue, particularly benefiting Curionópolis and neighboring municipalities.

The Project also presents environmental challenges, notably the suppression of native vegetation and changes to the local hydrological regime due to water reservoir construction. Mitigation strategies include a comprehensive set of environmental and social programs aligned with ESG principles, as well as a mine closure plan following national and international standards. The plan provides for environmental rehabilitation, structural stability, and long-term monitoring.

25.11 Capital and Operating Costs

The total Base Case Project CAPEX over the 17-year operational life is estimated at US$ 592.9 million, comprising US$ 495.8 million in Initial CAPEX and US$ 97.1 million in Sustaining CAPEX. These estimates include the following components:

- Site Preparation
- Processing Plant
- Infrastructure:
    - Transmission Line and Substation
    - Maintenance Facilities
    - Administrative Buildings
    - Internal Roads
    - External Roads
    - WRSF and DSF
- Mine Closure
- Indirect costs:
    - Engineering
    - Management
    - Spare Parts
    - Commissioning
    - Supervision
    - Initial Plant Charge

The total Base Case Project OPEX over the 17-year operational life is estimated at US$ 6,755 million. The average OPEX was estimated at US$ 40.86/t processed. These estimates include the following components:

- Mining
- Processing
- General and administrative
- Logistics

25.12 Economic Analysis

The preliminary economic analysis demonstrates that the Project has sufficient economic potential under the Base Case scenario to warrant advancing the Project towards a PFS or FS. At average prices of US$1,271/oz Pd, US$1,500/oz Pt, US$6,000/oz Rh, US$3,251/oz Au, and US$8.00/lb Ni, the after-tax NPV (8%) is estimated at US$ 1,249 million, an IRR of 49% and a payback period of 2.4 years. These results indicate a financially attractive opportunity, supporting continued advancement of the Project.

25.13 Alternate Case (Vertical Integration) – On-Site Smelting Scenario

A second scenario was developed, considering the on-site smelting of the flotation concentrate, with capacity to treat up to 200,000 tonnes wet basis of PGM-rich, nickel sulphide concentrate, up to 800,000 ounces of PGM, 16,000 of nickel, and 120,000 tonnes of sulphuric acid annually. The metallurgical process comprises a single line fluidized bed roaster and direct-current electric arc furnace producing a PGM-enriched, ferronickel alloy for further precious and base metal refining. Extracted sulphur will be processed to sulphuric acid in a sulphuric acid plant.

The total CAPEX for the on-site facilies is estimated at US$ 181.9 million, over and above the CAPEX estimate for the Base Case, including the following components:

- Auxiliary Equipment
- Shipping & Logistics
- Engineering, Construction & Installation

The annual incremental OPEX for the smelter facilies is estimated at US$ 54.9 million, over and above the OPEX for the Base Case, resulting in an average OPEX of US$ 274.50/t concentrate. These estimates do not include feed costs and byproduct credits, and include the following components:

- Energy and water
- Reagents and consumables
- Labor
- Maintenance and overheads

The Alternate Case preliminary economic analysis results in after-tax NPV (8%) estimated at US$ 1,861 million, an IRR of 49% and a payback period of 2.4 years.

25.14 Risks and Opportunities Analysis

The PEA is an initial study aimed at estimating the potential of a mineral project and should therefore be interpreted with appropriate caution, given its preliminary nature in the decision-making process. For the Luanga Project, specific risks and opportunities have been identified and are presented in Table 25-1 and Table 25-2.

Table 25-1 - Risk Analysis

Item	Risk
Geology and Resource Modelling	Mineral resource estimates inherently involve risks, and they are crucial for project success. Key risks include potential changes to the geological model, affecting mineralization continuity, and the risk of increased dilution during mining. The PEA incorporates Inferred Mineral Resources into the assumed production schedule. Additional work would be required to upgrade the confidence level of this material before it could be included in a Pre Feasibility Study (PFS). While it is reasonably expected that such upgrading would be successful, it is not certain that all such Inferred would be upgraded.
Mining	Inadequate drainage can cause water accumulation, limiting access to mining fronts and affecting pit stability. It may result in extra infrastructure costs and operational impacts. Geotechnical parameters may not be accurate to the operational features for all mining zones. More tests are recommended to confirm the considered assumptions.
Metallurgical Recoveries	The metallurgical performance was estimated based on locked cycle tests. Confirmation of the selected flowsheet in a continuous pilot plant is required.
CAPEX and OPEX	Capital (CAPEX) and operating (OPEX) cost estimates were developed at a Class 5 level of definition, in accordance with the guidelines established by AACE. These estimates are characterized by an expected accuracy range of approximately -30% to +50%, based on P10 to P90 levels, and were derived using benchmark data relevant to the scope and level of the studies conducted. This level of accuracy is considered appropriate for the current stage of the project.
Market & Economic Analysis	Metal prices fluctuation represents a risk to the economics of the project. The market for PGM-Ni concentrates is relatively limited in scale and participants. Bravo has not entered into any offtake agreements at this time, and market interest in and capacity to handle Luanga's contemplated concentrate production may not be available on the same terms and conditions as assumed in the PEA. The PEA assumes that the Luanga Project would be eligible for SUDAM tax benefits (see below), but Bravo has not yet made application for eligibility.
Environmental, Social and Permitting	The project is currently in the process of securing permits. While permitting in Brazil is complex and often presents regulatory challenges, the project site benefits from a favorable context due to the current level of anthropization in the area, which may facilitate the environmental licensing process.

Source: GE21, 2025.

Table 25-2 - Opportunity Analysis

Item	Opportunity
Geology and Resource Modelling	Increase resources with further drilling and exploration program, including potential targets.
Mining	Further Geotechnical studies may indicate steeper angles for the mining zones, with the opportunity of reducing strip ratio. A strategic mine planning may improve the project results.
Metallurgical Recoveries	Enhance metallurgical recovery through the implementation of more complex processing methods and technologies. There is potential for further improvements to metallurgical recoveries and optimization of processing reagent consumption during more detailed future study phases, which could involve more exhaustive and larger scale pilot plant testwork. Test work is continuing.
Processing of Stockpile	Approximately 30 Mt of sulphide mineralised material is contemplated to be stockpiled that is below the total cut-off grade but above the marginal treatment grade. This material could be supplied to the processing plant at the end of the production schedule which would extend the LOM to approximately 20 years.
Oxide Mineral Resources	Excluded from the PEA is processing of Oxide mineralized material, where approximately 13 Mt has been defined through a combination of surface trenches and drilling. Initial metallurgical testing suggests potential for economic recoveries, and more test work is planned. If incorporated into future studies, additional plant components would be required that are not considered in this PEA.
Toll Treatment Agreement	The Base Case Concentrate Sales scenario currently contemplates a traditional sales contract. However, certain third-party producers in Southern Africa also operate under toll treatment agreements. This may provide the Company with the opportunity to see further benefit in Net Smelter Return ("NSR") achieved. In a toll treatment scenario, Bravo would be responsible for the marketing and logistics of final metal products.
CAPEX and OPEX	Optimization of CAPEX and OPEX after Project's maturity and consequently reduce the costs contingency.
Verticalization Case	The on-site smelting facilities bring the opportunity to add value to the project.

Item	Opportunity
Processing and Export Free Trade Zone (ZPE)	The Economic Development Company of Pará (CODEC), in partnership with Bravo, has applied to the Brazilian Federal Government (MDIC) to include Bravo into the Processing and Export Free Trade Zone (ZPE) at the Port of Vila do Conde. If approved, Bravo would be permitted to construct its downstream processing facility within the ZPE, benefitting from the associated import and export, and other fiscal and taxation exemptions. In due course, Bravo may conduct a trade-off study to investigate the optimal location (on-mine vs ZPE) for such a downstream processing facility.
BNDES and FINEP Funding	Bravo's Luanga Project has been selected by the BNDES and the Federal Agency for Funding Authority for Studies and Projects in Brazil ('FINEP'), as one of the successful companies to receive significant potential funding to progress the referred Project and downstream facility. This initial round of funding from BNDES and FINEP aims to deploy BRL $5 billion (US $903 million) across leading strategic mineral projects in Brazil. Having now been formally selected, Bravo has initiated discussions with BNDES and FINEP to explore the potential funding structure, which includes economic grants, debt facilities and equity participation. The program aims to develop sustainable supply chains for critical minerals, including PGMs, within Brazil. The funding encompasses various forms of financial support to invest in a range of projects, including commercial-scale plants, pilot facilities, demonstration projects and necessary research studies, depending on the stage of the projects and technologies involved.
IOCG-Style Discovery	The PEA does not factor in the recent copper-gold discoveries on the Project, outside of the Luanga Deposit, which may provide other opportunities for production.

Source: GE21, 2025.

26 RECOMMENDATIONS

The PEA demonstrates that further development of the Luanga Project through additional engineering and de-risking is warranted. The following subitems describe the proposed recommendations, and the correspondent budgets, to advance the project through the Pre-Feasibility Study stage.

26.1 Mineral Resources

26.1.1 Luanga PGM + Au + Ni Deposit

GE21 recommends completion of the outstanding metallurgical work that was not completed in 2024, including final assays, review, interpretation and reporting. This may unlock modest gains in metallurgical recoveries, which could be applied to subsequent updates to the current 2025 MRE.

If a further MRE upgrade is deemed desirable, GE21 recommends future work targets:

- Drilling at depth in areas where the constraining pit that encapsulates the reported Luanga MRE is limited due to the absence of drill data at depth.
- Further refinement of the geological and mineralogical models, which may result in unlocking a modest gain in MRE metal grades. An update of the mineralization geological model adopting an approach with implicit modelling methods and reducing domain internal dilution.
- Completion of the outstanding metallurgical testwork noted above.

Resource estimation by the conditional simulation method defining the SMU to define the recoverable resource.

Collectively, the above work could deliver a further upgrade to the Luanga MRE, both in overall tonnage and, potentially, the average grade of the MRE.

Further, additional metallurgical programs are recommended, building off the 2024 test work results, to further optimize metallurgical recoveries and concentrate grades in various types of mineralization with a view to increasing potential payabilities at third part facilities.

Following on from the successful and significant upgrade to the Luanga MRE in the 2025 MRE (as compared to the maiden 2023 MRE), the next significant stage for the Company would be the completion of a Preliminary Economic Assessment (PEA) and/or a Pre-Feasibility Study (PFS) to define mineral reserves (given that most of the MRE is now classified in the Measured and Indicated categories).

GE21 Consultoria Mineral

26.1.2 Luanga Area Exploration Potential

The last results of Bravo's exploration, based on the previously completed HeliTEM Electromagnetic (EM) survey covering the entire Luanga property, has generated intersections of massive sulphide mineralization, including:

- Ni/Cu/PGM and Ni/Cu massive sulphides, probably related to the emplacement of the Luanga PGM+Au+Ni deposit.
- More interestingly, compelling evidence of Carajás style Iron Oxide Copper Gold mineralization, particularly in work reported from the T5 target.

GE21 recommends that further exploration work should focus on continuing the exploration for mineralization outside of the Luanga PGM+Au+Ni deposit, and particularly the potential for IOCG deposits given the results from lastest exploration works.

26.1.3 Luanga Carbon Capture Potential

The Luanga deposit is hosted almost entirely in ultramafic rocks which early works indicate the potential for permanent carbon sequestration in tailings and/or waste rock. This is an opportunity that can be investigated further, subject to test results and economic assessment, could be incorporated into future study phases with the potential to create a carbon negative operation in combination with other mitigation efforts such as use of hydroelectric power, mine electrification and reforestation.

26.2 Mining Methods

- Carry out geotechnical investigations to support pit and WRSF slope design and stability. It is recommended that further diamond drilling be performed to inform geotechnical studies.
- Conduct further fragmentation tests to better characterize rock hardness.
- Refine the mine plan and scheduling based on further geometallurgical characterization studies, and strategic mine planning.
- Evaluate alternative pit configurations and waste storage strategies to optimize strip ratio and costs.
- Conduct detailed hydrogeological investigations to characterize groundwater flow, aquifer properties, and seasonal variations.

26.3 Mineral Processing and Metallurgy

- Undertake pilot-scale metallurgical testing to validate the process flowsheet and recovery assumptions.
- Conduct variability testing on samples from different mineralized zones to assess metallurgical consistency.
- Investigate alternative processing routes or technologies to maximize efficiency and product quality.

26.4 Infrastructure and Engineering

- Complete engineering studies for plant layout, roads, and site infrastructure.
- Conduct topographic and hydrological surveys to support infrastructure siting and design.
- Carry out studies for the investigation of co-disposal of waste rock and filtered tailings.

26.5 Environmental, Social, and Governance

- Continue baseline environmental studies required for permitting.
- Develop an environmental management plan, including mitigation and compensation measures.
- Establish a stakeholder engagement strategy, focusing on local communities and authorities.
- Identify key environmental risks and opportunities for early mitigation or design adaptation.

26.6 Capital and Operating Costs

- Refine CAPEX and OPEX estimates with supplier quotations and updated engineering inputs.
- Develop project implementation schedule and investment timeline.
- Identify cost optimization opportunities and assess potential for strategic partnerships.

26.7 Permitting and Legal

- Advance the environmental licensing process.
- Secure formal agreements with landowners and local stakeholders.

26.8 Recommended Work Program

The recommended work program comprises:

PHASE 5A – Metallurgical testwork at Luanga

Completion of any outstanding metallurgical testwork and optimization:	
Estimate	US$0.50M
Continuation of carbon sequestration study:	
Estimate	US$0.05M
Continued Metallurgical testwork and optimization	
Estimate	US$0.20M
Sub-total – Phase 5A	***US$0.75M***

PHASE 5B – Prefeasibility Study ("PFS") following favorable results from a PEA

Completion of a PFS:	
Estimate	US$1.0M

Sub-total – Phase 5B	***US$1.0M***

PHASE 5C – Deep Drilling below the Luanga PGM+Au+Ni deposit
Deep drilling at the Luanga PGM+Au+Ni deposit.
8 holes @ ~500m = 4,000m @ US$450/m

Estimate	US$1.8M
Sub-total – Phase 5C	***US$1.8M***

Phase 5D – Regional Exploration
Exploration of new (IOCG and/or massive sulphide Ni/Cu/ PGM targets):
Geological, geophysical and drilling programs to evaluate the potential for the discovery of additional zones of mineralization:

Geophysics	US$0.1M
Drilling: 70 holes x 200m for 14,000m @ US$400m (all inclusive)	US$5.6M
Sub-total – Phase 5D	***US$5.7M***
TOTAL – PHASE 5	***US$9.25M***

PHASE 6 – Feasibility Study ("FS") following favorable results from a PFS
Completion of a FS, including any required infill drilling, geotechnical drilling:

Estimate	US$5.0M
TOTAL – PHASE 6	***US$5.0M***

Source: GE21, 2025.

These work programs and cost estimates are preliminary in nature and will be refined, adjusted and modified as additional information is compiled, contracts for the various aspects of the work program entered, and results from new work are received. This could result in some movement in funds between different categories.

27 REFERENCES

ARAÚJO, O. J. B.; MAIA, R. G. N. Projeto especial mapas de recursos minerais, de solos e de vegetação para a área do Programa Grande Carajás; Subprojeto Recursos Minerais; Folha SB.22-Z-A Serra dos Carajás - Estado do Pará: DNPM/CPRM, 1991.

ARAÚJO, O. J. B.; MAIA, R. G. N.; JOÃO, X.S.J.; COSTA, J. B. S. A megaestruturação arqueana da Folha Serra dos Carajás. Congresso Latino-Americano de Geologia, Belém Brazil, Anais, p. 324-338, 1988.

BARNES, S. J. et al. Sulfide-silicate textures in magmatic Ni-Cu-PGE sulfide ore deposits: Massive, semi-massive and sulfide-matrix breccia ores. Ore Geology Reviews, v. 101, p. 629-651, 2018.

BARNES, S. J. et al. The mineral system approach applied to magmatic Ni–Cu–PGE sulfide deposits. Ore geology reviews, v. 76, p. 296-316, 2016.

BARNES, S. J.; DAVID, A. H. Margaux Le Vaillant; Depósitos de minério de sulfeto magnético. Elementos 2017; 13 (2): 89–95.

BRANDT MEIO AMBIENTE LTDA. Estudo de Impacto Ambiental (EIA). Curionópolis (PA): June 2024.

BRANDT MEIO AMBIENTE LTDA. Relatório de Impacto Ambiental (RIMA). Curionópolis (PA): April 2024.

BRAVO MINERAÇÃO LTDA. Plano De Aproveitamento Econômico (PAE). Curionópolis (PA): 11 December 2023.

CABRI, L. J. A Mineralogical Study of Samples from the Luanga Layered PGE Deposit, Carajás, Brazil. Parte 5: Samples from the Luanga Orebody. Confidential Report 2003-07. 74p, 2003.

CONSENSUS ECONOMICS INC. Energy, Metals and Agriculture Consensus Forecasts. London (UK): June 2025.

DALL'AGNOL, R.; OLIVEIRA, M. A.; ALMEIDA, J. A. C.; ALTHOFF, F. J.; LEITE, A. A. S.; OLIVEIRA, D. C.; BARROS, C. E. M. Archean and Paleoproterozoic granitoids of the Carajás metallogenetic province, eastern Amazonian craton. Symposium on Magmatism, Crustal Evolution, and Metallogenesis of the Amazonian Craton, Abstracts Volume and Field Trips Guide, 150 pp, 2006.

DARDENNE, M. A; FERREIRA FILHO, C. F.; MEIRELLES, M. R. The role of shoshonitic and calc-alkaline suites in the tectonic evolution of the Carajás District, Brazil. Journal of South American Earth Sciences 1, 363–372, 1988.

DIELLA, V.; FERRARIO, A.; GIRARDI, V. A. V. PGE and PGM in the Luanga Mafic-Ultramafic Intrusion in Serra dos Carajás (Pará State, Brazil). 9p, 1995.

DOCEGEO - Rio Doce Geologia e Mineração. Revisão Litoestratigráfica da Província Mineral de Carajás, in, SBG-NNO, ed., 35º Congresso Brasileiro Geologia, Belém, Anais: p. 11-59, 1988.

FEIO, G. R. L.; DALL'AGNOL, R.; DANTAS, E. L.; MACAMBIRA, M. J. B.; SANTOS, J. O. S.; ALTHOFF, F. J.; SOARES, J. E. B. Archean granitoid magmatism in the Canaã dos Carajás area: Implications for crustal evolution of the Carajás province, Amazonian craton, Brazil. Precambrian Research 227, 157-185, 2013.

FERREIRA FILHO, C. F.; CANÇADO, F.; CORREA, C.; MACAMBIRA, E. M. B.; SIEPIERSKI, L.; BROD, T. C. J. Mineralizações estratiformes de EGP-Ni associadas a complexos 91 acamadados em Carajás: os exemplos de Luanga e Serra da Onça. In: Publitec Gráfica & Editora, Contribuições à Geologia da Amazônia, vol. 5, pp. 01-14. 2007.

FERREIRA FILHO, C.F. Petrography. Report on Luanga. 2022 to 2024.

GE21 Consultoria Mineral Ltda. Independent Technical Report for the Luanga PGM+Au+Ni Project, Pará State, Brazil. Report on Luanga, 2023.

GE21 Consultoria Mineral Ltda. NI 43-101 Independent Technical Report Luanga PGM + Au + Ni Project Pará State, Brazil. Belo Horizonte (MG): 2 April 2025.

GEOHYDROTECH ENGENHARIA. Estudo de Alternativas - Relatório Trade Off das Alternativas. [s.l.]: 18 September 2023.

GIBBS, A. K.; WIRTH, K. R.; HIRATA, W. K.; OLSZEWSKI Jr, W. J. Age and composition of the Grão Pará Group volcanics, Serra dos Carajás. Revista Brasileira de Geociências 16, 201-211, 1986.

HDK Engenharia. Caracterização de Amostras e Dimensionamento Preliminar de Circuito de Moagem – Depósito Luanga. 28p, 2003.

HOATSON, D. M.; JAIRETH, S.; JAQUES, A. L. Nickel sulphide deposits in Australia: Characteristics, resources, and potential: Ore Geology Reviews, v. 29, p. 177–241, 2006.

HOLDSWORTH, R. E.; PINHEIRO, R. V. L. The anatomy of shallow-crustal transpressional structures: insights from the Archean Carajás fault zone, Amazon, Brazil. Journal of Structural Geology 61, 1105-1123, 2000.

HUHN, S. R. B., SANTOS, A. B. S.; AMARAL, A. F.; LEDSHAM, E. J.; GOUVEIA, J. L.; MARTINS, L. P. B.; MONTALVÃO, R. M. G.; COSTA, V. C. O terreno granito-greenstone da região de Rio Maria-Sul do Pará. 35º Congresso Brasileiro de Geologia, Belém, Brasil, Anais, Sociedade Brasileira de Geologia, pp. 1438-1452, 1986.

LIGHTFOOT, P. C.; LAMSWOOD, D. E. Structural controls on the primary distribution of mafic–ultramafic intrusions containing Ni–Cu–Co–(PGE) sulfide mineralization in the roots of large igneous provinces. Ore Geology Reviews - Elsevier, v. 64, p. 354-386, 2015.

MACHADO, N.; LINDENMAYER, Z. G.; KROGH, T. E.; LINDENMAYER, D. U-Pb geochronology of Archean magmatism and basement reactivation in the Carajás area, Amazon shield, Brazil. Precambrian Research 49, 329-354, 1991.

MANSUR, E. T. Caracterização e Metalogênese do Depósito de Elementos do Grupo da Platina do Complexo Luanga, Província Mineral do Carajás. 158p, 2017.

MANSUR, E. T.; FERREIRA FILHO, C. F. Magmatic structure and geochemistry of the Luanga Mafic–Ultramafic Complex: Further constraints for the PGE-mineralized magmatism in Carajás, Brazil. Lithos, v. 266, p. 28-43, 2016.

MANSUR, E.T.; FERREIRA FILHO, C. F.; OLIVEIRA, D. P. L. The Luanga deposit, Carajás Mineral Province, Brazil: Different styles of PGE mineralization hosted in a medium-size layered intrusion. Ore Geology Reviews. 18p, 2020.

MEIRELES, E. M.; SILVA, A. R. B. Depósito de ouro de Serra Pelada, Marabá, Pará, in Schobbenhaus, C., and Coelho, C.E.S., eds., Principais depósitos minerais do Brasil – volume III: Brasília, Departamento Nacional da Produção Mineral, Companhia Vale do Rio Doce, p. 547–557, 1988.

MÜLLER, P. H. Preliminary Smelter Study – Luanga Project. [s.l.]: 01 April 2025

MÜLLER, P. H. Technical Report on Luanga PGE+Au Project, State of Para, Brazil, 2020.

MUNGALL, J. E.; NALDRETT, A. J. Ore deposits of the platinum-group elements. Elements, v. 4, n. 4, p. 253-258, 2008.

NALDRETT, A. J. Depósitos de elementos do grupo da platina (PGE). Depósitos de sulfeto magmático: Geologia, Geoquímica e exploração, p. 481-612, 2004.

NALDRETT, A. J. Secular variation of magmatic sulphide deposits and their source magmas: Economic Geology, v. 105, p. 669–688, 2010a.

NAVARRO, L. Avaliação Geotécnica e Modelo 3D. Curionópolis (PA): Luis Navarro Geologia e Serviços Eireli, May 2023.

PIDGEON, R. T.; MACAMBIRA, M. J. B.; LAFON, J. M. Th-U-Pb isotopic systems and internal structures of complex zircons from enderbite from the Pium Complex, Carajás Province, Brazil: evidence for the ages of granulite facies metamorphism and the protolith of the enderbite. Chemical Geology 166, 159-171, 2000.

PRINSLOO, C. F. Investigation into the Recovery of PGM Values from a Brazilian Ore Type for Companhia Vale do Rio Doce (CVRD). Mintek. 98p, 2002.

REPLACE CONSULTORIA, TSE ENERGY CONSULTING. Alternativas de conexão ao Sistema Interligado Nacional. Curionópolis (PA): September 2023.

RIBEIRO, V. E. Gênese e Controle das Mineralizações de Pt-Pd nos Alvos Luanga e Fênix – Complexo Máfico-Ultramáfico Luanga, Província Mineral de Carajás, Pará. 210p, 2020.

ROLIM, V. K. Geotectonic environment and Structural control of Luanga Complex. Report on Luanga, 2023.

ROSA, W. D. Complexos acamadados da Serra da Onça e Serra do Puma: Geologia e petrologia de duas intrusões Máfico-Ultramáficas com sequência de 94 cristalização distinta na Província Arqueana de Carajás, Brasil. Unpublished M.Sc. Thesis, Universidade de Brasília, Brazil, 65 p, 2014.

SANTOS, J. O. S. et al. Age and autochthonous evolution of the Sunsás Orogen in West Amazon Craton based on mapping and U–Pb geochronology. Precambrian Research, v. 165, n. 3-4, p. 120-152, 2008.

SGS Lakefield Research Limited. Applicability of the PLATSOL® Process to Five Samples from the Luanga Deposit. 64p, 2005.

SGS Lakefield Research Limited. Locked Cycle Flotation Products, Luanga Deposit, Brazil. 131p, 2004.

SIEPIERSKI, L.; FERREIRA FILHO, C.F. Spinifex-textured komatiites in the south border of the Carajas ridge, Selva Greenstone belt, Carajás Province, Brazil. Journal of South American Earth Sciences 66, 41-55, 2016.

SOUZA, Z. S.; DALL'AGNOL, R. Caracterização geoquímica e tectônica de rochas meta vulcânicas de "greenstone belts" arqueanos da região de Rio Maria, SE do Pará. Boletim. IG-USP, special publication, online, vol.18, pp. 97-101. ISSN 0102-6275, 1996.

SOUZA, Z. S.; POTREL, A.; LAFON, J. M.; ALTHOFF, F.J.; PIMENTEL, M. M., DALL'AGNOL, R.; OLIVEIRA, C. G. Nd, Pb and Sr isotopes in the Identidade Belt, an Archean greenstone belt of Rio Maria region (Carajás Province, Brazil): implications for the geodynamic evolution of the Amazonian Craton. Precambrian Research 109, 293-315, 2001.

TALLARICO, J. B. C. Caracterização de Amostras de Cromitito. Projeto Platina – Luanga. 14p, 2002.

TALLARICO, J. B. C. Projeto Platina, Carajás – Caracterização Mineralógica de Amostra de Minério tipo “Sulfeto Oxidado” (SO-1). 32p, 2003.

TALLARICO, J. B. C. Projeto Platina, Carajás – Caracterização Mineralógica de Quatro Amostras Iniciais (S1 a S4) de Minério Sulfetado de Luanga. 46p, 2003.

TALLARICO, J. B. C. Projeto Platina, Carajás: Petrografia de Amostras Testadas para WI do Minério. 13p, 2003.

TALLARICO, J. B. C.; SÁ, L. R. B. M. Projeto Platina, Carajás – Caracterização Mineralógica de Amostra Inicial (SB) de Minério Sulfetado. 43p, 2004.

TAVARES, F. M. Evolução geotectônica do nordeste da Província Carajás. Unpublish, Ph.D. Thesis, Universidade Federal do Rio de Janeiro, p. 115 p, 2015.

TEIXEIRA, A. S. Geologia, Petrologia e Geocronologia do Complexo Acamadado Lago Grande: Evidência para uma Suíte Magmática Mineralizada a EGP na Província Mineral de Carajás, Brasil. 216p, 2013.

TEIXEIRA, A. S.; FERREIRA FILHO, C. F.; GIUSTINA, M. E. S. D.; ARAUJO, S. M.; SILVA, H. H. A. B. Geology, petrology and geochronology of the Lago Grande layered complex: Evidence for a PGE-mineralized magmatic suite in the Carajás Mineral Province, Brazil. J. South Am. Earth Sci. 64, 116-138, 2015.

TEIXEIRA, J. B. G.; EGGLER, D. H. Petrology, Geochemistry, and Tectonic Setting of Archaean Basaltic and Dioritic Rocks from the N4 Iron Deposit, Serra dos Carajás, Pará, Brazil. Acta Geologica Leopoldensia 17, 71-114, 1994.

TEIXEIRA, N. A. et al. Carajás Mineral Province-Example of metallogeny of a rift above a cratonic lithospheric keel. Journal of South American Earth Sciences, v. 108, p. 103091, 2021.

TRUNFULL, E. F. et al. Critical assessment of geochronological data from the Carajás Mineral Province, Brazil: implications for metallogeny and tectonic evolution. Ore Geology Reviews, v. 121, p. 103556, 2020.

VALE. (Unknown date). Report Luanga. 159p.

VASQUEZ, M. L.; CARVALHO, J. M. A.; SOUSA, C. S.; RICCI, P. S. F.; MACAMBIRA, E. M. B.; COSTA, L. T. R. Geological map of the Pará state in GIS, Geological Survey of Brazil-CPRM, 2008.

VILLAS BOAS, J. M.; ARAÚJO, C. C. de. Projeto Especial. Mapas de Recursos Minerais, de Solos e de Vegetação para Área do Programa Grande Carajás. Subprojeto de Recursos Minerais. Açailândia. CPRM, 1999.

VILLAS, R. N., SANTOS, M. D. Gold deposits of the Carajás Mineral Province: deposit types and metallogenesis. Mineralium Deposita 36, 300–331, 2001.

XCALIBUR MULTIPHYSICS. Time Domain Electromagnetic and Magnetic Survey. Report on Luanga, 2023.

ZACHARIAS, O. Estudos de Flotação com Amostras de PGM Luanga – Relatório de Progresso 1. 24p, 2007.

ZACHARIAS, O.; TALLARICO, J. B. C. Caracterização de Amostras de Cromititos de Luanga – Estudo em Escala de Bancada. Relatório Final. 33p, 2002.

ZIENTEK, M. L. Magmatic ore deposits in layered intrusions-Descriptive model for reef-type PGE and contact-type Cu-Ni-PGE deposits. US Geological Survey, 2012.

APPENDIX A – CERTIFICATES OF QUALIFIED PERSON

GE21 Consultoria Mineral

QP CERTIFICATE OF PORFIRIO CABALEIRO RODRIGUEZ

I, **Porfirio Cabaleiro Rodriguez**, FAIG (#3708), as an author of the independent technical report titled “NI 43-101 Preliminary Economic Assessment (PEA) Independent Technical Report for the Luanga PGM + Au + Ni Project Pará, Brazil” dated August 20th, 2025, with an effective date of July 7, 2025 (Technical Report), prepared for Bravo Mining Corp. (Issuer), do hereby certify that:

1. I am a Mining Engineer and Director for GE21 Consultoria Mineral Ltda., which is located on Avenida Afonso Pena, 3130, 9th floor, Savassi, Belo Horizonte, MG, Brazil - CEP 30130-910.
2. I am a graduate of the Federal University of Minas Gerais, located in Belo Horizonte, Brazil, and hold a Bachelor of Science Degree in Mining Engineering (1978). I have practised my profession continuously since 1979.
3. I am a professional enrolled with the Australian Institute of Geoscientists (AIG) Fellow (FAIG #3708).
4. I am a professional Mining Engineer, with over 40 years of relevant experience in Mineral Resource and Mineral Reserves estimation, including numerous mineral properties in Brazil, including PGM and other polymetallic properties.
5. I have read the definition of “qualified person” set out in National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* (NI 43-101) – and certify that, by reason of my education, affiliation with a professional association and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I personally inspected the property that is the subject of the Technical Report from July 4th to July 7th, 2023, 3rd October to 6th October 2023, and January 27th to January 31st, 2025.
7. I have no prior involvement with the property that is the subject of the Technical Report other than as an author of the technical report titled ‘Independent Technical Report on the Luanga PGM + Au + Ni Project, Pará State, Brazil’, with an effective date of February 18, 2025; and the previous technical report titled “Independent Technical Report on Resources estimate for Luanga PGM + Au + Ni Project Pará State, Brazil” dated December 1, 2023 with an effective date of October 22, 2023.
8. I am responsible for Sections 2, 3, 4, 5, 6, 13, and 14, with co-responsibility for Sections 1, 11, 12, 25, 26 and 27 of the Technical Report.
9. As for the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections in the Technical Report that I have authored and am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I have no personal knowledge of any material fact or material change which is not reflected in this Technical Report as of the date of this certificate.
11. I am independent of the Issuer, applying all tests in section 1.5 of NI 43-101.
12. I have read NI 43-101 and Form 43-101F1, and this Technical Report has been prepared in compliance with that instrument form documents.

Belo Horizonte, Brazil, August 20th, 2025.

<Signed and sealed in the original>

____

Porfirio Cabaleiro Rodriguez

GE21 Consultoria Mineral

QP CERTIFICATE OF BERNARDO HORTA DE CERQUEIRA VIANA

I, **Bernardo Horta de Cerqueira Viana**, FAIG (#3709), as an author of the independent technical report titled "NI 43-101 Preliminary Economic Assessment (PEA) Independent Technical Report for the Luanga PGM + Au + Ni Project Pará, Brazil" dated August 20th, 2025, with an effective date of July 7, 2025 (Technical Report), prepared for Bravo Mining Corp. (Issuer), do hereby certify that:

1. I am a Geologist and Director for GE21 Consultoria Mineral Ltda., which is located on Avenida Afonso Pena, 3130, 9th floor, Savassi, Belo Horizonte, MG, Brazil - CEP 30130-910.
2. I am a graduate of the Federal University of Minas Gerais, located in Belo Horizonte, Brazil, and hold a Bachelor of Science Degree in Geology (2002). I have practised my profession continuously since 2002.
3. I am a professional enrolled with the Australian Institute of Geoscientists (AIG) Fellow (FAIG #3709).
4. I am a professional Mining Engineer, with over 20 years of relevant experience in Mineral Resource and Mineral Reserves estimation, including QA/QC, geological exploration and economic geology, which includes numerous mineral properties in Brazil, including PGM and other polymetallic properties.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* (NI 43-101) – and certify that, by reason of my education, affiliation with a professional association and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am responsible for Sections 7, 8, 9, 10, 20 and 23, with co-with responsibility for Sections 1, 11, 12, 25, 26 and 27 of the Technical Report.
7. I personally inspected the property that is the subject of the Technical Report from 3rd October to 6th October 2023 and from January 27th to January 31st, 2025.
8. I have no prior involvement with the property that is the subject of the Technical Report other than as an author of the technical report titled 'Independent Technical Report on the Luanga PGM + Au + Ni Project, Pará State, Brazil', with an effective date of February 18, 2025; and the previous technical report titled "Independent Technical Report on Resources estimate for Luanga PGM + Au + Ni Project Pará State, Brazil" dated December 1, 2023 with an effective date of October 22, 2023.
9. As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections in the Technical Report I have authored and am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I have no personal knowledge of any material fact or material change which is not reflected in this Technical Report as of the date of this certificate.
11. I am independent of the Issuer, applying all tests in section 1.5 of NI 43-101.
12. I have read NI 43-101 and Form 43-101F1 – Technical Report, and the Technical Report has been prepared in compliance that instrument and form.

Belo Horizonte, Brazil, August 20th, 2025.

<Signed and sealed in the original>

____

Bernardo Horta de Cerqueira Viana

GE21 Consultoria Mineral

QP CERTIFICATE OF PAULO ROBERTO BERGMANN MOREIRA

I, Paulo Roberto Bergmann Moreira (“FAusIMM”, B.Sc.), as an author of the independent technical report titled “NI 43-101 Preliminary Economic Assessment (PEA) Independent Technical Report for the Luanga PGM + Au + Ni Project Pará, Brazil” dated August 20th, 2025, with an effective date of July 7, 2025 (Technical Report), prepared for Bravo Mining Corp. (Issuer), do hereby certify that:

1. I am a Mining Engineer for GE21 Consultoria Mineral Ltda., which is located on Avenida Afonso Pena, 3130 − 12º floor, Belo Horizonte, MG, Brazil, CEP 30.130-910.
2. I am a graduate of the Federal University of Minas Gerais, located in Belo Horizonte, Brazil, and hold a Bachelor of Science Degree in Mining Engineering (1983). I have practiced my profession continuously since 1983.
3. I am a professional enrolled with the Australasian Institute of Mining and Metallurgy (AusIMM) Fellow - (FAusIMM) #333121.
4. I am a professional Mining Engineer, with more than 42 years’ relevant experience in Mineral Processing and Mineral Reserves estimation, including numerous mineral properties in Brazil, as well as PGM and other polymetallic properties.
5. I have read the definition of “qualified person” set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that, by reason of my education, affiliation with a professional association as defined in NI 43-101, and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
6. I am responsible for Sections 17 and 24, with co-with responsibility for Sections 1, 21, 25, 26 and 27 of the Technical Report.
7. As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report that I have authored and am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
8. I have no personal knowledge, as of the date of this certificate, of any material fact or material change which is not reflected in this Technical Report.
9. I am independent of the Issuer, applying all the tests in section 1.5 of NI 43-101.
10. I have read NI 43-101 and Form 43-101F1 – Technical Report and, in my opinion, the Technical Report has been prepared in compliance with such instrument and form.

Belo Horizonte, Brazil, August 20th, 2025.

<signed & sealed in the original>

Paulo Roberto Bergmann Moreira

GE21 Consultoria Mineral

QP CERTIFICATE OF JULIANO FELIX DE LIMA

I, Juliano Felix de Lima, MAIG, as an author of the independent technical report titled "NI 43-101 Preliminary Economic Assessment (PEA) Independent Technical Report for the Luanga PGM + Au + Ni Project Pará, Brazil" dated August 20th, 2025, with an effective date of July 7, 2025 (Technical Report), prepared for Bravo Mining Corp. (Issuer), do hereby certify that:

1. I am employed as a Chief Technology Innovation Officer for GE21 Consultoria Mineral Ltda., which is located on Avenida Afonso Pena, 3130 − 12º floor, Belo Horizonte, MG, Brazil, CEP 30.130-910.
2. I am graduated from Universidade Federal de Ouro Preto with a bachelor's in science in Geological Engineering on 20th August 1999.
3. I am a professional enrolled with the Australian Institute of Geoscientists (AIG) Member - (MAIG) #9180.
4. I am a professional Geological Engineer, with more than 25 years' relevant experience in exploration, resource and reserves estimation and open pit and underground mining, including numerous mineral properties in Brazil.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association as defined in NI 43-101, and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am responsible for the preparation of Section 18, with co-with responsibility for Sections 1, 21, 25, 26 and 27 of the Technical Report.
7. As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report that I have authored and am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
8. I have no personal knowledge, as of the date of this certificate, of any material fact or material change which is not reflected in this Technical Report.
9. I am independent of the Issuer, applying all the tests in section 1.5 of NI 43-101.
10. I have read NI 43-101 and Form 43-101F1 – Technical Report and, in my opinion, the Technical Report has been prepared in compliance with such instrument and form.

Belo Horizonte, Brazil, August 20th, 2025.

<signed & sealed in the original>

____

Juliano Felix de Lima

GE21 Consultoria Mineral

QP CERTIFICATE OF EDUARDO DEQUECH DE CARVALHO

I, Eduardo Dequech de Carvalho, MAusIMM, as an author of the independent technical report titled "NI 43-101 Preliminary Economic Assessment (PEA) Independent Technical Report for the Luanga PGM + Au + Ni Project Pará, Brazil" dated August 20th, 2025, with an effective date of July 7, 2025 (Technical Report), prepared for Bravo Mining Corp. (Issuer), do hereby certify that:

1. I am a mining engineer for GE21 Consultoria Mineral Ltda., which is located on Avenida Afonso Pena, 3130, 9th, 12th and 13th floor, Savassi, Belo Horizonte, MG, Brazil - CEP 30130-910.
2. I am a graduate of the Federal University of Minas Gerais, located in Belo Horizonte, Brazil, and hold a Bachelor of Science Degree in Mining Engineering (2018). I have practised my profession continuously since 2018.
3. I am a Professional enrolled with the Australasian Institute of Mining and Metallurgy (AusIMM) Member - (MAusIMM) #3113310.
4. I am a professional Mining Engineering with over 7 years of experience in mineral reserve estimation and mining planning, including dealing with various commodities and types of deposits, including several polymetallic properties.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association as defined in NI 43-101, and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am responsible for Sections 16, 19, and 22, with co-with responsibility for Sections 1, 21, 25, 26 and 27 of the Technical Report.
7. As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report that I have authored and am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
8. I have no personal knowledge, as of the date of this certificate, of any material fact or material change which is not reflected in this Technical Report.
9. I am independent of the Issuer, applying all the tests in section 1.5 of NI 43-101.
10. I have read NI 43-101 and Form 43-101F1 – Technical Report and, in my opinion, the Technical Report has been prepared in compliance with such instrument and form.

Belo Horizonte, Brazil, August 20th, 2025.

<signed & sealed in the original>

Eduardo Dequech de Carvalho