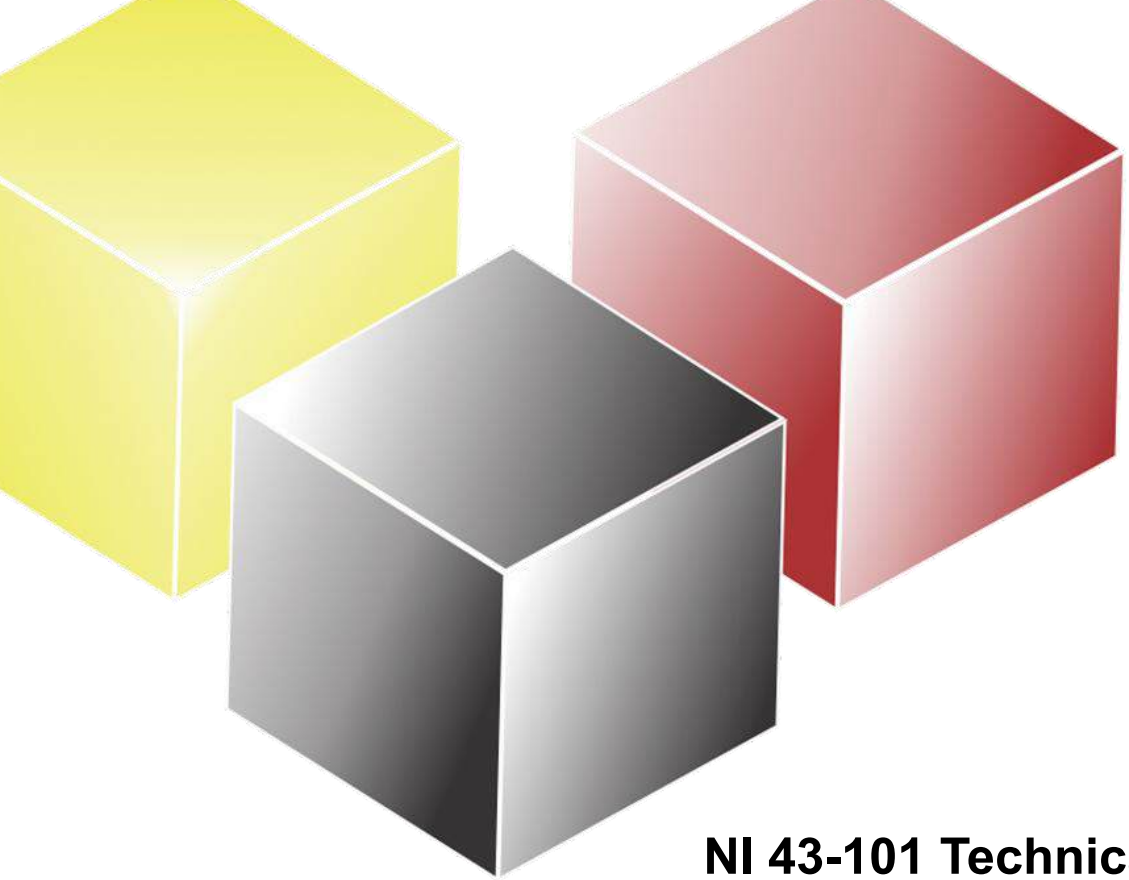


NI 43-101 Technical Report Pre-Feasibility Study on the Luanga Project Pará State, Brazil

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FORWARD LOOKING STATEMENTS

This report contains “forward-looking information” within the meaning of applicable Canadian securities laws. Forward-looking information is not comprised of historical facts and is generally, but not always, identified by words such as “plans”, “targets”, “expects”, “budget”, “schedule”, “estimates”, “forecasts”, “outlook”, “intends”, “anticipates”, “believes”, “assumes”, “potential”, “opportunity”, variants of these words and other similar words, phrases, or statements that certain events or conditions “may”, “could”, “should”, “will”, “would” and similar expressions.

Forward-looking information in this report includes, among other things, statements regarding: the results, assumptions and conclusions of the PFS, included PEAs, and the Mineral Reserve estimate, and the Mineral Resource estimate on which these are based; the potential future economic and technical performance of the Luanga Project; projected production, timing, recoveries, operating and capital costs, cash costs, AISC, revenues, cash flows, NPV, IRR and payback; the potential future development, construction, commissioning and operation of the mine, concentrator and proposed Barcarena ZPE smelter; the availability and benefits of the ZPE fiscal, customs and administrative regime and the SUDAM provisions in respect of Luanga; the sourcing of renewable electricity; the production and sale of metal alloy and sulphuric acid; the timing and outcome of environmental licensing and other regulatory approvals; the timing, results of and completion of future engineering and a Definitive Feasibility Study; the timing, results and impact of ongoing and future drilling and the targeted Q1 2027 Mineral Resource update; the potential conversion of Inferred Mineral Resources to Measured and Indicated Mineral Resources and ultimately to Mineral Reserves; the potential for mine-life extension and Mineral Resource and Mineral Reserve growth; the potential economic contribution of copper, self-mining and other Project opportunities; the results supporting or otherwise, and potential implementation of, Alternate Case A, Alternate Case B and the Opportunity Case; the availability and terms of potential project financing, including potential funding from Brazilian development institutions; commodity prices, exchange rates and other economic assumptions; and the timing of, and the Company’s ability to advance the Project toward, a future construction decision.

Forward-looking information is based on management’s current expectations, estimates, assumptions and beliefs, including assumptions regarding the accuracy of the PFS and included PEAs, and Mineral Resource and Mineral Reserve estimates; metallurgical recoveries; metal prices and exchange rates; capital and operating costs; availability of equipment, labour, power, water, infrastructure and financing; the ability to obtain and maintain required permits and approvals; the continued availability of ZPE and other fiscal incentives, including SUDAM eligibility; the results of future drilling and its impact on future mineral resource and mineral reserve estimates, engineering and metallurgical test work; the results of future exploration to expand the current mineral resource estimate, and from other targets identified in this Report, and their impact on the economic analyses contained in this Report; and general business, economic and market conditions. Although the Company considers these assumptions reasonable as of the

date of this report, they are inherently subject to significant business, economic, technical, regulatory and competitive uncertainties and contingencies.

Forward-looking information involves known and unknown risks, uncertainties and other factors that may cause actual results, performance or achievements to differ materially from those expressed or implied by such information. These risks include, without limitation, risks relating to the accuracy of Mineral Resource and Mineral Reserve estimates; the speculative nature of Inferred Mineral Resources; changes to modifying factors and PFS assumptions; exploration, drilling, metallurgical, engineering, construction and commissioning results; cost escalation and inflation; commodity price and foreign exchange fluctuations; financing availability and terms; permitting compliance and permitting regulatory delays; environmental and social risks; the design, construction and operation of the proposed smelter; availability of infrastructure, power and other inputs; changes to tax, customs, ZPE or other applicable regimes; marketability and payability of concentrates, alloy products, bi-products; and the other risks inherent in mineral exploration, development and mining.

There can be no assurance that forward-looking information will prove to be accurate, as actual results and future events could differ materially from those anticipated. Readers are cautioned not to place undue reliance on forward-looking information. The Company undertakes no obligation to update or revise any forward-looking information, whether as a result of new information, future events or otherwise, except as required by applicable securities laws.

UNITS, SYMBOLS, AND ABBREVIATIONS

Abbreviations	
3D	Three-Dimensional
3E	Three Elements: Palladium + Platinum + Gold (Pd + Pt + Au)
3SD	Three-Stage Dilution
4E	Four Elements: Palladium + Platinum + Rhodium + Gold (Pd + Pt + Rh + Au)
AACE	Association for the Advancement of Cost Engineering
AAS	Atomic Absorption Spectroscopy
ABNT	Brazilian Association of Technical Standards *Associação Brasileira de Normas Técnicas*
AC	Alternating Current
ADA	Directly Affected Area *Área Diretamente Afetada*
AFRMM	Additional Freight Charge for the Renewal of the Merchant Navy *Adicional ao Frete para Renovação da Marinha Mercante*
AI	Bond Abrasion Index
AIG	Australian Institute of Geoscientists
AIPL	Americas Investments & Participation Ltd.
AISC	All-in Sustaining Cost
AMSA	Amazônia Mineração S.A.
ANEEL	Brazilian National Electric Energy Agency *Agência Nacional de Energia Elétrica*
ANM	National Mining Agency of Brazil *Agência Nacional de Mineração*
AusIMM	Australasian Institute of Mining and Metallurgy
BAIP	Brazil Americas Investments and Participation Mineração Ltda.
BHEM	Borehole Electromagnetic
BLUE	Best Linear Unbiased Estimator
BNDES	Brazilian Development Bank *Banco Nacional de Desenvolvimento Econômico e Social*
BWI	Bond Work Index
CA	Environmental Offset *Compensação Ambiental*
CAPEX	Capital Expenditure
CBS	Contribution on Goods and Services *Contribuição sobre Bens e Serviços*
CCTV	Closed-Circuit Television
CETEM	Centre for Mineral Technology *Centro de Tecnologia Mineral*
CFEM	Financial Compensation for the Exploitation of Mineral Resources *Compensação Financeira pela Exploração de Recursos Minerais*
CGI	Copper–Gold–Iron
CIF	Cost, Insurance and Freight
CIL	Carbon-in-Leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIT	Innovation and Technology Centre *Centro de Inovação e Tecnologia*
CLT	Consolidation of Labour Laws *Consolidação das Leis do Trabalho*
CMC	Carboxymethyl Cellulose
CMM	Companhia Meridional de Mineração
CMP	Carajás Mineral Province
CNRH	National Water Resources Council *Conselho Nacional de Recursos Hídricos*
CODIM	Companhia de Desenvolvimento de Indústrias Minerais
CoG	Cut-off Grade
CONAMA	Brazilian National Environment Council *Conselho Nacional do Meio Ambiente*
CRM	Certified Reference Material

Abbreviations	
CSA	Canadian Securities Administrators
CSLL	Social Contribution on Net Profit *Contribuição Social sobre o Lucro Líquido*
CV	Coefficient of Variation
CVRD	Companhia Vale do Rio Doce (currently Vale)
CZPE	Export Processing Zone Council *Conselho Nacional das Zonas de Processamento de Exportação*
DC	Direct Current
DCDA	Double-Contact Double-Absorption
DDH	Diamond Drill Hole
DEM	Digital Elevation Model
DFS	Definitive Feasibility Study
DIC	Direct Installed Cost
DNPM	National Department of Mineral Production (former name of ANM) *Departamento Nacional de Produção Mineral*
DOCEGEO	Rio Doce Geologia e Mineração
DTH	Down-the-Hole
EBITDA	Earnings Before Interest, Taxes, Depreciation and Amortization
EDA	Exploratory Data Analysis
EFC	Carajás Railway *Estrada de Ferro Carajás*
EIA	Environmental Impact Study *Estudo de Impacto Ambiental*
EM	Electromagnetic
EPCM	Engineering, Procurement and Construction Management
ESG	Environmental, Social and Governance
ETF	Exchange-Traded Fund
EV	Electric Vehicle
FFAH	FFA Holding e Mineração Ltda.
FLTEM	Fixed-Loop Transient Electromagnetic
G&A	General and Administrative
GE21	GE21 Consultoria Mineral Ltda.
GISTM	Global Industry Standard on Tailings Management
GNSS	Global Navigation Satellite System
GPS	Global Positioning System
HARD	Half Average Real Difference
HDPE	High-Density Polyethylene
HeliTEM	Helicopter-borne Time-Domain Electromagnetic
HPAL	High-Pressure Acid Leach
HPGR	High-Pressure Grinding Rolls
IBGE	Brazilian Institute of Geography and Statistics *Instituto Brasileiro de Geografia e Estatística*
IBS	Tax on Goods and Services *Imposto sobre Bens e Serviços*
ICMS	Tax on the Circulation of Goods and Services *Imposto sobre Circulação de Mercadorias e Serviços*
ICOLD	International Commission on Large Dams
ICP	Inductively Coupled Plasma
ICP-AES	Inductively Coupled Plasma Atomic Emission Spectroscopy
ICP-MS	Inductively Coupled Plasma Mass Spectrometry
ICP-OES	Inductively Coupled Plasma Optical Emission Spectrometry
ID	Identification
IEC	International Electrotechnical Commission
IFRS	International Financial Reporting Standards
II	Import Duty *Imposto de Importação*
INCRA	Brazilian National Institute for Colonization and Agrarian Reform *Instituto Nacional de Colonização e Reforma Agrária*
INSG	International Nickel Study Group

Abbreviations	
IOCG	Iron Oxide Copper-Gold
IP	Induced Polarization
IPI	Industrial Products Tax *Imposto sobre Produtos Industrializados*
IQD	Inverse of Quadratic Distance
IRPJ	Corporate Income Tax *Imposto de Renda da Pessoa Jurídica*
IRR	Internal Rate of Return
ISCG	Iron Sulphide Copper-Gold
ISO	International Organization for Standardization
ISS	Service Tax *Imposto sobre Serviços*
IT	Information Technology
JV	Joint Venture
K80	80% Passing Size
LBMA	London Bullion Market Association
LCT	Locked-Cycle Test
LFP	Lithium Iron Phosphate
LI	Installation Licence *Licença de Instalação*
LIDAR	Light Detection and Ranging
LIP	Large Igneous Province
LME	London Metal Exchange
LO	Operating Licence *Licença de Operação*
LOM	Life of Mine
LP	Preliminary Licence *Licença Prévia*
LSZ	Low Sulphide Zone
MASU	Massive Sulphide Zone
MCC	Motor Control Centre
MDF	Medium-Density Fibreboard
MDIC	Ministry of Development, Industry, Trade and Services *Ministério do Desenvolvimento, Indústria, Comércio e Serviços*
MG	Minas Gerais State
MIBC	Methyl Isobutyl Carbinol
MME	Ministry of Mines and Energy *Ministério de Minas e Energia*
MRE	Mineral Resource Estimate
MSZ	Main Sulphide Zone
MUM	Mafic-Ultramafic
MZ	Mafic Zone
NA	Not Applicable
NBR	Brazilian Standard *Norma Brasileira*
NI 43-101	National Instrument 43-101
NMC	Nickel-Manganese-Cobalt
NN	Nearest Neighbour
NPI	Nickel Pig Iron
NPV	Net Present Value
NRM	Mining Regulatory Standards *Normas Reguladoras de Mineração*
NSR	Net Smelter Return
OEM	Original Equipment Manufacturer
OK	Ordinary Kriging
ONS	National Electric System Operator *Operador Nacional do Sistema Elétrico*
OPEX	Operational Expenditure
OWS	Oil–Water Separator

Abbreviations	
P60	60% Passing Size
P80	80% Passing Size
P90	90% Passing Size
PA	Pará State
PAE	Economic Utilization Plan *Plano de Aproveitamento Econômico*
PAX	Potassium Amyl Xanthate
PCFM	Conceptual Mine Closure Plan *Plano Conceitual de Fechamento de Mina*
PdEq	Palladium Equivalent
PDF	Probability Distribution Function
PEA	Preliminary Economic Assessment
PFS	Pre-feasibility Study
PG	Primary Grind
PGC	Grande Carajás Project *Projeto Grande Carajás*
PGE	Platinum Group Element
PGM	Platinum Group Metals
PIS/COFINS	Social Integration Program and Contribution to Social Security Financing *Programa de Integração Social e Contribuição para o Financiamento da Seguridade Social*
PNMA	Brazilian National Environmental Policy *Política Nacional do Meio Ambiente*
PNSB	Brazilian National Dam Safety Policy *Política Nacional de Segurança de Barragens*
PPE	Personal Protective Equipment
PVC	Polyvinyl Chloride
pXRF	Portable X-Ray Fluorescence
QA/QC	Quality Assurance and Quality Control
QemSCAN	Quantitative Evaluation of Minerals by Scanning Electron Microscopy
QP	Qualified Person
QQ	Quantile–Quantile
RC	Reverse Circulation
RGB	Red, Green and Blue
RIMA	Environmental Impact Report *Relatório de Impacto Ambiental*
RL	Reduced Level
RMR	Rock Mass Rating
ROM	Run-of-Mine
RPEEE	Reasonable Prospects for Eventual Economic Extraction
RQD	Rock Quality Designation
RTK	Real-Time Kinematic
SAD69	South American Datum 1969
SAG	Semi-Autogenous Grinding
SCA	Environmental Offset Calculation System *Sistema de Cálculo de Compensação Ambiental*
SD	Standard Deviation
SEDAR+	System for Electronic Document Analysis and Retrieval+
SEMAS/PA	Pará State Secretariat for Environment and Sustainability *Secretaria de Estado de Meio Ambiente e Sustentabilidade do Pará*
SENAI	National Service for Industrial Training *Serviço Nacional de Aprendizagem Industrial*
SG&A	Selling, General and Administrative
SI	International System of Units
SIN	National Interconnected System *Sistema Interligado Nacional*
SIRGAS 2000	Geocentric Reference System for the Americas 2000 *Sistema de Referencia Geocéntrico para las Américas 2000*
SMU	Selective Mining Unit

Abbreviations	
SUDAM	Superintendency for the Development of the Amazon *Superintendência do Desenvolvimento da Amazônia*
SZ	Sulphide Zone
TBRC	Top-Blown Rotary Converter
TCCA	Environmental Offset Agreement *Termo de Concordância de Compensação Ambiental*
TDH	Total Dynamic Head
TEM	Transient Electromagnetic
TFRM	Pará State Mineral Resources Inspection Fee *Taxa de Controle, Acompanhamento e Fiscalização das Atividades de Pesquisa, Lavra, Exploração e Aproveitamento de Recursos Minerais*
TRC	Trench Channel
TSF	Tailings Storage Facility
TSX.V	TSX Venture Exchange
TZ	Transition Zone
UC	Conservation Unit *Unidade de Conservação*
UTM	Universal Transverse Mercator
UZ	Ultramafic Zone
VTEM	Versatile Time-Domain Electromagnetic
WPIC	World Platinum Investment Council
WRSF	Waste Rock Storage Facility
WSS	Water Supply System
WTP	Water Treatment Plant
XRF	X-Ray Fluorescence
ZPE	Export Processing Zone *Zona de Processamento de Exportação*

Units and Symbols	
%	percentage
°	degree (angle, azimuth)
″	inch
°C	degree Celsius
cm	centimetre
cm³	cubic centimetre
ft	foot
g	gram
g/cm³	gram per cubic centimetre (density)
g/t	gram per tonne
Ga	billion years
GWh	gigawatt-hour
h	hour
ha	hectare
HP	horsepower
k	thousand
kg	kilogram
kg/t	kilogram per tonne
klb	thousand pounds
km	kilometre
km²	square kilometre
km/h	kilometre per hour
koz	thousand troy ounces
kt	thousand tonnes
ktpa	thousand tonnes per year
kV	kilovolt
kVA	kilovolt-ampere
kW	kilowatt

Units and Symbols	
kW/m²	kilowatt per square metre
kWh/t	kilowatt-hour per tonne (specific energy, Bond Work Index)
L	litre
lb	pound
line-km	line kilometre
m	metre
m²	square metre
m³	cubic metre
m³/h	cubic metre per hour
M	million
Mm³	million cubic metres
Moz	million troy ounces
MPa	megapascal
Mt	million tonnes
Mtpa	million tonnes per annum
MVA	megavolt-ampere
MW	megawatt
MWh	megawatt-hour
MWh/t	megawatt-hour per tonne
Nm³/h	normal cubic metre per hour
oz	troy ounce
ppb	parts per billion
ppm	parts per million
R$	Brazilian real
t	tonne
t/h	tonne per hour
t/h·m²	tonne per hour per square metre
t/m³	tonne per cubic metre
t/t	tonne per tonne (dimensionless ratio, e.g. strip ratio)
tpd	tonne per day
USD or US$	United States dollars
US$/lb	United States dollars per pound
US$/oz	United States dollars per troy ounce
US$/t	United States dollars per tonne
W	watt
µm	micrometre

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1 EXECUTIVE SUMMARY

1.1 Introduction

GE21 Consultoria Mineral Ltda. (GE21) is an independent mineral consulting company based in Belo Horizonte, Brazil, specializing in Mineral Resource and Reserve declarations under NI 43-101 standards.

Bravo Mining Corp. (Bravo) is a mineral exploration and development company based in Canada and Brazil, focused on the Luanga Project in the Carajás Mineral Province. Its head office is located in Rio de Janeiro and its registered and records office is located in Vancouver.

Bravo commissioned GE21 to prepare a Pre-feasibility Study (PFS) for the Luanga Project in compliance with NI 43-101 guidelines. The results of the PFS are summarized in this report (Technical Report or Report).

1.2 Qualifications, Experience and Independence

The Qualified Persons (QPs) responsible for the PFS and this Report are presented in Table 1-1 below.

Table 1-1: Report items and assigned QP responsibilities

Qualified Person (Authors)	Company	Site Visit Date	Responsible for Sections of this Report
Porfirio Cabaleiro FAusIMM | FAIG	GE21	July 4 to 7, 2023; October 3 to 6, 2023; January 27 to 31, 2025; February 10 to 13, 2026.	14, 15, 16, 19, 21, 22, 23, 24 and co-responsibility for 1, 18, 25, 26 and 27
Bernardo Viana FAIG	GE21	October 3 to 6, 2023; January 27 to 31, 2025.	2, 3, 4, 5, 6, 7, 8, 9, 10, 11, 12, and co-responsibility for 1, 25, 26 and 27
Paulo Bergmann FAusIMM	GE21	-	13, and co- responsibility for 1, 17, 18, 25, 26 and 27
Rodney Jones FSAIMM	Independent Consultant	-	Co-responsibility for Sections 1, 17, 25, 26 and 27
Branca Horta MAIG	GE21	-	20, and co- responsibility for 1, 25, 26 and 27

Source: GE21, 2026.

Neither GE21 nor the Authors of this Technical Report have or had any material interest invested in Bravo or any of its related entities. GE21's and the Authors' relationships with Bravo are 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.

1.3 Reliance on Other Experts

The Authors of this Report are Qualified Persons (QPs) as defined under NI 43-101, with relevant expertise in mineral exploration, data validation, and resources and reserves estimation.

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

Bravo engaged Linneu de Albuquerque Mello, a qualified lawyer in Brazil, to provide a title opinion on the Luanga Mineral Rights, in respect of their validity and good standing as of April 02, 2026.

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 Licence Nº.1961 (ANM.851.966/1992) and covers 7,810.02 hectares.

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. These agreements are renewable every two years.

Although the original exploration licence 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 Licence 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.

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

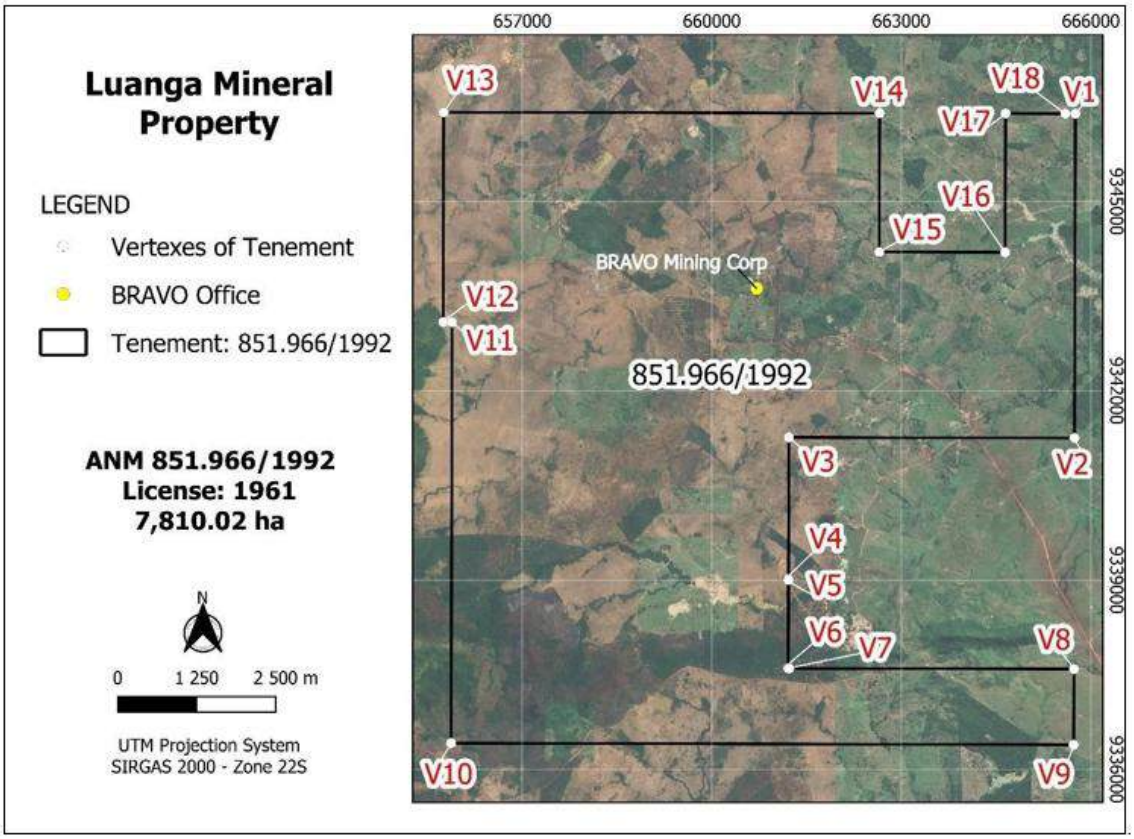


Figure 1-1: Luanga Project tenement map

Source: GE21, 2025.

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

The Luanga Project is located in the Curionópolis region, 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 500-700 m), 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.

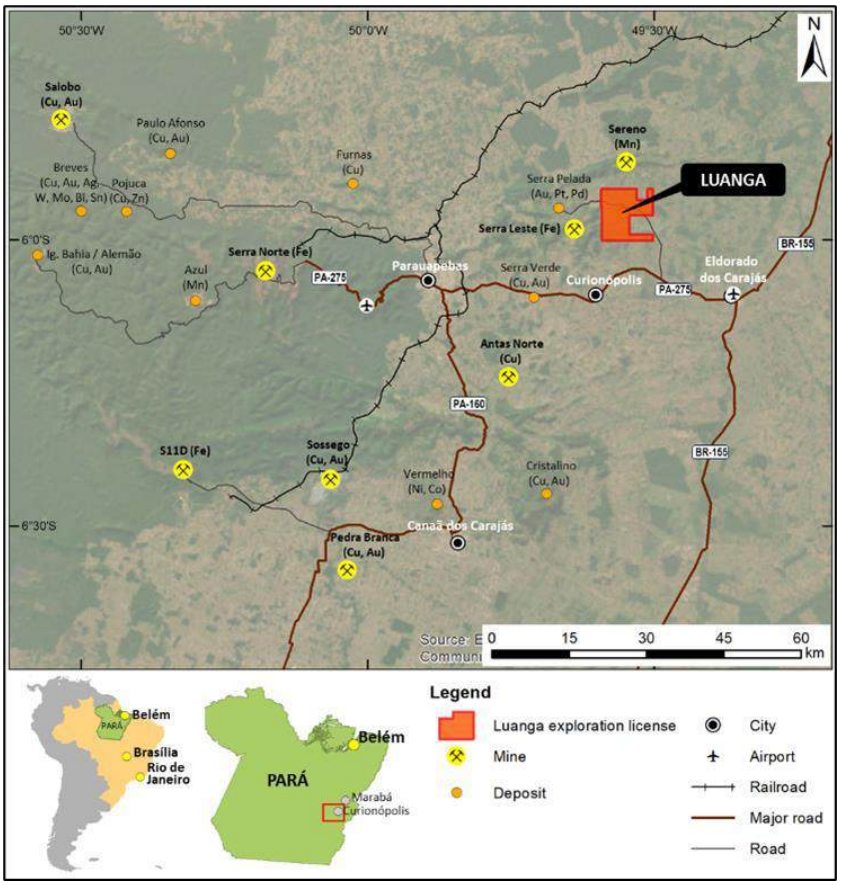


Figure 1-2: Access map for Luanga Project

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 Union Carbide subsidiary, 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 Amazônia Mineração S.A. (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 the Carajás iron mineralization. By 1977, Vale acquired the remaining 49% of CMM for US$55 million, gaining full control over the Carajás Mineral Province.

In 1978, Vale began construction of the Carajás Railroad, which links 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 and ultramafic rocks within 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, totalling 256 drill holes (50,786.74 m):

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 2023 MRE was superseded by an updated MRE, also conducted by GE21 in 2025. The 2025 MRE is the basis for the development of the present PFS study, and comprises:

- 36 Mt grading 2.00 g/t PdEq, for a total of 2.3 Moz of PdEq in the Measured category;
- 122 Mt grading 2.06 g/t PdEq, for a total of 8.1 Moz of PdEq in the Indicated category;
- 78 Mt grading 2.01 g/t PdEq for 5.0 Moz of PdEq in the Inferred category.

See Section 14 for details, including grades by individual metals.

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.

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 to the east and south by the Araguaia Neoproterozoic Belt and overlain to the west by Paleoproterozoic sequences. 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 host major Ni laterite deposits (e.g., Onça-Puma, Vermelho, Jacaré) and PGM deposits (e.g., Luanga, Lago Grande). The magmatic structures 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 8.1 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 m higher than the surrounding flat areas, where gabbroic rocks prevail. The layering within the complex forms an arc-shaped structure that mirrors the region's overall morphology.

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 metres 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.

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 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 TZ and the 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 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 (USA) 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

Since acquiring the mineral rights in November 2021, Bravo has explored the Luanga Project along two parallel fronts: the definition and extension of the stratiform PGM+Au+Ni mineralization of the Luanga Mafic-Ultramafic (MUM) Complex, which hosts the Mineral Resource, and generative exploration for genetically distinct mineralization styles across the wider property, principally magmatic Ni-Cu (± PGM) sulphides and iron sulphide/oxide copper-gold (ISCG/IOCG) systems.

Work has comprised resampling of 202 historical Vale drill holes (40,065 m), reprocessing of historical soil, IP and magnetic data, detailed geological mapping at 1:10,000 and 1:5,000 scales, new soil grids analysed by pXRF, 58 trenches totalling 10,901 m, petrographic and mineralogical studies, and an in situ density programme. Geophysical coverage includes borehole and fixed-loop electromagnetic surveys, ground magnetometry and micro-gravity over 18.7 km², a property-wide helicopter-borne time-domain electromagnetic survey (HeliTEM, 771.2 line-km) and a 50 line-km deep induced polarization (IP) programme initiated in June 2026.

The Luanga Complex extends for approximately 8.1 km and remains open at depth along essentially its whole strike length. Bravo's drilling has extended the mineralization from the 100 m to 150 m depth tested historically by Vale to approaching 350 m in the Central Sector and approximately 200 m in the North Sector, with equal or greater thicknesses and grades at depth, as illustrated by 51.0 m at 3.67 g/t PGM+Au in hole DDH26LU312. In the Southwest Sector, the constraining pit shell of the 2025 MRE reaches the limit of the drilling data, and the Main Sulphide Zone has been shown to maintain continuous thickness. Trenching has also shown a wider lateral extent of the shallow oxide PGM+Au mineralization than interpreted from drilling.

Mapping within the northern extension of the Luanga Mafic-Ultramafic (MUM) Complex has identified chromitite and orthopyroxenite outcrops and float blocks in an area previously mapped as gneissic terrain. This suggests that the North Sector mineralized limit can extends

further north than currently recognized, beyond the existing drilling grid, and represents significant potential to expand the current mineral resource estimate for the Luanga deposit

Additional potential within the deposit area includes a magmatic Ni-Cu-PGM massive sulphide zone (LU330 Target, 13.44 m at 1.55% sulphide Ni in hole DDH26LU347), a hydrothermal massive sulphide zone at the Babylon Target (11.04 m at 2.04% Ni and 1.23% Cu in hole DDH22LU047), supported by a deep IP chargeability anomaly, and the Crescent Zone, where 2026 drilling intercepted PGM mineralization comparable to that of the deposit. The Luanga South MUM Complex, with 6.5 km of the same stratigraphy and historical PGM+Au intercepts, is a further prospect for new PGM Mineral Resources.

The 2023 HeliTEM survey generated 17 priority targets outside the deposit. Drilling in 2024 discovered high-grade Cu-Au mineralization at the T5 Target, including 11.48 m at 14.27% Cu and 3.33 g/t Au in hole DDH2405T002, within a 250 m long, near-vertical sulphide body interpreted as ISCG-type. Massive to brecciated sulphides were also intersected at T16 (6.54 m at 2.25% Cu and 0.55% Ni) and T6, and encouraging results were obtained at several other targets. A Phase 1 programme of mapping, geochemistry, deep IP, fixed-loop and borehole TEM surveys and approximately 6,000 m of drilling is under way to test the most prospective targets. No Mineral Resources have been estimated outside the Luanga deposit.

1.10 Drilling

Approximately 150,161 m have been drilled in 707 drill holes on the Project since 1992. Of these, 256 (50,787 m) are diamond drill holes (DDH) executed by DOCEGEO (Vale) and include the Luanga's North, Central, and Southwest geological targets. Bravo's drilling totals 99,374.19 m and 451 holes, representing 64% 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, resource growth and 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 program implemented by Bravo was performed using a mixture of HQ and NQ2 diameters.

During 2025/2026 Bravo completed approximately 26,551 m of further resource and exploration drilling, including drilling associated with the ongoing copper exploration program. Drill holes associated with these drilling campaigns were completed post the cut-off date for the current MRE and thus do not contribute to its estimation.

Table 1-2: Drilling summary for Luanga

Year	Drill Type	Drill Holes	Total Metres	Company	Contractor
1992	DDH	4	643.69	DOCEGEO	DOCEGEO
2001	DDH	86	14,584.35		Geosol
2002	DDH	71	15,423.25		Geosol

2003	DDH	95	20,135.45		Geosol/Rede
2022	DDH	135	23,258.20	Bravo	Servdrill Perfuração e Sondagem (Servdrill)
2023	DDH	116	30,296.60		Servdrill
2024	DDH	94	19,268.65		Servdrill
2025	DDH	29	4,861.85		Servdrill
2026	DDH	77	21,689.15		Servdrill
TOTAL		**707**	**150,161.19**		

Source: Bravo, 2026.

1.11 Sample Preparation, Analyses and Security

Bravo's quality assurance/quality control (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 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.

The QA/QC program comprises 27,257 control samples (20,612 from Bravo's program, 3,556 from Vale's program and 3,089 historical samples resampled by Bravo), representing approximately 16.7% of all samples analyzed.

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 dataset.

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

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 industry best practices, and the Luanga Project database is considered suitable for the current Mineral Resource Estimation.

1.12 Data Verification

Since 2023, 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; January 27-31, 2025; and February 10-13, 2026.

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 current 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

Metallurgical investigations completed on the Project demonstrate that the sulphide mineralization is amenable to comminution and flotation processing, using fine grinding and Jameson technology flotation cells. Historical and recent testwork programs have evaluated the metallurgical performance of sulphide, transitional, and oxide mineralization, including comminution characterization, flotation optimization, hydrometallurgical testwork, solid-liquid separation studies, and preliminary downstream processing investigations.

Comminution testing indicates that the mineralization exhibits moderate to high hardness characteristics and is suitable for conventional, proven technologies and equipment commonly used in commercial mining operations, such as high pressure grinding rolls (HPGR) and ball milling circuits. The ore displays relatively low abrasivity and no significant processing constraints have been identified from the comminution program.

Flotation testwork conducted on samples representing the principal mineralized domains, consistently produced saleable grade concentrates containing platinum group metals, gold, and nickel. Locked-cycle and batch flotation programs demonstrated that recoveries are influenced by mineralization style, mineralogy, flotation reagent selection, and grinding size, with improved performance generally achieved at finer grind sizes. Testwork utilizing advanced flotation technologies, including hydrodynamic cavitation and Jameson Cell applications, demonstrated potential enhancements in fine and ultra-fine particle recovery.

Based on the available metallurgical database, recoveries adopted for fresh Mineral Resource estimation are 77% for palladium, 81% for platinum, 51% for rhodium, 48% for gold, and 50% for nickel. These values are supported by multiple stages of laboratory metallurgical testing and are considered reasonable for the current level of study.

Transitional mineralization exhibits lower flotation recoveries than fresh sulphide material but remains responsive to conventional flotation. The flotation of Oxide mineralization appeared to be maily impacted in zones hosting high concentrations of altered and clay-rich material (talc). Consequently, additional hydrometallurgical testwork was undertaken on oxide samples. Cyanide

batch leaching tests achieved high recoveries of gold and palladium, indicating that alternative processing routes may provide future development opportunities for oxide resources.

Preliminary smelting investigations further demonstrated that flotation concentrates are suitable for downstream metallurgical treatment and can achieve high recoveries of payable metals.

In the opinion of the Qualified Person, the metallurgical testwork completed to date demonstrates that the sulphide mineralization is amenable to processing through a flotation-based flowsheet. The results provide reasonable support for the metallurgical assumptions applied in the Mineral Resource Estimate and indicate favourable potential for future economic extraction.

1.14 Mineral Resource Estimates

GE21 carried out the 3D geological modelling, statistical and geostatistical studies, and grade estimate for the Luanga Project, assessing a set of factors, including the amount and spacing of available data, interpreted mineralization controls, mineralization style, and the quality of data used.

The Mineral Resource was classified per CIM Standards (2014) and follows CIM Best Practices Guidelines (2019), utilizing geostatistical and classical methods, along with economically- and mining-appropriate parameters relevant to the deposit type.

With the intention to simplify the Mineral Resource statement, the additional variable 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, and Ni price of US$ 7.10/lb.

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

Where:

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

Where P = Metal Price and 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 (Table 1-3).

Table 1-3: MRE statement at a cut-off of 0.5 g/t PdEq

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), FAIG, director of GE21 Consultoria Mineral Ltda., an independent Qualified Person (QP) under NI 43-101. 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.
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):
   a. Metallurgical recovery in sulphide material of 77% Pd, 81% Pt, 51% Rh, 48% Au, 50% Ni to a saleable Ni-PGM concentrate.
   b. Metallurgical recovery in oxide material of 81% Pd, 23% Pt, 54% Rh, 90% Au to a saleable PGM ash residue (Ni not applicable).
   c. Metallurgical recovery in high-talc sulphide material of 51% Pd, 55% Pt, 27% Rh, 27% Au, 0% Ni to a saleable Ni-PGM concentrate.
   d. Independent Geotechnical Testwork – Overall pit slopes of 40 degrees in oxide and 50 degrees in Fresh Rock.

e. 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.
f. External downstream payability has not been included, as the base case MRE assumption considers internal downstream processing.
g. Payable royalties of 2%.

7. 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, and 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:

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

$_P$ = Metal Price.

$_R$ = Metallurgical Recovery.

8. Costs are taken from comparable projects in GE21's extensive database of mining operations in Brazil, which includes not only operating mines, but also recent actual costs from what could potentially be similarly sized operating mines in the Carajás. Costs considered a throughput rate of ca. 10 Mtpa:
   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.
9. The current MRE supersedes and replaces the Previous Estimate (2023), which should no longer be relied upon.
10. 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.
11. Totals may not sum due to rounding.

Source: GE21, 2025.

1.15 Mineral Reserve Estimates

The Initial Mineral Reserve estimate for Luanga is based on the Mineral Resource Statement with an effective date of February 18, 2025. Mineral Resources are inclusive of Mineral Reserves.

Table 1-4: Luanga Project – Mineral Reserve estimates

Category	Mass	Pd	Pt	Rh	Au	Ni	Pd Eq	Pd	Pt	Rh	Au	Ni	Pd Eq
	(Mt)	(ppm)						(koz)				(t)	(koz)
Proven Reserves	17.7	0.95	0.82	0.12	0.08	1,765	2.99	541	466	70	44	31,239	1,703
Probable Reserves	69.0	0.93	0.65	0.10	0.07	1,786	2.72	2,062	1,442	226	161	123,237	6,028
Total Reserves	**86.7**	**0.93**	**0.68**	**0.11**	**0.07**	**1,781**	**2.77**	**2,603**	**1,908**	**295**	**205**	**154,476**	**7,731**

Notes:
1. Mineral Reserves are reported in accordance with the CIM Standards and Best Practices Guidelines, as required by National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).
2. Mineral Reserves are the economic portion of Measured and Indicated Mineral Resources. No Inferred Resource was counted as Mineral Reserve.
3. Mineral Reserves are accounted at plant ROM pad;
4. The Mineral Reserve Estimates have been prepared by Mr. Porfirio Cabaleiro Rodriguez, B.Sc Min. Eng., FAIG, FAusIMM, director of GE21, an independent QP under NI 43-101.
5. The effective date of the Mineral Reserve Estimates is September 16, 2026.
6. Mineral Reserves block model using 5 m x 5 m x 5 m SMU.
7. Mining recovery factor of 95%. Mining dilution of 7.4% applied in the block model.
8. Defined cut-off grade of 0.87 g/t Pd_Eq.
9. Pd_Eq formula: Pd_Eq = Pd + Pt*1.11733 + Rh*2.60558 + Au*1.58174 + Ni*0.000394
10. The Mineral Reserve Estimates are reported within a designed pit, developed based on a selected pit shell generated by GEOVIA Whittle® software, using the following assumptions:
    - Metallurgical recovery of 77% Pd, 81% Pt, 51% Rh, 50% Au, 62% Ni to a saleable Ni-PGM concentrate.
    - Independent Geotechnical Testwork – Overall pit slopes of 33° in oxide, 48° in poor quality fresh rock, 51° in medium quality fresh rock, and 55° in good quality fresh rock.
    - The Mineral Reserves are reported on a dry density basis.
    - Smelting payability of 99% for Pd and Pt, 95% for Rh and Au, and 98% for Ni.
    - Payable royalties of 2%.
11. Costs:
    a. Mining costs: US$2.10/t oxide, US$2.90/t waste rock, and US$3.20/t ROM.
    b. Processing costs: US$7.64/t.
    c. Other costs: US$5.00/t processed for General & Administration, US$1.00/t processed for grade control, US$1.00/t processed for rehabilitation, US$4.00/t processed for thickening and filtration.
    d. Smelting and refining: US$251.00/t concentrate for smelting and refining, and US$80.00/t concentrate for smelter G&A.
12. Metal selling prices, based on Consensus Long-Term nominal prices: US$1,271.00/toz Pd, US$1,350.00/toz Pt, US$5,000.00/toz Rh, US$3,096.00/toz Au, and US$9.07/lb Ni.
13. The responsible QP is not aware of political, environmental, legal, title, taxation, socioeconomic, marketing, or other relevant risks that would materially affect the estimation of Mineral Reserves other than those discussed in this Technical Report.
14. Totals may not sum due to rounding.

Source: GE21, 2026.

1.16 Mining Methods

The Project would be mined by conventional open pit methods over a mine life of 10 years, delivering 86.7 Mt of Proven and Probable Mineral Reserves to the processing plant at a strip ratio of 6.6:1.

Pit optimization was performed in GEOVIA Whittle on Measured and Indicated Mineral Resources only, applying economic and metallurgical modifying factors and a cut-off grade of 0.87 g/t PdEq. The operational final pit was designed with 10 m benches, 6 m berms, 25 m ramps at a maximum gradient of 10% and a 45 m minimum bottom width. Overall slope angles range from 33° in weathered material to between 48° and 55° in fresh rock, depending on rock mass

quality, as defined by the geotechnical model and structural analysis. A mining recovery of 95% was applied, and mining dilution of 7% is incorporated in the block model.

The final pit contains 17.7 Mt of Proven and 69.0 Mt of Probable Mineral Reserves, averaging 2.77 g/t PdEq (0.93 g/t Pd, 0.68 g/t Pt, 0.11 g/t Rh, 0.07 g/t Au and 1,781 ppm Ni). Total material movement is 662.8 Mt, including 527.4 Mt of waste, 18.9 Mt of oxide material and 12.3 Mt of marginal material. Oxidized mineralization and marginal material are not treated as plant feed at this stage and would be stockpiled separately, for potential future use.

The mine schedule considers 10 Mtpa of plant feed after a one-year ramp-up at 60% of capacity in Year 1. Higher-grade, lower strip ratio zones are prioritized early to support payback, with plant feed grades of 3.0 to 3.4 g/t PdEq in the first three years. A temporary run-of-mine (ROM) stockpile near the plant accumulates up to 4.7 Mt of lower-grade ore, which is reclaimed in Years 9 and 10 to sustain plant feed as the pit is exhausted.

Mining would be carried out by a contractor using conventional drill, blast, load and haul cycle. The primary fleet comprises 70 t hydraulic excavators loading 55 t trucks for ROM, and 120 t excavators loading 120 t off-highway trucks for waste, supported by drill rigs, bulldozers, motor graders, front-end loaders and water trucks. All units are standard models with established dealer support in Brazil.

Waste rock would be placed in a dedicated waste rock storage facility, while marginal, oxide and ROM materials would be held in separate stockpiles.

Recommendations for the next study phase include optimizing the waste and stockpile deposition sequence to reduce haul distances, trade-off studies on larger-class equipment and dedicated haul roads, and a drill and blast study for the oxide and transition zones.

1.17 Recovery Methods

The Pre-feasibility Study (PFS) for the Luanga Project incorporates significant process optimization relative to the previous Preliminary Economic Assessment (PEA), resulting in an expanded processing capacity to 10 Mtpa and an accelerated production ramp-up strategy, with full throughput capacity achieved during the second year of operation. The proposed process flowsheet has been developed for the treatment of fresh sulphide mineralization, while oxidized saprolitic material is planned to be stockpiled for potential future processing.

Metallurgical studies confirmed that the Luanga orebody, containing platinum group metals, gold, and nickel sulphides, presents a complex mineralogical association characterized by the intimate occurrence of valuable minerals with ferromagnesian silicate gangue, particularly talc. Extensive testwork programs were completed to address the impact of talc on flotation performance and to optimize metallurgical recovery.

The selected comminution circuit consists of primary gyratory crusher, secondary cone crushers, HPGR, ball milling, and vertical stirred milling. The circuit was designed to produce an fine grind size of approximately P80 26–30 µm, required to achieve adequate liberation of PGMs

and nickel-bearing minerals. Comminution testing indicates that the ore exhibits moderate to high hardness characteristics, with the Central Zone representing the most competent material. Abrasion indices are low across all deposit sectors, supporting the use of conventional, industry-proven equipment and technologies.

The flotation plant envisages a Jameson Cell-based flowsheet comprising scalper, rougher, scavenger, and cleaner stages. This configuration was selected based on extensive laboratory and pilot-scale metallurgical testwork demonstrating improved recovery of fine and ultra-fine PGM-bearing particles. The process design assumes an overall mass pull of approximately 1.6%, producing a high-value flotation concentrate. Metallurgical recoveries adopted for the PFS economic evaluation are 81.5% for palladium, 80.0% for platinum, 55.0% for rhodium, 88.0% for gold, and 70.6% for nickel. The final concentrate is expected to contain approximately 100 g/t 3E PGM and 8.44% Ni.

The process plant incorporates high-capacity concentrate and tailings thickening systems, concentrate filtration facilities, and conventional tailings handling infrastructure. Concentrate would be thickened, filtered, and stored for transportation, while tailings would be thickened prior to disposal in the Tailings Storage Facility (TSF), maximizing process water recovery.

The estimated industrial mass balance indicates that the flotation circuit can produce approximately 21.3 t/h of final concentrate from a plant feed of approximately 1,332 t/h representing a concentration ratio of ~62.6while achieving PGM, gold and nickel recoveries consistent with metallurgical testwork results.

The proposed plant layout occupies approximately 568,000 m², with an additional 40,000 m² allocated to administrative and support facilities. Major equipment sizing was developed in collaboration with leading technology suppliers, including Weir and Glencore Technology, and is supported by metallurgical testing and benchmarking against comparable operations. Additional validation work is planned during the Definitive Feasibility Study (DFS) stage.

The water management strategy is based on extensive water recycling, incorporating reclaim water from process thickeners and the TSF.

Electrical power for the operation would be supplied through a dedicated connection to Brazil's National Interconnected System (SIN). The operation is expected to require approximately 58 MW of nominal power, with the preferred development option consisting of a dedicated 230 kV transmission line connecting the project to the Carajás Substation. Total annual energy consumption is estimated at approximately 350 GWh, with grinding circuits representing the largest power consumers.

Project infrastructure also includes a centralized analytical laboratory designed to process approximately 500 samples per day, together with a dedicated metallurgical laboratory supporting geometallurgical programs, process development activities, comminution testing, and flotation optimisation studies.

In the opinion of the Qualified Persons, the proposed process plant, water management system, and supporting infrastructure are based on conventional and well-established mining technologies, are technically achievable, and provide a suitable basis for the PFS-level evaluation of the Project. The selected design supports the planned production rate of 10 Mtpa and provides an appropriate framework for advancement to the DFS stage.

1.18 Project Infrastructure

The Project infrastructure occupies approximately 5,000 ha and comprises the open pit, waste rock storage facility and stockpiles, site access and haul roads, power supply, tailings and water storage facilities, and the civil, maintenance and administrative facilities required to support a 10 Mtpa operation.

A total of 24.4 km of internal access and haul roads would be developed progressively over the mine life, combining upgrades to existing unpaved roads and new sections.

The Project would connect to the National Interconnected System (SIN) at 230 kV via a dedicated transmission line to the Carajás Substation, selected from three alternatives on cost, electrical losses and regulatory complexity. The main substation, adjacent to the process plant, includes two 60 MVA 230/13.8 kV transformers in a redundant configuration, each able to carry the full load, with provision for future expansion and stand-by diesel generation for critical units such as fire prtection and thickeners etc.

The Tailings Storage Facility accommodates the full life-of-mine tailings in a single structure. The TSF comprises the B8 embankment and the auxiliary B8A embankment, which protects the toe of the waste rock storage facility.

The Water Storage Dam provides a regulated yield of approximately 7.3 Mm³/year. Preliminary water balances indicate that plant demand can be met from reclaim water and catchment runoff, with no abstraction from external sources.

Potable water would be supplied from wells within the Project area and treated on site for a peak workforce of approximately 1,200 people. Mine drainage would collect runoff and groundwater in pit sumps for reuse, with excess water discharged once it meets environmental standards. Sanitary and industrial effluent and solid waste treatment systems, a fire water network, and a duplictated fibre-optic communication network are included.

Site facilities include the truck workshop, plant workshop, warehouse, change house, main office, cafeteria, training center, control room, medical clinic and fire station laboratory and explosives magazine, all of conventional masonry or steel-frame construction.

1.19 Market Studies and Contracts

The PFS Base Case assumes that the Project's Ni-PGM flotation concentrate would be smelted in Brazil, at a Bravo-owned facility in the Barcarena Export Processing Zone (ZPE), into

a PGM-nickel alloy sold to an international precious-metal refiner, with sulphuric acid as a by-product.

Smelting and refining capacity for PGM-bearing concentrates is concentrated in South Africa, Russia and a small number of European refineries, exposing a concentrate-export model to high freight costs and a limited number of offshore counterparties. The Barcarena option addresses these constraints. Bravo Metals Ltda. was named anchor project of the Barcarena ZPE, established by presidential decree in January 2026, and received operating authorization (CZPE Resolution No. 128) in August 2026 to manufacture nickel-PGM mattes and alloys. The ZPE regime provides 20 years of preferential tax and customs treatment.The requirement for the various environmental and construction permits for the smelter is an ongoing process and will only be dfined after the next level of studies. The Project has also been recognized by the federal government as a Strategic Minerals Project. Concentrate and alloy would move through the existing Carajás logistics corridor to the ports of Barcarena/Vila do Conde approximately 500 km from the mine

Palladium demand is dominated by gasoline autocatalysts, with supply concentrated in Russia and South Africa. After deficits every year since 2012, Johnson Matthey forecasts a modest surplus in 2026. Platinum recorded its third consecutive deficit in 2025, and the World Platinum Investment Council forecasts a fourth in 2026, with above-ground stocks falling below three months of demand. Rhodium, a by-product of platinum and palladium mining, moved from deficit toward a small forecast surplus in 2026. In contrast, nickel remains in structural surplus, driven by Indonesian supply growth and the shift toward nickel-free battery chemistries. PGM prices were highly volatile through 2026, and bank forecasts diverge widely.

PGM supply remains concentrated in South Africa, Russia and Zimbabwe, where producers face rising costs, declining grades and restructuring, with few development projects able to add meaningful new supply. As one of the few advanced PGM development projects outside these jurisdictions, and the only one of scale in Brazil, the Project represents a geopolitically distinct potential source of supply.

Given the short-term volatility, the economic analysis adopts long-term consensus prices rather than spot prices: US$ 1,245/oz Pd, US$ 1,700/oz Pt, US$ 8,000/oz Rh, US$ 3,500/oz Au and US$ 17,000/t Ni. The sulphuric acid price of US$ 650/t CIF Barcarena is based on a three-year trailing average of CRU benchmark prices, adjusted for delivery to northern Pará and excluding the recent period of conflict-driven volatility. It is conservative relative to regional prices of US$ 1,200 to 1,400/t in the first half of 2026.

No offtake, toll-treatment or metal sales agreements have been executed, which is consistent with industry practice at PFS stage. Typical concentrate and matte contracts are built on payabilities, treatment and refining charges, penalties for deleterious elements, and LME or LBMA pricing references over a defined quotational period. For the economic evaluation, alloy toll refining terms of approximately US$ 25 per 4E ounce, plus a nickel refining charge, were based on disclosed terms between a major South African smelter and a European refiner. Through the

Definitive Feasibility Study, Bravo will need to engage both toll smelters and trading houses, as well as engineering and technology partners for the Barcarena facility.

1.20 Environmental Studies, Permitting and Social or Community Impact

The environmental and locational feasibility of the Project was certified by SEMAS/PA through Preliminary Licence (LP) No. 2062/2025, issued on 03 March 2025 and valid until 26 February 2030.

Environmental permitting follows the Brazilian three-phase model established by CONAMA Resolution No. 237/1997: LP, Installation Licence (LI) and Operating Licence (LO). Licensing jurisdiction lies with the State of Pará, through the Pará State Secretariat for Environment and Sustainability (SEMAS/PA). The LP application was filed in June 2024, supported by an Environmental Impact Study and Report (EIA/RIMA), and a public hearing was held in Curionópolis in December 2024. The Environmental Offset Agreement (TCCA) was signed in February 2025, setting the environmental offset at R$ 29.3 M. The developer has met the 30-day and 180-day LP conditions within the established deadlines.

The Directly Affected Area (ADA) covers approximately 7,810 ha within the municipality of Curionópolis, in the Carajás Mineral Province and the Amazon Biome. The area has been historically disturbed by cattle ranching and small-scale mining and retains fragmented forest remnants with reduced ecological connectivity. There is no overlap with Conservation Units, Indigenous lands, Quilombola territories, the National Institute for Colonization and Agrarian Reform (INCRA) rural settlements or other legally protected areas. The main environmental aspects to be managed are vegetation removal, landscape change, potential interference with the local water regime, dust and noise emissions, and effects on fauna and flora. Agreements are in place with the landowners within the Project boundary, and community programs cover communication, environmental education, training, local hiring and support for economic diversification.

Following the LP, the Project underwent an engineering review that introduced a revised mining and processing concept, a Tailings Storage Facility in place of the originally licensed disposal system, a new mine closure plan and an updated power demand of approximately 58 MW. Elimination of the oxide processing plant reduces water and reagent consumption, tailings generation and the footprint of processing and disposal facilities. Because these changes alter the configuration assessed under LP No. 2062/2025, they must be formally submitted to SEMAS/PA, and parts of the environmental assessment remain based on the originally licensed scenario.

A Conceptual Mine Closure Plan (PCFM) prepared in 2025 provides for progressive closure, including decommissioning of structures, slope reconfiguration, physical and chemical stabilization, revegetation, and post-closure monitoring and maintenance.

The QP, Ms. Branca Horta de Almeida Abrantes, concluded that no environmental, social or permitting factors have been identified that, individually or collectively, could materially affect the ability to develop and extract the Mineral Resources and Mineral Reserves, subject to compliance with legal requirements. Continuation of the Project depends on meeting the LP conditions, submitting the design changes to SEMAS/PA and obtaining the LI and LO.

1.21 Capital and Operating Costs

The Base Case total life-of-mine capital cost (CAPEX) is estimated at US$ 908.7 M, and CAPEX summary is presented in Table 1-5.

Table 1-5: Capital cost estimate summary – Base Case

CAPEX Item	US$ M
Pre-Production Mine Site and Plant	
Mining and pre-production	19.7
Process plant, mechanical equipment	280.6
Civil works, buildings and earthworks	51.6
Electrical, substation and distribution	65.3
Tailings storage facility (starter stage)	14.5
Water storage dam	15.2
Indirect costs (freight, imports, first fill)	30.2
Owner's costs	47.7
EPCM	71.6
Financial costs (insurance and taxes)	7.2
Subtotal – mine site and plant growth capital	603.6
Pre-Production Smelter	
Barcarena ZPE smelter – growth capital	181.3
Total Pre-Production CAPEX	784.9
Sustaining Capital (LOM)	
Mine site and plant sustaining capital	83.9
Smelter sustaining capital	14.3
Total Sustaining Capital	98.2
Closure	
Mine closure and rehabilitation (LOM)	25.6
Total CAPEX (LOM)	908.7

Note: all values include an embedded 15% contingency. Totals may not add due to rounding.
Source: GE21, Onix and Bravo, 2026; PFS financial model.

Total operating cost (OPEX) is estimated at US$ 4,124.2 M, and OPEX summary is presented in Table 1-6.

Table 1-6: Operating cost summary – Base Case

Description	US$ M (LOM)	Unit cost
Mining		
Ore mining	362.8	
Waste rock mining	2,039.3	
Marginal material mining	46.8	
Oxide / weathered material mining	67.3	
Grade control, geology and mine management	24.5	
Total mining cost	2,540.7	US$ 3.83/t moved
Processing		
Labour	55.1	
Power	124.0	

Description	US$ M (LOM)	Unit cost
Reagents (flotation and others)	159.3	
Laboratory	20.8	
Media and liners	197.9	
Plant maintenance supplies	59.3	
Plant maintenance services	65.2	
Equipment rental	15.6	
Tailings and water dams	37.9	
Other costs (10% of total)	68.2	
Total processing cost	803.3	US$ 9.26/t processed
Smelting (Barcarena ZPE)		
Energy – electricity and natural gas	70.9	
Consumables – electrodes, reductant, flux, oxygen, refractories	34.7	
Labour	46.3	
Maintenance – materials and contracted services	27.3	
Smelter G&A – insurance, environmental, legal, IT, travel	11.4	
Concentrate logistics – mine to ZPE	50.2	
Total smelting cost	240.8	US$ 164.20/t concentrate
General and Administration		
G&A – mine site	316.8	
G&A – Bravo Mining corporate / head office	84.0	
G&A – Bravo Metals ZPE entity	35.0	
Community and social investment	9.3	
Exploration expensed during operations	30.0	
Total G&A	475.1	US$ 5.48/t processed
Other		
Irrecoverable taxes (PIS/COFINS and ICMS) on operating costs	64.3	US$ 0.74/t processed
Total Operating Cost (LOM)	4,124.2	US$ 47.56/t processed

Note: totals may not add due to rounding.
Source: GE21, Onix and Bravo, 2026; PFS financial model.

1.22 Economic Analysis

The Base Case returns an after-tax NPV at an 8% real discount rate of US$ 1,448.7 M and an after-tax IRR of 35.1%, with a payback of 2.0 years from the start of production.

The analysis is a discounted cash flow model of after-tax, unlevered free cash flow, in constant 2026 US dollars, with a valuation date of 1 January 2028 and a mid-period discounting convention. The production schedule is based solely on Proven and Probable Mineral Reserves; Inferred Mineral Resources are excluded. The model consolidates the mining and smelting entities, and costs incurred before the valuation date are treated as sunk.

Over a 10-year life, the concentrator processes 86.7 Mt at an average head grade of 2.77 g/t PdEq, producing 1,466.7 kt of concentrate that is smelted at Barcarena into 146.7 kt of PGM-nickel alloy and 878.1 kt of sulphuric acid.

Payable production totals 3.94 Moz of 4E (Pd, Pt, Rh, Au), an average of 394 koz per year, and 98.2 kt of nickel. At the PFS price deck, equivalent to a 4E basket price of US$ 1,798/oz, gross revenue is US$ 9,322.9 M, of which 94% is from alloy. Net revenue after refining charges, alloy freight, royalties and production levies is US$ 8,346.4 M.

Table 1-7 presents the economic results summary.

Table 1-7: Base Case economic results

Item	Unit	Value
Returns		
After-tax NPV at 8% (real)	US$ M	1,448.7
After-tax IRR (real)	%	35.1
Payback period, post-tax, from start of production	years	2.0
Peak funding requirement	US$ M	877
NPV / peak funding	x	1.65
Project free cash flow (LOM, undiscounted, from Year 0)	US$ M	2,815
Revenue and Cost		
Gross revenue	US$ M	9,322.9
of which alloy	US$ M	8,752.1
of which sulphuric acid	US$ M	570.8
Refining charges, alloy freight and insurance	US$ M	201.9
Royalties and production levies	US$ M	774.6
Net revenue	US$ M	8,346.4
Net revenue per payable 4E ounce	US$/oz	2,119.5
Total operating cost	US$ M	4,124.2
EBITDA (LOM, after closure provision)	US$ M	4,196.6
EBITDA margin	%	50.3
C1 cash cost, net of by-product credits	US$/oz 4E	478
All-in sustaining cost (AISC)	US$/oz 4E	706
Capital		
Growth capital	US$ M	784.9
Sustaining capital	US$ M	98.2
Closure and rehabilitation	US$ M	25.6
Total capital	US$ M	908.7
Taxation and Royalties		
CFEM, TFRM, ANM fees and private royalties	US$ M	774.6
Income tax (IRPJ and CSLL) after SUDAM	US$ M	511.0
SUDAM benefit realised	US$ M	628.3

Note: C1 cash cost and AISC are non-IFRS measures. C1 comprises total operating costs (mining, processing, smelting, G&A and irrecoverable indirect taxes) net of nickel and sulphuric acid revenue, per payable 4E ounce; AISC adds royalties and production levies, sustaining capital and closure. Refining charges and alloy freight are netted in revenue and are in neither measure. Totals may not add due to rounding.
Source: Bravo and GE21, 2026; PFS financial model.

The Project is most sensitive to metal prices: a 10% change in prices changes the NPV by approximately US$ 455 M, compared with approximately US$ 215 M for operating costs and US$ 70 M for capital costs. The NPV remains positive with metal prices 30% below the Base Case. At discount rates between 5% and 12%, the NPV ranges from US$ 1,858.5 M to US$ 1,032.7 M.

The Base Case economics depend primarily on PGM prices, and also on securing the SUDAM incentive, the ZPE regime and acceptable alloy offtake terms.

1.23 Adjacent Properties

Two significant mineral deposits lie within 10 km of the Project, neither of which shares the geology of the Luanga mafic-ultramafic intrusion.

The Serra Pelada Au + PGM deposit, 8 km west of the Project, is held under a Mining Licence by Serra Pelada Companhia de Desenvolvimento Mineral. In the 1980s it was the largest gold mine in Brazil, worked by hand by tens of thousands of artisanal miners, with a recorded

production of 1.04 Moz Au. The Serra Leste high-grade hematite iron ore mine, approximately 8.5 km southwest of the Project, is an active iron ore operation held by Vale. Several minor gold occurrences, mostly worked by artisanal miners, also lie to the west.

There is no open ground for new exploration claims around Project tenement, and Vale is the principal holder of exploration claims in the region. The QP has not verified the information on adjacent properties, which is not necessarily indicative of the mineralization on the Project.

1.24 Other Relevant Data and Information

Three alternative development cases were evaluated in the PFS financial model alongside the Base Case:

- **Alternate Case A** keeps the Base Case mine and concentrator but stages the smelter, with first alloy in Year 3. Concentrate produced in the first two years is sold to third-party smelters on indicative terms. Alternate Case A was developed to a PFS level of confidence.
- **Alternate Case B** applies the Base Case configuration at 5 Mtpa, recovering the same Mineral Reserve over 20 years with lower up-front capital. It assumes renewal of the SUDAM incentive, which the Base Case does not. Alternate Case B was developed to a PEA level of confidence.
- The **Opportunity Case** applies the Base Case modifying factors to the full Mineral Resource model. Its schedule contains 123.3 Mt, of which 27% is Inferred, extending the mine life by about three years and adding approximately US$ 330 M to the NPV. No allowance was made for the additional TSF, waste rock storage and closure capacity that the larger tonnage would require. Alternate Case B was developed to a PEA level of confidence.

Alternate Case B and the Opportunity Case are based on the Association for the Advancement of Cost Engineering (AACE) Class 5 estimates. The Opportunity Case includes Inferred Mineral Resources, which are considered too speculative geologically to have economic considerations applied to them that would allow them to be categorized as Mineral Reserves. There is no certainty that the Opportunity Case will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 1-8 presents the summary results for the base case and the developed alternate cases.

Table 1-8: Comparison of the Base Case and alternative development cases

Indicator	Base Case	Alternate Case A	Alternate Case B	Opportunity Case
Description	Mine, concentrator and smelter built concurrently	Base Case mine and concentrator; staged smelter, first alloy in Year 3	Base Case configuration at 5 Mtpa	Base Case modifying factors applied to M, I and Inferred
Study class (AACE)	Class 4	Class 4	Class 5	Class 5
Study status	PFS	PFS	PEA	PEA
Mine life (years)	10	10	20	13
Ore processed, LOM (Mt)	86.7	86.7	86.8	123.3
Waste mined, LOM (Mt)	575.9	575.9	544.0	781.9
Strip ratio (t/t)	6.64	6.64	6.27	6.34
Peak mining rate (Mtpa)	81.4	81.4	41.6	104.0
Peak plant throughput (Mtpa)	10.03	10.03	5.00	10.00
Payable Pd, LOM (koz)	2,088.9	2,031.8	2,069.8	2,912.1
Payable Pt, LOM (koz)	1,507.5	1,463.9	1,495.2	1,985.2

Indicator	Base Case	Alternate Case A	Alternate Case B	Opportunity Case
Payable Rh, LOM (koz)	160.7	155.7	159.6	207.1
Payable Au, LOM (koz)	180.9	174.6	181.0	229.4
Payable Ni, LOM (kt)	98.2	93.7	99.2	128.0
Average annual payable 4E (koz)	393.8	382.7	195.3	410.3
Growth capital (US$ M)	784.9	603.6	491.1	784.9
Sustaining capital, LOM (US$ M)	98.2	265.2	98.2	98.2
Closure and rehabilitation (US$ M)	25.6	25.6	25.6	25.6
Total capital, LOM (US$ M)	908.7	894.4	614.9	908.7
After-tax NPV8 (US$ M)	1,448.7	1,213.5	1,151.2	1,777.1
After-tax IRR (%)	35.1	31.5	30.6	32.0
Payback, post-tax (years)	2.0	2.5	2.5	2.5
Peak funding (US$ M)	877	718	547	948
C1 cash cost (US$/oz 4E)	478	526	488	509
AISC (US$/oz 4E)	706	797	718	729

Note: sustaining capital excludes closure, which is shown separately. In Alternate Case A, sustaining capital includes the US$ 181.3 M smelter capital spent after the start of production. Payback is from the start of production. Alternate Case B and the Opportunity Case are PEAs; see the cautionary statement in Section 24.1.1.
Source: Bravo and GE21, 2026; PFS financial model.

The PFS identifies the following opportunities, none of which is included in the Base Case:

- Mineral Resource growth through the 2026 programme of 28,000 m of drilling, with mineralization open at depth along the 8.1 km strike of the Complex and a Mineral Resource update targeted for Q1 2027;
- A copper credit, with copper recoveries of 89% to 95% to concentrate, potentially worth approximately US$ 290 M to US$ 430 M over the life of mine;
- Spare smelter capacity (200 ktpa against a peak of about 170 ktpa), allowing a lower concentrate grade target for higher recovery, or toll treatment of third-party concentrates;
- An Owner-operated or hybrid mining fleet, which could reduce mining unit costs by up to 43%, for additional capital;
- Oxide Mineral Resources of 10 Mt at 1.51 g/t PdEq (Measured and Indicated), that would be already mined and stockpiled, subject to further metallurgical testwork.

Beyond resource extension in the Central, North and Southwest Sectors, drilling has intersected magmatic Ni-Cu-PGM massive sulphide at the LU330 and Babylon targets, and PGM mineralization at the Crescent Zone. The Luanga South Complex, with 6.5 km of the same stratigraphy, has not yet been explored by Bravo. A property-wide HeliTEM survey also defined a pipeline of copper-gold targets, including the high-grade T5 discovery within approximately 1 km of the deposit. These targets are conceptual in nature, and no Mineral Resource has been estimated for them.

1.25 Interpretation and Conclusions

Based on the assumptions and parameters presented in this Report, the PFS study shows positive economics, with an after-tax NPV at 8% real discount rate of US$ 1,448.7 M and an after-tax IRR of 35.1%, with a payback of 2.0 years from the start of production.

The QPs conclude that the exploration data, Mineral Resource and Mineral Reserve estimates, and engineering and cost inputs are adequate to support this Pre-feasibility Study. The Base Case shows positive economics and is resilient to cost variations. The Project is most exposed to PGM prices, the availability of fiscal incentives, and the execution and permitting of

the Barcarena smelter. None of the risks identified is considered a fatal flaw, are typical of projects at this stage and can be reduced through the work recommended in Section 26.

The QPs conclude that the results support advancing the Project to a Definitive Feasibility Study.

1.26 Recommendations

Table 1-9 summarizes the recommendations described in this section, and the corresponding estimated budgets proposed to advance the project through the DFS stage.

Table 1-9: Proposed budget to advance the Project to the Definitive Feasibility Study stage

PHASE 5A – Metallurgical testwork at Luanga	
Completion of any outstanding metallurgical testwork and optimization:	
Estimate	US$ 0.50 M
Continuation of carbon sequestration study:	
Estimate	US$ 0.05 M
Continued Metallurgical testwork and optimization	
Estimate	US$ 0.20 M
Sub-total – Phase 5A	**US$ 0.75 M**
PHASE 5B – Pre-feasibility Study (PFS) following favorable results from a PEA	
Completion of a PFS:	
Estimate	US$ 1.0 M
Sub-total – Phase 5B	**US$ 1.0 M**
PHASE 5C – Deep Drilling below the Luanga PGM+Au+Ni deposit	
Deep drilling at the Luanga PGM+Au+Ni deposit.	
8 holes @ ~500 m = 4,000 m @ US$ 450/m	
Estimate	US$ 1.8 M
Sub-total – Phase 5C	**US$ 1.8 M**
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$ 0.1 M
Drilling: 70 holes x 200 m for 14,000 m @ US$ 400/m (all inclusive)	US$ 5.6 M
Sub-total – Phase 5D	**US$ 5.7 M**
PHASE 5E – Deep Drilling outside the Luanga PGM+Au+Ni deposit	
Deep drilling at the Luanga PGM+Au+Ni deposit.	
3 holes @ ~800 m = 2,400 m @ US$ 450/m (all inclusive)	US$ 1.08 M
Sub-total – Phase 5E	**US$ 1.08 M**
PHASE 5F – Extension and Infill Drilling at Luanga	
Drilling to convert existing MRE Inferred resources to MRE Indicated resources, and define new MRE Inferred Resources:	
Drilling: 33 holes x 300 m for 9,900 m @ US$ 400/m (all inclusive)	US$ 3.96 M
Sub-total – Phase 5F	**US$ 3.96 M**
TOTAL – PHASE 5	**US$ 14.29 M**
PHASE 6 – Definitive Feasibility Study (DFS) following favorable results from a PFS	
Completion of a FS, including any required infill drilling, geotechnical drilling:	
Estimate	US$ 5.0 M
TOTAL – PHASE 6	**US$ 5.0 M**

Source: GE21, 2026.

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.

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 commissioned GE21 to prepare a Pre-feasibility Study (PFS) Technical Report for the Luanga Project in Pará, Brazil, in accordance with the requirements of NI 43-101.

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.

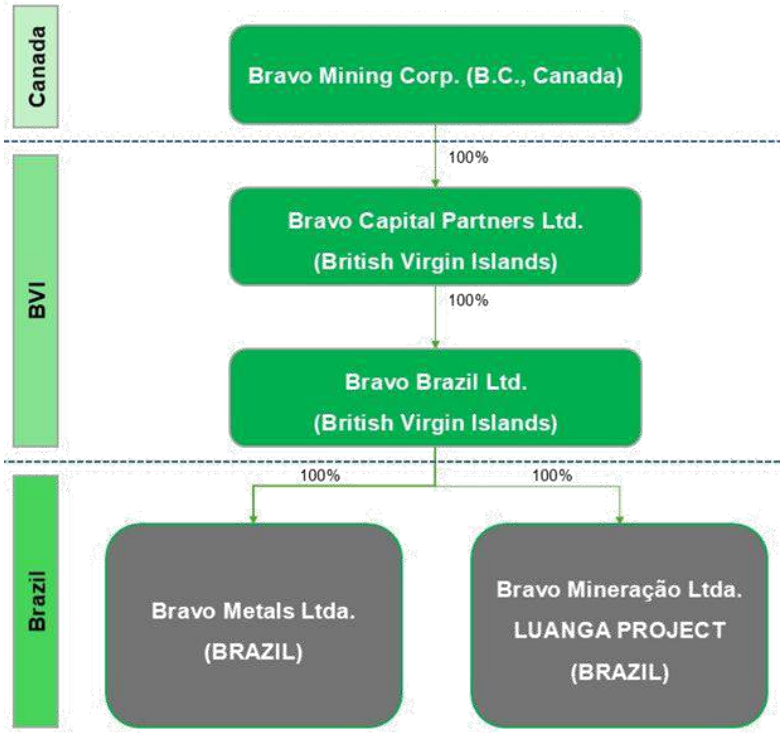


Figure 2-1: Bravo organization chart

Source: Bravo, 2025.

2.1 Terms of Reference

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

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.

The effective date of this Report is September 16, 2026, based on the receipt date of the Project database, while the effective date for the Mineral Resource Estimate within this Report remains as February 18, 2025.

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 Qualified Persons (QPs) 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 Authors' relationships with Bravo are 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. Rodney Jones, and Mrs. Branca Horta.

Mr. Porfirio Cabaleiro Rodriguez is one of the QPs regarding the objectives of this Report. Mr. Porfirio is responsible for Sections 14, 15, 16, 19, 21, 22, 23 and 24, with co-responsibility for Sections 1, 18, 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, mineral resourses and reserves, and mining projects.

Mr. Bernardo Viana is also on of the QPs responsible for portions of this Report. Mr. Viana is responsible for sections 2, 3, 4, 5, 6, 7, 8, 9, 10, 11, 12 and co-responsible for Sections 1, 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 to be 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 a QP responsible for portions of this Report. Mr. Moreira is responsible for Section 13, with co-responsibility for Sections 1, 17, 18, 25, 26 and 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. Rodney Trevor Jones is also a QP responsible for portions of this Report. Mr. Jones was co-responsible for Sections 1, 17, 25, 26 and 27. Mr. Jones is an independent consulting pyrometallurgist, and FSAIMM, with over 40 years of pyrometallurgical experience in a wide variety of processes that have been taken from concept to industrial implementation, included in this time is a period of approximately twenty years focused on the production of platinum group metals.

Mrs. Branca Horta is also a QP responsible for portions of this Report. Mrs. Horta is responsible for Section 20, with co- responsibility for Sections 1, 25, 26 and 27. Mrs. Horta is a professional environmental consultant, with over 19 years of experience in the mining industry, geography and environmental analysis. Mrs. Horta's experience relevant to this Report includes environmental studies, permitting, and planning.

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

Table 2-1: Report sections and assigned QP responsibilities

Qualified Person	Company	Site Visit Date	Responsible for Sections of this Report
Porfirio Cabaleiro FAusIMM | FAIG	GE21	July 4 to 7, 2023; October 3 to 6, 2023; January 27 to 31, 2025; February 10 to 13, 2026.	14, 15, 16, 19, 21, 22, 23, 24 and co-responsibility for 1, 18, 25, 26 and 27
Bernardo Viana FAIG	GE21	October 3 to 6, 2023; January 27 to 31, 2025.	2, 3, 4, 5, 6, 7, 8, 9, 10, 11, 12, and co-responsibility for 1, 25, 26 and 27
Paulo Bergmann FAusIMM	GE21	-	13, and co- responsibility for 1, 17, 18, 25, 26 and 27
Rodney Jones FSAIMM	Independent Consultant	-	Co-responsibility for Sections 1, 17, 25, 26 and 27
Branca Horta MAIG	Harpia	-	20, and co- responsibility for 1, 25, 26 and 27

Source: GE21, 2026.

2.4 Site Visits and Details of Inspection

Mr. Rodriguez visited the property from July 4 to 7, 2023, October 3 to 6, 2023, January 27 to 31, 2025, and February 10 to 13, 2026.

Mr. Viana visited the property from October 3 to 6, 2023, and January 27 to 31, 2025.

During the site visits, 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.

Mr. Moreira, Mr. Jones and Mrs. Horta did not visit the property.

2.5 Effective Dates

The effective date of this Report is September 16, 2026. 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 PFS 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 requirements of NI 43-101.

GE21 also conducted a Preliminary Economic Assessment (PEA) for Luanga Project. This technical report is titled “NI 43-101 Preliminary Economic Assessment (PEA) Independent Technical Report for the Luanga PGM+Au+Ni Project Pará, Brazil” dated November 28, 2025 with an effective date of July 7, 2025. This Report supersedes the previously issued PEA technical report, which should no longer be relied upon.

The current PFS uses the February 18, 2025 MRE as a foundation for evaluating the potential economic viability of the Project.

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$) unless otherwise indicated. An exchange rate of Brazilian reais (R$) 5.50/US$ 1.00 was used based on parameters adopted in international projects.

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

3 RELIANCE ON OTHER EXPERTS

The Authors of this Report are QPs, as defined under NI 43-101, with relevant experience in mineral exploration, data validation, mineral resources and reserves estimation, metallurgy, and capital and operating cost 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 April 2, 2026, the Luanga Mineral Rights were valid and in good standing at that time.

4 PROPERTY DESCRIPTION AND LOCATION

Luanga is an advanced-stage mineral project 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).

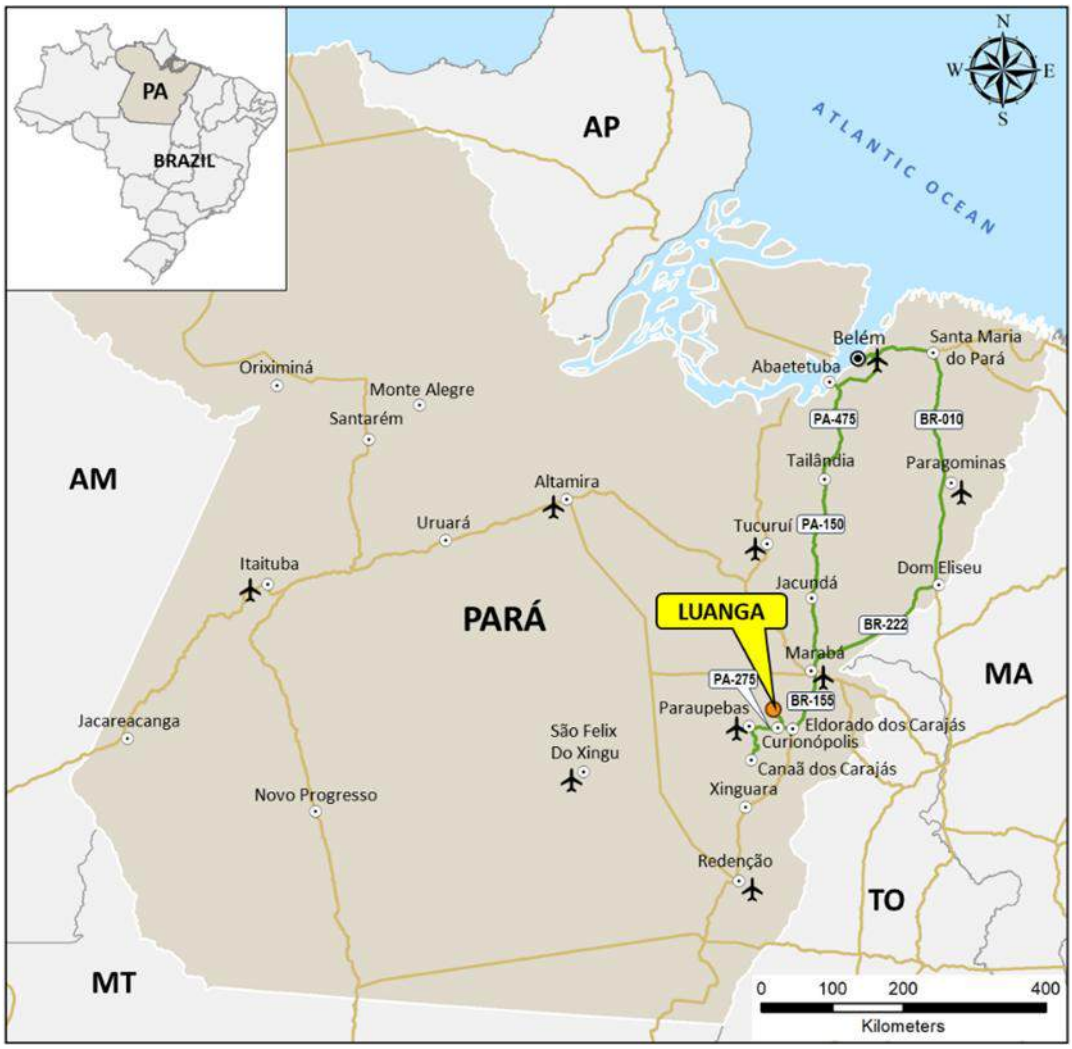


Figure 4-1: Regional location of Luanga Project in Pará State, Brazil

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.

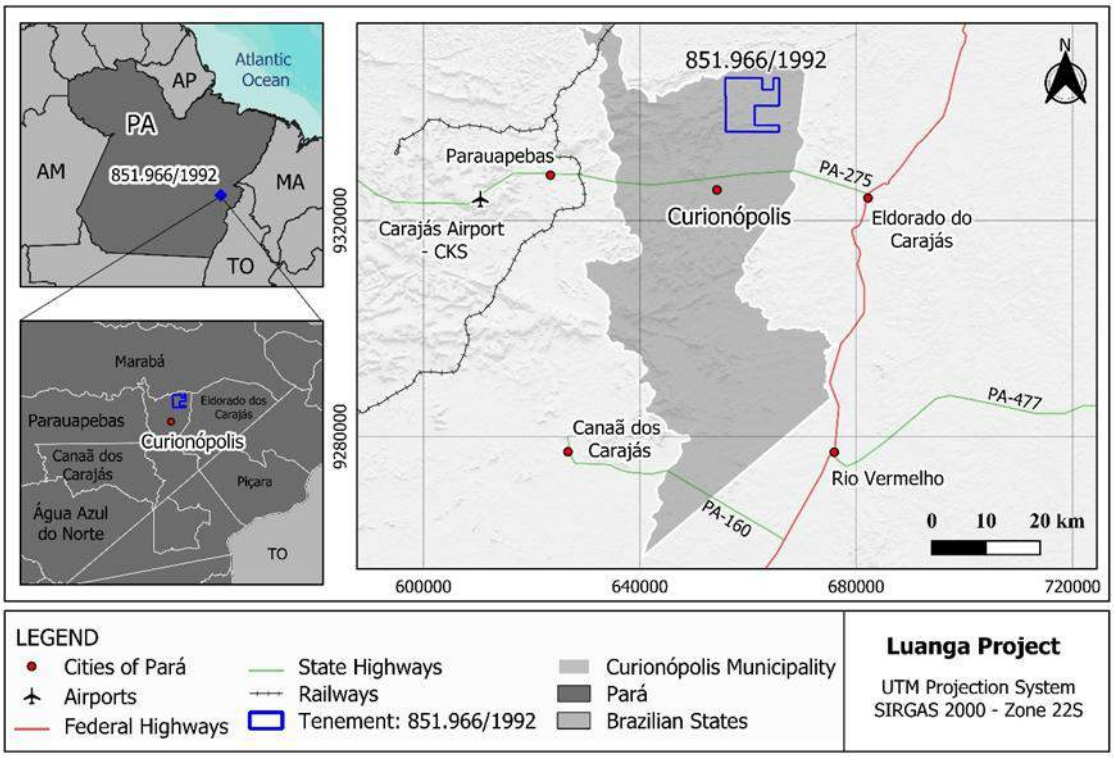


Figure 4-2: Luanga Project location map

Source: GE21, 2025.

4.1 Mineral Tenure

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

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

Table 4-1: Mineral tenement summary

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

Notes:
Comments: Mining Licence pending.
ANM = National Mining Agency.
Source: ANM, 2026.

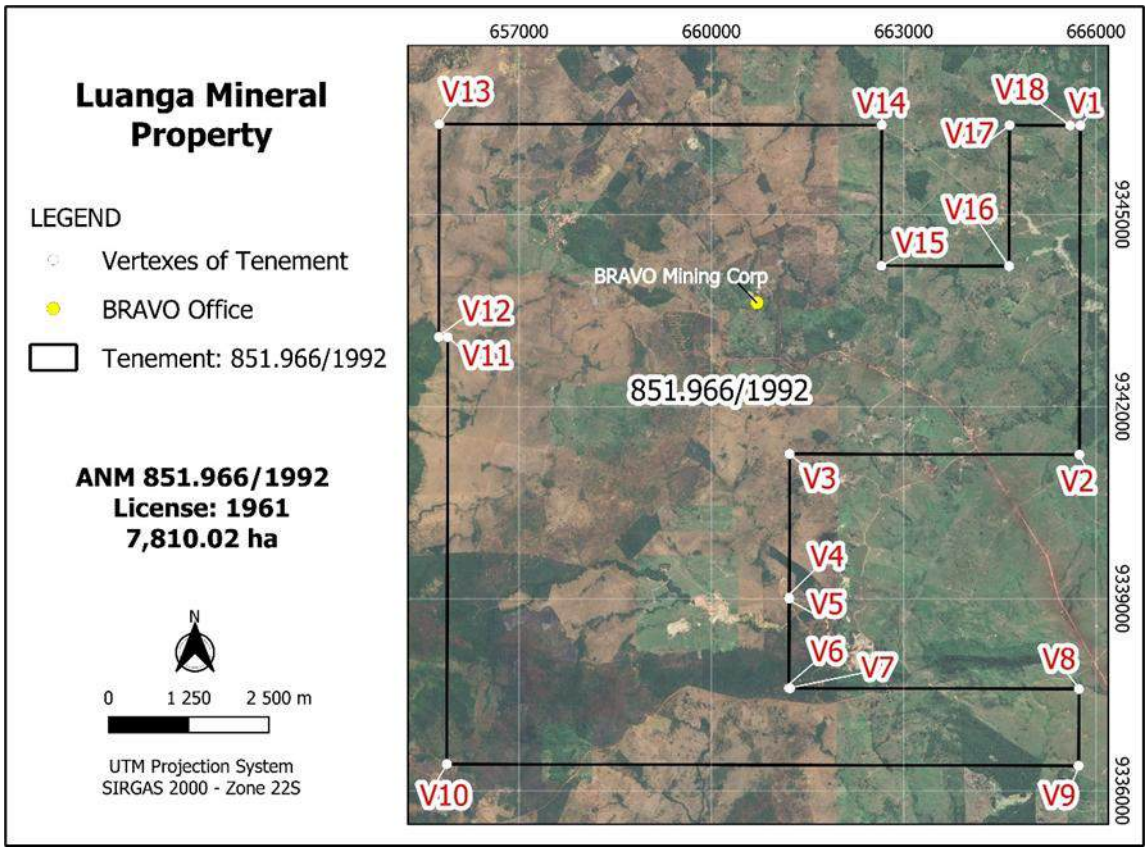


Figure 4-3: Luanga Project tenement map

Source: GE21, 2025.

The Luanga mineral property is centred at coordinates -05º57'24.34" S/-49º32'51.00" W. Bounding coordinates of Exploration Licence 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

Note: Exploration Licence Nº 1961, ANM.851.966/1992 – Datum SIRGAS 2000
Source: ANM, October 2023.

Although the original exploration licence 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 Licence 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.

- USD 300 k paid on 7th December 2021.
- USD 500 k paid on 9th November 2022.
- USD 500 k paid on 19th November 2023.
- Total: USD 1.3 M

On 24th January 2022, Bravo´s wholly owned subsidiary acquired 100% of the shares owned by 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 Licence. The current land use at the Project is solely agricultural cattle grazing. There are no significant rivers within the property. As of the date of this Report, Bravo has planted more than 50,000 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 concurrently rehabilitates 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).
- 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 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 Report.

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 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.

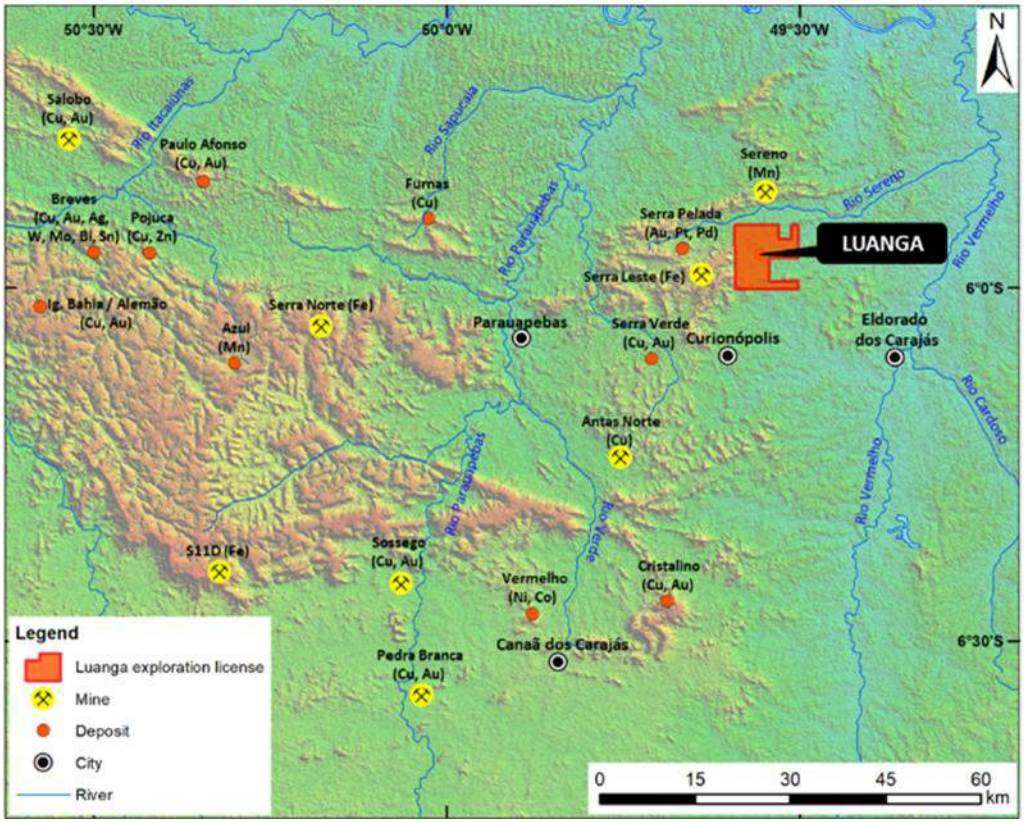


Figure 5-1: Physiography of Carajás region

Source: GE21, 2023.

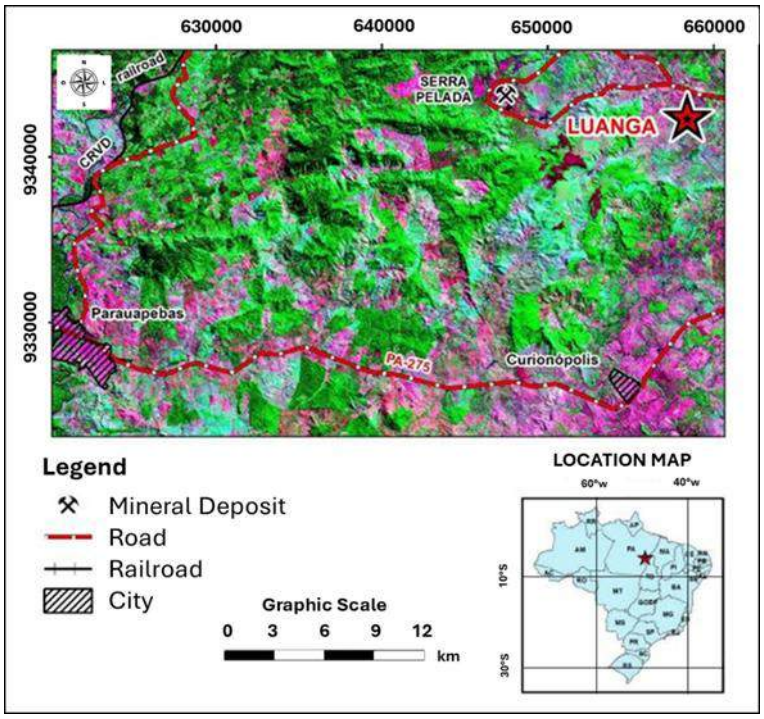


Figure 5-2: Sat image (RGB composition 342) with relief and vegetation of Carajás region

Source: GE21, 2025.

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.

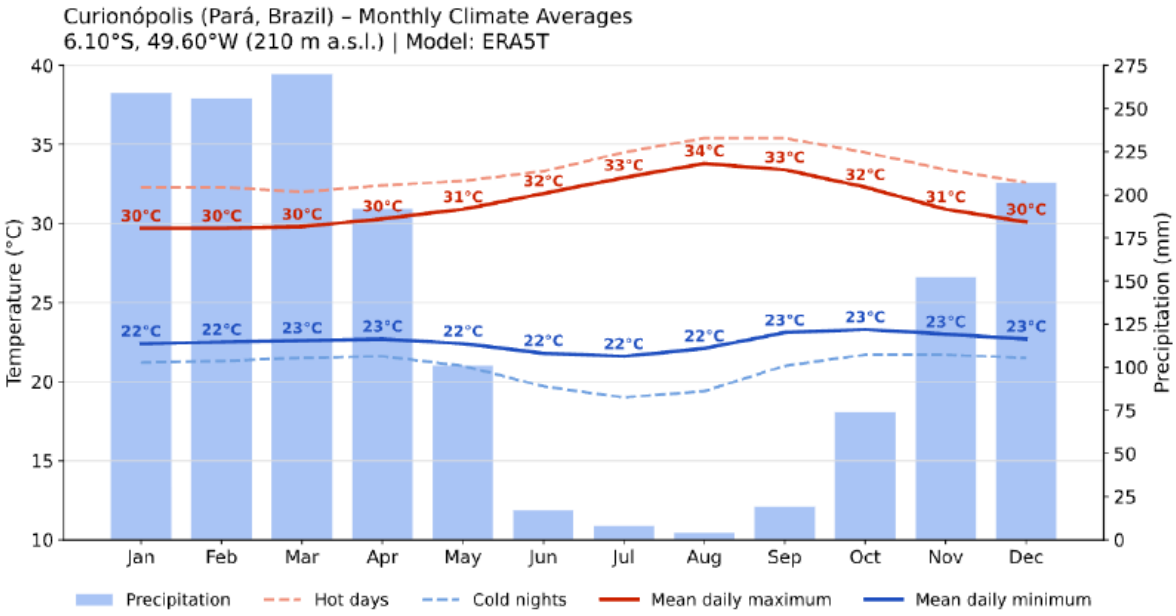


Figure 5-3: Average monthly temperature and rainfall at Curionópolis

Source: Meteoblue.com, 2025, adapted by GE21, 2026.

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).

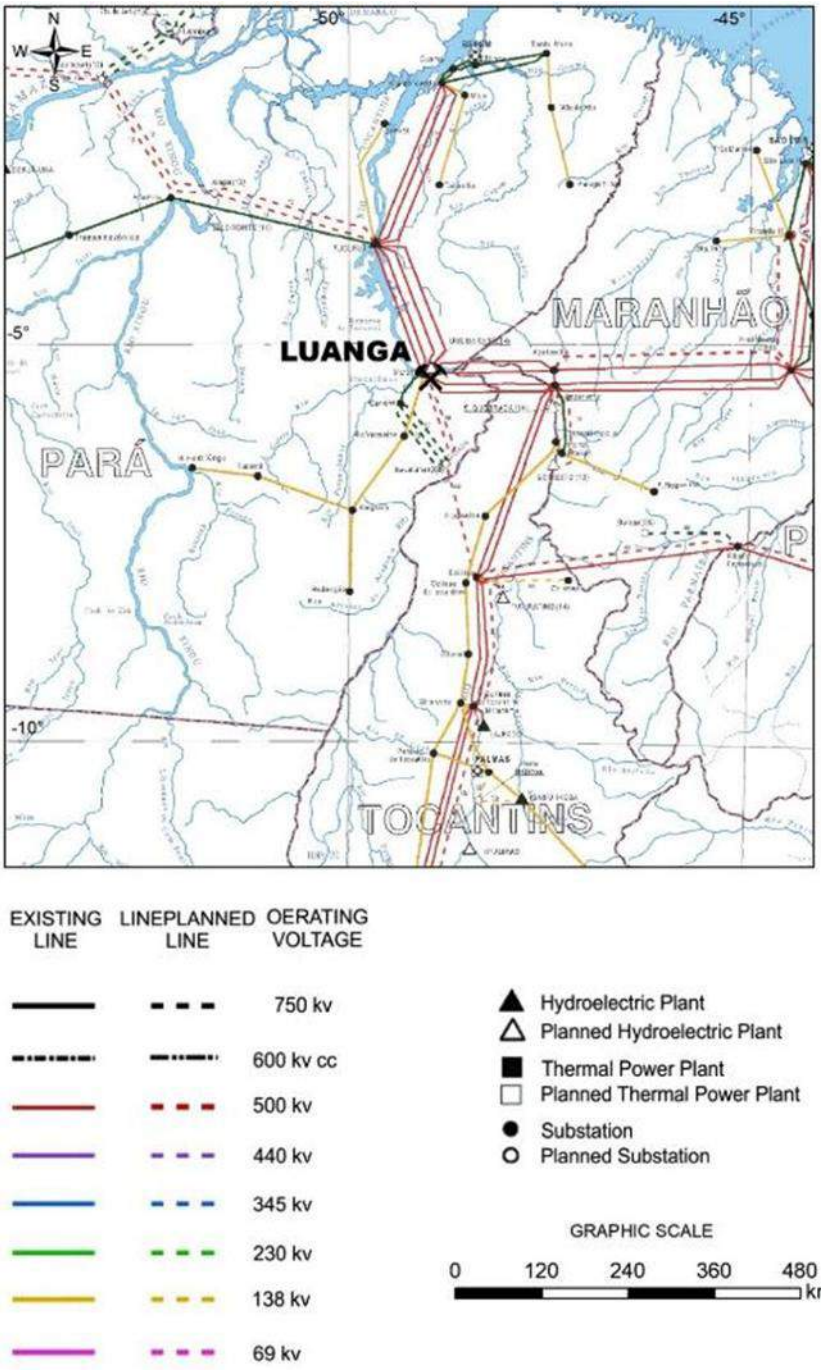


Figure 5-4: Power transmission lines in the region of Luanga Project

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 56,000 trees (or approximately 124 trees per hole drilled). As of the Effective Date, the Company has over 17,000 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 Carajás Mineral Province (CMP)

The history of the Carajás region is essential to contextualize the discovery and subsequent evaluation of the Luanga deposit. Until the 1960s, geological work in the Carajás region was restricted by limited access.

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 at Serra do Sereno. In July 1967, Companhia Meridional de Mineração (CMM) discovered high-grade iron ore averaging 66% Fe. In April 1970, the Brazilian Government created a joint venture (JV) company, Amazônia Mineração S.A. (AMSA), led by Companhia Vale do Rio Doce (CVRD, currently Vale), which owned 51%, with the remaining 49% owned by CMM.

In 1977, Vale bought from CMM the rights for mineral exploration and development of the entire Carajás Mineral Province (CMP).

6.2 Luanga Project

Mafic-ultramafic rocks of the Luanga Complex were identified in 1983 during regional exploration carried out by Rio Doce Geologia e Mineração (DOCEGEO) in the Serra Leste region. After discovering up to 2 metrethick chromitites, DOCEGEO conducted geological mapping, a soil geochemistry survey (400 × 40 m grid), and a ground magnetic survey in the Luanga Complex. Four diamond drill holes were drilled, but drilling results were not favourable for chromium mineralization, although they intersected anomalous concentrations of Pt and Pd, including 9 m at 2.57 ppm of Pt + Pd (drill hole PPT-LUAN-FD0004).

In 1997, a JV between DOCEGEO and Barrick Gold conducted a stream-sediment campaign over the Luanga Complex area, which identified Au anomalies.

In 2000, Vale conducted a new soil geochemical survey to test the Au anomalies outlined by Barrick Gold. The sampling grid defined a 1 km-long trend of Pt and Pd anomalies, followed by an additional soil geochemistry survey in the northern portion of the Luanga Complex, which identified another 1 km-long Pd and Pt anomalous trend. The geochemical survey was extended, adding a further 2 km, now forming a continuous Pt-Pd anomalous trend along the entire layered intrusion.

In 2001, Vale started a systematic geological and structural mapping using integrated RADARSAT and Landsat-TM5 data, along with an airborne geophysical survey, which led to the discovery of several additional layered mafic intrusions.

6.3 Vale/DOCEGEO Drilling

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

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 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.

Table 6-1: DOCEGEO drilling summary

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

Legend: DDH = diamond drill hole.
Source: Bravo, 2025.

6.4 Historical Metallurgical Test Work

The development of the mineral processing strategy for the Luanga Project represents a fundamental technical and strategic evolution, transforming a conventional concept into a processing solution tailored to the complexity of the deposit. This section summarizes the historical metallurgical test work and its role in reducing process risk.

The transition from a complex, reagent-intensive flowsheet to Jameson Cell technology reflects the maturing of mineralogical understanding, which has enabled the Project to evolve to a level of high energy efficiency, superior physicochemical selectivity, and improved process performance. Luanga's technical trajectory consists of successive milestones: the definition of the historical baseline by Vale, the "technological narrowing" program led by Bravo, and the final validation at a mini-plant scale at BML (Canada). Each phase refined the ore's response to the critical challenge of talc gangue and the mandatory need for ultrafine grinding to liberate platinum group metals (PGMs).

The following analysis details this evolution, beginning with the premises established during the Vale management phase.

6.4.1 Vale Phase (1992–2020)

6.4.1.1 Mineral Exploration and Drilling (1992–2003)

Although the rocks of the Luanga Complex were identified in 1983, intensive exploration took place between the 1990s and the early 2000s.

- Drilling Campaigns: Vale conducted systematic campaigns, totaling 256 diamond drill holes (~50,787 m).
  - 1992: 4 initial holes (643.69 m).
  - 2001: 86 holes (14,584.35 m) focused on the Luanga target.
  - 2002: 71 holes (15,423.25 m).
  - 2003: 95 holes (20,135.45 m), including holes for metallurgical testing and exploration at the Luanga South target.
- Geochemical Surveys: In 2000, a new soil geochemical survey identified a trend of platinum (Pt) and palladium (Pd) anomalies approximately 1 km long in the southern portion, which was later extended to a continuous trend throughout the entire complex.

6.4.1.2 Experimental Development and Metallurgy (2002–2007)

Metallurgical test work began in 2002, with the aim of defining the processing route for sulfide ore (fresh rock).

- Key institutions and studies:
  - CABRI (Canada, 2003): Conducted detailed mineralogical characterization studies.
  - MINTEK (South Africa, 2002): Conducted flotation and characterization tests on a laboratory and mini-plant scale, based on routes used in South African platinum group metal plants.
  - SGS Lakefield (Canada, 2003–2004): Locked-cycle flotation tests to validate the processing scheme.
  - HDK Engenharia (2003): Preliminary sizing of the grinding circuit.
- Historical Results: Vale's tests confirmed the technical feasibility of producing a commercial Ni-Pt-Pd-Au concentrate, achieving average grades of 83.58 g/t (3E/4E) and PGM + Au recoveries of approximately 73.8%. However, the flowsheet was considered complex and associated with high operating costs at the time.

6.4.1.3 Regulatory Steps and Mining Tenements

Vale proceeded with the regularization of the asset with the National Mining Agency (Agência Nacional de Mineração – ANM) under process no. 851.966/1992.

- Final Exploration Report: Following the renewal of the licence in 2005, Vale submitted the Final Exploration Report to the ANM (former Departamento Nacional de Produção Mineral – DNPM) in April 2008.

- Mining Licence Application: In April 2013, the company filed the “Mining Licence Application.” The ANM proceeded with the technical review in May 2018, requesting additional documents that were submitted in September of that year.

6.4.1.4 Transition to Bravo Mining (2020–2022)

The process of transferring the asset from Vale to Bravo took place in strategic stages:

- June 2020: Vale, FFA Holding, and Brazil Americas Investments (BAIP) signed an acquisition agreement, with Bravo as the beneficiary.
- September 2020: The Bravo team conducted its first fieldwork, collecting samples of historical drill core stored at Vale’s facilities in Parauapebas for validation.
- January 2022: Bravo consolidated a 100% interest in the Project after acquiring the shares owned by Americas Investments & Participation Ltd (AIPL).
- March 2022: Bravo formally began its own systematic drilling and exploration program at the site.

6.4.1.4.1 Limitations of the Conventional Process Route

The proposed process route was based on an intensive two-stage grinding and flotation scheme. The re-engineering of the process route now recognizes that the original configuration exhibited low selectivity and extreme sensitivity to mineralogical alteration. The use of massive doses of reagents in a non-staged manner resulted in excessive mechanical mass entrainment, making the process vulnerable to grade variations and the presence of altered materials (saprolites). From a strategic consulting perspective, Vale’s process route resulted in a project with marginal profitability due to high operating costs (OPEX) and concentrate instability.

6.4.1.5 Summary of Metallurgical Results (Vale Phase)

Table 6-2 summarizes the metallurgical results of the Vale Phase.

Table 6-2: Metallurgical results (Vale Phase)

Parameter	Cleaner III Concentrate (Lakefield)	Average Concentrate (Historical)	Best Test (Mintek/Vale)
3E/4E Feed	-	~3.0 g/t	4.8 g/t
Ni Grade (%)	8.98%	4.52%	-
Pt grade (g/t)	35.10	34.10	-
Pd grade (g/t)	51.90	47.30	-
Au grade (g/t)	3.44	2.18	-
3E/4E grade (g/t)	90.50	83.58	123.00
3E + Au Recovery (%)	-	-	73.80%

Source: Bravo, 2026.

Louis Cabri’s research (2003) provided the conclusion for the paradigm shift: the degree of PGM liberation was the limiting factor. Conventional grinding was insufficient, demonstrating that Luanga’s success would depend on aggressive particle size reduction to expose ultrafine particles trapped in the mineral matrix. This technical barrier paved the way for the technological restructuring led by Bravo.

6.4.1.5.1 General Comments on the Vale Phase

The initial phase, led by Vale (then CVRD), established the project's historical baseline through mineralogical characterization (QemSCAN) and flotation campaigns at institutions such as Lakefield (Canada) and Mintek (South Africa). These studies were crucial for identifying the association of platinum group metals (PGMs) with sulphides (pentlandite) and nickel silicates.

6.4.2 Bravo Metallurgical Development

Upon taking over the Project, Bravo adopted a "technological narrowing" strategy, prioritizing physicochemical selectivity over high reagent dosages. Technologies ineffective for the prevaling mineralogy, such as gravity concentration, flotation columns, and CavTube, were discarded after tests indicated inability to efficiently work the in the high talc environment due to talc's natural hydrophobicity and prevalence to entrainment and report to the concentrate stream.

6.4.2.1 Multi-laboratory Testing Program

- CETEM: CETEM developed the "Bravo Route." The focus was on reagent optimisation, notably the introduction of the Floticor (Clariant) dispersant, which enabled a reduction of up to 47% in CMC consumption. The talc depression regime was set with PE26 (CMC) in the critical range of 450–780 g/t to control mass entrainment without suppressing the kinetics of slow-floating sulphides.
- CIT/SENAI: The pilot tests processed 3.8 metric tons of material. Phase 1 (2.55 metric tons) validated the enrichment potential (185 g/t 4E), while Phase 2 (1.30 metric tons) confirmed the stability of the continuous cleaner circuit. A notable result was the rhodium (Rh) performance, increased by 15% during Phase 2, reinforcing the project's metallurgical value.
- BML (Canada): This stage marked the transition to Jameson technology. BML used three-stage dilution tests (3SD) as a necessary precursor to parameterize the L150 mini-plant. The Jameson Cell validated the capture of fine particles (<30 µm) and the rejection of talc, achieving grades of up to 204 g/t 3E in the Central Zone.

6.4.2.2 Notable Comparative Results

In total, more than 330 tests (bench-scale, pilot-scale, and mini-plant) were conducted at three institutions, which supported the project's progress and confirmed the feasibility of producing concentrate to commercial specifications (>80 g/t 3E/4E and Ni >4%), with operational stability and recoveries in the range of >81% for PGM and >70% for Ni (Table 6-3).

Table 6-3: Notable comparative results

Institution	Test ID	Objective	4E Grade (g/t)	Recovery (%)
CETEM	TC-01	Bravo Route Validation	82.00	74.7%
CIT/SENAI	Pilot I (Sample E)	Maximum Enrichment	185.03	89.5% (Rougher)
BML	LCT34 (Central)	Locked Cycle (Conv.)	201.97	68.0%
BML	T20B (North)	Jameson L150 (Scalper)	136.14	18.1%
BML	T20B (North)	Jameson L150 (Global)	15.85	84.5%

Source: Bravo, 2026.

6.4.3 Oxidized Ore (Saprolite)

Since acquiring the Luanga Project in 2022, Bravo has identified opportunities to extract value from the saprolite and has initiated a metallurgical process for its beneficiation. This development was concentrated at Testwork in Nova Lima (MG), with an initial focus on hydrometallurgical process routes. The studies progressed from exploratory carbon-in-leach (CIL) tests—which indicated promising recoveries of gold and palladium (>70%)—to addressing challenges related to the solubilization of platinum and rhodium and high reagent consumption.

The following are the milestones of this development at Testwork:

- Phases I and II (2022–2024): Confirmed the technical feasibility of cyanidation but revealed high geometallurgical variability among holes and trenches.
- Optimization and Alternatives (2024–2025): Given the mineralogical complexity, the studies sought to reduce CAPEX and OPEX through processes such as vat leaching (leaching in tanks or pools) and heap leaching. To make these options feasible, agglomeration tests were conducted using cement, lime, and gypsum to ensure the physical stability of the material.
- Innovation and Reagents: Alternative methods were tested, such as the use of glycine to reduce cyanide consumption and the carbon ashing process (incineration of loaded carbon), proposed to produce metal briquettes.

6.4.4 Strategy for the Pre-feasibility Study (PFS)

Despite encouraging advances, Bravo has not included oxidized material within the current scope of the PFS. This decision is based on current limited evidence of economic viability compared to the fresh rock mineralisation. The strategy for the PFS is therefore to stockpile the saprolite making it available for future treatment as and when viable.

6.4.5 Current Status and Definition of the Fresh Rock Processing Route

The analysis of the multiple testing programs has resulted in a highly efficient metallugical flowsheet configuration achieving good recovery of fine particles while controlling the talc gangue.

6.4.5.1 Talc and Geometallurgy

Talc can account for up to 60% of the ore mass in some areas and is a major factor affecting selectivity. Grinding to a particle size between 28 and 34 µm is required for efficient liberation of the PGM and pentlandite particles. Ore mineralolgy variability dictates that the flowsheet necessary to handle this hydrophobic gangue requires both physio-mechanical and chemical/reagent regime flexibility for optimum performance across the whole deposit.

6.4.5.2 Comminution

Tests conducted by Metso and BML identified a critical heterogeneity in hardness. While the average Bond Work Index (BWI) is 17.4 kWh/t, the Central Zone reaches 20.5 kWh/t. This

value represents the Project's maximum energy intensity/consumption and dictates the sizing of the comminution equipment to achieve the target grind size.

6.4.5.3 Mineralogical Sensitivity and Grade Protection

The PFS focuses on the fresh rock and is where the Jameson technology demonstrates superior performance. Selectivity using the Jamison flotation cells has been demonstrated to be effective in the hydrosilicate gangue environment. Selectivity over gangue mineralogy is key to achiving acceptable concentrate grades. Therefore ore blending is recommended as a "geometallurgical safeguard" to reduce variances in the mill feed hydrosilicate concentration. Blending will provide a feed with controlled hydrosilicate levels, providing greater assurance of producing and maintaining target concentrate grades.

6.4.5.4 Thickening and Filtration

Westech tests confirmed that the extremely fine particle size (P80 ~25 µm) poses challenges for dewatering. The selection of filter media requires fabrics with a mesh opening of 7 µm to prevent the loss of suspended solids and ensure clarification of the recovered filtrate waters.

6.4.5.5 Dewatering Parameters (Westech – Summary Values)

Table 6-4: Dewatering parameters

Parameter	Tailings sample	Concentrate sample
Solids Density (t/m³)	2.50	2.86
Cake Moisture (%)	32.0%–33.8%	35.0%
Filtration Rate (t/h·m²)	0.440	0.700
Underflow Solids Concentration	46.85%	45.81%

Source: Bravo, 2026.

6.4.5.6 Justification for Jameson Technology

The choice of the Jameson Cell is based on its high-intensity downcomer mechanism, which maximizes surface energy to capture particles <30 µm—where conventional cells are less efficient due to lower bubble–particle collision probability. The wash water system creates a physical barrier that rejects talc through mechanical entrainment. Published literature indicates that the technology can reduce the plant's footprint by 76% and energy consumption by 59%, providing meaningful operational and economic upside.

6.4.6 Conclusion

Luanga's metallurgical history documents the transition from conventional flotation cell designs to a Jameson Cell-based flowsheet. The Jamison addresses the hydrosilicate challenge and the prevaling fine grind. Tests confirmed the production of concentrates exceeding 80 g/t 3E/4E and Ni > 4%.

6.5 Historical Mineral Resource

6.5.1 Historical Mineral Resource (2020)

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

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

This disclosure is made as per Section 2.4 of NI 43-101, including parts 1 to 6:

1. Besides Mansur et al., other Vale geologists were co-authors of the “2020 Historical Estimate”.
2. No breakdown of the individual metals contributing to the 2020 Historical Estimate was published, and no related technical report is available to the authors.
3. The 2020 Historical Estimate did not use Mineral Resource categories.
4. A QP has not done sufficient work to classify the 2020 Historical Estimate as a current Mineral Resource.
5. The work required for the 2020 Historical Estimate to be classified as a current Mineral Resource was outlined in Section 26 of the 2022 Technical Report, filed in conjunction with Bravo’s initial public offering, and was subsequently undertaken.
6. Bravo is not treating the 2020 Historical Estimate as a current Mineral Resource.

GE21 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 (MRE) detailed in the 2023 Technical Report and, subsequent to that, the MRE contained in the 2025 Technical Report and used in 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.

6.5.2 Previous Mineral Resource Estimate (2023)

After 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 NI 43-101, for the Luanga Project, which was estimated by GE21 and disclosed by Bravo in 2023 (Table 6-5). This MRE was superseded by the 2025 MRE and, as a result, the 2023 MRE is not a current estimate and should no longer be relied upon. Nevertheless, Bravo believes the information is relevant to the reader.

Table 6-5: Previous Mineral Resource estimate – 2023

Mineral Resource Classification	Weathering	Average Grades and Contained Metal Estimates												
		Tonnes	PdEq		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:

- *The MRE has been prepared by Porfírio Cabaleiro Rodriguez, Mining Engineer, BSc (Mine Eng), MAIG, director of GE21 Consultoria Mineral Ltda., an independent Qualified Persons (QP) under NI 43-101. The effective date of the MRE is 22 October 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 National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).*
- *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.*
- *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 Whittle software, using the following assumptions:*
  - *Generated from work completed by Bravo and historical test work:*
    - *Phase 1 and 2 Metallurgy testwork – Metallurgical recovery in sulphide material of 80% Pd, 88% Pt, 59% Rh, 56% Au, 50% Ni to a saleable Ni-PGM concentrate.*
    - *Phase 1 and 2 Metallurgy testwork– Metallurgical recovery in oxide material of 73% Pd, 24% Pt, 61% Rh, 95% Au to a saleable PGM ash residue (Ni not applicable).*
    - *Independent Geotechnical Testwork – Overall pit slopes of 40 degrees in oxide and 50 degrees in Fresh Rock.*
    - *Densities are based on 26,898 relative density sample measurements. Averages are 1.58 t/m3 oxide, 2.71 t/m3 Saprock and 2.85 t/m3 fresh rock.*
    - *External downstream payability has not been included, as the base case MRE assumption considers internal downstream processing.*
    - Payable royalties of 2%.
    - *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.*
  - The PdEq equation is: PdEq = Pd g/t + F1 + F2 + F3 + F4,
    - Where: $\boldsymbol{F1}=(\boldsymbol{Ptp}*\boldsymbol{PtR})(\boldsymbol{Pdp}*\boldsymbol{PdR})\boldsymbol{Ptt}\ \boldsymbol{F2}=(\boldsymbol{Rhp}*\boldsymbol{RhR})(\boldsymbol{Pdp}*\boldsymbol{PdR})\boldsymbol{Rht}\ \boldsymbol{F3}=(\boldsymbol{Aup}*\boldsymbol{AuR})(\boldsymbol{Pdp}*\boldsymbol{PdR})\boldsymbol{Aut}\ \boldsymbol{F4}=(\boldsymbol{Nip}*\boldsymbol{NiR})(\boldsymbol{Pdp}*\boldsymbol{PdR})\boldsymbol{Nit}$ P = Metal Price R = 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. 10 mtpa: ▪ 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.*
- *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.*
- *The current MRE supersedes and replaces the Historical Estimate, which should be no longer relied upon.*
- *The QP is not aware of political, environmental, or other risks that could materially affect the potential development of the Mineral Resources.*
- *Totals may not sum due to rounding.*

Source: GE21, 2023.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Amazon Craton is one component of the Brazilian Shield. It is subdivided into seven major geological provinces, each with distinct geochronological and tectonic characteristics Figure 7-1:

- Carajás (3.0–2.5 Ga)
- Central Amazon (>2.3 Ga):
- 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)

These provinces reflect the complex evolution of the craton, from the formation of early Archean crust to the Neoproterozoic orogenic events. The CMP (Figure 7-2), in particular, is a globally significant region for iron, copper-gold, and nickel-platinum group element (PGE/PGM) mineralization (Santos et al., 2008).

The CMP is one of the most economically important regions in the Amazon Craton. It is divided into two main structural domains:

- Rio Maria Domain (South): Characterized by typical Mesoarchean granite-greenstone terrains.
- Carajás Domain (North): Composed of Neoarchean metavolcanic-sedimentary sequences and mafic-ultramafic complexes.

The Luanga Project is located within the CMP, in the southeastern margin of the Archean Amazonian Craton (Figure 7-1). The CMP is composed mostly of granites and greenstone belts and 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 CMP.

The Luanga Complex is intruded near the edge of the Amazon Craton, within a dilational splay at the end of the Cinzento Shear.

The Serra Leste Magmatic Suite comprises small- to medium-sized layered intrusions emplaced during the Neoarchean magmatic event. These intrusions are spatially and temporally associated with the Carajás Domain and are considered to be part of a broader magmatic system that includes the Luanga Complex.

The Luanga Complex is hosted within a dilational splay of the Cinzento Shear Zone, a tectonic environment favourable for the accumulation of sulphide-rich magmas. The mineralization is primarily controlled by the magmatic layering and the structural framework of the

intrusion. The PGM mineralization is typically associated with chromitite layers and disseminated sulphide phases (Trunfull et al., 2020).

7.2 Local Geology

The Luanga Complex is a Neoarchean, tectonically overturned, layered mafic-ultramafic intrusion with an approximate surface extent of 8.1 km in length and 3.5 km in width (total area of ~18 km²). The intrusion is cut by Proterozoic-age dolerite dykes and is hosted within the Serra Leste Magmatic Suite. The complex comprises three main stratigraphic zones, each with distinct lithological and mineralogical characteristics (Figure 7-1 and Figure 7-3).

7.2.1 Stratigraphy

- Ultramafic Zone (UZ): The deepest and most mafic portion of the complex, with an estimated thickness of 800 m. It is composed primarily of wehrlites, dunites, and clinopyroxenites. The UZ is the primary host for chromitite pods and disseminated sulphide mineralization.
- Transition Zone (TZ): A 500-metre-thick zone that marks the transition from ultramafic to mafic lithologies. It is characterized by interbedded orthopyroxenites, harzburgites, and norites. The TZ hosts the Main Sulphide Zone (MSZ), the most economically significant mineralized horizon.
- Mafic Zone (MZ): The uppermost zone, composed predominantly of norites with minor orthopyroxenites. The MZ is less mineralized than the UZ and TZ.

The overall stratigraphy of the Luanga Complex is interpreted as reflecting a cumulate sequence, with the UZ representing the earliest-crystallized phases and the MZ the latest. The complex has undergone significant tectonic inversion, resulting in the overthrusting of the UZ over the MZ.

7.2.2 Tectonic Setting and Metamorphism

The Luanga Complex was emplaced during a period of oblique rifting and transcurrent faulting, associated with the development of the Cinzento and Canaã shear systems. The intrusion is interpreted to have formed in a transtensional regime, where mantle plume activity facilitated the emplacement of mafic magmas into extensional crustal settings (Figure 7-3).

Following emplacement, the complex underwent regional metamorphism, reaching greenschist- to amphibolite-facies conditions. This metamorphism was accompanied by two major episodes of hydrothermal alteration: a K-Fe-Mg-Cl metasomatic event and a later Fe-Ca metasomatic event. These alteration events are spatially associated with the regional Iron Oxide-Copper-Gold (IOCG) systems of the CMP and may have contributed to the remobilization and redistribution of PGMs within the complex.

7.2.3 Mineralization Styles

The Luanga Complex hosts a diverse suite of PGM + Au + Ni mineralization styles that are spatially and genetically related to the intrusion's stratigraphy and tectonic evolution. Six

distinct mineralization styles have been identified, each with unique petrographic and geochemical characteristics (Figure 7-2):

- MSZ – It is the most significant mineralized horizon within the Luanga Complex and is hosted in the Transition Zone (TZ). It has an average thickness of 10–50 m and is characterized by disseminated sulphides (pyrrhotite, pentlandite, chalcopyrite) with high Ni and PGE contents. Local assays have recorded PGE values up to 100 ppm, with a Pd/Pt ratio of approximately 1.5. The sulphides are typically fine to medium-grained and exhibit cumulus textures, indicating a magmatic origin.
- Low Sulphide Zone (LSZ) – It is a minor mineralized horizon, occurring as thin lenses (up to 30-metre-thick) within the TZ. It is characterized by low visible sulphide content but high PGE concentrations, particularly in the Pt-Pd fraction. The LSZ is interpreted to represent a secondary enrichment zone, possibly formed by late-stage magmatic or hydrothermal processes.
- Chromite-associated Zone (Chr-PGM) – This zone is composed of chromitite pods within the UZ and is characterized by high PGE concentrations and a distinct Pt/Pd ratio of ~4. The mineralization is closely associated with chromite and is interpreted to have formed during the intrusion's early crystallization. The PGMs (Platinum-Group Minerals) are typically found in association with sulphides and are hosted in chromite-rich layers.
- Ni-Rh Sulphide Zone – This zone is a localized mineralized horizon within the TZ and is characterized by fine-grained, net-textured sulphides rich in Ni and Rh, but relatively poor in Pt. The zone is interpreted to have formed during late-stage magmatic fractionation and is spatially associated with the MSZ. The Ni-Rh zone is less economically significant compared to the MSZ but may represent a secondary enrichment event.
- Sulphide Zone (SZ) – It is a narrow, N-S-trending mineralized structure within the TZ, characterized by fine-grained sulphides with low PGE concentrations and a low Pt/Pd ratio. The SZ is interpreted to have formed during late-stage hydrothermal activity and is spatially associated with shear zones within the complex.
- Massive Sulphide Zone (MASU) – It is a localized zone of massive sulphide mineralization within the TZ and is interpreted to have formed by hydrothermal processes. It is characterized by high Ni (up to 2.04%) and Cu (up to 1.23%) contents and is spatially associated with alteration zones. The MASU is considered to be a secondary enrichment zone, possibly formed by the remobilization of magmatic sulphides during hydrothermal alteration.

The Luanga Complex is a Neoarchean, tectonically overturned, layered mafic-ultramafic intrusion that hosts a diverse suite of PGM + Au + Ni mineralization styles. The mineralization is primarily magmatic in origin, with the MSZ representing the most economically significant horizon. The deposit exhibits strong stratigraphic and structural controls, with mineralization concentrated in the TZ and UZ.

The geological setting of the Luanga Complex is analogous to that of the Bushveld (South Africa) and Stillwater Complexes (Montana, USA)—two of the world's premier layered mafic-ultramafic intrusions hosting reef-type PGM-Ni-Cu mineralization—in terms of magmatic evolution, stratigraphy, and mineralization style. The presence of reef-type mineralization,

cumulus textures, and multiple episodes of hydrothermal alteration further support a magmatic-hydrothermal origin for the deposit.

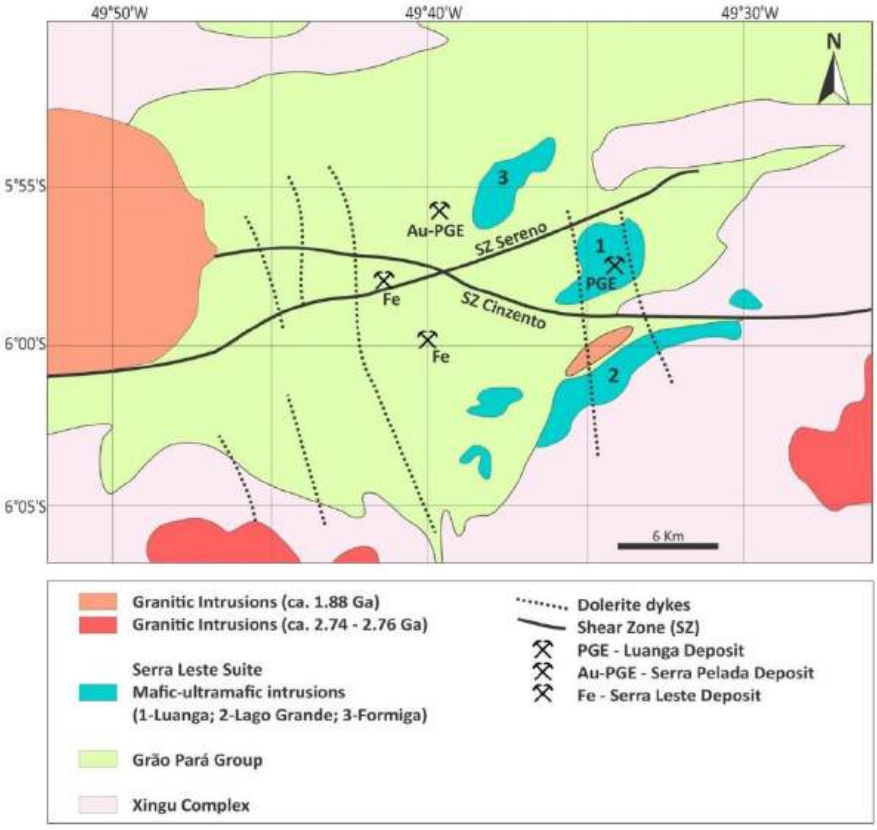


Figure 7-1: Major structures controlling Luanga intrusion

Source: Teixeira & Ferreira Filho, 2015.

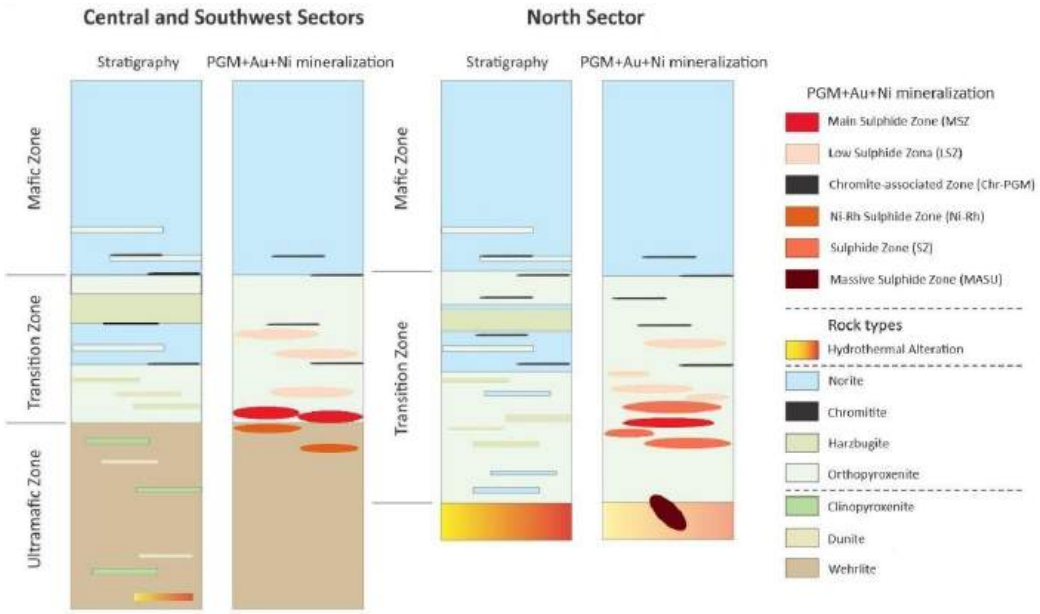


Figure 7-2: Schematic stratigraphic column locating the different styles of PGM mineralization on Luanga Complex

Source: Bravo, 2025.

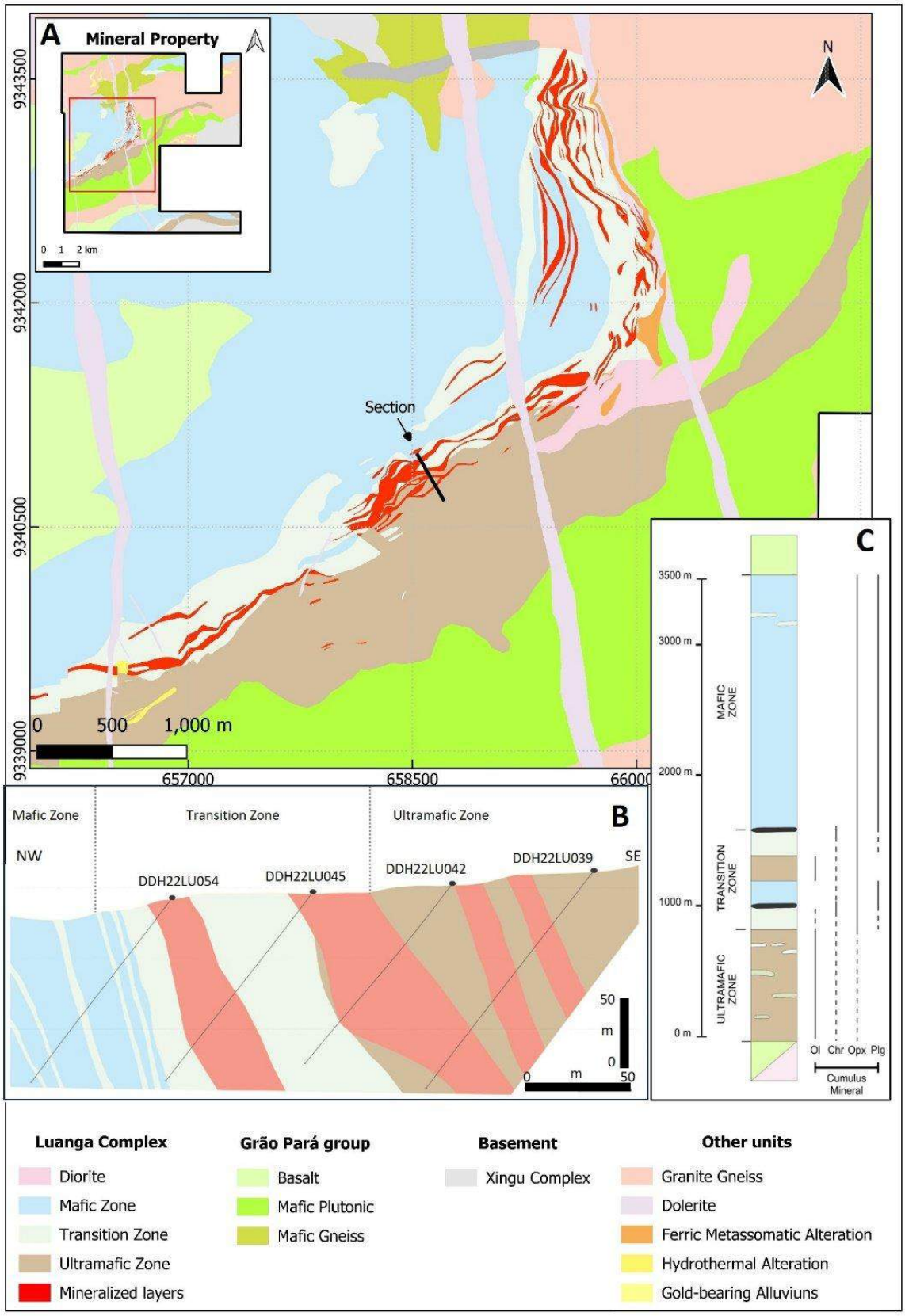


Figure 7-3: Luanga Complex

Legend: A) geological map; B) section of the Central sector; C) stratigraphic column.
Source: Bravo, 2025.

8 DEPOSIT TYPES

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 & 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 that originated as a separate immiscible liquid from the parental magma.

"PGM reefs" are strata-bound 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.

A schematic model of an ideal layered intrusion is presented in 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 PGM deposits with differing magmatic affinities can occur in similar positions within an intrusive system.

Magmatic Ni-Cu-PGM sulphides 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 across distinct stratigraphic horizons of the Luanga Complex, including different metal tenors and Pt/Pd and Rh/Pd ratios, indicate that several mineralization events 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).so

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 across scales (from mineral crystals to zones several hundred metres thick) and largely preserves the primary textures and compositions of cumulate rocks and PGM-mineralized zones, magmatic features can be recognized throughout the layered intrusion.

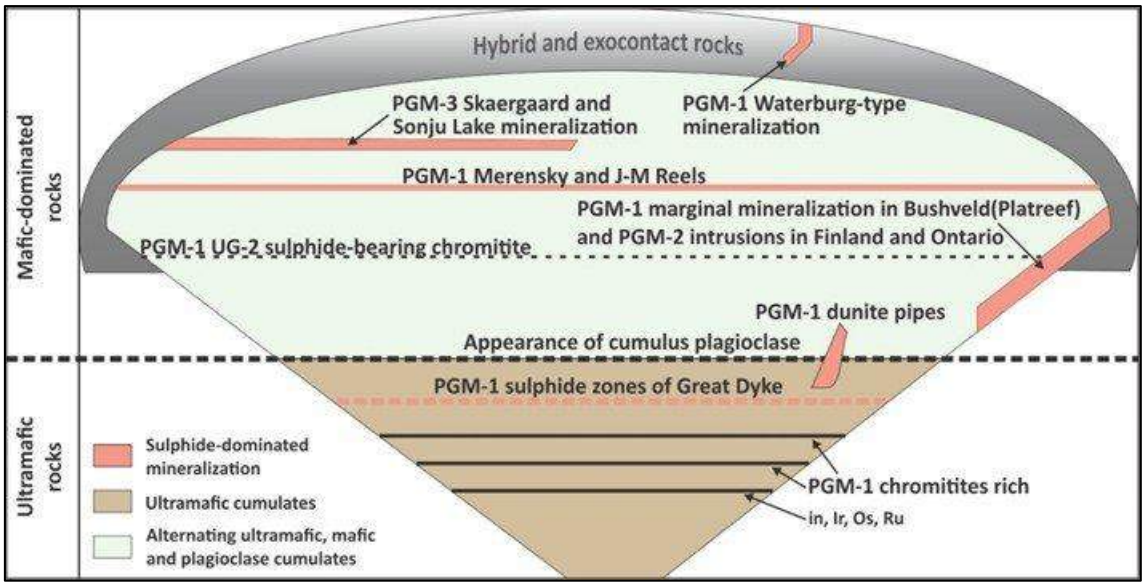


Figure 8-1: Schematic model of a large igneous province (LIP) related layered intrusion

Source: Naldrett, 2010.

9 EXPLORATION

Bravo has followed a systematic exploration approach at the Luanga PGM + Au + Ni Project since acquiring the mineral rights in November 2021. Exploration has proceeded on two parallel fronts: (1) definition and extension of the stratiform palladium-platinum-rhodium-gold-nickel mineralization of the Luanga Mafic-Ultramafic (MUM) Complex, which hosts the Mineral Resource reported in Section 14 and which underpins the economic analysis in this Report; and (2) generative exploration for genetically distinct mineralization styles within the wider Luanga property, principally magmatic nickel-copper (± PGM) sulphides and iron oxide copper-gold (IOCG) systems in the surrounding country rocks.

This Section describes the exploration methods applied and the exploration potential that remains. Work programmes are described in sub-sections 9.1 (grids and surveys), 9.2 (geological mapping), 9.3 (geochemical sampling), 9.4 (geophysics) and 9.5 (petrology, mineralogy and research). The exploration potential itself, and the drill results that define it, are summarised in sub-section 9.6 and described prospect-by-prospect in sub-sections 9.6.1 to 9.6.3.

9.1 Grids and Surveys

All Bravo survey data are held in the SIRGAS 2000 datum, UTM Zone 22S projection. Historical Vale's collars were originally sited on the Instituto Brasileiro de Geografia e Estatística (IBGE) base datum and surveyed in SAD69 using a Topcon GTS 229 total station at the end of each drilling campaign. Bravo re-surveyed most of the Vale collars in SIRGAS 2000; where the original collar could not be located in the field, the recorded SAD69 coordinates were transformed to SIRGAS 2000. Historical collars were capped with cement blocks incorporating a PVC tube and an aluminium plate stamped with the hole identifier, coordinates, elevation, dip, azimuth and final depth.

All Bravo drill collars are geolocated by real-time kinematic (RTK) GNSS to geodetic accuracy, initially by the contractor RR Topografia & Engenharia and subsequently by Bravo's own survey team. Trench channel-sample positions are likewise surveyed by RTK GNSS. Downhole deviation surveys are run on all Bravo holes with a Reflex Gyro Sprint-IQ instrument, supplemented where practical by guided runs using a Reflex ACT3 device. Of the historical holes, 240 (approximately 95% of the population) have downhole surveys at 3 m intervals.

Topographic control has been established by two drone campaigns. In 2022 RR Topografia & Engenharia completed orthophotography and a digital elevation model (DEM) over the deposit area using commercial drone survey equipment, with ground survey used to control accuracy, positioning and georeferencing. In June 2023 Sul Pará completed a second drone survey covering 26.96 km² over the Luanga Complex and adjacent ground, delivering a 1 m DEM and an ortho-mosaic image.

9.2 Geological Mapping

Bravo has completed detailed geological mapping over the Luanga Complex at 1:10,000 scale, along north-south profiles spaced at 100 m. In structurally or lithologically complex areas mapping was carried out at 1:5,000, with drilling data and ground geophysics used to support the interpretation. The resulting map underpins the lithological and stratigraphic framework described in Section 7 of this Report, including the subdivision of the Complex into its Ultramafic Zone, Transition Zone and Mafic Zone and the recognition of the six mineralization styles.

Semi-regional mapping over the whole mineral property was also conducted by Bravo, as well as detailed mapping over the advanced Cu-Au and Ni-Cu prospects.

9.3 Geochemical Sampling

9.3.1 Verification Sampling of Historical Core

Bravo's earliest work on the Project, between 2 and 23 September 2020, was a verification exercise: Company staff visited Vale's core facilities and collected five quarter-core samples from mineralized intervals in four historical drill holes that had previously been sampled by Vale. The samples were cut, bagged, identified with the original sample numbers and shipped by Bravo personnel directly to ALS Brasil Ltda. in Belo Horizonte. All samples were weighed, dried, crushed, split and pulverised to 85% passing 75 µm, then analysed for platinum, palladium and gold by fire assay with ICP-AES finish, for rhodium by fire assay with ICP-MS finish, and for 48 further elements by four-acid digest with ICP-MS finish.

In 2021, Bravo conducted a detailed review of the Vale dataset and took delivery of 234 complete historical diamond drill holes at its core yard. Of these, 202 holes—all the historical drilling within the Luanga deposit, representing 40,065 m or 87% of the 46,029 m received—were relogged geologically and specific intervals were resampled. Core boxes, markings and labels were repaired and cleaned before relogging; the core was then cleaned and photographed before resampling. For the resampling programme, remaining half core was cut in half again with a standard core saw, and where only a quarter remained it was sampled in its entirety. Certified reference materials, blanks and standards were inserted at a ratio of one of each in every twenty samples, giving a quality-control sample after every ten primary samples. Standards were sourced from OREAS in Australia and AMIS in South Africa, selected to span the relevant grade range and to provide the closest available matrix match to Luanga mineralization. Samples were submitted to ALS Brasil's reception facility in Parauapebas.

The resampling results show the expected positive correlation with the historical Vale assays for the platinum-group metals. Minor variations between the original and repeat assays are attributed to updated preparation and analytical methods and to improved assaying techniques for rhodium and nickel. The nickel resampling data resolve into two populations, interpreted as reflecting silicate-hosted and sulphide-hosted nickel respectively—the distinction that underlies Bravo's subsequent practice of reporting sulphide nickel only.

9.3.2 Soil Geochemistry

A total of 5,241 soil samples were collected by Vale during its exploration of Luanga. All were analysed for a 16-element suite in parts per million (ppm)—silver, arsenic, beryllium, bismuth, cerium, cobalt, chromium, copper, lanthanum, nickel, lead, antimony, tin, tellurium, tungsten and zinc—by inductively coupled plasma mass spectrometry at three laboratories (Nomos, Lakefield and SGS), and for gold, platinum and palladium by fire assay with atomic absorption spectrometry finish. The laboratory responsible for the fire-assay work is not recorded in the historical database.

Bravo reprocessed the historical soil results. Chromium and nickel anomalies correlate closely with the mafic and ultramafic rocks of the Luanga and Luanga South Complexes, while palladium and platinum anomalies are spatially coincident with the Transition Zone of both Complexes—the stratigraphic interval that hosts the Main Sulphide Zone.

New soil grids have been established over the outlying targets, sampled at 25 m stations along gridlines spaced at 100 m and locally 50 m. Samples of approximately 3 kg are dried and sieved to minus 1 mm at the Luanga field camp, and the minus 1 mm fraction is analysed for 33 elements by portable X-ray fluorescence (pXRF). Anomalous results are then verified by an independent commercial laboratory.

9.3.3 Rock Chip and Auger Sampling

Rock chip sampling at Lizard Target returned copper grades between 0.20% and 0.78%, with one sample assaying 1.39% copper. An auger drilling programme of 30 holes totalling 269.70 m was completed over the main Lizard copper-in-soil anomaly, with samples analysed by pXRF at the Luanga camp.

9.3.4 Trenching and Channel Sampling

Trenching began in the fourth quarter of 2022 to characterise the mineralized zones at surface, where saprolite-hosted oxide PGM+Au mineralization in the Luanga deposit, and locally in Taurus and Lizard targets. Currently, 58 trenches totalling 10,901 m had been completed. Trenching procedures follow NBR 9604, the Brazilian standard for trench excavation and sampling. Trenches are 80 cm wide—the width of the backhoe bucket—and to a maximum depth of 120 cm. All trenches are geologically described, mapped and sampled, and their channel samples surveyed by RTK GNSS. Geological description is logged on site into Micromine GeoBank For Field Teams, linked directly to the main project database.

Channel samples are cut to a 1 m standard length, varying between 0.7 m and 1.2 m to respect geological contacts, and are taken 40 to 50 cm above the trench floor to sample saprolite and avoid soil. Samples are collected with chisel and sledgehammer onto an aluminium tray positioned beneath the channel. Trenching across the strike of the deposit has been completed in all three sectors along the full 8.1 km strike length. Results have consistently shown a greater lateral extent to the shallow oxide PGM+Au mineralization than had been interpreted from drilling

alone, extending across the topographic highs, and have confirmed supergene enrichment in the saprolite. Reported trench results from Luanga deposit include 175.7 m at 1.71 g/t PGM+Au in TRC23LU024; 97 m at 4.86 g/t PGM+Au including 24 m at 14.36 g/t PGM+Au; 146.8 m at 2.05 g/t PGM+Au including 50.0 m at 4.18 g/t PGM+Au; 145 m at 1.42 g/t PGM+Au including 11 m at 4.45 g/t PGM+Au; and 123 m at 2.33 g/t PGM+Au.

9.3.5 Sample Security and Analytical Laboratories

Sample security, chain of custody and quality assurance and quality control procedures for drill core and channel samples are described in full in Sections 11 and 12 of this Report and are not repeated here. In summary, samples are prepared and analysed at SGS Geosol in Parauapebas, Pará (historically also ALS Brasil), under a chain-of-custody regime in which certified reference materials, blanks and duplicates are inserted blind into every batch, and in which a failed control sample triggers investigation and, where warranted, re-assay of the batch. Current SGS Geosol methods are PRPCLI preparation (85% passing 200 mesh); FAI515 and FAI34V for platinum, palladium and gold; FAI30V for rhodium; AA04B for sulphide nickel and sulphide copper; and ICP40B for trace elements. Portable XRF results from soil, and auger programmes are used for reconnaissance only and are systematically verified against commercial laboratory assays before being relied upon.

9.4 Geophysics

9.4.1 Reprocessing of Historical Data (2021)

Geophysical processing and interpretation was completed for Bravo in 2021 by SGC and by Southernrock. Southernrock reprocessed the historical induced polarization (IP) data and SGC reprocessed the historical magnetic data; the reprocessed IP resistivity was inverted in three dimensions and overlain on the reprocessed magnetic image to guide the first phase of Bravo drilling.

9.4.2 Borehole and Ground Electromagnetics

Borehole electromagnetic (BHEM) and surface electromagnetic surveys were conducted by Geomag S.A., totalling 4,265 line-metres. Fixed-loop transient electromagnetic (FLTEM) surveys were subsequently completed over the Central and North Sectors on 56 transverse lines totalling 54.62 km. The raw data were processed and several conductive plates modelled; some have been drilling tested and others remain to be tested. The value of that practice is demonstrated by drill hole DDH26LU347, which was sited to test a high-conductivity plate modelled from BHEM data acquired in hole DDH26LU330—the plate was interpreted as dipping 48° towards approximately 270°, and the hole intersected massive to semi-massive sulphide close by 200 m as predicted.

9.4.3 Ground Magnetometry and Micro-Gravity

A ground geophysical survey comprising magnetometry and micro-gravity on a 100 m line-spacing grid was acquired by the Bravo team during 2023 and 2024 over approximately 18.7 km², covering the ultramafic and transition zones of the Luanga Complex. Magnetic readings were taken continuously along the lines; micro-gravity stations were spaced at 50 m. Raw data were delivered to Southern Geoscience Consultants (SGC) for quality control, correction and processing. The magnetic data define a linear anomalous feature trending southwest-northeast across the Southwestern and Central Sectors of the Luanga Complex, with a flank running south-north through the North Sector and a further flank trending northwest-southeast at the eastern extreme of the grid. The Bouguer image derived from the micro-gravity data shows the highest-density rocks in the Southwestern and Central Sectors, associated with the wehrlites of the basal Ultramafic Zone, and resolves the contact between those rocks and the meta-mafic plutonic rocks of the Grão Pará Group. In the Babylon Target, the survey identified a gravity high closely associated with a high-amplitude magnetic anomaly and coincident with the helicopter-borne time-domain electromagnetic and magnetic survey (HeliTEM) conductor.

9.4.4 Airborne Time-Domain Electromagnetic and Magnetic Survey (HeliTEM)

A HeliTEM was flown by Xcalibur Multiphysics over 99.72 km², covering the whole of the Luanga mineral property. The survey comprised 768 line-km, of which 697.1 km were northwest-southeast transverse lines spaced at 150 m and 70.9 km were southwest-northeast tie lines spaced at 1,500 m. SGC was engaged to model the principal conductors in three dimensions; the exercise identified 17 Priority 1 and 16 Priority 2 targets, ranked on their electromagnetic response together with the ground micro-gravity and magnetic data, detailed geological and structural interpretation, soil geochemistry and the existing drilling database. The HeliTEM dataset, together with regional structural analysis and the historical geochemistry, was the basis of the Cu-Au and magmatic Ni-Cu targets. The dataset has since been reprocessed to produce a Magnetisation Vector Inversion three-dimensional model and Self-Organising Maps, which are being used to re-rank the target pipeline.

9.4.5 Deep Induced Polarization Survey (2026)

A ground IP programme of approximately 50 line-km was commenced on 16 June 2026 across the prioritised copper-gold and magmatic Ni-Cu targets. Approximately 7.6 km on three lines had been completed by 24 August 2026. One of the completed traverses is a 2 km east-west line across the Babylon target. The preliminary interpretation of that line shows strong chargeability with coincident low resistivity at depth, interpreted as corresponding to the Babylon sulphide mineralization and indicating that it may extend deeper, and at a steeper angle, than the drilling completed to date. The IP data will be integrated with the existing geological, drilling and electromagnetic datasets at Babylon to design a deep drill test.

9.5 Petrology, Mineralogy and Research

Petrographic and mineralogical characterization has been central to Bravo's work at Luanga, both because the Complex has been variably metamorphosed and hydrothermally altered—so that primary cumulate textures are commonly pseudomorphed—and because the metal grade of each mineralization style is controlled by the sulphide assemblage rather than by the host lithology alone. Petrographic study supported the renaming of several lithological units originally defined by Vale and the recognition of four mineralization styles additional to the two previously described, as set out in Section 7 of this Report. The six styles now recognised are the Main Sulphide Zone, the Low Sulphide Zone, the Chromite-associated Zone, the Ni-Rh Sulphide Zone, the Sulphide Zone and the Massive Sulphide Zone.

Reflected-light and transmitted-light petrography, supported by magnetic susceptibility measurement with a KT-20 S/C instrument on all logged core, has established the diagnostic characteristics of each style. In the Main Sulphide Zone, mineralization is associated with sulphide blebs up to a few centimetres' interstitial to cumulus orthopyroxene and olivine or their pseudomorphs, the sulphide fraction comprising pyrrhotite and pentlandite with minor variably distributed chalcopyrite, and pentlandite forming 30% to 50% of the sulphide fraction as fine-grained crystals below 1 mm with rare exsolutions in pyrrhotite. In the Ni-Rh Sulphide Zone, disseminated to net-textured sulphides occur in interlayered wehrlite and dunite with pentlandite at 30% to 40% of the sulphide fraction as coarser 1–2 mm crystals, indicating a very high nickel tenor with correspondingly lower platinum and palladium tenor. In the Massive Sulphide Zone, sulphides comprise 80% to 90% pyrrhotite and 10% to 20% pentlandite with local chalcopyrite-rich domains, amphibole-biotite-quartz-magnetite assemblages are enclosed within the sulphide, and sulphide-amphibole intergrowths are common. Reported maxima for the Massive Sulphide Zone are below 6.0% nickel, 6.0 g/t palladium, 2.5 g/t platinum and 0.2 g/t rhodium, indicating lower tenors than those calculated for the Main Sulphide Zone.

In drill hole DDH26LU347 (North Sector), polished thin section established that the sulphide assemblage is pyrrhotite, pentlandite and minor chalcopyrite, with pentlandite present as fine to medium-grained crystals and as minor exsolution lamellae in pyrrhotite, and that euhedral chromite crystals enclose sulphide blebs and show partially corroded faces. The Company interprets that texture as recording the coexistence of chromite with an immiscible nickel-PGE-bearing sulphide liquid, and therefore as evidence that a sulphide liquid pool segregated and concentrated in that part of the Luanga intrusion—establishing a magmatic origin for the mineralization and a direct genetic link to the Luanga Complex rather than to a later hydrothermal event.

9.5.1 In Situ Density Sampling

Bulk density in the regolith profile was determined by an in-situ programme in which Bravo's field team collected 112 samples of saprolite and soil. The regolith horizon to be sampled is first identified and the pedological and geological characteristics of the material described. The

sample site is cleaned to expose a flat vertical face, and a cylinder 15 cm high and 6 cm in internal diameter is driven perpendicularly into the weathered material with a small sledgehammer. The cylinder is then excavated with a spatula, and excess material removed from both open ends without loss of sample. Samples are sealed and labelled in plastic bags and transported to the core shed, where they are weighed, dried at 105°C for 24 hours and reweighed. Dry density is calculated as dry weight divided by the cylinder volume of 423.9 cm³, and moisture content as the difference between wet and dry weight expressed as a percentage of dry weight. Sample coordinates, experimental data and the calculated densities and moisture contents are recorded in the project database. These determinations support the density model applied to the oxide and transition domains of the Mineral Resource estimate.

9.6 Exploration Potential

The Luanga MUM Complex has a strike length of approximately 8.1 km and is divided, from north to south, into the North, Central and Southwest Sectors. Mineralization remains open at depth along essentially the whole of that strike length. Drilling by Vale between 1992 and 2003 tested principally the upper 100 m to 150 m of the deposit, while Bravo's drilling since March 2022 has extended the drill-defined mineralization to approximately 300 m below surface in the Central Sector, to approximately 200 m in the North Sector, and to shallower depths again in the Southwest Sector. In the Central Sector, drilling has repeatedly encountered mineralized thicknesses equal to or greater than, and PGM+Au grades equal to or higher than, those of the shallower drilling above, commonly with higher sulphide nickel grades.

Beyond the disseminated stratiform PGM+Au+Ni deposit, exploration has established two further mineralization styles associated with massive to semi-massive sulphide zones within the deposit area. A magmatic nickel-copper (± PGM) massive to semi-massive sulphide zone (LU330 Target) occurs in drill holes DDH26LU330 and DDH26LU347 located in the southern portion of the North Sector and a hydrothermal massive sulphide zone (Babylon Target) first recognised in drill hole DDH22LU047.

Numerous copper-gold and magmatic Ni-Cu targets were generated in 2023 after a HeliTEM survey that covered the whole mineral property. Currently, the most advanced are T5 and T16 targets.

In addition, the mineral property also covers a significant portion of the Luanga South MUM Complex where the past operators conducted a scout drilling programme which indicated the same stratigraphy of Luanga MUM Complex and the presence of PGM+Au mineralization. At this stage, Bravo have not started the exploration on this prospective area.

9.6.1 Potential for Expansion of Luanga PGM+Au+Ni Deposit

The exploration targets identified for the expansion of the Mineral Resource Estimate within the Luanga Deposit are summarised in Table 9-1 and shown in Figure 9-1.

Table 9-1: Exploration target inventory for expansion of the MRE within the Luanga deposit

Target / area	Mineralization style	Favourable indicators	Stage
Luanga Central Sector (depth)	Stratiform PGM+Au+Ni	MSZ open below ~300 m within potential open-pit depths	Resource extension
Luanga North Sector (depth)	Stratiform PGM+Au+Ni	MSZ open below ~200 m; grades and widths improving at depth	Resource extension
Luanga Southwest Sector (depth)	Stratiform PGM+Au+Ni	Shallowest drill coverage of the three sectors; open at depth	Resource extension
Hole LU330 Target	Magmatic Ni-Cu-PGM massive sulphide	Massive sulphide with chromitite in Transition Zone; BHEM plates	Drilling / BHEM
Babylon Target	Hydrothermal Ni-Cu (± PGM)	Magnetic high + EM conductor + gravity high; deep IP anomaly	Drilling / BHEM
Crescent Zone	Stratiform PGM+Au	Repeated (folded) MUM stratigraphy; identified 2025, first drilled 2026	Early drill testing

Source: Bravo, 2026.

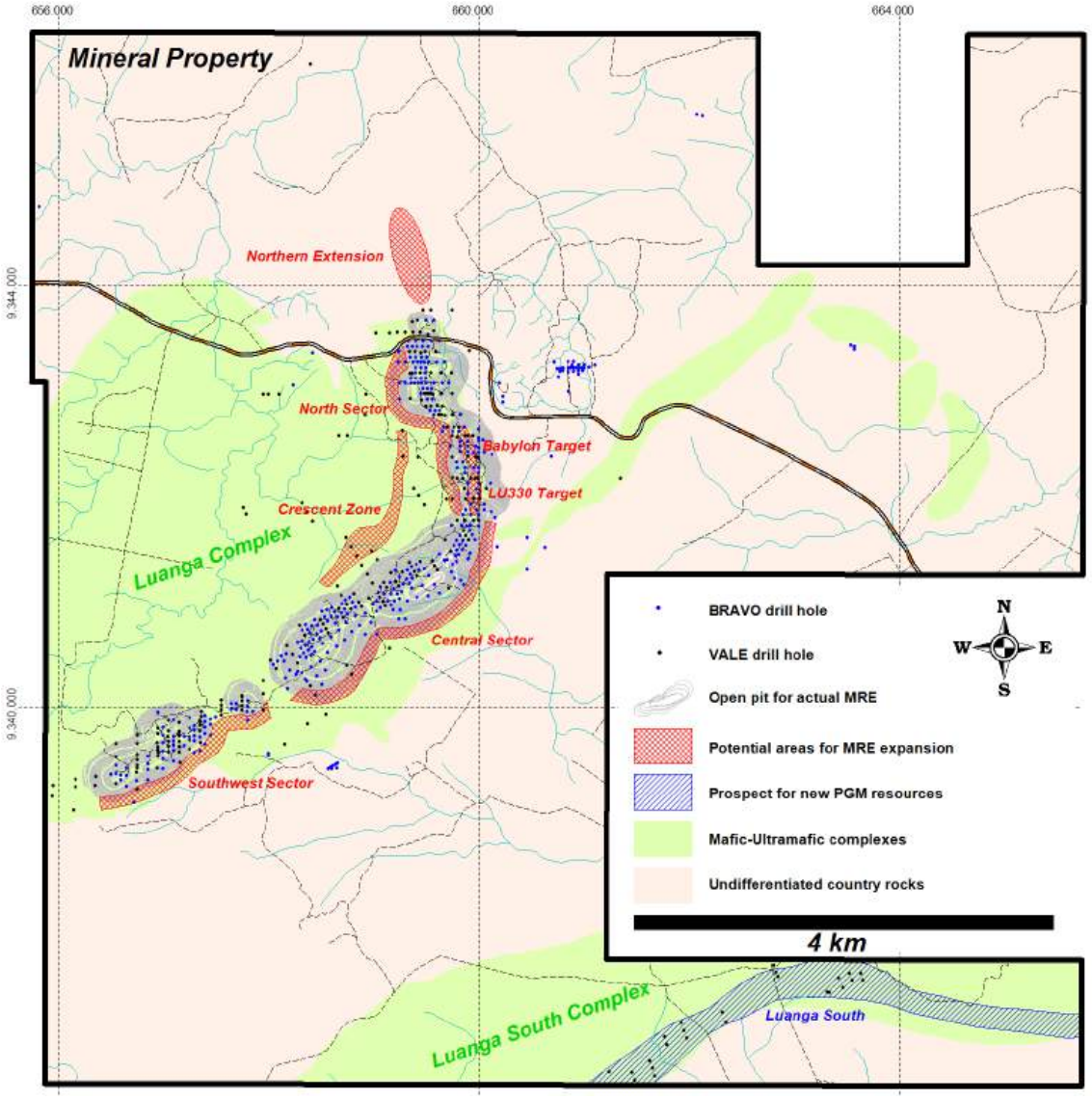


Figure 9-1: Luanga deposit showing exploration targets to expand Mineral Resources

Source: Bravo, 2026.

9.6.1.1 Luanga Central Sector

The Luanga Central Sector contains the largest proportion of the Mineral Resource tonnage and is the most deeply drilled of the three sectors. Mineralization is dominated by a single thick, laterally continuous Main Sulphide Zone (MSZ) that persists for kilometres along strike. Bravo's 2026 infill and extensional programme has extended the drill-defined MSZ from approximately 200 m to approaching 350 m below surface and is confirming the trend of equal or greater thickness, equal or higher PGM+Au grade, and increasing sulphide nickel grade with depth. Mineralization remains open below the current base of drilling and, on the Company's interpretation, within depths reachable by open-pit extraction (Table 9-2 and Figure 9-2).

Table 9-2: Significant drill intercepts, Central sector

Hole ID	Dip / Az (°)	From (m)	To (m)	Interval (m)	PGM+Au (g/t)	Rh (g/t)	Ni (%) sulph.	Type	Rank
DDH26LU312	-60 / 330	259.25	310.25	51.00	3.67	0.14	0.33	FR	187
including		*285.25*	*308.25*	23.00	*5.59*	*0.21*	*0.44*	*FR*	–
DDH26LU316	-60 / 330	296.60	346.60	50.00	2.55	0.09	0.20	FR	128
DDH24LU288	-60 / 330	122.30	178.30	56.00	1.97	0.05	0.24	FR	110
DDH24LU289	-60 / 330	75.70	143.80	68.10	1.55	0.06	0.19	FR	105
including		*100.70*	*136.70*	36.00	*2.21*	*0.09*	*0.25*	*FR*	–
DDH24LU286	-60 / 330	118.00	157.00	39.00	2.34	0.06	0.32	FR	91

Notes: All From, To and Interval values are downhole depths. Rank = interval x PGM+Au grade, used as a ranking device only, not a resource estimate. Type: FR = fresh rock; Ox = oxide. Ni grades are sulphide nickel and exclude non-recoverable silicate nickel. Given the orientation of drilling and mineralization, intercepts are estimated by the Company at 108% to 140% of true thickness in the Central Sector.
Source: Bravo, 2026.

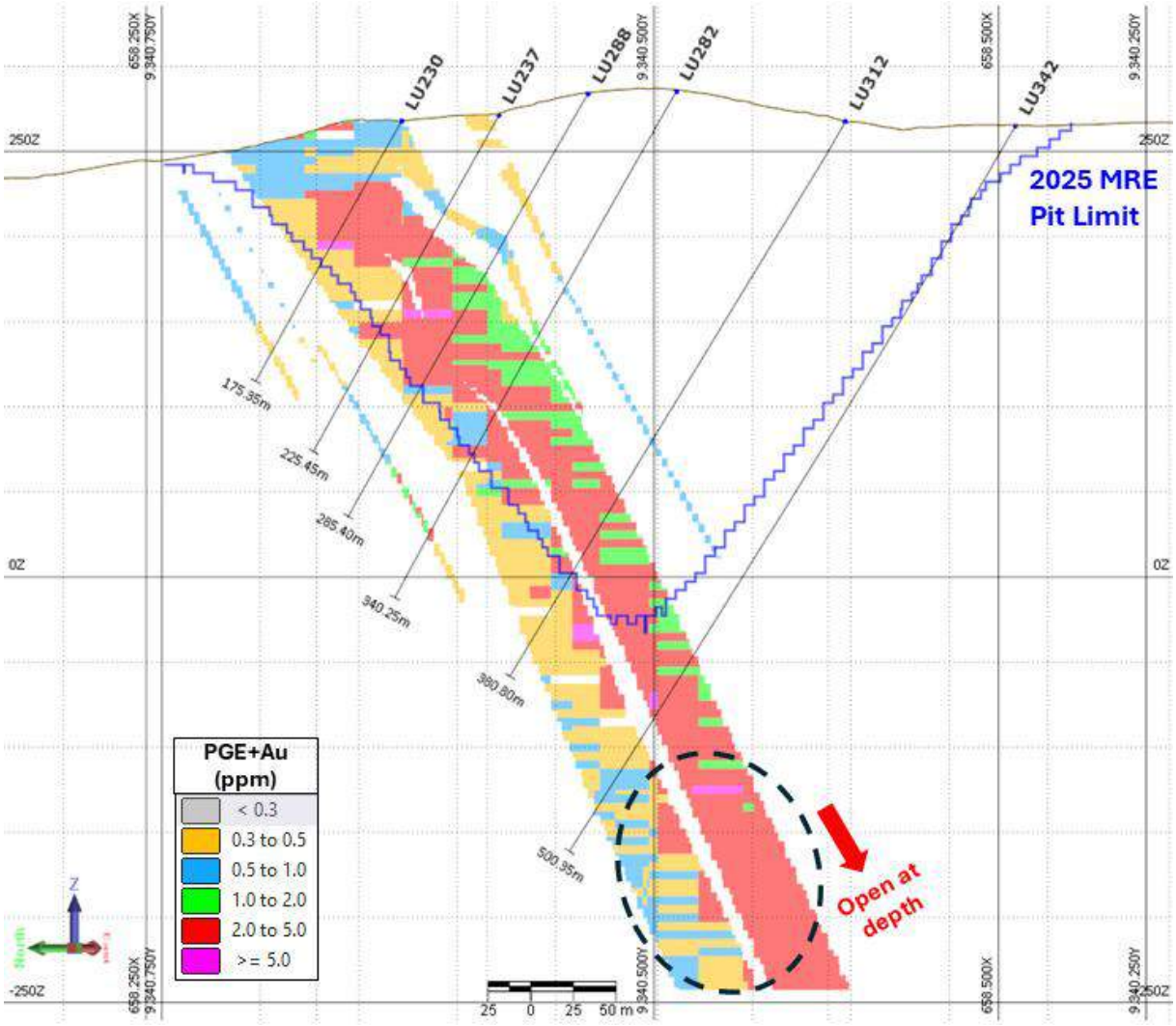


Figure 9-2: Cross section on the Central sector showing potential expansion of the Mineral Resources at depth

Source: Bravo, 2026.

9.6.1.2 Luanga North Sector

The Luanga North Sector is at an earlier stage of definition than the Central Sector, with mineralization currently defined from surface to approximately 200 m depth. Because the Central Sector has been shown to carry mineralization to more than 350 m, the North Sector is regarded as carrying substantial potential for Mineral Resource growth at comparatively shallow depths. The North Sector also differs from the other two sectors in showing localised high-grade rhodium, narrow zones of copper sulphide at depth, and the separate magmatic/hydrothermal massive sulphide styles (Table 9-3 and Figure 9-3).

Table 9-3: Significant drill intercepts, North sector

Hole ID	Dip / Az (°)	From (m)	To (m)	Interval (m)	PGM+Au (g/t)	Rh (g/t)	Ni (%) sulph.	Type	Rank
DDH24LU235	-60 / 090	21.50	79.80	58.30	3.88	0.25	0.12	FR	226
including		*49.80*	*64.80*	*15.00*	*6.41*	*0.37*	*0.11*	*FR*	–
and		93.80	139.50	45.70	3.60	0.26	0.08	FR	165
including		*102.85*	*116.50*	*13.65*	*8.47*	*0.56*	*0.13*	*FR*	–
DDH23LU224	-60 / 090	118.20	147.15	28.95	2.40	0.27	0.10	FR	69
including		*124.20*	*127.20*	*3.00*	*3.36*	*1.41*	*0.03*	*FR*	–
DDH24LU234	-60 / 090	54.40	92.40	38.00	1.79	0.12	0.06	FR	68
DDH23LU219	-60 / 090	236.93	259.30	22.37	1.82	0.10	0.14	FR	41
and		271.50	280.65	9.15	2.22	0.08	0.06	FR	20
DDH24LU231	-60 / 090	9.00	44.90	35.90	1.03	0.05	n.a.	Ox	37

Notes: All From, To and Interval values are downhole depths. Rank = interval x PGM+Au grade, used as a ranking device only, not a resource estimate. Type: FR = fresh rock; Ox = oxide. Ni grades are sulphide nickel and exclude non-recoverable silicate nickel. Given the orientation of drilling and mineralization, intercepts are estimated by the Company at 125% to 140% of true thickness in the North Sector. n.a. = not applicable (oxide material is not assayed for sulphide nickel).
Source: Bravo, 2026.

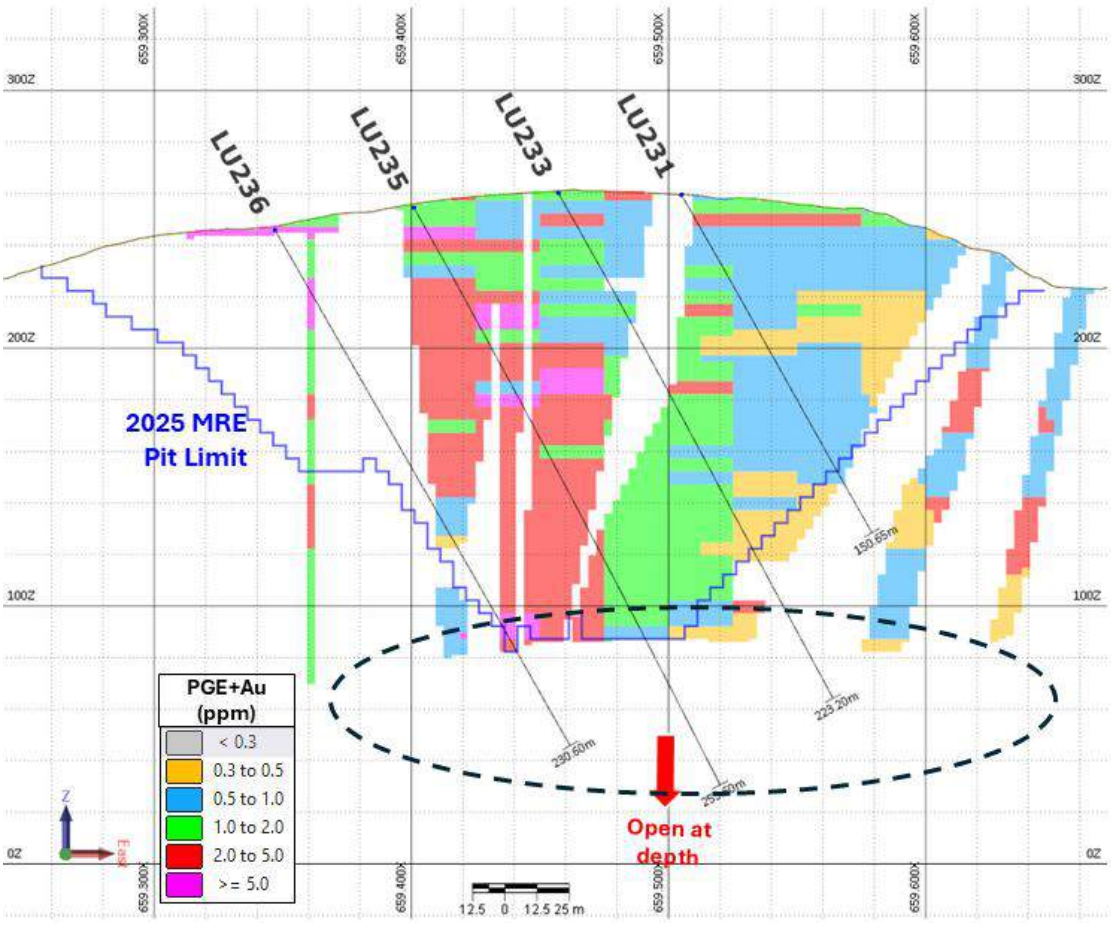


Figure 9-3: Cross section on the North sector showing potential expansion of the Mineral Resources at depth

Source: Bravo, 2026.

9.6.1.3 Luanga Southwest Sector

The Southwest Sector has the shallowest drill coverage of the three sectors, and, on the 2025 Mineral Resource Estimate, the constraining pit shell reaches the limit of the drilling data, with additional mineralized zones present stratigraphically higher in the sequence. Drilling has confirmed that the MSZ, previously interpreted to narrow in this sector, in fact maintains substantial and continuous thickness (Table 9-4 and Figure 9-4).

Table 9-4: Significant drill intercepts, Southwest sector

Hole ID	Dip / Az (°)	From (m)	To (m)	Interval (m)	PGM+Au (g/t)	Rh (g/t)	Ni (%) sulph.	Type	Rank
DDH23LU142	-60 / 360	0.00	50.00	50.00	1.79	0.05	n.a.	Ox	90
and		157.00	158.00	1.00	0.04	0.01	1.51	FR	–
DDH23LU147	-60 / 360	66.30	99.30	33.00	1.36	0.05	0.08	FR	45
DDH23LU149	-60 / 360	84.70	103.70	19.00	2.12	0.08	0.15	FR	40
including		*91.70*	*97.70*	*6.00*	*3.91*	*0.16*	*0.17*	*FR*	–
DDH23LU144	-60 / 360	79.70	105.70	26.00	1.11	0.05	0.17	FR	29
DDH23LU145	-60 / 360	102.10	116.30	14.20	1.72	0.08	0.06	FR	24

Notes: All From, To and Interval values are downhole depths. Rank = interval x PGM+Au grade, used as a ranking device only, not a resource estimate. Type: FR = fresh rock; Ox = oxide. Ni grades are sulphide nickel and exclude non-recoverable silicate nickel. Given the orientation of drilling and mineralization, intercepts are estimated by the Company at 110% to 130% of true thickness in the Southwest Sector. n.a. = not applicable (oxide material is not assayed for sulphide nickel)
Source: Bravo, 2026.

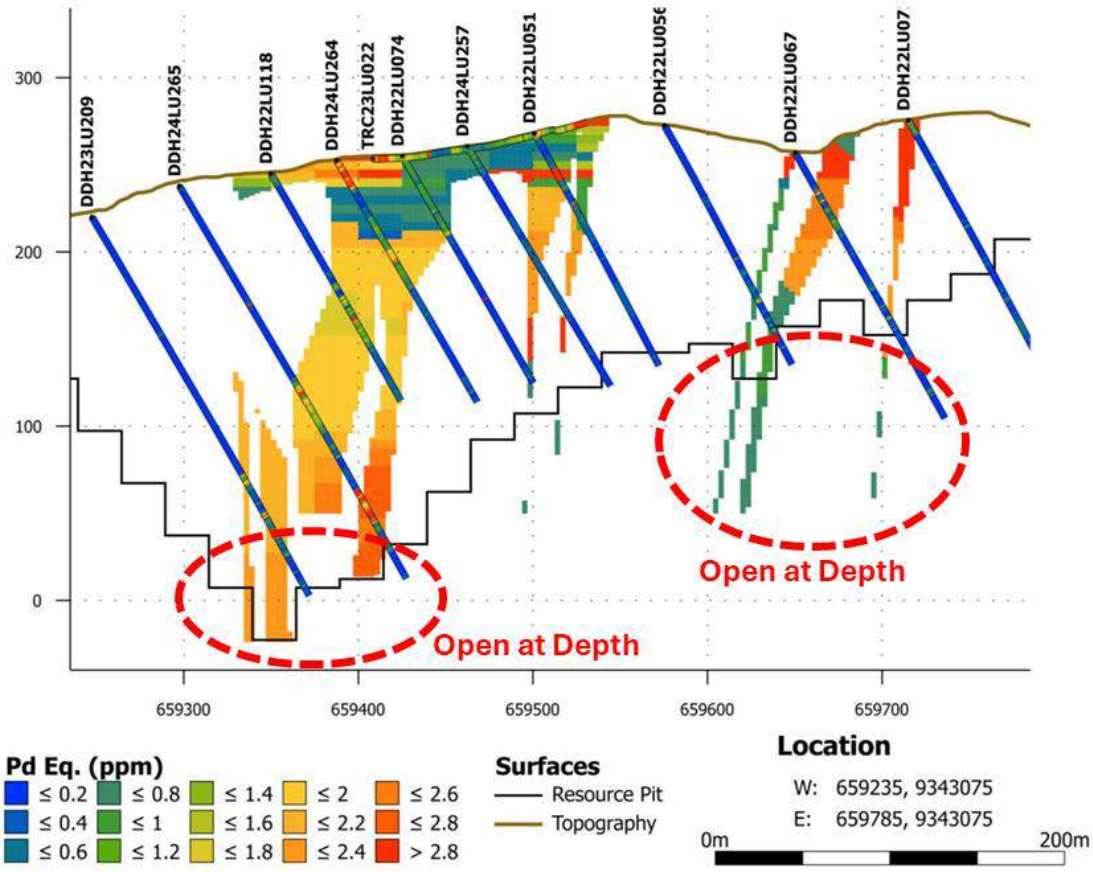


Figure 9-4: Section Southwest sector, showing MRE constraining pit shell reaching the limit of drilling data, and the presence of additional mineralized zones indicating potential expansion zones for the MRE

Source: Bravo, 2026.

9.6.1.4 Magmatic Massive Sulphide Ni-Cu-PGM Mineralization – North Sector – Hole LU330 Target

The magmatic massive to semi-massive Ni-Cu-PGM sulphide mineralization is spatially associated with, but genetically separate from, the stratiform PGM+Au+Ni mineralization that constitutes the actual Mineral Resource Estimate. It was recently recognised in 2026 on drill holes DDH26LU330 and DDH26LU347, associated with a chromitite interval hosted in orthopyroxenite of the Transition Zone. Euhedral chromite crystals enclosing sulphide blebs, with partially corroded faces, indicate the coexistence of chromite and an immiscible Ni-PGE-bearing sulphide liquid, and therefore a magmatic origin directly related to the Luanga intrusion (Table 9-5).

Table 9-5: Significant drill intercepts, North sector – hole LU330 target

Hole ID	Dip / Az (°)	From (m)	To (m)	Interval (m)	Ni (%) sulph.	Cu (%) sulph.	PGM+Au (g/t)	Rank
DDH26LU347	-60 / 090	205.58	219.02	13.44	1.55	0.33	2.02	20.8
including		*205.58*	*212.30*	*6.72*	*2.25*	*0.55*	*3.14*	–
and		122.70	128.70	6.00	0.01	0.01	6.81	–
DDH26LU330	-60 / 090	215.33	220.30	4.97	1.00	0.37	1.28	5

Hole ID	Dip / Az (°)	From (m)	To (m)	Interval (m)	Ni (%) sulph.	Cu (%) sulph.	PGM+Au (g/t)	Rank
including		*215.33*	*218.00*	*2.67*	*1.75*	*0.68*	*2.11*	–
and		202.55	204.35	1.80	1.41	0.29	1.30	2.5

Notes: All From, To and Interval values are downhole depths. Rank = interval x Nickel grade, used as a ranking device only, not a resource estimate.
Source: Bravo, 2026.

9.6.1.5 Hydrothermal Massive Sulphide Ni-Cu-PGM Mineralization – North Sector - Babylon Target

The Babylon target lies immediately adjacent to, and stratigraphically below, the North Sector of the Luanga deposit. It was generated to test the nickel-copper potential of a large, high-amplitude magnetic anomaly coincident with a strong electromagnetic conductor and a gravity high defined by the ground micro-gravity survey completed in 2024. The actual diamond drilling programme tested the basal zone of the Luanga MUM Complex.

Drilling intersected numerous sulphide-rich intervals, locally reaching semi-massive and massive textures, predominantly as sulphide-matrix breccias. The mineralization is hosted on a Fe-Ca-K metasomatic zone which was superimposed to the basal zone of the Luanga Complex.

In August 2026 a Deep Induced Polarization traverse across Babylon identified strong chargeability with coincident low resistivity at depth, interpreted by the Company to correspond to the Babylon sulphide mineralization extending deeper and potentially at a steeper angle than the drilling completed to date (Table 9-6 and Figure 9-5).

Table 9-6: Significant drill intercepts, North sector – Babylon target

Hole ID	Dip / Az (°)	From (m)	To (m)	Interval (m)	Ni (%) sulph.	Cu (%) sulph.	PGM+Au (g/t)	Rank
DDH22LU047	-60 / 090	131.11	142.15	11.04	2.04	1.23	4.24	22.5

Notes: All From, To and Interval values are downhole depths. Rank = interval x Nickel grade, used as a ranking device only, not a resource estimate.
Source: Bravo, 2026.

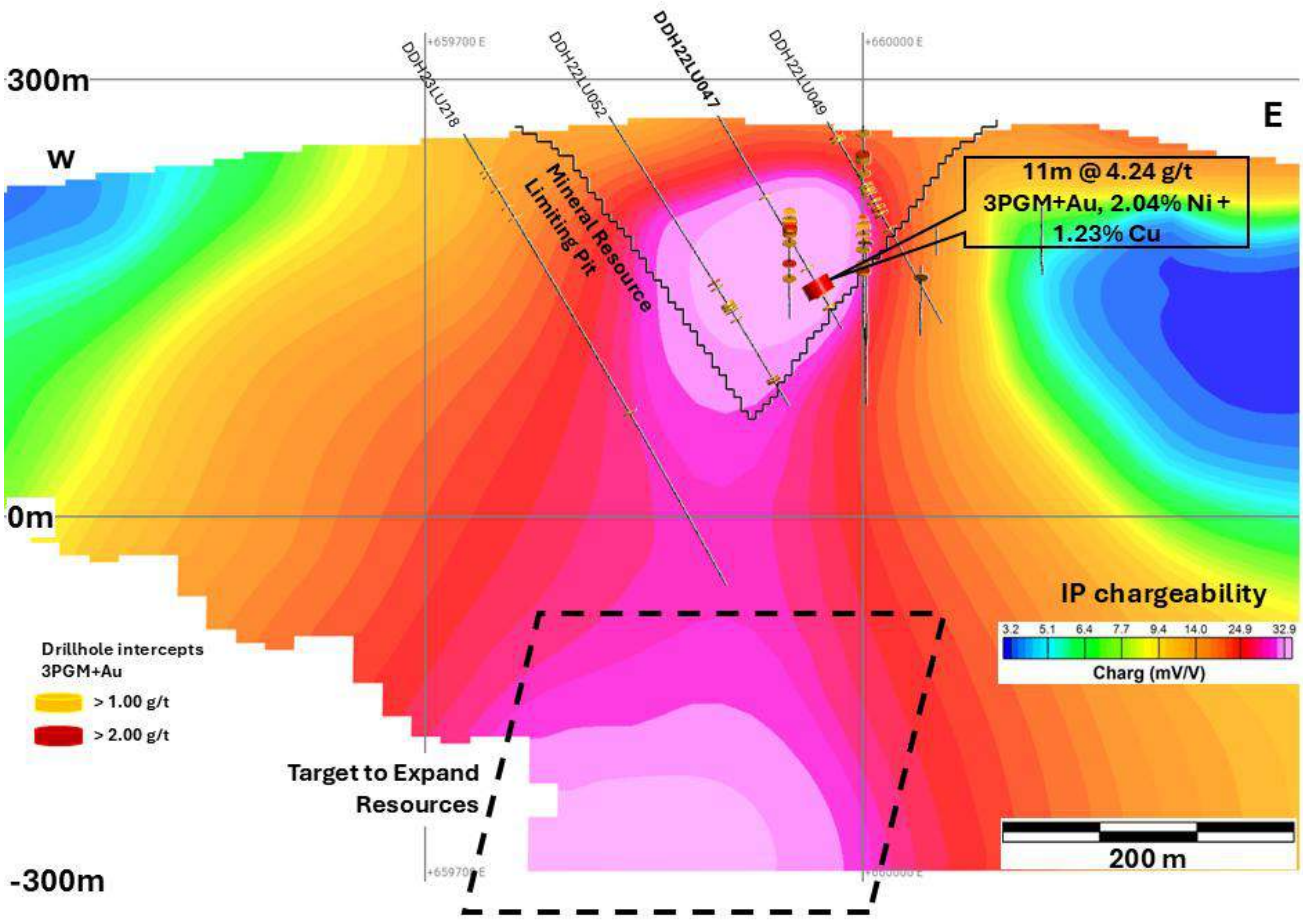


Figure 9-5: Cross section on the North sector (Babylon target) showing potential for hydrothermal massive sulphide mineralization

Source: Bravo, 2026.

9.6.1.6 Crescent Zone

Crescent is a new target area defined by a repetition of the Transition Zone (orthopyroxenites, harzburgites and chromitites) that occurs in the central east of the Luanga Complex within the norites of the Mafic Zone. Drilling in 2026 intercepted PGM mineralization comparable to that delineated within the deposit area, representing a potential opportunity to increase the current Mineral Resources.

9.6.1.7 Northern Extension of Luanga North Sector

Mapping within the northern extension of the Luanga Mafic-Ultramafic (MUM) Complex has identified chromitite and orthopyroxenite outcrops and float blocks in an area previously mapped as gneissic terrain. This suggests that the North Sector mineralized limit can extends further north than currently recognized, beyond the existing drilling grid, and represents significant potential to expand the current mineral resource estimate for the Luanga deposit

9.6.2 Prospects for New PGM Mineral Resources

9.6.2.1 Luanga South MUM Complex

Luanga South has the same stratigraphy as the Luanga MUM Complex and occurs along 6.5 km of strike in the south-southeastern portion of Bravo's mineral property. The area was subject to limited exploration by the past operator (Vale), which carried out a limited IP survey followed by a short scout drilling programme. Twelve of twenty-four drill holes returned PGM+Au mineralization, with average grades of 0.29 g/t to 14.17 g/t PGM+Au (Figure 9-6). Bravo is planning to start an exploration programme on this prospective target, which has the potential to add significant tonnage and metal to the current mineral resource inventory.

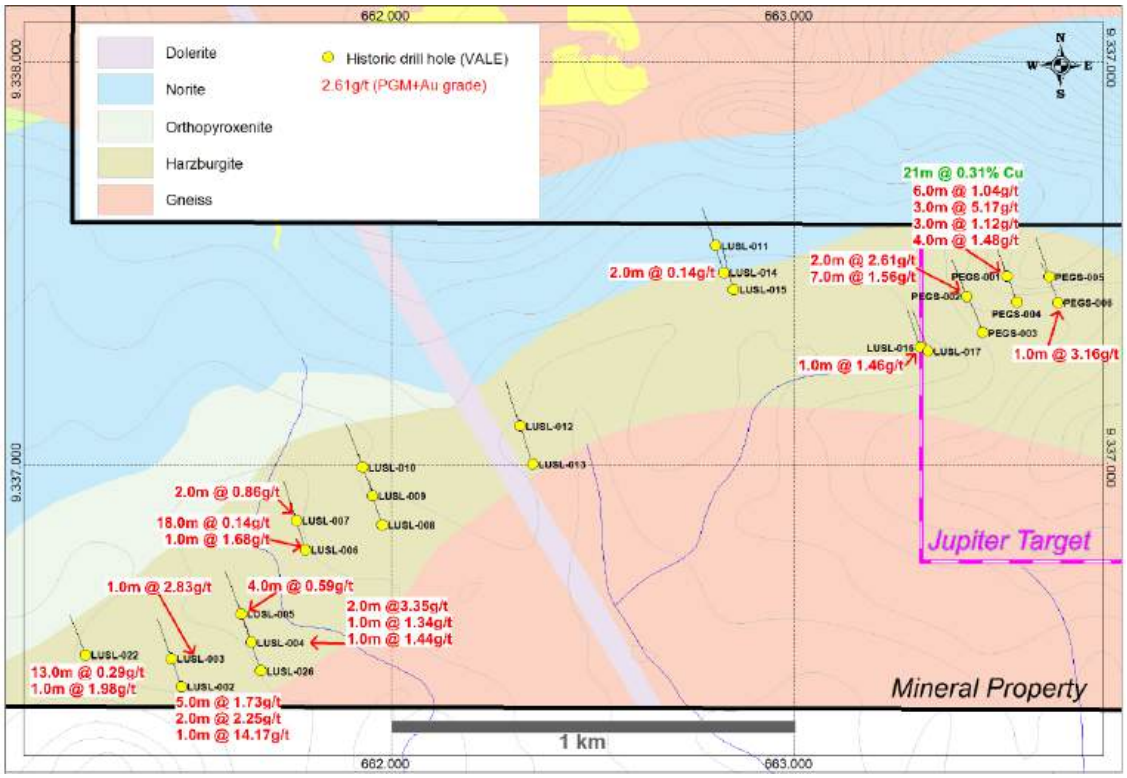


Figure 9-6: Historical drilling results on the Luanga South MUM Complex

Source: Bravo, 2026.

9.6.3 Prospects for Cu-Au and Magmatic Ni-Cu Deposits

9.6.3.1 Introduction

A property-wide helicopter-borne Time-Domain Electromagnetic (HeliTEM) survey carried out in 2023, targeting magmatic massive-sulphide mineralization, generated 17 priority-one and 16 priority-two electromagnetic (EM) anomalies within the Luanga property, located both within or adjacent to the Luanga complex and in other parts of the property. This data was integrated with historical structural and geochemical data resulting in the definition of 17 priority exploration targets for follow up (Figure 9-7).

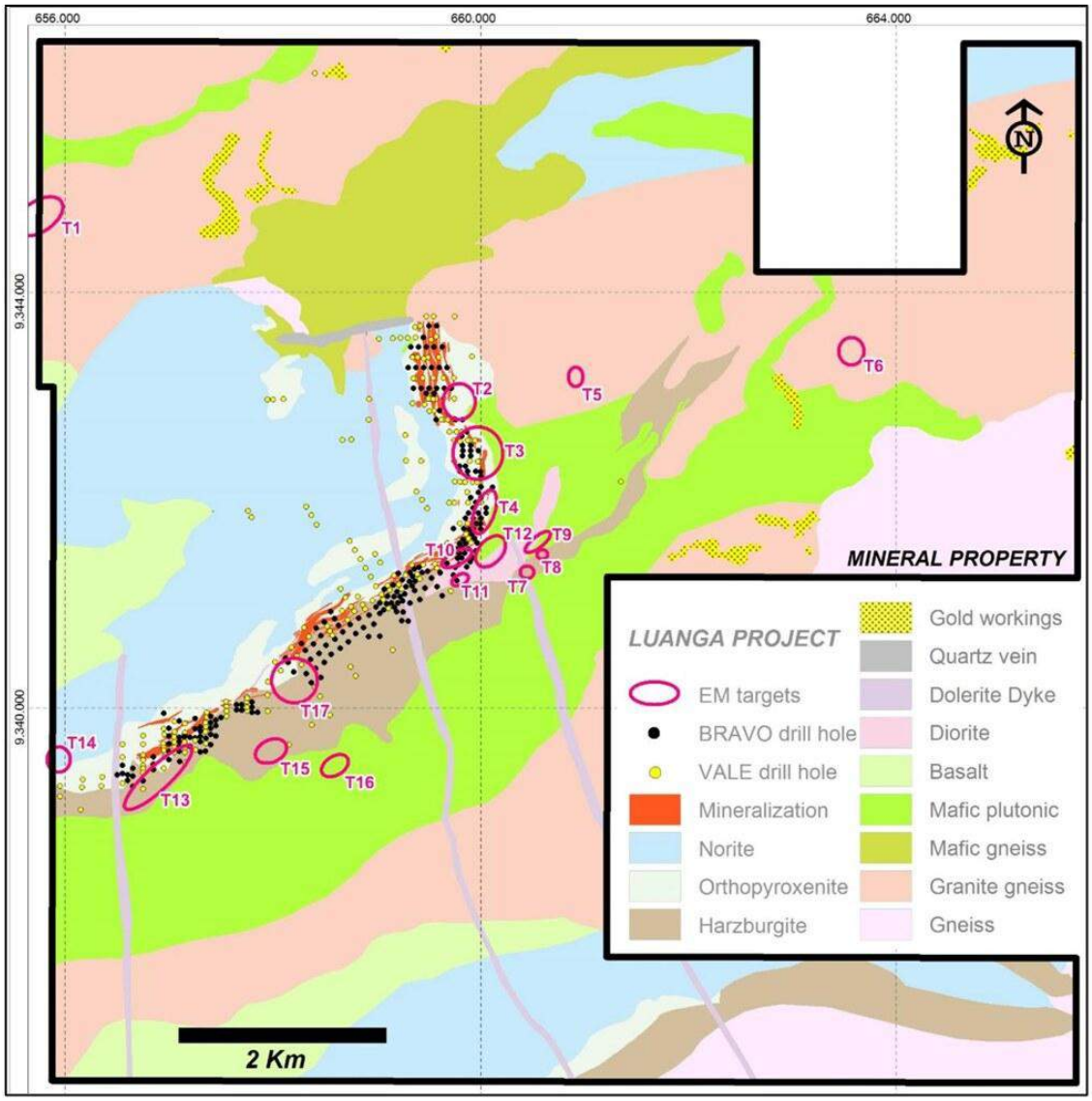


Figure 9-7: 2023 HeliTEM survey – 17 first-priority EM targets shown over Luanga geology

Source: Bravo, 2026.

Drill testing of HeliTEM targets in 2024 returned encouraging results for different styles of mineralization outside of the known Luanga PGM+Au+Ni deposit. The highlight of this programme was the discovery of high-grade Cu-Au mineralization at T5, with drill hole DDH2405T002 returning 11.48 m at 14.3% Cu, 3.3 g/t Au. Mineralization at T5 occurs as a 250 m long, E-W oriented, 150 m deep, near vertical, tabular, massive to brecciated sulphide orebody. As per its alteration assemblage and textures, the mineralization at T5 was initially interpreted as being of IOCG affiliation, later revised to an iron sulphide copper-gold (ISCG) affiliation. Massive to brecciated sulphide mineralization was also defined at the T6 and T16 targets, with moderate Cu and Ni grades and negligible PGM+Au.

Additional exploration work (mapping, soil geochemistry, ground geophysics, trenching and diamond drilling) continued outside the Luanga PGM+Au+Ni deposit in 2025, focused on the then recently demonstrated potential for IOCG-style mineralization as well as aiming at magmatic

Ni/Cu potential. Additional targets were selected from the HeliTEM survey, combined with regional structures, aeromagnetic anomalies and Cu-in-soil anomalies (Figure 9-8).

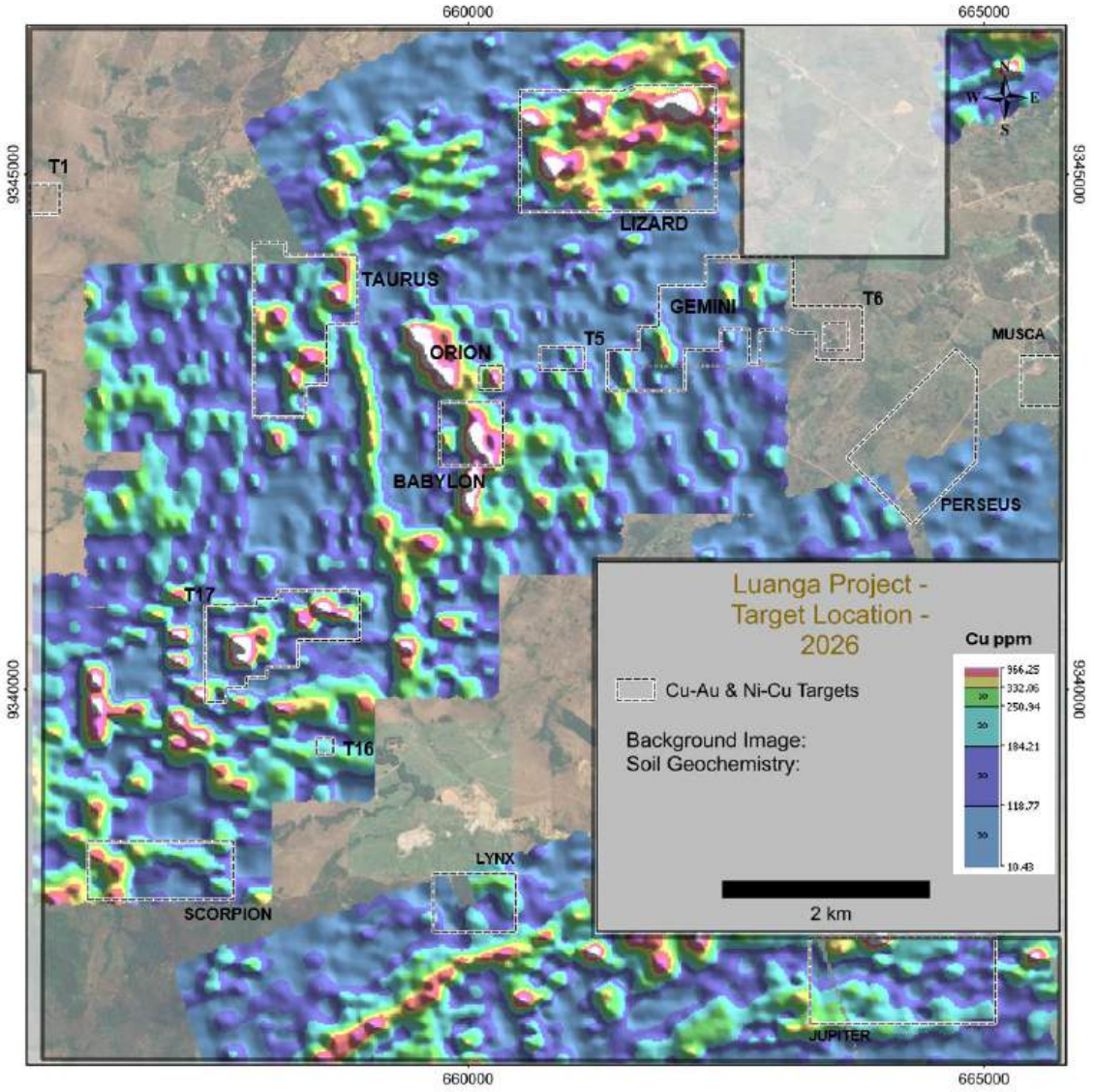


Figure 9-8: Luanga soil geochemistry – Cu (ppm)

Source: Bravo, 2026.

Exploration in 2025 focused on two main areas, the structural corridor that hosts the high-grade T5 IOCG mineralization and included several HeliTEM exploration targets, and over five Cu-Au targets selected for their potential to host large-scale IOCG-style disseminated sulphide mineralization, being those Lizard, Scorpion, Jupiter, Lynx and Perseus. Additional drilling was performed at T5, T16 and T17. During the year, limited trenching and drilling were executed at the Lizard, Taurus and Orion targets.

In late 2025, a review of the geological characteristics of the Luanga property indicated that the Cu-Au mineralization outside the Luanga deposit can be classified as belonging to the ISCG deposit type. This classification would encompass the targets containing massive sulphide drilled by then, including T5, T6 and T16.

ISCGs are a closely related deposit type to IOCGs and part of the broader Copper–Gold–Iron (CGI) mineral system. A key distinction in the ISCG deposits is the absence of Fe oxides (magnetite or hematite), which are replaced by Fe sulphides, notably pyrrhotite and pyrite. ISCGs are also characterised by more abundant quartz and tourmaline, which is a feature observed at Luanga. The ISCG breccias at Luanga contain abundant quartz fragments together with fragments of other alteration minerals in a matrix of sulphides.

A Phase 1 exploration programme was designed in early 2026 to better define and test the Cu–Au targets (Figure 9-9). The programme includes initial target refinement through geological mapping, additional geochemical sampling of soils and rock chips, followed by geophysical surveys and drilling.

Anomalous soil samples from several targets, initially analysed by pXRF, were selected and the pulps sent for reanalysis at SGS Geosol. Soil samples are being prepared by drying, disaggregation and sieving at 120 mesh, and analysed by methods FAA313 (Au by Fire Assay with AAS finish) and ICM14B (aqua regia digestion and ICP OES / ICP MS finish – 51 elements).

A 50 line-km of deep-Induced Polarization (Deep-IP) surveying is ongoing across multiple targets. IP is a well-established method for targeting disseminated and stockwork sulphide mineralization, which constitutes the bulk of mineralization in IOCG/ISCG systems. The survey is designed to investigate depths greater than 300 m from surface, beyond the limit of conventional IP methods. The Deep-IP survey will also cover portions of the Luanga deposit to assess the potential for deep extensions and additional occurrences of PGM+Au+Ni and ISCG/IOCG mineralization.

Additional geophysical surveys, including fixed-loop and borehole TEM, as well as structural work, will be performed on targets of known mineralization such as T5 and T16 to identify continuation of the high-grade mineralized shoots identified to date along strike and at depth. A Phase-1 8,000 m drill programme is planned to test the most prospective targets generated from the aforementioned exploration work.

Table 9-7 summarises the main features of the current exploration targets in the Luanga Project, and their distribution is shown in Figure 9-9, which also depicts the current Phase 1 exploration programme. It is important to note that the Luanga PGM+Au+Ni deposit is included as a reference location and does not constitute part of the actual Cu-Au exploration programme described herein.

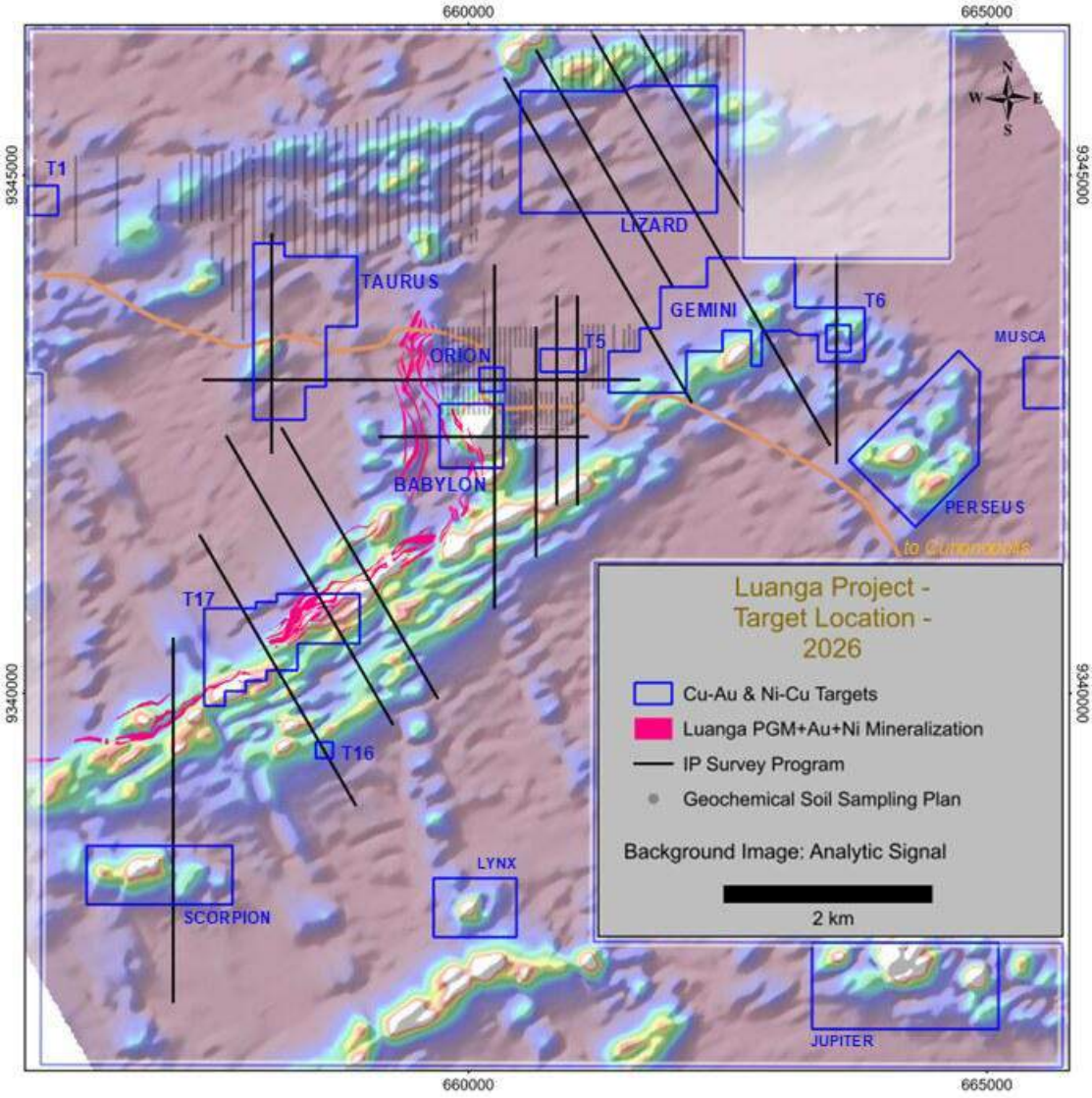


Figure 9-9: Location of IOCG/ISCG and magmatic Ni-Cu exploration targets in the Luanga Project, Phase 1 geochemical sampling programme and IP survey

Source: Bravo, 2026.

Table 9-7: Summary of Cu-Au IOCG/ISCG and Ni-Cu magmatic targets in the Luanga Project

Target ID	Style	Favourable Elements	Exploration Stage
T1 Target	Cu-Au IOCG/ISCG	EM conductor	In progress
		Regional shear structure	
T5 Target	Cu-Au ISCG	EM conductor	In progress
		Cu-oxide mineral at surface	
T6 Target	Cu-Au ISCG / Ni-Cu-PGM Magmatic	EM conductor Magnetic and gravity anomaly	In progress
T16 Target	Cu-Au ISCG	EM conductor Magnetic and gravity anomaly	In progress
T17 Target	Ni-Cu-PGM Magmatic / IOCG/ISCG	EM conductor	In progress
		Historic Cu in-soil anomaly	
Lizard	Cu-Au IOCG/ISCG	Historic Cu in-soil anomaly	In progress
		Regional shear structure	

Target ID	Style	Favourable Elements	Exploration Stage
Scorpion	Cu-Au IOCG/ISCG	Magnetic high	*To be started*
		Regional shear structure	
Taurus	Cu-Au IOCG/ISCG Ni-Cu-PGM Magmatic	EM conductors	In progress
		Regional shear structure	
Gemini	Cu-Au IOCG/ISCG	EM conductors	In progress
		Magnetic anomaly	
		Regional structures	
Babylon	Ni-Cu-PGM Magmatic / IOCG/ISCG	EM conductor	In progress
		Magnetic anomaly	
		Gravity anomaly	
Orion	Cu-Au IOCG/ISCG	Cu-oxide mineral at surface	In progress
Lynx	Cu-Au IOCG/ISCG	Magnetic anomaly	*To be started*
		Regional shear structure	
Perseus	Cu-Au IOCG/ISCG	Magnetic anomaly	*To be started*
		Regional structures	
Jupiter	Cu-Au IOCG / Cu-Ni-PGM Magmatic	Magnetic anomaly	*To be started*
		Regional shear structure	
Musca	Cu-Au IOCG/ISCG	EM conductors	*To be started*

Source: Bravo, 2026.

The relevant exploration results obtained to date on the most advanced targets will be briefly summarised in the next sections.

9.6.3.2 T5 Target

The T5 Target is located outside and approximately 1 km to the east of the Luanga Complex. It is currently an exploration stage target where Cu-Au mineralization was intercepted in several drill holes carried out to test an HeliTEM conductor.

To date, Bravo has drilled 28 drill holes at T5, totalling 5,926.25 metres. Sulphide mineralization was intercepted in 19 drill holes.

Table 9-8 and Figure 9-10 illustrate the significant mineralized intercepts and the location of drill holes, respectively.

Table 9-8: Main drill intercepts at the T5 target

Hole ID	From (m)	To (m)	Thickness (m)	Cu (%) sulph.	Au (g/t)
DDH2305T001	212.30	213.00	0.70	1.98	0.04
DDH2405T002	165.62	177.10	11.48	14.27	3.33
including	*167.50*	*170.36*	*2.86*	*22.91*	*3.62*
DDH2405T003	157.91	163.10	5.19	3.10	1.12

Hole ID	From (m)	To (m)	Thickness (m)	Cu (%) sulph.	Au (g/t)
DDH2405T004	153.60	162.35	8.75	9.48	2.08
DDH2405T005	43.50	48.50	5.00	0.59	0.19
DDH2405T009	148.90	166.32	17.42	3.49	0.95
including	*155.88*	*159.40*	*3.52*	*13.14*	*3.87*
DDH2405T010	140.63	144.90	4.27	4.66	0.97
DDH2405T011	155.50	156.50	1.00	0.30	0.10
And	323.55	324.50	0.95	1.19	0.50
DDH2405T012	156.00	159.20	3.20	8.00	1.83
DDH2405T013	107.05	109.65	2.60	2.79	0.57
And	152.95	157.85	4.90	0.93	0.68
DDH2405T014	162.00	165.00	3.00	1.63	0.68
DDH2405T015	121.70	135.09	13.39	2.93	0.39
including	*132.93*	*135.09*	*2.16*	*10.23*	*1.38*
DDH2405T016	174.45	178.25	3.80	1.06	0.35
DDH2405T017	152.55	154.60	2.05	0.92	0.23
And	180.63	182.10	1.47	0.78	0.19
DDH2405T018	54.40	56.40	2.00	1.20	0.48
DDH2405T021	139.25	142.55	3.30	1.09	0.26
DDH2505T024	114.25	120.90	6.65	2.57	0.55
including	*117.10*	*119.00*	*1.90*	*5.45*	*1.42*
DDH2505T025	91.79	94.70	2.91	6.01	0.43
DDH2505T026	65.55	73.86	8.31	1.65	0.15
And	99.36	105.56	6.20	2.07	0.48
Note: drill holes DDH2405T006, DDH2405T007, DDH2405T008, DDH2405T019, DDH2405T020, DDH2405T022, DDH2505T023, DDH2505T027 and DDH2505T028 returned no significant Cu-Au intersections.					

Source: Bravo, 2026.

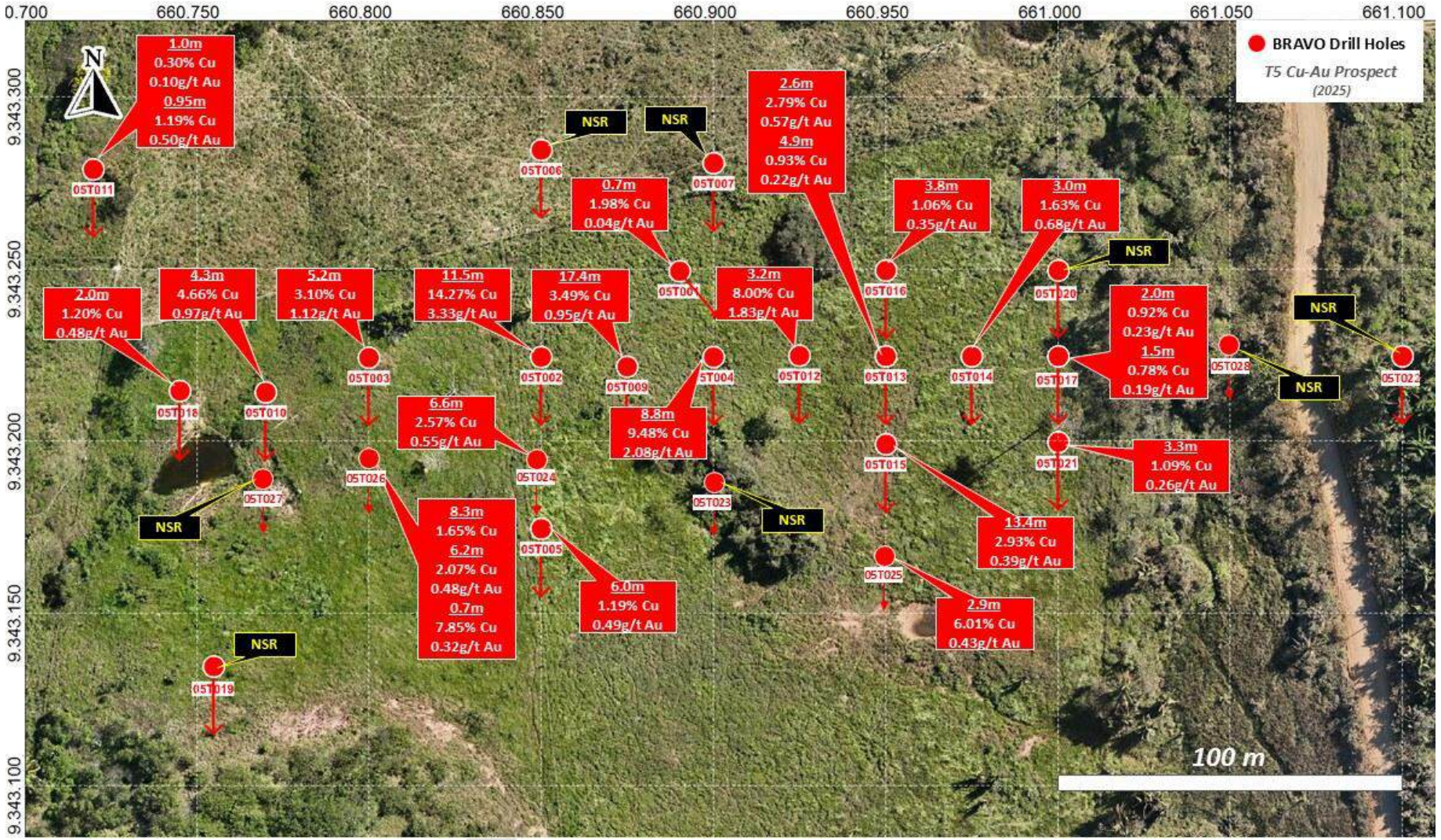


Figure 9-10: Drill hole location and Cu-Au intercepts at the T5 target

Note: Drill hole DDH2405T006 is not shown because it is located far from the mineralized zone. “NSR” = No significant result.
Source: Bravo, 2026.

The Cu-Au sulphide mineralization at T5 occurs primarily as chalcopyrite with subordinate pyrrhotite and 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 enveloped by a hydrothermal zone associated with the development of biotite, actinolite, scapolite, chlorite, apatite, tourmaline, carbonate and silica (Figure 9-11).

The whole system is controlled by a WNW-ESE fault structure steeply dipping to NNE that was generated under a brittle regime (brecciation with incipient foliation). The host rock is a tonalitic intrusion with a Th+U radiometric signature on the airborne survey. The hydrothermal assemblage, together with the structural control, suggest that the T5 Cu-Au mineralization can be ascribed to the ISCG deposit type.

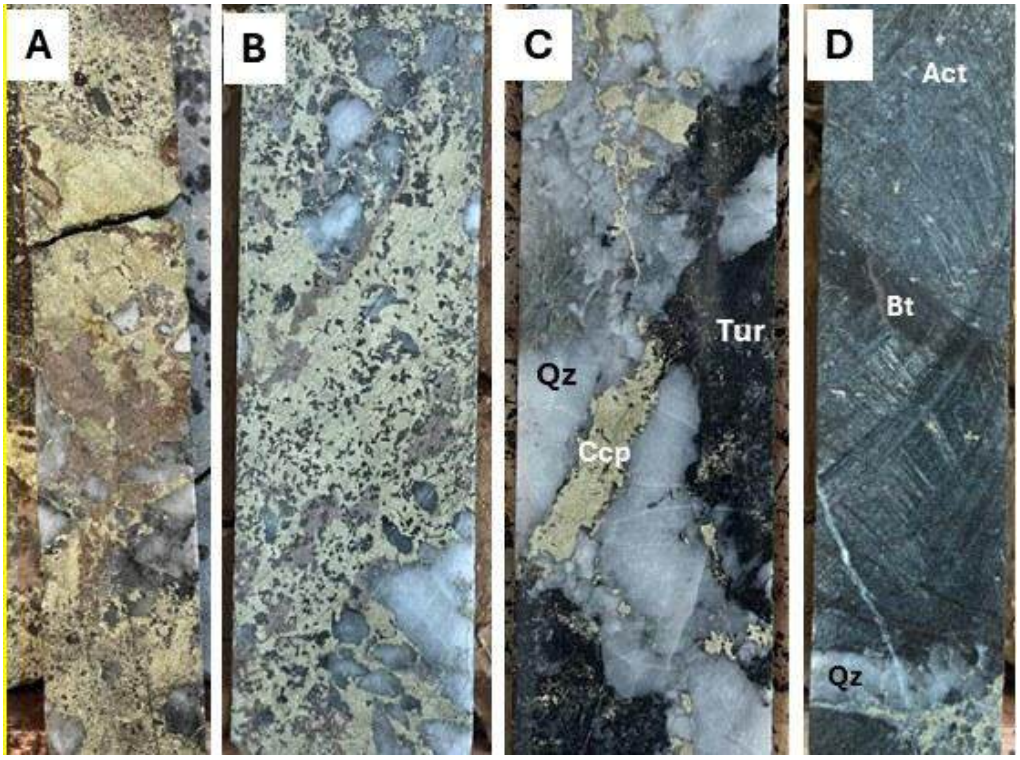


Figure 9-11: Core samples from T5 – mineralization and hydrothermal alteration

A and B: Sulphide breccia (chalcopyrite+pyrrhotite) with quartz clasts [DDH2405T002]. **C)** Breccia crosscutting quartz (Qz) and tourmaline clusters (Tur) with chalcopyrite (Ccp) [DDH2405T010]. **D)** Actinolite (Act) crosscut by biotite (Bt) and quartz+chalcopyrite veinlets [DDH2405T012].

Source: Bravo, 2026.

Exploration activities continue on the T5 Cu-Au target aiming to generate new drill targets to expand the existing mineralization with geological mapping, additional geochemical sampling and the planned geophysical surveys (IP and TEM). The soil grid is being extended to the NE and SW to connect T5 to Orion and Babylon. To date, no mineral resources have been estimated for this area.

9.6.3.3 T16 Target

T16, a strong electromagnetic (EM) conductor situated to the south and outside the Luanga mafic-ultramafic Complex, was the object of an exploratory drilling programme in 2023 and 2024 and follow-up drilling in 2025. Two drill holes completed in 2023 and 2024 intersected brecciated sulphide mineralization, revealing promising Cu and Ni grades, associated with low platinum-group metal (PGM) values. The best drill intercepts included 1.29 m at 2.39% Cu, 1.27% Ni, from 94.59 m downhole (DDH2316T001) and 3.0 m at 0.87% Cu, 0.48% Ni, from 99.30 m downhole (DDH2416T002) (Figure 9-12).

Five drill holes, comprising 761 linear metres were drilled at T16 during 2025. DDH2516T005 was drilled 25 m to the northeast of previous drill hole DDH2316T001 to test the lateral continuity of the Cu-Ni mineralization. The drill hole intercepted a 7.8 m wide hydrothermal system comprising biotite, actinolite, scapolite and almandine. Massive to semi-massive sulphide zones occur (up to 70% sulphides) within this alteration zone, composed of pyrrhotite, pyrite and subordinate chalcopyrite, locally showing a brecciated texture with abundant quartz fragments plus fragments of biotite, actinolite and carbonate enveloped by the sulphide. Results include 6.54 m at 2.25% Cu, 0.55% Ni, from 82.96 m downhole. Except for drill hole DDH2516T003, the other drill holes managed to intercept Cu-Ni mineralization, albeit generally of lower grade and in narrower intercepts (Table 9-9).

The hydrothermal system is hosted within basement rocks of dioritic to tonalitic composition. At surface, a NW silicification zone is observed, with some outcrops of quartz veins with black tourmaline aggregates. The hydrothermal alteration observed at T16 and the textural features of mineralization indicate an ISCG affiliation for the target and similarities to the mineralization observed at T5.

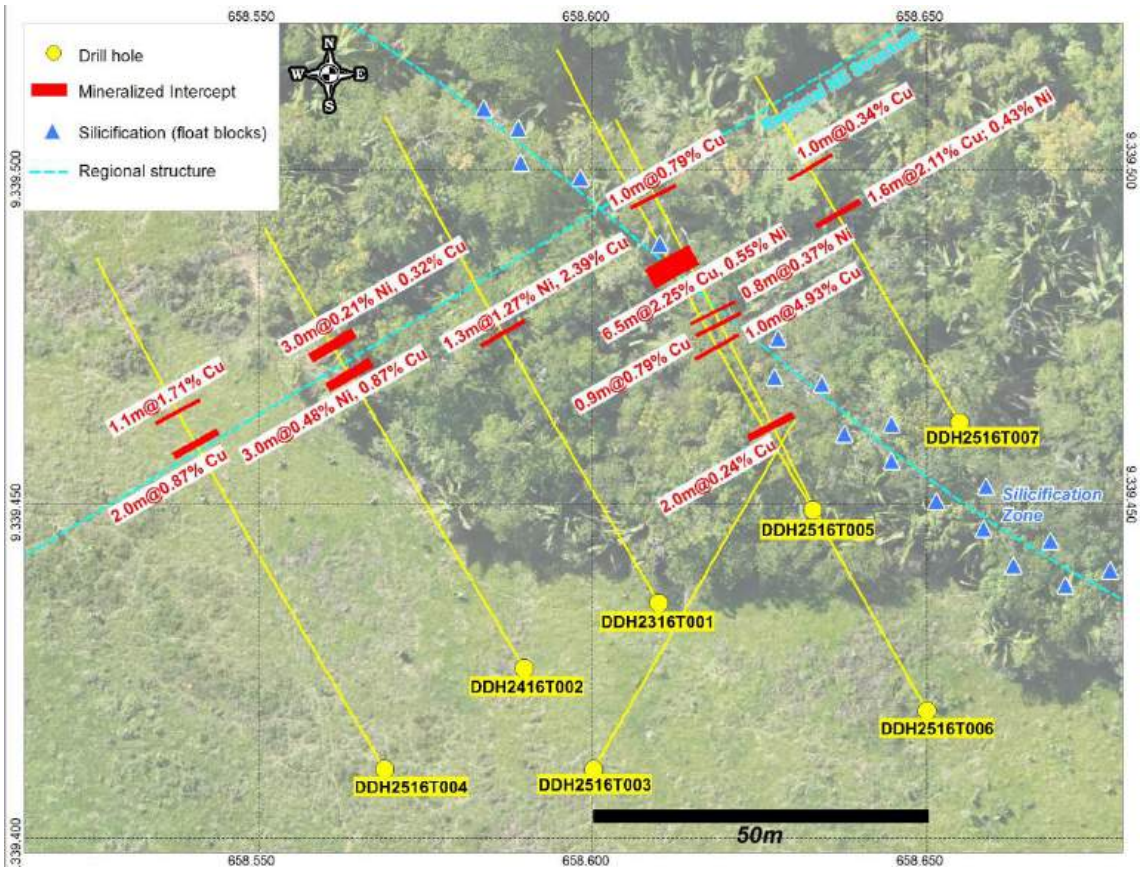


Figure 9-12: Drilling at the T16 target

Source: Bravo, 2026.

Table 9-9: Main drill intercepts at the T16 target

Hole ID	From (m)	To (m)	Interval (m)	Ni (%) sulph.	Cu (%) sulph.
DDH2316T001	94.59	95.88	1.29	1.27	2.39
DDH2416T002	99.30	102.30	3.00	0.48	0.87
and	109.30	112.25	2.95	0.21	0.32
DDH2516T004	108.75	110.75	2.00	0.07	0.87
and	119.95	121.00	1.05	0.04	1.71
DDH2516T005	57.68	58.68	1.00	0.02	4.93
and	82.96	89.50	6.54	0.55	2.25
DDH2516T006	92.10	94.10	2.00		0.24
and	125.40	126.35	0.95		0.79
and	129.15	130.00	0.85	0.37	
and	166.20	167.20	1.00		0.79
DDH2516T007	71.30	72.85	1.55	0.43	2.11
and	87.70	88.70	1.00		0.34

Source: Bravo, 2026.

An IP line is planned to cross-cut the T16 target which will be integrated with additional mapping and geochemical sampling to generate new drilling targets.

9.6.3.4 T6 Target

The T6 target consists of a strong and discrete EM anomaly located in the eastern part of the Gemini target. This anomaly also coincides with discrete magnetic and gravimetric anomalies.

Four drill holes were drilled at the T6 target to date. The mineralization apparently responsible for the strong T6 conductor was intercepted in drill hole DDH2406T002 and consists of around 6 m of massive/semi-massive/breccia sulphides, composed predominantly of pyrrhotite. This sulphide zone was the best intercept in the target assaying 6.27 m at 0.41% Cu, 0.10% Ni from 54.09 m. In this drill hole, the massive to brecciated sulphide lens occurs within a hydrothermally altered zone with biotite and amphibole at the contact of an ultramafic rock (wehrlite) with a tonalitic sequence. The sulphide locally shows a brecciated texture with fragments of quartz enveloped by the sulphide matrix. Drill hole DDH2406T004 returned 2.26 m at 0.37% Cu from 34.9 m (Figure 9-13).

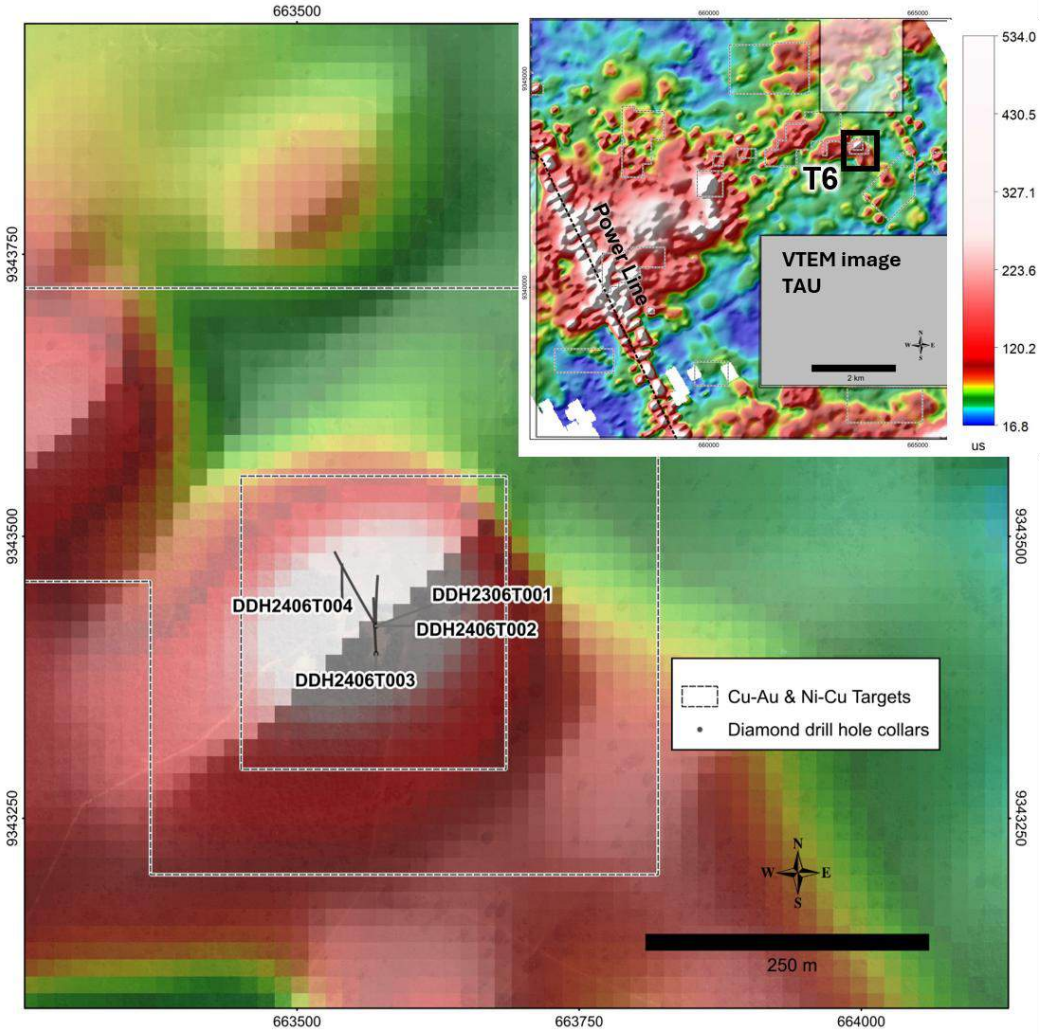


Figure 9-13: T6 target diamond drill hole locations over the VTEM survey image

Source: Bravo, 2026.

Given the intercept of massive sulphides, the occurrence of strong and coincident magnetic, electromagnetic and gravity anomalies, further work is warranted at the T6 target. The target is part of the current deep-IP survey and additional mapping and geochemical sampling are planned.

9.6.3.5 T1 Target

The T1 target consists of a strong HeliTEM anomaly located at the NW edge of the Luanga property. Two drill holes were drilled in 2024 at the T1 target, totalling 177.75 linear metres. Drill hole DDH2401T002 intercepted a narrow semi-massive (20%) sulphide interval with pyrite and subordinated chalcopyrite from 93.98 to 95.70 m, hosted by a hydrothermal alteration zone composed of biotite-amphibole-carbonate plus quartz veining. This zone assayed 0.19% Cu, 0.13% Ni. The host rock is a sequence of granitic rocks with some mylonitic passages, intercalated with gabbros. The HeliTEM conductor at T1 has a very strong response and the occurrence of sulphides in the preliminary drilling suggests that this target warrants additional work. Further mapping, geochemical sampling and fixed loop TEM are planned for T1.

9.6.3.6 T17 Target

The T17 target is located within the Central Sector of the Luanga PGM+Au+Ni deposit, close to a complex structural zone that offsets this sector from the Southwestern Sector. Malachite-bearing gossans occur as float blocks at surface. Geological mapping indicates that the distribution of the gossanous blocks is controlled by this NW-SE regional structure, which crosscuts the Luanga PGM deposit.

Three drill holes were carried out to test two individual EM plates. Drill hole DDH2317T001 intersected 1 metre grading 0.35% Ni, 0.26% Cu and drill hole DDH2317T002 intersected 1 metre grading 0.91% Ni, 0.11% Cu. DDH2517T004 (120 m depth) was drilled to investigate Cu potential along the interpreted structure (Figure 9-14). The drill hole intercepted 2.0 m at 0.51% Cu immediately below the base of oxidation, 23.35 m downhole. All drill holes reported low PGM and Au grades. Detailed mapping and structural analysis are planned. An IP line will also traverse the target.

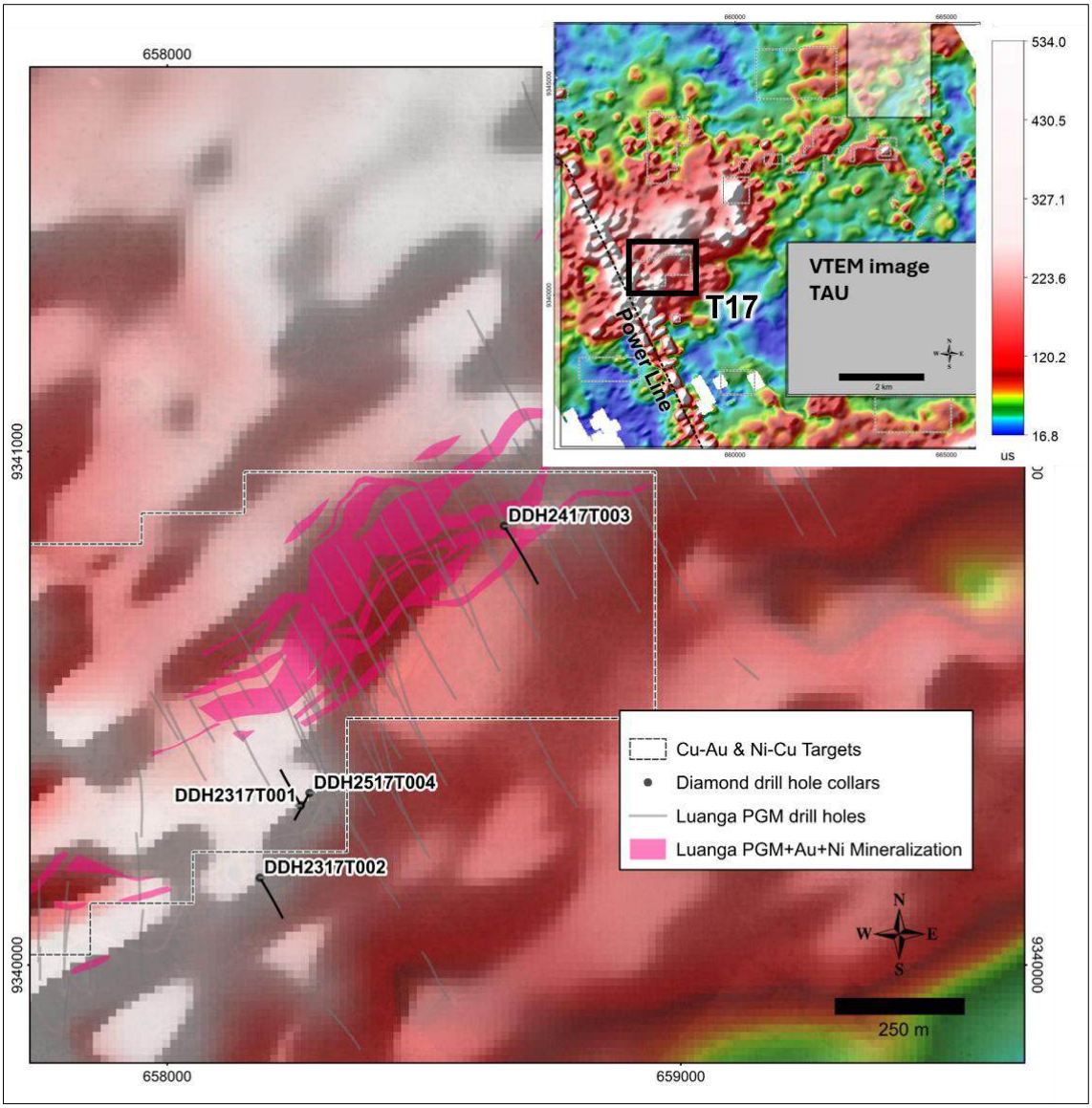


Figure 9-14: T17 target drill hole locations over the VTEM image

Source: Bravo, 2026.

9.6.3.7 Lizard Target

The Lizard target is located approximately 2,500 metres northwest of the T5 target and about 3,000 metres from the North Sector of the Luanga PGM+Au+Ni deposit. The target covers the Cinzento shear zone.

In 2025, a soil grid was conducted along 100 m-spaced gridlines (locally 50 m-spaced), with stations spaced 25 m apart covering approximately 192 hectares and totalling 837 soil samples. Soil samples (~3 kg) were dried and sieved to <1 mm at Luanga field camp. The <1 mm fraction was analysed with a portable XRF (pXRF) for 33 elements. In 2026, pulps from 353 soil samples over anomalous zones were selected and sent for reanalysis at the SGS Geosol laboratory (multi-element ICP and Au).

Mapping at Lizard shows that the target is dominated by metamafic rocks (metagabbros/metabasalts) forming an E-W belt, with tonalites occurring on a restricted area in the southeastern portion of the geochemical grid. Between these rock sequences, a large corridor of hydrothermally-altered and deformed rocks and FeOx-rich siliceous breccias occurs. Siliceous breccia zones with FeOx are predominantly hosted within chlorite-rich hydrothermally altered rocks. Supergene Cu minerals (e.g., malachite) occur on the surface in these breccias and surrounding chlorite schists. Rock-chip samples of the breccias locally reported Cu grades (pXRF) from 0.20 to 0.78% Cu, with one sample assaying 1.39% Cu.

Lizard is characterised by a broad soil geochemical anomaly for copper in soils above 400 ppm, covering approximately 1,800 m vs 600 m along an E-NE trend. A stronger and well-defined anomalous zone approximately 850 m vs 450 m occurs in the target's NE sector. This anomaly, which is EW aligned, was the focus of more detailed exploration work with auger drilling, trenching and diamond drilling.

The auger drilling programme comprised 30 drill holes and 269.70 metres. Samples were analysed by pXRF and are indicative. The location of the auger holes with the best intercepts suggested that the primary mineralization is aligned along a WNW-ESE trend.

Best auger intercepts are:

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

One N-S trench (133.30 metres long) followed the auger drilling in this area. This programme generated 141 channel samples which were analysed by fire assay (Au) and ICP-OES (multi-elements) at SGS Geosol.

Trench mapping revealed that both the mineralized zone and its host sheared schists exhibit a consistent N70-80W strike, dipping 70-80° towards the north. Main Cu intersections in the trench are shown in Table 9-10 and trench and drilling location with respect to the soil geochemistry and auger drilling results are presented on Figure 9-15.

Table 9-10: Copper intersections on trench TRC25LZ001

Trench ID	From (m)	To (m)	Length (m)	Cu (%)	Cu intersection
TRC25LZ001	49.55	54.85	5.30	0.25	**5.30 m at 0.35% Cu**
	54.85	58.35	3.50	0.12	3.50 m at 0.13% Cu
	58.35	65.10	6.75	0.33	**6.75 m at 0.40% Cu**

Source: Bravo, 2026.

Two drill holes were performed aiming to intercept the mineralization observed in the trench and auger drilling, but failed to crosscut any copper mineralization and also did not cut any of the FeOx-rich siliceous breccias associated with oxidised copper seen at the top of the hill to the south. Ongoing mapping and structural work should help in better defining mineralization and the structural framework of the target.

Four deep IP lines are being surveyed at Lizard and this information will be integrated with the expanded geochemical dataset and new detailed mapping information to generate new drill targets.

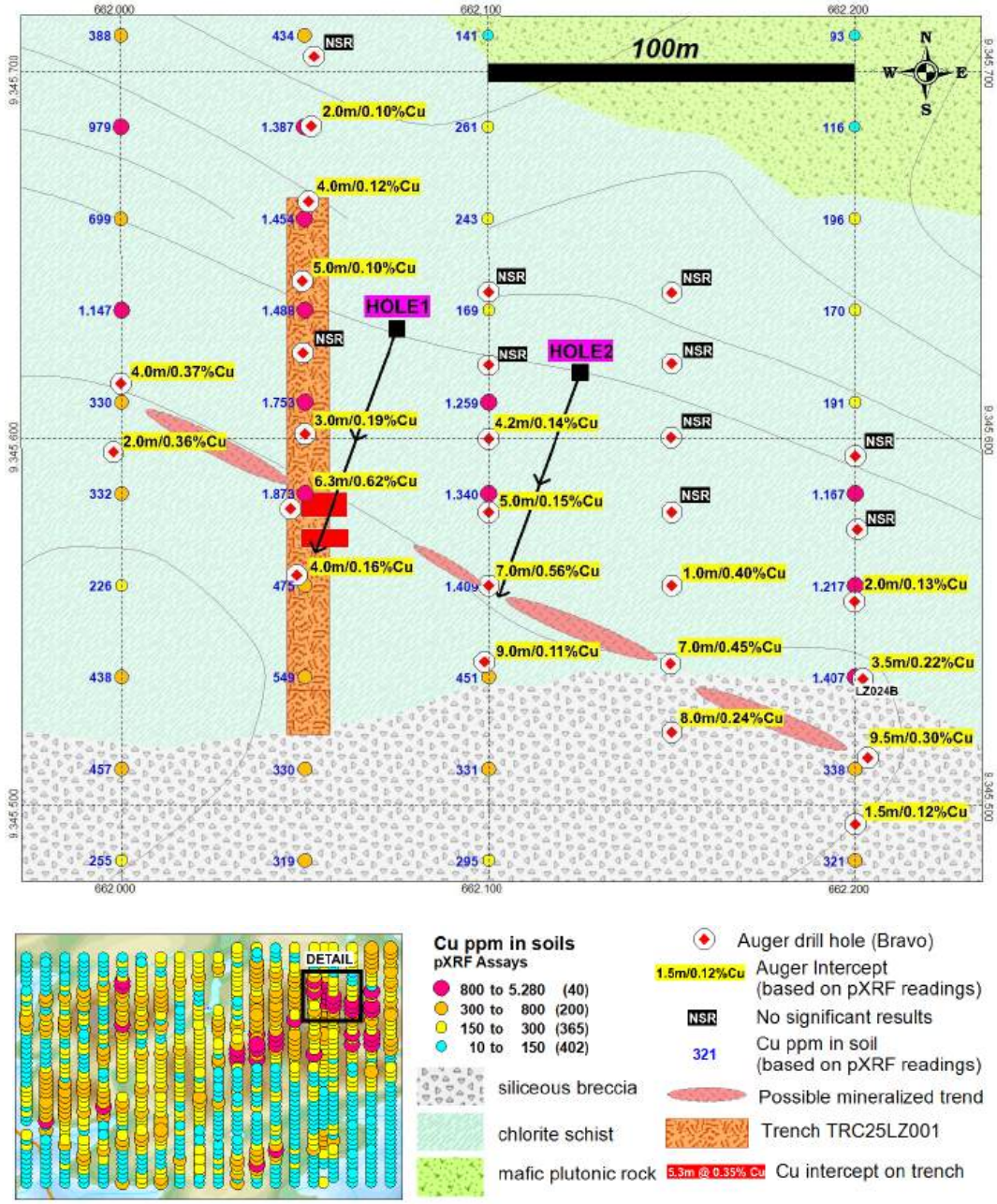


Figure 9-15: Soil geochemistry, trenching, auger and diamond drilling on the main zone at the Lizard target

Source: Bravo, 2026.

9.6.3.8 Gemini Target

The Gemini target covers several (T39 to T45) anomalies defined by the HeliTEM survey which are located from 500 m up to 1,500 m east/northeast of the T5 Cu-Au target. Exploration work in 2025 encompassed a geochemical grid covering approximately 133 hectares, followed by detailed mapping, soil sampling and pXRF analysis of soils. The main objective was to identify anomalous Cu-Au-Ni zones that could be tested by diamond drilling.

Five Cu anomalous zones were identified considering values above 120 ppm, with values above 400 ppm Cu frequent. These zones generally showed NE-SW and E-W trending controls (Figure 9-16), which were also confirmed by the regional HeliTEM survey.

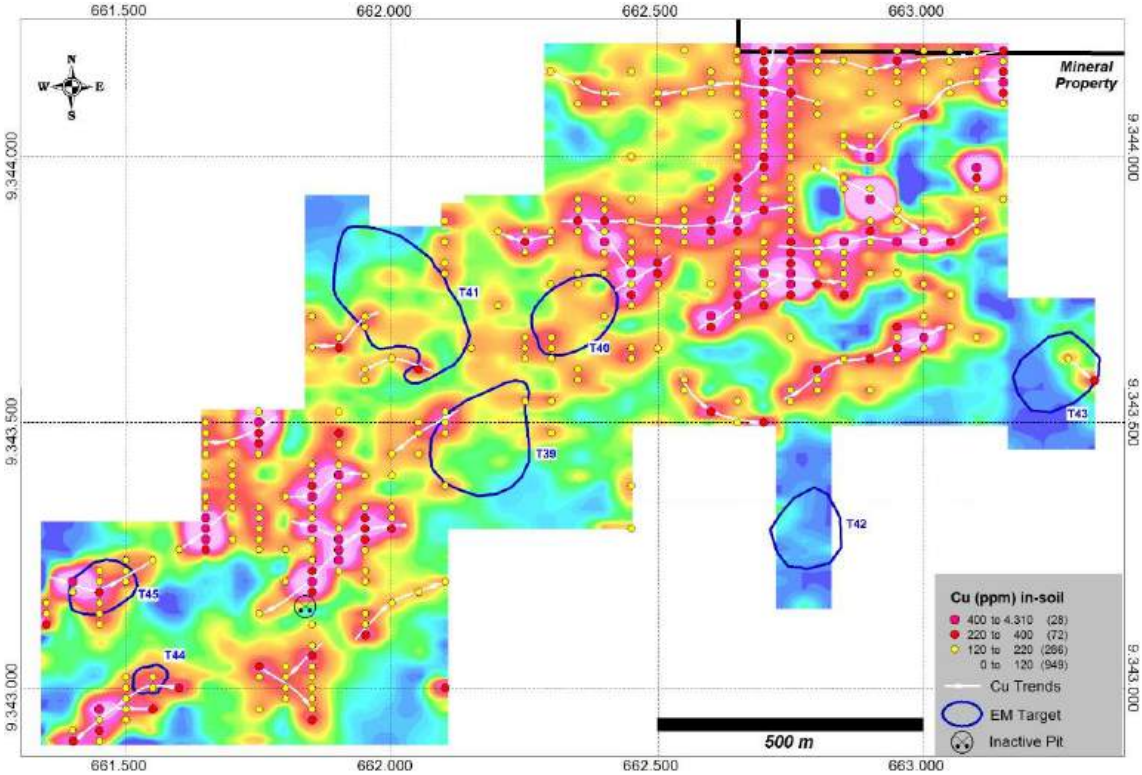


Figure 9-16: Cu (ppm) in soils and HeliTEM conductors at the Gemini target

Note: Soil results obtained from pXRF equipment. Best Cu results reanalysed by independent commercial laboratory.

Source: Bravo, 2026.

In 2026, two IP lines are planned at the Gemini target. About 100 soil samples previously analysed by pXRF over the anomalous areas were sent for reanalysis at SGS Geosol. Detailed mapping of the main anomalies and additional geochemical sampling are planned. This data will be integrated for the generation of drilling targets.

9.6.3.9 Taurus Target

The Taurus target is located from 2,300 to 3,000 metres to the west of the T5 target and on the northern edge of the Luanga mafic-ultramafic layered intrusion. In 2025, a geochemical soil grid covering approximately 128 hectares was established to investigate three EM conductors (T48, T49 and T51). Soil samples were analysed by pXRF and anomalous Cu-Au-Ni results were verified at the SGS Geosol laboratory.

Four areas of interest, named as Zones 1, 2, 3 and 4, were delineated based on Cu in soil values greater than 220 ppm Cu, with internal zones greater than 440 ppm Cu (Figure 9-17). The most expressive anomalies are Zone 1 with a 750 m long by ~400 m wide anomaly and Zone 2 with a 750 m long by 450 m wide anomaly.

In terms of geochemical signature, Zones 1, 3 and 4 are basically defined by anomalous Cu values while Zone 2 shows a clear Cu-Au association where Au values are locally greater than 100 ppb (Figure 9-18).

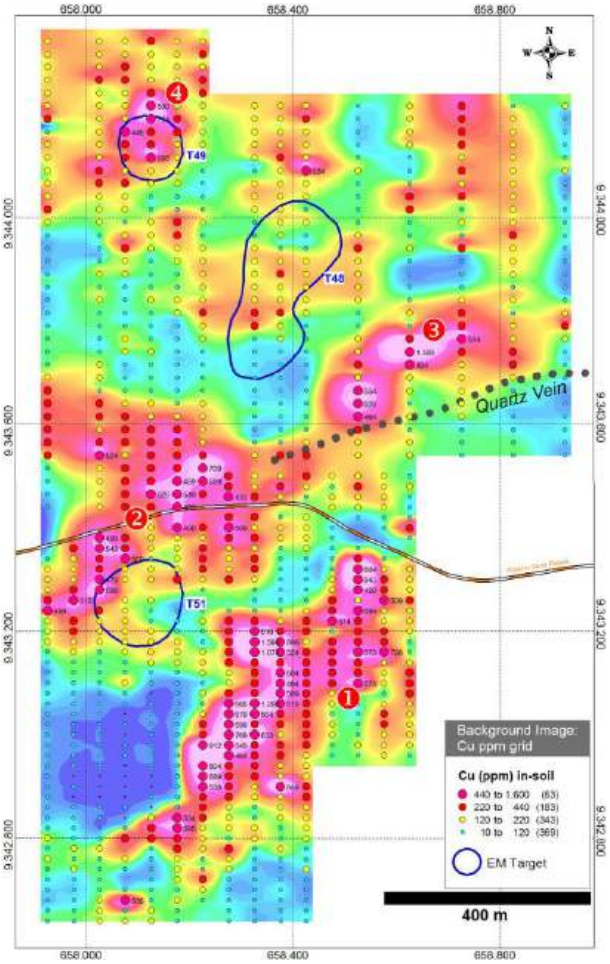


Figure 9-17: Cu (ppm) in soil results over Cu grid image at the Taurus target

Note: Soil results obtained from pXRF. Highest Cu-Au-Ni results reanalysed by SGS Geosol.

Source: Bravo, 2026.

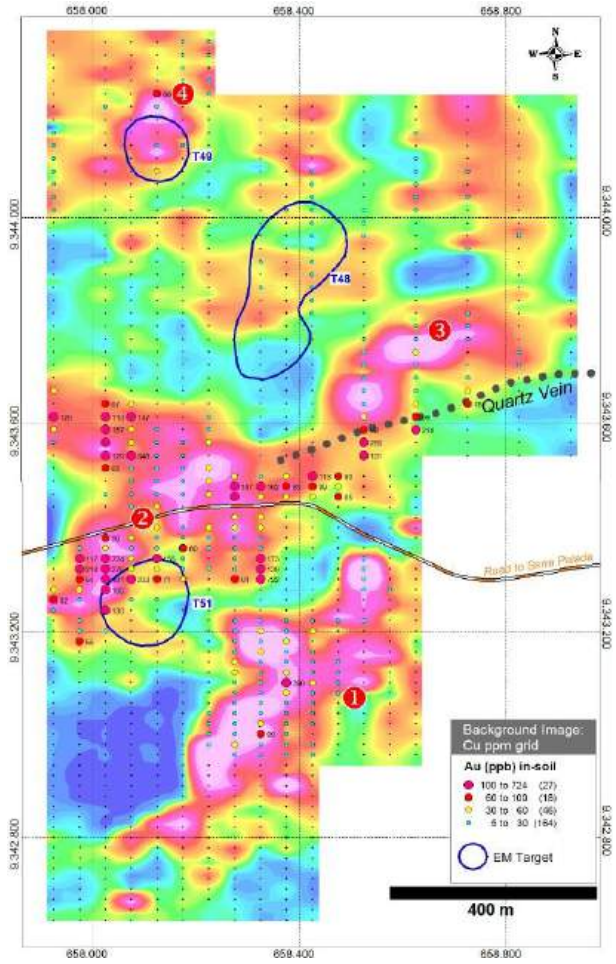


Figure 9-18: Au (ppb) in soil results over Cu grid image at the Taurus target

Note: Soil results obtained from pXRF. Highest Cu-Au-Ni results reanalysed by SGS Geosol.

Source: Bravo, 2026.

Four trenches were opened at Zone 1 and one trench at Zone 2 (Figure 9-19). All trenches returned low-grade copper and gold intercepts, with the best intercept at trench 3, with 14.9 m at 0.46% Cu, 0.10 g/t Au (Table 9-11).

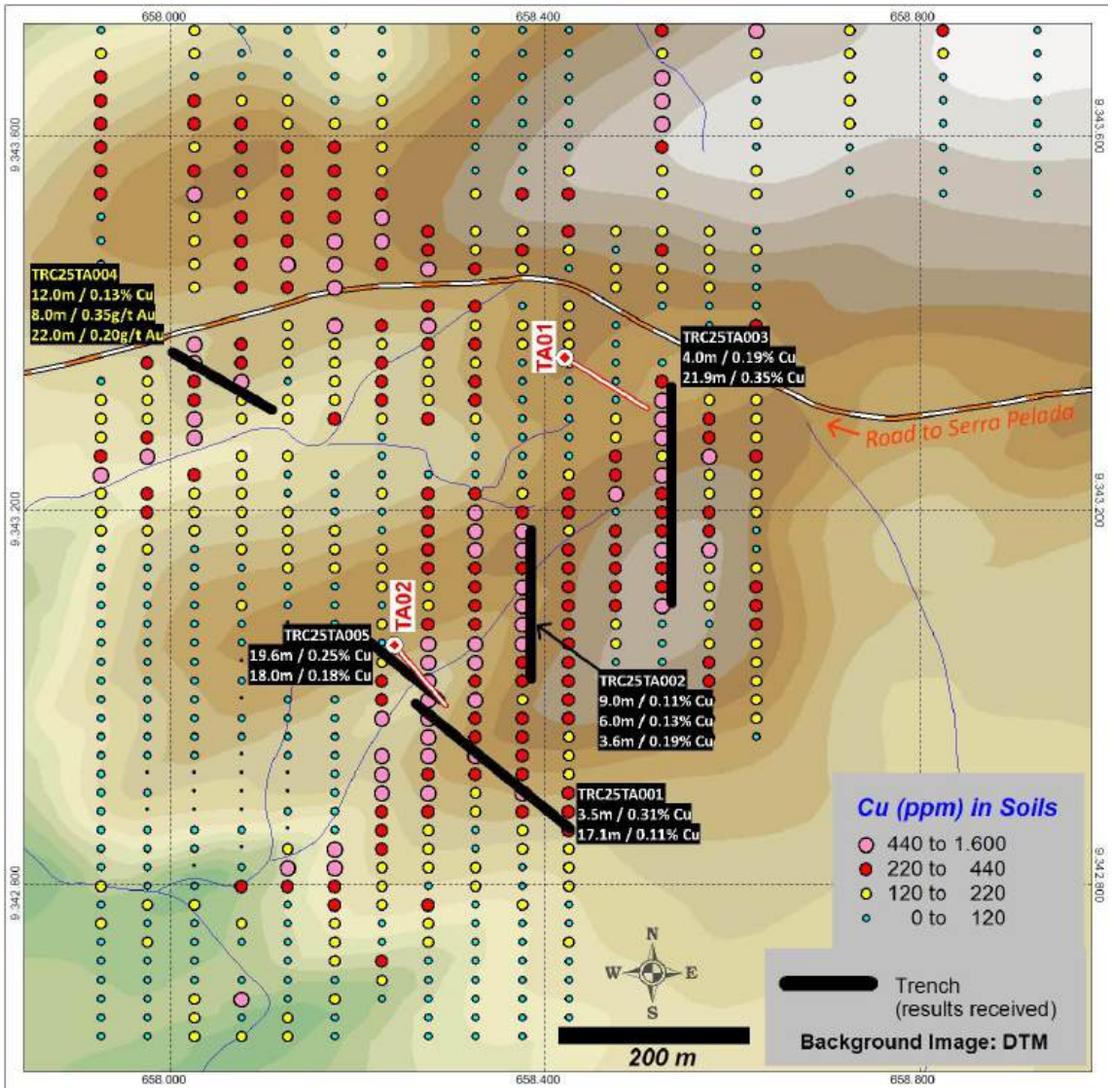


Figure 9-19: Trenches at the Taurus target

Source: Bravo, 2026.

Table 9-11: Trench copper and gold intercepts at the Taurus target

TRENCH_ID	From (m)	To (m)	Thickness (m)	Cu (%)	Au (g/t)
TRC25TA001	63.70	67.15	3.45	0.31	0.04
	199.95	217.05	17.10	0.11	0.04
TRC25TA002	89.90	98.90	9.00	0.11	0.03
	108.90	114.90	6.00	0.13	0.02
	141.30	144.90	3.60	0.19	0.05
TRC25TA003	36.80	40.80	4.00	0.19	0.01
	55.90	56.90	1.00	0.52	0.00
	198.60	220.50	21.90	0.35	0.08
(including)	*198.60*	*213.50*	*14.90*	*0.46*	*0.10*
TRC25TA004	61.00	73.00	12.00	0.13	0.05
	76.00	84.00	8.00	0.07	0.35
	87.00	109.00	22.00	0.06	0.20
TRC25TA005	0.00	19.60	19.60	0.25	0.03
	35.50	53.50	18.00	0.18	0.02

Source: Bravo, 2026.

The geology at Taurus is dominated by a package of talc-schists to the south and east of the target. In the central and northern part of the target there is a predominance of chlorite-schists. Both schist packages show some intense crenulation and quartz veinlets and localized black tourmaline alteration in the form of aggregates. The chlorite-schists show hydrothermal alteration zones a few metres wide where Fe oxides are abundant (magnetite and hematite). A coarse-grained isotropic gabbro/norite with some sheared passages was intercepted in trench 3. The northeast of the target is cross-cut by an EW zone of quartz-veining, with localized quartz breccia with an oxidised sulphide matrix.

Two drill holes were drilled at Taurus in 2025, which failed to return any significant Cu-Au intercept, with the best intercept being 5 m at 0.41% Cu from 29 m at drill hole DDH25TA002 (Table 9-12). The drill hole intersected predominantly chlorite-talc schists, containing dm to m-wide passages with strong black-tourmaline alteration and subordinate chlorite, quartz, calcite and rhodochrosite. Rare mm-wide chalcopyrite venulations were observed associated with the quartz phase of the alteration.

Table 9-12: Main drill intercepts at the Taurus target

Hole ID	From (m)	To (m)	Interval (m)	Cu (%) sulph.	Au (g/t)
DDH25TA001	31.30	33.75	2.45		0.24
and	103.45	107.45	4	0.26	
DDH25TA002	29.00	34.00	5	0.41	
and	127.20	129.11	1.91	0.81	0.14

Source: Bravo, 2026.

The ongoing programme at Taurus contemplates additional geochemical work with rock sampling and the reanalysis of existing anomalous soil pulps. The soil grid will be extended to the

north, NE and NW of Taurus in an area of alluvial gold mining that covers the Sereno shear zone and extends to the T1 target. Two IP lines are currently programmed for the target. Based on the new results, additional drilling will be planned.

9.6.3.10 Orion Target

The Orion target is located 800 m to the east of the T5 target. The area is characterised by the occurrence of very intense quartz veining float, which is locally brecciated with the matrix being composed of Fe hydroxides and boxwork textures, from sulphides. Some malachite can be locally observed. The quartz vein outcrops in a small hill in the north of the target. From the drill holes, the geology of the area is dominated by an intensely altered and deformed tonalite, locally mylonitised, with intercalations of fine-grained rocks altered to biotite and scapolite. Silicification is observed, but the massive brecciated quartz vein was not intercepted by drilling.

Bravo has drilled three diamond drill holes at the Orion target (Figure 9-20), with the main intercept being DDH25OR002 with 8.50 m at 0.58% Cu from 4.1 m (Table 9-13).

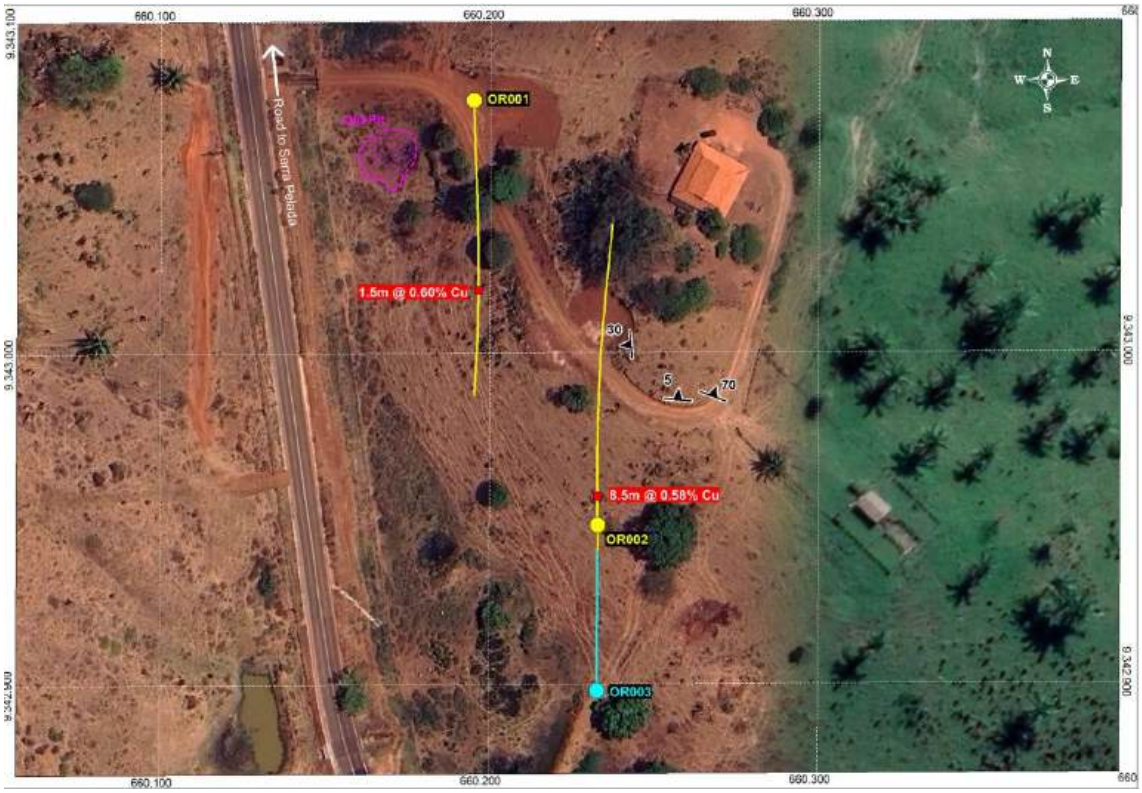


Figure 9-20: Drill hole location at the Orion target

Source: Bravo, 2026.

Table 9-13: Main drill intercepts at the Orion target

Hole ID	From (m)	To (m)	Interval (m)	Cu (%) sulph.	Au (g/t)
DDH25OR001	116.83	118.30	1.47	0.81	0.14
DDH25OR002	4.10	12.60	8.50	0.58	0.01
DDH25OR003	11.60	13.60	2.00	0.26	<0.01
and	61.50	64.54	3.04	0.38	0.10
including	*62.50*	*63.50*	*1.00*	*0.92*	*0.10*

Source: Bravo, 2026.

As part of the 2026 programme, Orion will be integrated into an expanded T5-Orion target and covered by a soil grid. Mapping and rock-chip sampling will ensue. The area will also be covered by two IP lines, one E-W and one N-S. Data will then be integrated to generate new drill targets.

9.6.3.11 Additional Targets

Additional targets with potential for IOCG/ISCG and magmatic Ni-Cu-(PGM) deposits have been defined at the Luanga property and require additional exploration work for their potential to be assessed.

The Jupiter target is located in the SE part of the Luanga property and constitutes the eastern extension of the Luanga south target where magnetic anomalies coincide with anomalous copper in soils. The area, which is characterised by an ultramafic rock, was subject to exploration by past operators (Vale) who carried out soil sampling in grids and an IP survey followed by scout drilling. One of the drill holes returned 21 m at 0.31% Cu. The area requires further investigation and is also prospective for PGMs (Figure 9-21).

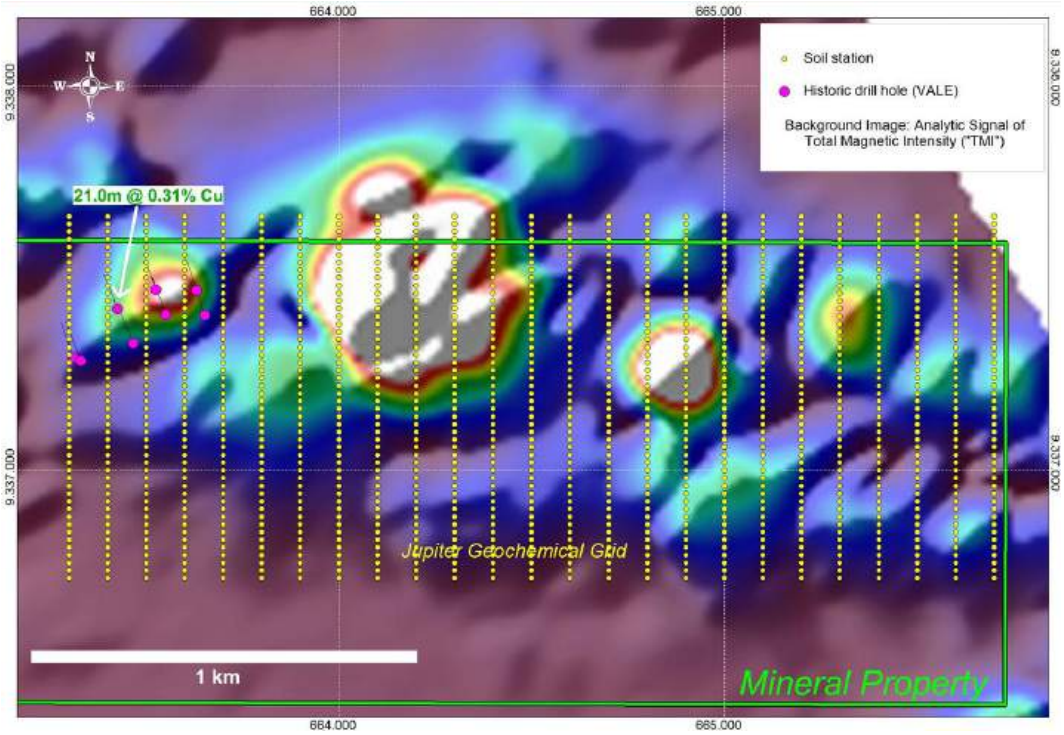


Figure 9-21: Jupiter target – soil grid and diamond drilling over magnetics

Source: Bravo, 2026.

The Scorpion target is located in the southwestern portion of the property and corresponds to a strong magnetic response associated with the Cinzento shear zone and crosscut by a NNW-SSE faulting system. Banded Iron Formations were mapped in the area. The Lynx target is located in the central-southern portions of the Luanga property and corresponds to a strong magnetic anomaly over the Cinzento shear zone. Perseus consists of a group of strong to moderate magnetic anomalies associated with a NE-SW faulting zone (Figure 9-22). The Musca target is located to the east of Perseus and was defined by the presence of EM anomalies.

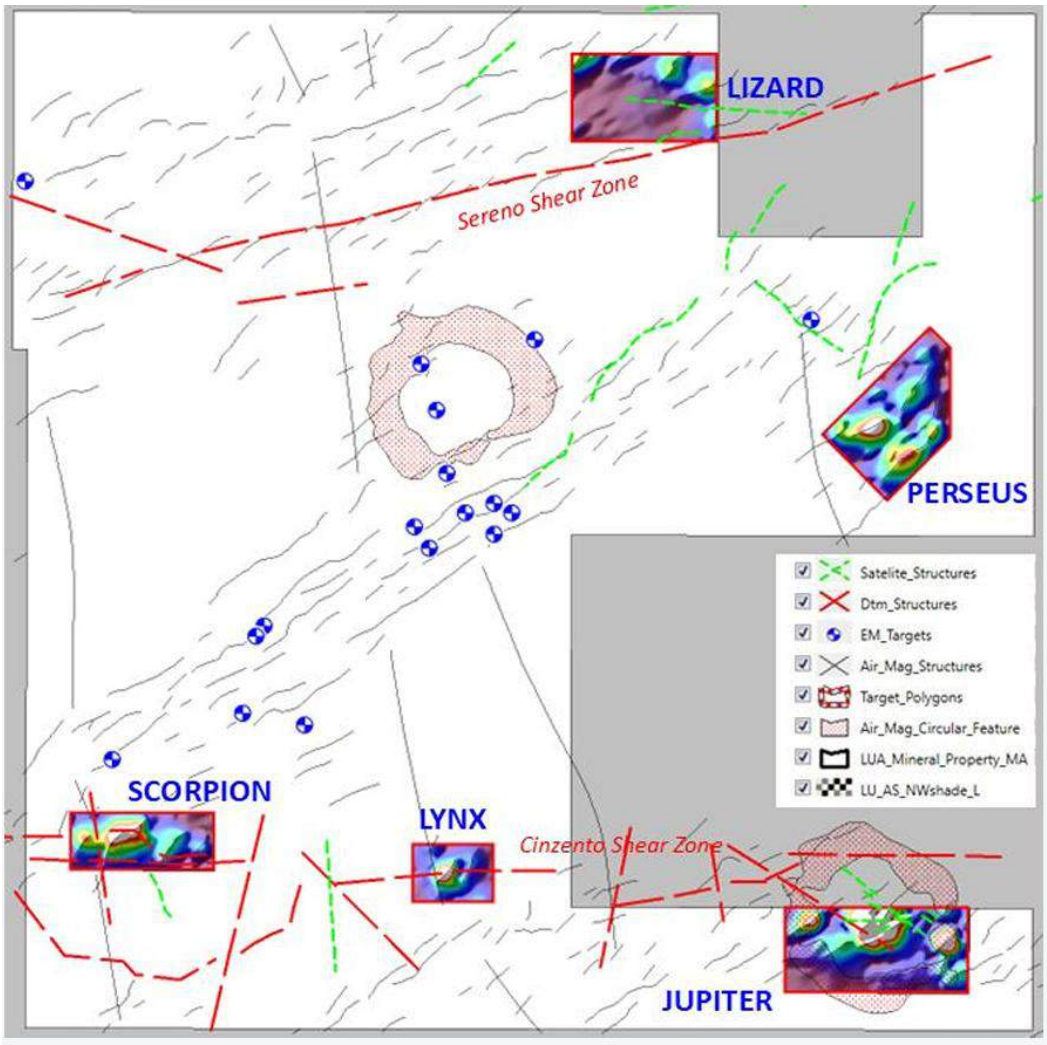


Figure 9-22: Location of additional IOCG targets

Source: Bravo, 2026.

10 DRILLING

As of September 16, 2026, a total of 150,161 m has been drilled in 707 drill holes on the Project 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 corresponds to 99,374.19 m and 451 drill holes, representing 64% of the Project's drilled length.

Details of the various drilling programs are summarized in Table 10-1, and drill hole locations are shown in Figure 10-1. Table 10-1 also includes drilling executed in the *exploration targets*, presented in Section 9 of this Report.

The drilling by Vale in 1992 was for early-stage exploration, while the drilling between 2001 and 2003 was for exploration-focused programs and for initial Mineral Resource estimates. Holes were drilled using HQ, NQ, BQ, and ZWF diameters.

Drilling by Bravo commenced in March 2022 and has continued since then. The program was designed primarily for infill drilling, resource growth and 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 program implemented by Bravo was performed using a mixture of HQ and NQ2 diameters.

During 2025/2026 Bravo continued completed approximately 26,551 m of further resource and exploration drilling, including drilling associated with the ongoing copper exploration program. Drill holes associated with these drilling campaigns were completed post the cut-off date for the current MRE and thus do not contribute to its estimation.

Table 10-1: Drilling summary for Luanga

Year	Drill Type	Drill Holes	Total Metres	Company	Contractor
1992	DDH	4	643.69	DOCEGEO	DOCEGEO
2001	DDH	86	14,584.35		Geosol
2002	DDH	71	15,423.25		Geosol
2003	DDH	95	20,135.45		Geosol/Rede
2022	DDH	135	23,258.20	Bravo	Servdrill Perfuração e Sondagem (Servdrill)
2023	DDH	116	30,296.60		Servdrill
2024	DDH	94	19,268.65		Servdrill
2025	DDH	29	4,861.85		Servdrill
2026	DDH	77	21,689.15		Servdrill
TOTAL		**707**	**150,161.19**		

Source: Bravo, 2026.

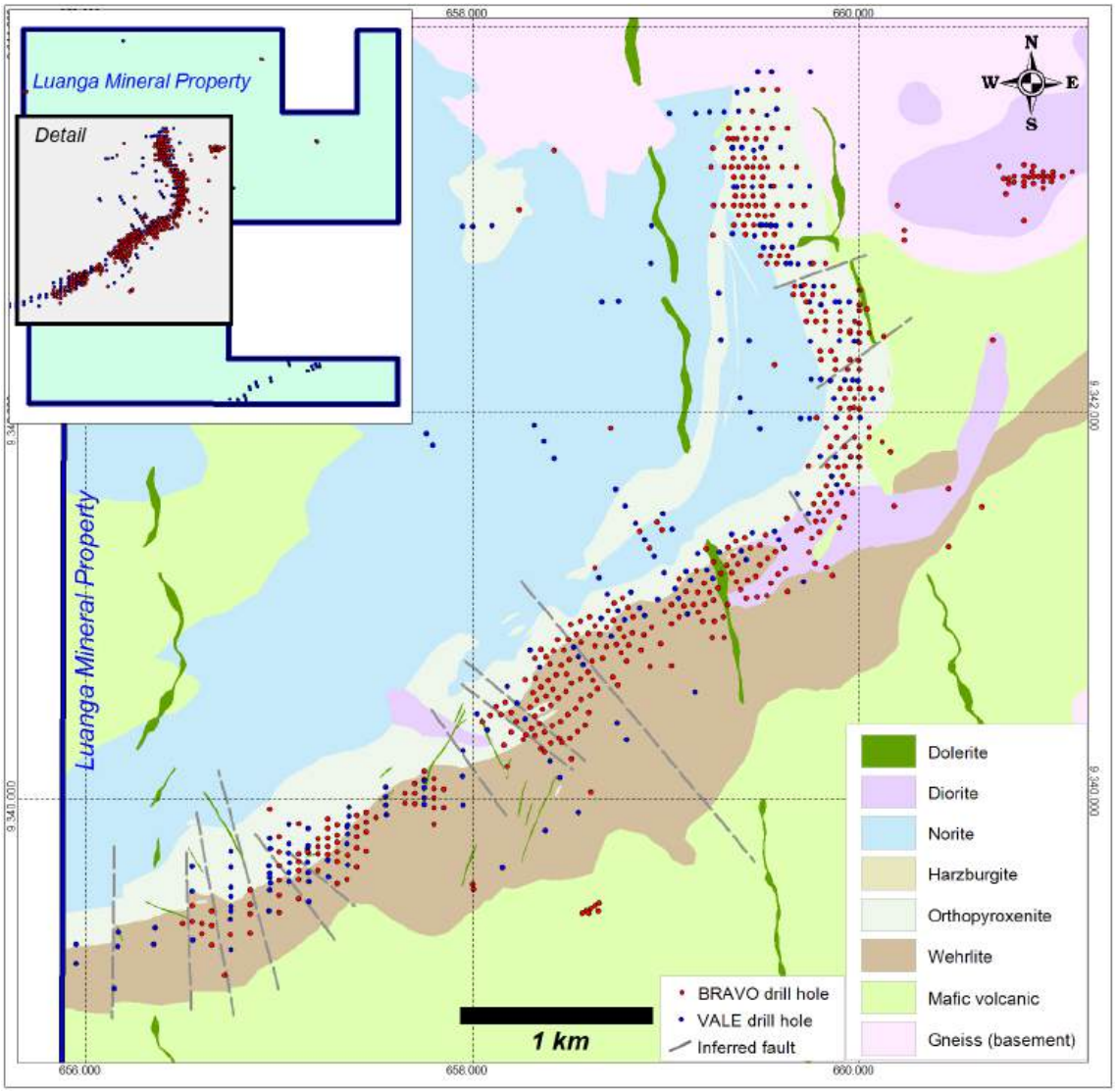


Figure 10-1: Drill hole locations at Luanga

Source: Bravo, 2026.

10.1 Bravo Drilling Program

Bravo has been carrying out its diamond drilling program using equipment from the third-party company Servdrill, with up to 6 drill rigs operating simultaneously. Drill inspection is carried out by Bravo's own employees.

10.1.1 Bravo Drill Collar Survey

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

All drill hole collars were capped with cement blocks, each containing a PVC tube and aluminum plates, bearing drill hole identification codes (hole ID). Information related to the drill hole, such as coordinates, elevation, dip, azimuth, and final depth, is included on the collar marks on the aluminum plates.

10.1.2 Bravo Downhole Survey

Once the drill hole is finished, deviation surveys are conducted for all holes by the drilling company itself using the REFLEX GYRO SPRINT-IQ device. In addition, runs are guided with the Reflex ACT3 device whenever possible.

10.1.3 Core Transport

At the end of each drilling shift, the wooden boxes containing the drill cores are taken to the drilling shed by the outsourced drilling company or by Bravo employees. The boxes are always transported securely tied and covered.

10.1.4 Core Logging

In the core shed, the boxes are placed on racks to check depth, advance, and recovery information, as well as metre-by-metre marking. Then, magnetic susceptibility measurements are taken with the KT-20 S/C device. Marking of the oriented intervals is carried out to proceed with structural measurements using the IQ Logger device.

A geotechnical description of the holes is completed, focusing on obtaining the Rock Quality Designation (RQD) measurement, which is calculated by adding the lengths of all core fragments greater than 10 cm within the same run and dividing this sum by the run length. Information about the strength and weathering of the rocks is also collected.

In the geological description, the following information is collected: granulation, texture, colour, mineralogy, magnetism, geological contacts, lithology, geological structures (fractures, faults, veins, and crenulations), among others. Magnetic susceptibility and conductivity are measured in the core at 0.5 m intervals using a handheld KT-20 S/C magnetic susceptibility/conductivity metre.

After the geological description, a sampling plan is drawn up. Then the sample numbers are marked on the boxes. A marking is also made to guide the sawing of the core into two equal halves. Photographs are taken of all core boxes, which are then sawed and sampled.

Given the orientation of the holes and the mineralization, the intercepts are estimated to range from approximately 130 to 100% of true thickness.

10.2 Twin Holes

A twin-hole program was designed to validate the historical drilling (Table 10-2). This program included drilling 8 twin holes in line with historical drilling, following coordinates surveyed in the field in SIRGAS 2000 format.

Table 10-2: 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

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)
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 130% to 100% 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

Bravo's QA/QC Policy establishes procedures to ensure the integrity of exploration data, analytical results, and project databases.

Core boxes and RC samples are transported from the drill site to the Company's core facility under a documented chain-of-custody process approved by the QP. Sample transfers are recorded, verified upon receipt, and any discrepancies are reported to the QP. Access to core storage, logging facilities, and project databases is restricted to authorized personnel. The drill core is cleaned, logged, photographed, and sampled by qualified staff in accordance with Company procedures. Core cutting and sample preparation are conducted under supervision to ensure representative sampling and maintain sample integrity.

Samples designated for analysis are placed in pre-labelled bags, secured with tamper-evident seals, and shipped to the analytical laboratory together with appropriate submission documentation. Laboratory instructions require immediate notification if any shipment security seal is damaged or compromised.

The remaining core is retained in secure storage, and its location is recorded. Removal of retained core for re-assaying, metallurgical testing, or other purposes requires prior approval from the QP and must be fully documented.

11.2 Sampling

The sample intervals are generally 1-metre-long, ranging from 0.70 m to 1.20 m, and are adjusted to honour lithological contacts, weathering boundaries, and intervals of poor core recovery. Following sample selection, intervals are marked in the core boxes, photographed, and sampled from the right-hand half of the core. Samples are placed in sealed and labelled bags, weighed, and recorded in the sampling database.

Sample submissions are prepared by the responsible geologist and accompanied by chain-of-custody documentation, including sample identification and seal records. Samples are transported by Company personnel to either ALS or SGS laboratories.

Sample preparation is conducted at ALS in Parauapebas, PA, or SGS in Vespasiano, MG, and includes drying, crushing, splitting, and pulverizing. Analytical determinations for Au, Pt, Pd, Rh, and Ni are performed using industry-standard fire assay, nickel sulphide collection, ICP-AES, and ICP-MS methods. Since January 2023, SGS has served as the primary analytical laboratory and ALS as the secondary laboratory. Both laboratories are independent commercial facilities

accredited to ISO/IEC 17025. Bravo's drilling and sampling campaign procedures are described in the flowchart in Figure 11-1.

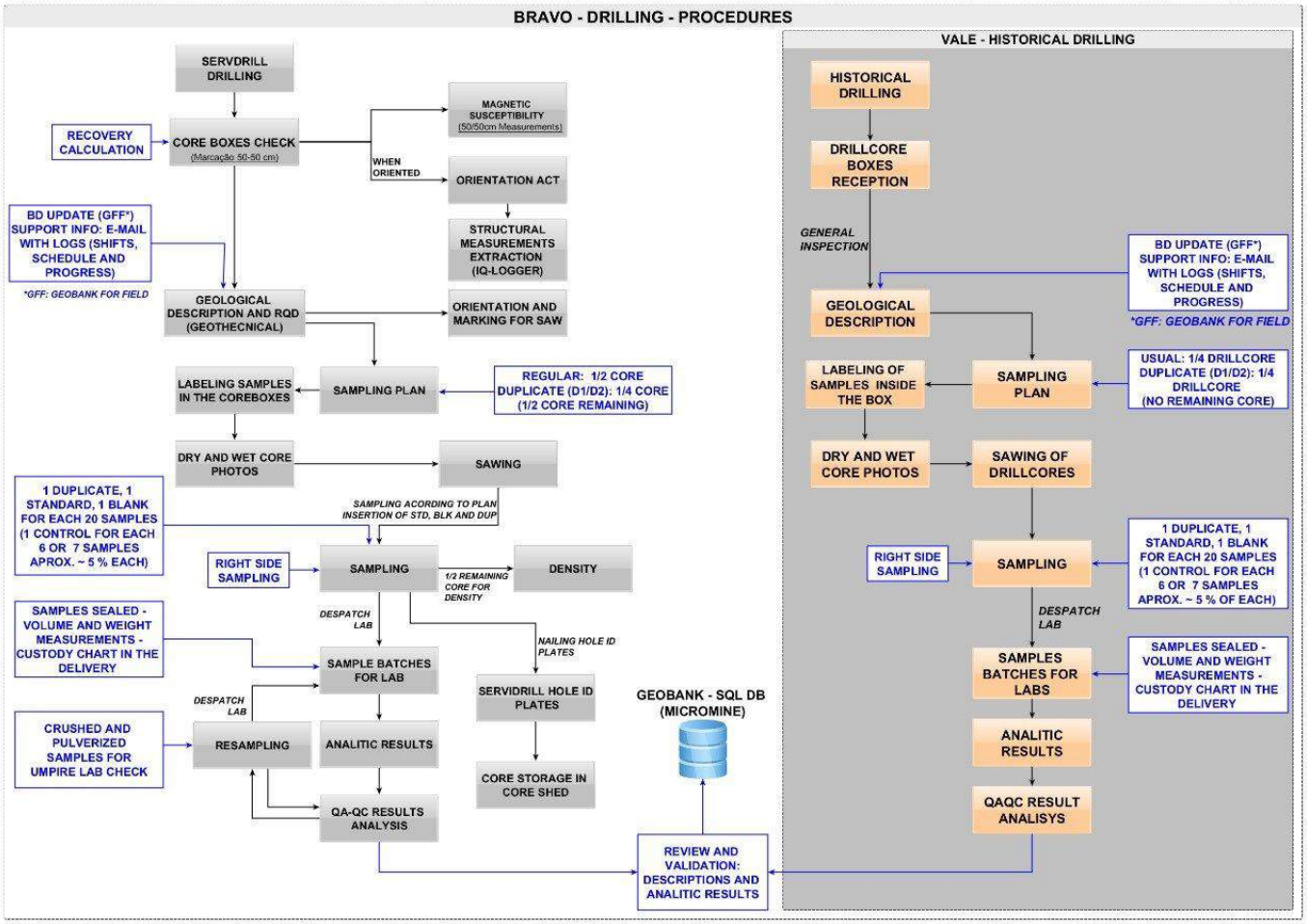


Figure 11-1: Bravo's drilling and sampling procedures

Source: Bravo, 2023.

The assay database used in the Project accounts for both Vale and Bravo samples. Vale samples, which Bravo reanalyzed, have their original grades replaced with the new reanalyzed grades in the database.

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

Table 11-1: Assay proportions by company

Company	# Samples	% Samples
Bravo	92,365	68%
Vale	43,686	32%
Total	**136,051**	**100%**

Source: Bravo, 2026.

11.3 Density Sampling

Density sampling in the drill cores executed by Bravo's technical team is conducted after sawing and sampling for chemical analysis. A half-length drill core sample is collected every

metre, corresponding to half the remaining core in the box. The selected samples are 10 to 35 cm long and are marked with the sample number and core orientation. Subsequently, the samples are submerged in water and weighed on a 0.001 kg precision scale to obtain the wet weight.

The samples are then dried in an oven at 105 °C for 24 hours and weighed again to obtain the dry weight. After the dry weight measurement, the samples are wrapped in cling film and subjected to volume measurement using the water displacement method. Finally, the samples are returned to the core box. As of the Effective Date of this Report, the Project database contains a total of 29,470 density measurements, of which 26,508 were performed by Bravo and 2,962 are historical Vale measurements.

Drill core density statistics for the Luanga Project are summarized in Table 11-2. Density samples were classified as mineralization where the length-weighted 4E (Pt + Pd + Rh + Au) grade of the overlapping assay intervals is equal to or greater than 0.14 g/t, and assigned to the closest target (North, Central or Southwest); all remaining samples, including those without assay results, were classified as host rock. All weathering types are included.

Table 11-2: Drill core density results

Target	Number of Samples	Length (m)	Mean	Minimum	Median	Maximum	Standard Deviation
			g/cm³				
North Mineralization	3,527	1,010.2	2.82	1.30	2.80	4.47	0.24
Central Mineralization	2,977	906.8	2.85	1.66	2.86	5.31	0.19
Southwest Mineralization	1,666	546.1	2.79	1.43	2.79	3.85	0.18
Host Rock	21,300	4,882.6	2.78	1.25	2.77	4.87	0.21
TOTAL	**29,470**	**7,345.7**	**2.79**	**1.25**	**2.78**	**5.31**	**0.21**

Source: Bravo, 2026.

11.4 Quality Assurance and Quality Control (QA/QC)

The Company's QA/QC program includes the routine insertion of Certified Reference Materials (CRMs), blank samples, field duplicates, and umpire assays in accordance with the Company's Technical Assurance procedures. Control samples are inserted at approximately 1 in 20 regular samples and are sourced from accredited suppliers.

The QA/QC program comprises 27,257 control samples (20,612 from Bravo's program, 3,556 from Vale's program and 3,089 historical samples resampled by Bravo), representing approximately 16.7% of all samples analyzed. In addition, a resampling program was completed to validate historical assay data and evaluate the relationship between total nickel and recoverable sulphide nickel grades. A total of 3,089 historical sample intervals were reanalyzed, representing approximately 6.5% of the historical database. Twin-hole drilling was also conducted to assess the reliability of historical drilling, sampling, and analytical procedures.

QA/QC performance is reviewed through regular internal monitoring of laboratory results. Depending on the nature and significance of any control sample failures, selected batches or portions of batches may be reanalyzed. Re-assay priorities are based on the performance of

CRMs, blanks, and duplicate samples, with particular attention given to Pt and Pd analytical results.

Based on the review of QA/QC data, the QP considers the analytical results to be of acceptable quality for exploration and Mineral Resource estimation purposes. 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	5,102
	Blank	5,058
	Field Duplicate	5,032
	Umpire Check	5,420
	Total Bravo QC Samples	20,612
	Total Bravo Samples	92,365
	Total Bravo Database	112,977
	Total QC Samples | Bravo Database (%)	**18.2%**
Vale	Duplicate	2,836
	Blank	720
	Total Vale QC Samples	3,556
	Total Vale Samples	43,686
	Total Vale Database	47,242
	Total QC Samples | Vale Database (%)	**7.5%**
Vale –	Resampled by Bravo	3,089
	Resampled by Bravo | Vale Database (%)	6.5%
	Vale QC + Resampled by Bravo | Vale Database (%)	**14.1%**
Total	Total Samples	136,051
	Total QA/QC	27,257
	Total	163,308
	Total QA/QC percentage	**16.7%**

Source: Bravo, 2026.

11.4.1 Blank Samples

The blank samples used in the Luanga Project consist of six types of commercial materials. Table 11-4 presents the number of blank samples used. 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	2,313
BLK	9
BLK1	302
OREAS 22d	166
BLP2	1,424
AMIS0793	844
Total Blank Samples	5,058

Source: Bravo, 2026.

GE21 has generated control charts from the blank samples' data, using 3x the Detection Limit as the acceptance level. The data were divided into SGS and ALS analyses for constructing the control charts. Considering all blank sample analyses for PGMs, Au, and Ni, the results show that more than 98% of the assay grades are below the 3x Detection Limit 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 CRM, 36 in total. Of these, 23 are from AMIS, 11 from OREAS, and 2 from CDN. The variety of Standard Materials aims to cover the low-, medium-, and high-grade ranges for the main elements analyzed: Pt, Pd, Rh, Au. Table 11-5 presents the quantitative information of CRM included in the QA/QC program.

Table 11-5: Bravo's CRM samples summary

CRM	Number of Samples
AMIS0279	33
AMIS0382	30
AMIS0486	406
AMIS0502	291
AMIS0504	204
AMIS0564	24
AMIS0606	342
AMIS0695	13
AMIS0723	49
AMIS0749	278
AMIS0759	365
AMIS0760	120
AMIS0768	133
AMIS0769	326
AMIS0771	473
AMIS0822	285
AMIS0853	51
AMIS0854	25
AMIS0874	150
AMIS0876	528
AMIS0890	94
AMIS0893	30
AMIS0909	29
OREAS 085	95
OREAS 086	98
OREAS 503 e	22
OREAS 508	23
OREAS 523	23
OREAS 680	33
OREAS 681	135
OREAS 682	149
OREAS 683	147
OREAS 684	29
OREAS 908 b	21
CDN-ME1310	25
CDN-PGMS29	23
Total CRM Samples	**5,102**

Source: Bravo, 2026.

GE21 has generated control charts for all the CRM 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 standard deviations for Pb Collection Fire Assay and NiS Fire Assay.

Nickel control charts have shown inconsistent behaviour because none of the CRMs used by Bravo have certified grades for sulphide nickel analysis, 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. ALS conducted an internal investigation. Bravo also conducted its own investigation into the matter. Both investigations concluded that the high content of refractory oxides (i.e., Al_2O_3, Fe_2O_3, and Cr_2O_3) in the material requires higher fusion temperatures to achieve complete metal recovery.

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.

The AMIS standards data used by Bravo were revised for QA/QC programs; new reference values were generated. No significant biases were found after the correction of reference values. Bravo discontinued using 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 duplicates with both the original and duplicate grades above 3x the Detection Limit of the analytical method evaluated. Table 11-6 summarizes the generated control charts. GE21 carried out an additional Exploratory Data Analysis (EDA) of field duplicates.

Table 11-6: Bravo's duplicate samples summary

Laboratory	Sample Count	Element:	Assay Method
ALS	1,178	Au	PGM-ICP27
		Pd	PGM-ICP27
		Pt	PGM-ICP27
		Rh	Rh-MS25
		Ni	Ni-ICP05
		Ni	ME-ICP61

Laboratory	Sample Count	Element:	Assay Method
		Ni	ME-ICP81
		Ni	ME-OG46
		Ni	ME-OG62
SGS	3,973	Au	FAI515
		Au	FAI34V
		Au	FAA525
		Pd	FAI515
		Pd	FAI34V
		Pt	FAI515
		Pt	FAI34V
		Rh	FAA35J
		Rh	FAI30V_RH
		Ni	AAS04B
		Ni	ICP40B
		Ni	ICP40B_S
		Ni	GE_ICP40Q
		Ni	ICM40B
		Ni	ICM90A

Source: Bravo, 2026.

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 the inclusion of coarse and fine duplicates as control samples of its QA/QC program. Coarse and fine duplicates might be collected as aliquots from the material after the mass-reduction processes employed following crushing and milling, respectively. This measure should help to evaluate the quality of the sample preparation processes.

11.4.4 Umpire Check

The Luanga Project's primary laboratory was ALS until January 2023, when 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 in which original and duplicate grades exceeded 3x the Detection Limit of the evaluated analytical methods. This measure is taken to remove analytical artifacts that occur near the detection limit. Palladium and platinum presented the best results, with practically 90% of the sample data below the 15% HARD limit.

Rhodium assays showed that nearly 60% of samples were below the 15% HARD limit. Gold check assays exhibit the greatest dispersion compared to the PGMs. This element has resulted in approximately 80% of the sample pairs being below the 15% HARD limit.

11.5 Validation of DOCEGEO (Vale) Diamond Drilling Data

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

11.5.1 Twin holes

Historical drilling is verified through relogging and resampling of historical drill holes, and by eight 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	657,149.3	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 observed spatial correlation indicates that the data acquired by Vale are of sufficient quality to characterize the deposit and its geochemical features.

11.5.2 Resampling – Vale Samples

Bravo previously relogged and resampled the historical DDHs. After the geological/structural description and the photographs of core boxes and magnetic susceptibility measurements, the geologist prepares the sampling plan, respecting lithological contacts, the weathering profile, drill core diameter, and drilling recovery. Relogging and resampling activities follow the same operational procedures and QA/QC protocols as Bravo's DDH campaign. The historical drill core sample data and Bravo's resampling of the historical core show an expected positive correlation for the assessed PGM. Correlated assay data show spread due to differences in preparation and analytical laboratory methods; the resampled Vale samples have their grades updated to the reanalyzed grades in the database.

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 samples with original and reanalyzed grades exceeding 3x the Detection Limit of the evaluated analytical methods. Due to differences in the 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 that limit. The SGS reanalysis data account for only 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 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.

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. Figure 11-2 to Figure 11-5 present the correlation for Au, Pd, Pt, and Rh after the transformation of Vale data.

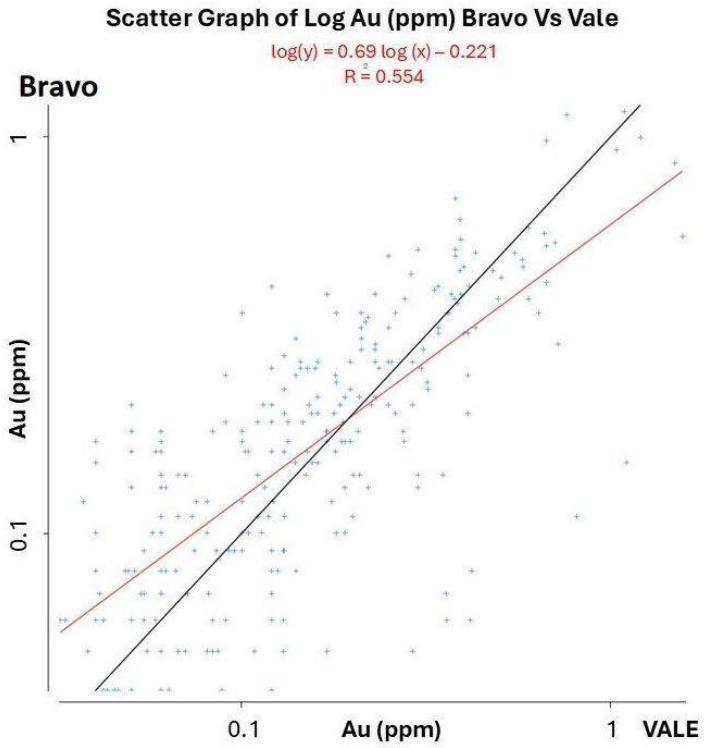


Figure 11-2: Chart correlation of Au assays from Bravo x Vale samples

Source: GE21, 2025.

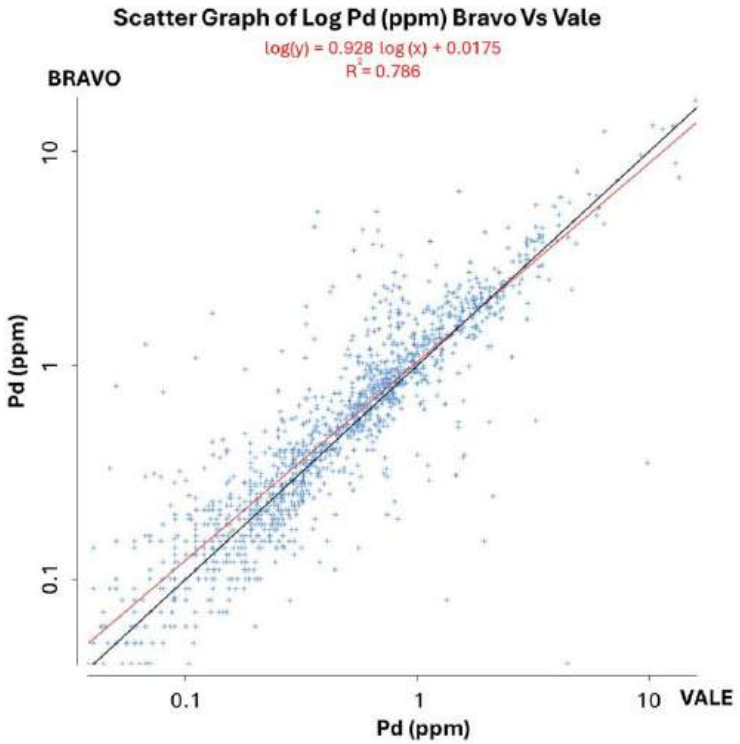


Figure 11-3: Chart correlation of Pd assays from Bravo x Vale samples

Source: GE21, 2025.

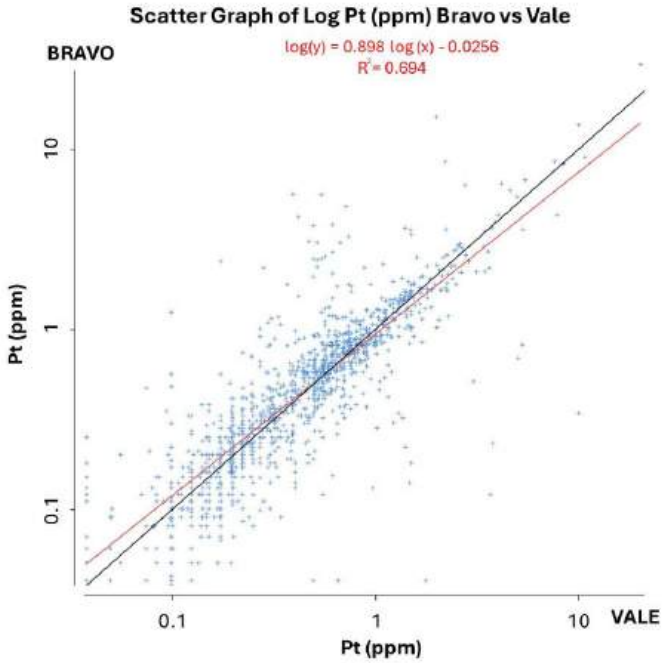


Figure 11-4: Chart correlation Pt samples Bravo x Vale

Source: GE21, 2025.

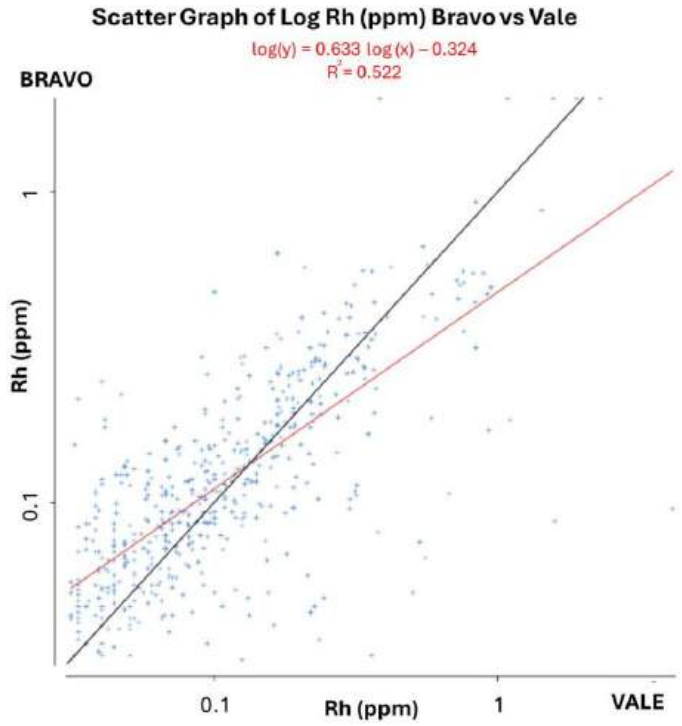


Figure 11-5: Chart correlation Rh samples Bravo x Vale

Source: GE21, 2025.

The nickel grades analyzed by Vale account for total nickel: silicate Ni, oxide Ni, and sulphide Ni. In contrast, the Bravo grades correspond only to sulphide Ni. 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 to both 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 were removed from the datasets. This grade corresponds to the higher detection limit used in the resampled pairs.
- The data constrained between the Positive Bias and the adjusted Negative Bias lines were used for a third linear regression. This regression was used to model the relationship between the Vale and Bravo grades.

Figure 11-6 presents the final linear regression for Ni and the Ni grade distribution after transforming the Vale data.

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

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

Element	Correlation Equation	R2
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.

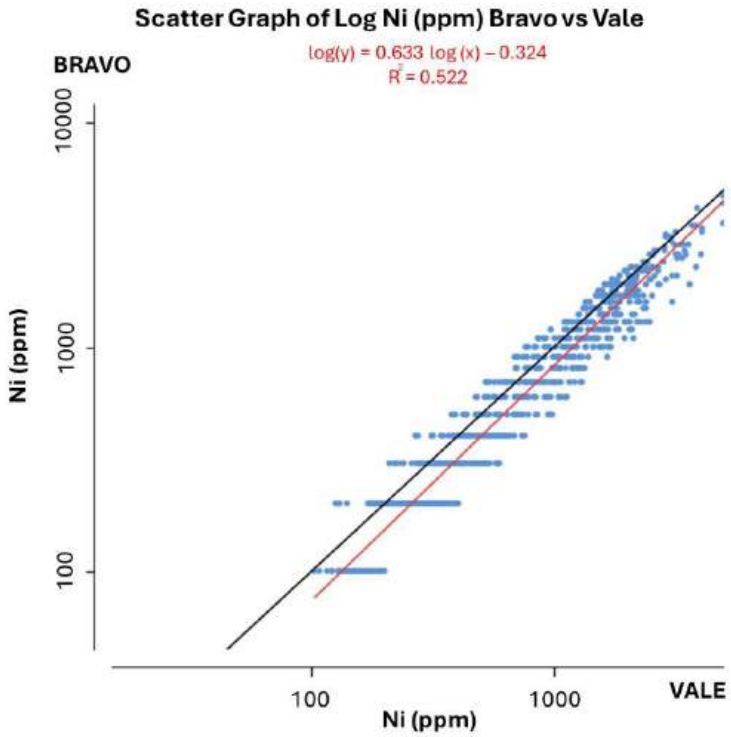


Figure 11-6: Chart correlation Ni samples Bravo x Vale

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 the database valid for Mineral Resource estimation work. The Vale QA/QC program did not include any CRM insertion. GE21 and Bravo have unsuccessfully attempted to obtain the internal QA/QC results from Vale's laboratory (SGS).

QA/QC procedures, sampling methodology, and analytical methods used by Bravo align with industry best practices. The QP responsible for this Report, considering the data presented in this section, believes that the Luanga Project's database is suited for a Mineral Resource estimation work.

Recommendations:

- Acquisition of CRM 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 at Luanga since 2022 to verify the company's infrastructure, the procedures in place, and the results of activities carried out by Bravo staff (See Item 2).

Mining engineer Mr. Porfirio Cabaleiro Rodriguez is an independent consultant and has conducted field visits to the Project in 2023, 2025 and 2026. Mr. Rodriguez has accompanied Bravo personnel during the development of the Mineral Resource estimate activities.

Mr. Bernardo Viana and Mr. Rodriguez have conducted field visits to the Project in 2023, 2025 and 2026.

12.1 Site Visit

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

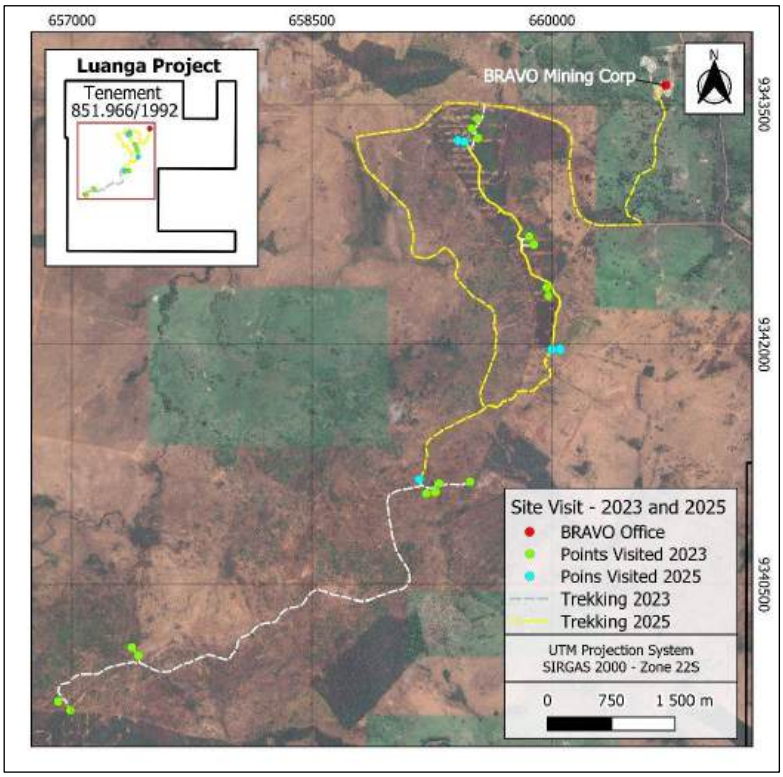


Figure 12-1: Points visited during January 2025

Source: GE21, 2025.

12.1.1 Density Test Laboratory

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

Discussions with on-site geologists confirmed that procedures were adequately applied. More details about Bravo procedures are available in Sections 10 and 11 of this Report.

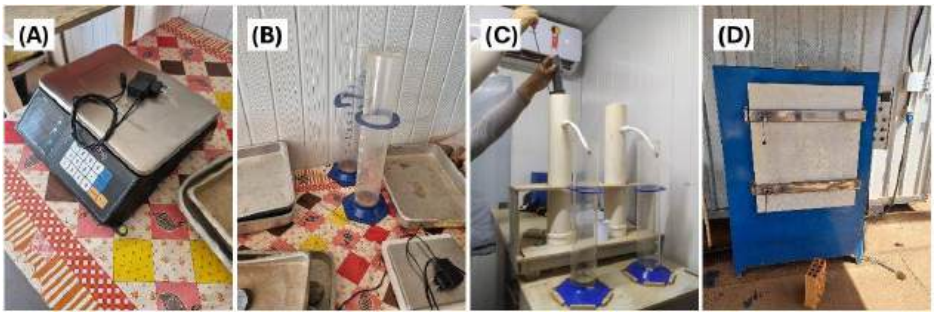


Figure 12-2: Density test equipment

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:

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

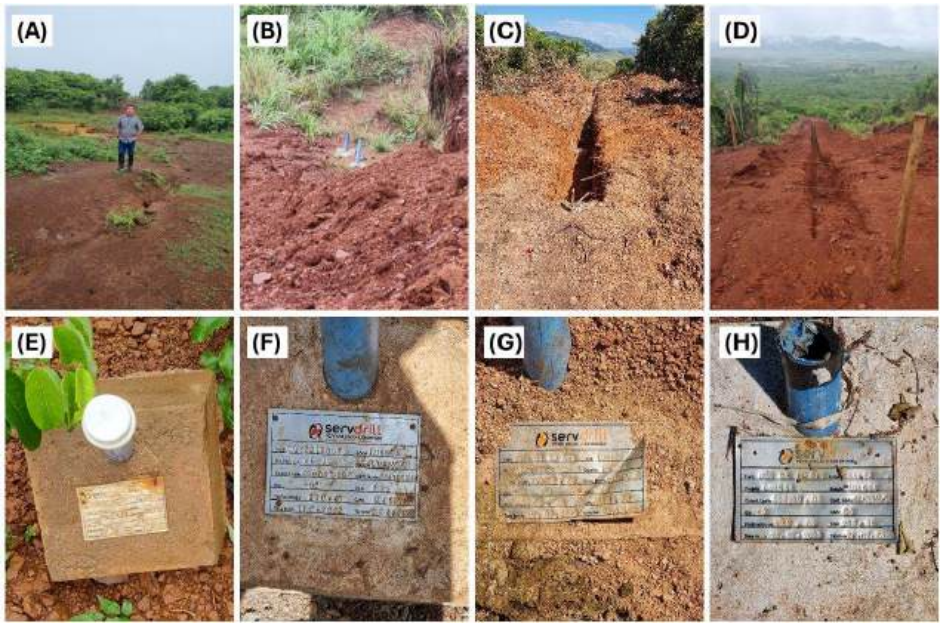


Figure 12-3: Drill hole location evidence

Legend: A) Mr. Viana on 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; 2025.

Drill hole collars have a physical identification marker. The markers consist of a concrete pad with a metal plate that designates the drilling contractor, drill hole number, drilling area,

orientation, coordinate location, start and end drilled dates, 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 of the Luanga Project (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.



Figure 12-5: Sampling and QA/QC procedures

Legend: A) Labelled drill core samples; B) LABELLED drill core sample batches; C) standards for QA/QC program; D) coarse blank for QA/QC program.

Source: GE21, 2023.

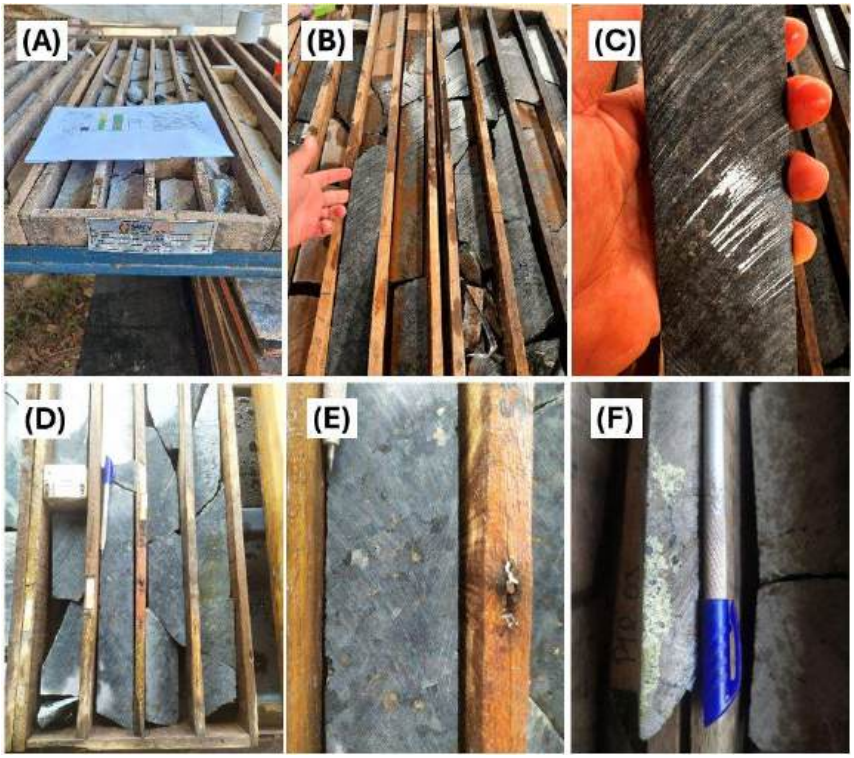


Figure 12-6: Checking drill holes

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 first-period visit in 2023, Mr. Rodriguez collected a sample from among the previously checked drill holes as a witness sample.

The sample under 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 for chemical analysis to the certified laboratory SGS in Vespasiano and received the result shown 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: 37.362.942/0002-39

REFERÊNCIA DO LOTE DE AMOSTRAS

REF. CLIENTE: ACES-BRAVO-0050-23

QTE. AMOSTRAS: 1

RECEBIDO: 11/07/2023

PROJETO: TESTEMUNHO

COMPLETADO: 07/08/2023

PROJETO: Luanga

EMITIDO: 07/08/2023

REFERÊNCIA ANALÍTICA

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

PRPC1J: Preparação Física conforme definição do cliente

LEGENDA SIGLAS

L.D.= Limite de Detecção | BLK = Branco | REP = Replicata | DUP = Duplicata

L.N.R. = Lotado e não Recebido | I.S. = Amostra Insuficiente | N.A. = Não Analisado | STD = Padrão

I.N.F. = Não reportado devido a interferentes | UNR = Não Analisado devido ao alto teor

SGS GEOSOL LABORATÓRIOS LTDA.

CERTIFICADO DE ANÁLISES

GQ2308925

ANÁLISES	Peso_bruto	Ni	Cu	Au	Rh	Pd	Pt
MÉTODO	PRPC1J	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 January 2025 visit, GE21 QPs, Mr. Viana and Mr. Rodriguez, collected three sample reserves from previously sampled trenches and sent them 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 that there is a good correlation across all witness samples.

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	5,102	2,958	0.749

Source: GE21, 2025.

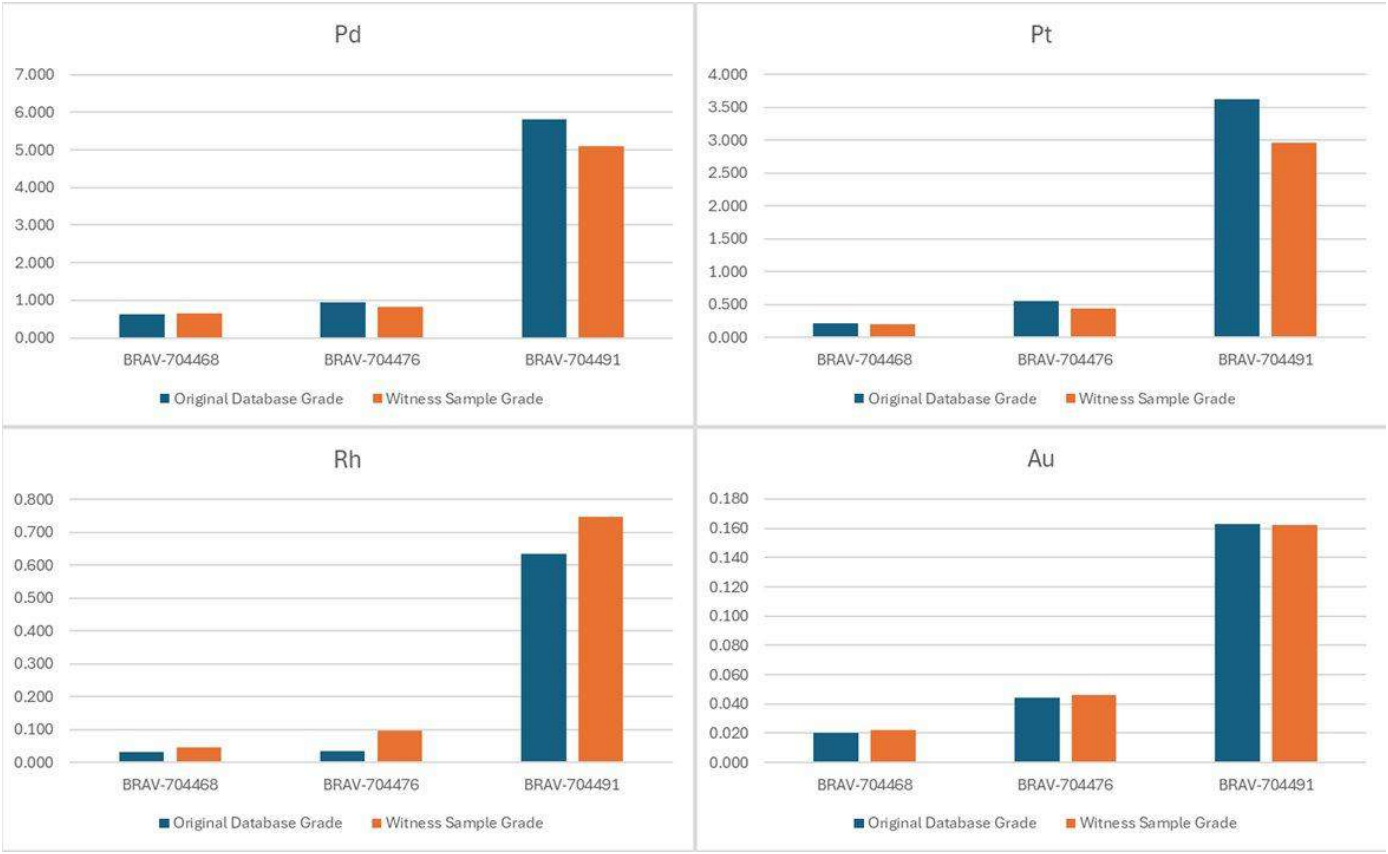


Figure 12-8: Comparison between original database grades and witness sample grades

Source: GE21, 2025.



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
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

manual signature omitted

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

Based on the data verification performed, it is the QP's opinion that the collar coordinates, downhole surveys, densities, lithologies, and assay results are considered suitable to support the Mineral Resource Estimation work. 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 standard industry practice logging procedures, which have been standardized by Bravo. GE21 reviewed logging procedures for randomly selected drill cores and verified the completeness of the logs. Considering all the evaluated content, Bravo has demonstrated understanding of the geology, and some omissions are not considered significant.
- Database: recent data is stored in a standard commercial database. Historical Vale records are well-managed and are currently being migrated to the Bravo database. Data storage procedures at Bravo are considered within standard industry practice. As part of the validation process, GE21 verified eight drill 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 by which density data is obtained is considered 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 Disclaimer

In this Report, the term “ore”, as used in certain contexts, refers to mineralized material that can be economically mined from the Measured and Indicated Resources and meets the definition of a Mineral Reserve, in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) guidelines.

The term “Platinum Group Metals (PGM) 3E” is used to refer to Pt, Pd, and Rh, but it is sometimes used to refer to Pt, Pd and Au, or to PGM 4E – Pt, Pd, Rh and Au. Strictly speaking, gold is not a member of the platinum-group elements (PGE) series.

13.2 Introduction

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

- Historical test work completed for Vale (Section 13.3).
- Extensive comminution and flotation test work on fresh (sulphide) rock samples was conducted for Bravo from 2022 to 2026, including work to review and validate 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, and parameter optimization tests conducted for Bravo between 2022 and 2025.
- Preliminary tests were conducted to evaluate the downstream treatability of Luanga concentrates on site or nearby.

13.3 Review of Historical Metallurgy Work

Historical metallurgical testing of 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 and pilot plant scales 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.

Table 13-1 summarizes the historical metallurgical efforts reported for Luanga.

Table 13-1: Historical test work

Date	Laboratory	Object	Process	Sample
2002	Mintek, Africa do Sul	Preliminary test, P_{60} 75µm,	Mill>float>mill>float bench and locked-cycle	100 kg of a 4.8 g/t PGM + Au; main bearing minerals sperrylite, bismuth tellurides, and stibiopaladinite. Concentrate LC 150 g/t PGM + Au, recovery 75%.
2003	CDM, Brasil	Confirm previous Mintek results	Locked-cycle, bench tests flotation, mini plant tests, aggressive reagents recipe	4 composites grade ranging between 1.07 g/t and 4.14 g/t PGM + Au. Concentrate grades Pt – 42.9 g/t, recovery 79%; Pd – 54.3 g/t, recovery 71%; Au – 3.6g/t, recovery 47%; total PGM + Au, 105 g/t, recovery 73%; Ni 7.5%, recovery 44.6%.
2003 2004	Lakefield, Canadá	Validation of CDM trials	Same CDM	200 kg sample; Mass pull, 0.78%, PGM + Au concentrate grade 132 g/t; recovery 70%

Source: Bravo, 2023.

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 [80-154 g/t] and PGM recoveries in the order of 74%. The closed-circuit (locked-cycle test – LCT) study obtained a (single) bulk concentrate for Luanga comprising PGM + Ni + Au.

13.4 Bravo Sample Selection

The mineralogical study was completed on drill core samples from Vale's previous drill programs. Samples for initial metallurgical studies were taken from the same intervals and distributed along the strike of the orebody, including the North, Central and Southwestern sectors, as shown in Figure 13-1.

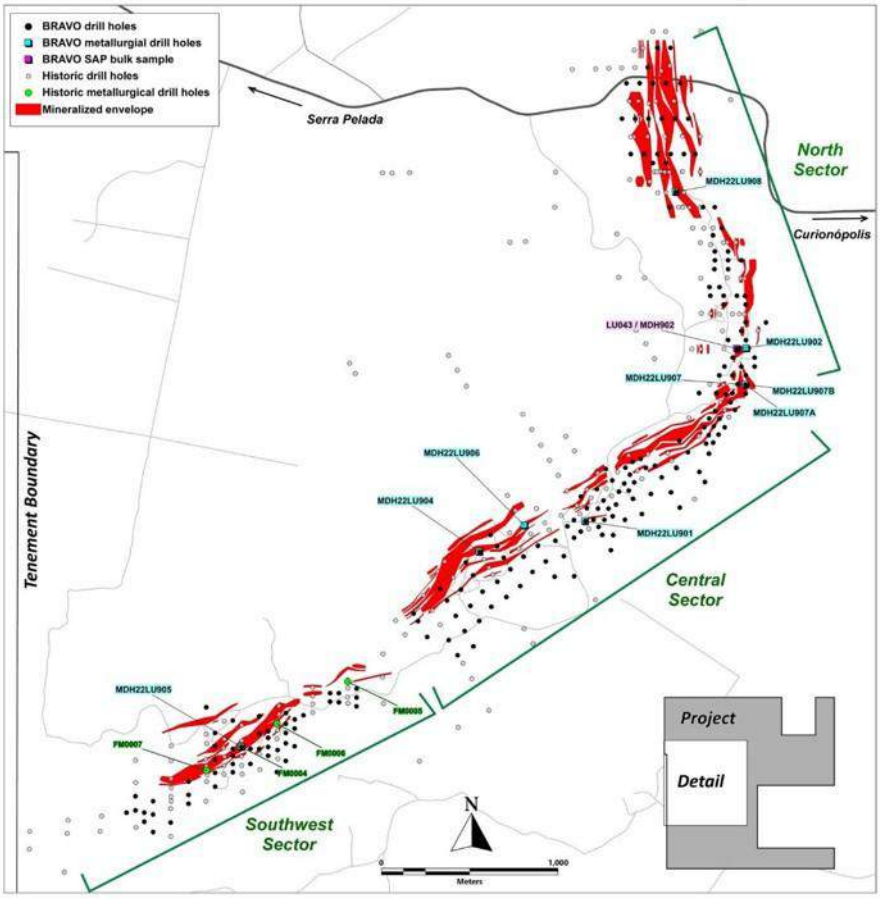


Figure 13-1: Location of metallurgical samples

Source: Bravo, 2023.

13.4.1 Sample Variance Selection

Twin drill core samples for the flotation and leaching test program were selected to be representative of the Luanga deposit, including fresh sulphide, transitional and oxide samples. Grade variance was considered, with sampling grades ranging between 0.5-11 g/t of PGM.

A bulk sample of oxidized material for leaching tests was collected at the northern edge of the Central sector of Luanga, which is considered to contain most oxide mineral resources by attributable tonnes.

13.5 Bravo Metallurgical Program 2022/2023

Bravo submitted approximately 3 t of fresh-sulphide metallurgical samples and 150 kg of oxide samples to the CETEM and Testwork laboratories, respectively, for metallurgical studies.

13.5.1 Sulphide Material

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

13.5.1.1 Bond Ball Mill Grindability Tests

Two individual BWi tests on composite blends from the Central and Southwest sectors were performed to determine preliminary energy requirements. One composite of fresh sulphide mineralized material and one sample of transitional mineralized material were tested. The preliminary BWi results indicate a “medium” classification for both samples (Table 13-2). Table 13-3 shows the sample composite details used in the BWi tests.

Table 13-2: BBWi results

Test	Wi
	kWh/t
1 Fresh	13.94
2 Transition	10.29

Source: Bravo, 2025.

Table 13-3: Sample composite – BBWi test

Sulphide composite samples	Drill hole ID	Zone	Sample depth
600014	MDH22LU901	Central	106-112 m
600015	MDH22LU901	Central	112-119 m
600030	MDH22LU905	Southwest	49-55 m
600031	MDH22LU905	Southwest	56-62 m
Transition composite samples	**Drill hole ID**	**Zone**	**Sample depth**
600006	MDH22LU904	Central	26-27 m
600028	MDH22LU905	Southwest	29-34 m

Source: Bravo, 2025.

13.5.1.2 Flotation Tests

Flotation tests were carried out with different reagents, grind sizes and flotation configurations, including pre-flotation of talc, fast rougher, staged roughers, scavenger-roughers, 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. It is believed that the presence of clays in the oxide sample inhibits flotation selectivity and explains the poor results. No other tests were performed on oxide samples.

13.5.1.3 Exploratory and Characterization Flotation Tests (Series T, TD)

Table 13-4 summarizes the results from exploratory test work.

Table 13-4: 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.

13.5.1.4 Comparative Flotation Tests (Series TC)

The test was carried out to compare results to the historical closed-circuit arrangement, achieving results that included 82 g/t concentrate grade with 75% recovery (TC-01) and 189 g/t concentrate grade with 78% recovery (TC-02). The particle size target was 80% passing 30 µm, and feed grades ranging between 2.6 g/t and 4.6 g/t PGM, respectively.

13.5.1.5 Preliminary Process Circuit Tests (Series TB, TH)

The TB test series represented a circuit configuration with a six-stage rougher, with combined rougher concentrates reporting to a five-stage cleaner. Table 13-5 summarizes the results from these flotation tests.

Table 13-5: 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.

13.5.1.6 Fines Flotation Tests

Previous test work indicated that a significant component of the PGM grain size distribution is concentrated in the 3-15 µm range, which can affect flotation.

Bravo submitted a 30 kg sample to Mintek in South Africa for hydrodynamic cavitation testing using the Mach reactor (Figure 13-2).

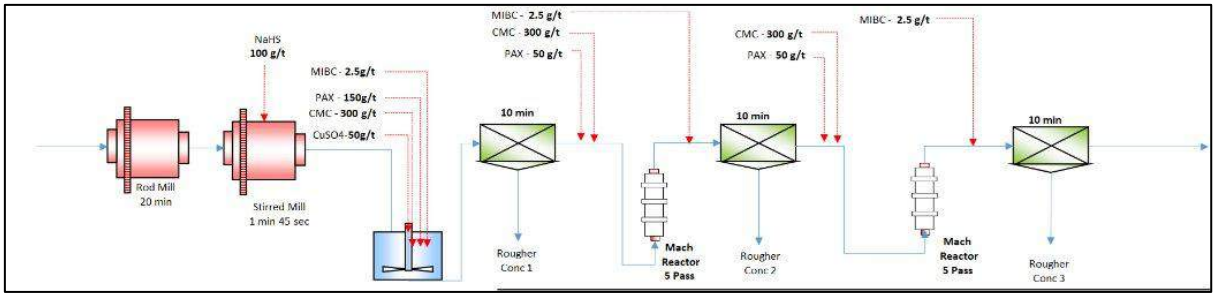


Figure 13-2: Test flowsheet and parameters used by Mintek

Source: Bravo, 2023.

Three control tests and five active Mach reactor tests were performed using a 2.66 g/t sample feed. Test 4 demonstrated a 6.1% improvement in recovery, with similar mass pull and concentrate grade metrics for the three-stage rougher circuit.

Results in Table 13-6 demonstrate the potential improvements achievable by incorporating the Mach Reactor for Luanga ores, particularly for lower-grade ores where fine PGM particles predominate.

Table 13-6: Results from fines cavitation 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 × 10 Pass + N CMC dosage	11.5	19	80.5	80.5	6.1

Source: Bravo, 2023.

13.5.1.7 Associated Base Metals

Two locked-cycle tests (LCTs) performed in the past by Vale produced adequate concentrate to examine the performance of nickel-sulphide recovery in the flotation process (Table 13-7).

Table 13-7: Nickel sulphide flotation results

CDM Vale LCT tests 2003	Feed grade Ni (%)	Conc grade Ni sulphide (%)	Ni sulphide recovery (%)
LCT1	0.19	7.5	47.3
LTC2	0.19	9.8	45.9

Source: Bravo, 2023.

13.5.2 Transition Material

Twenty-two transition flotation tests of transitional material were performed on samples with grades ranging from 0.38 g/t to 7.95 g/t PGM. Table 13-8 shows the results, where T1 to T3 are results of fresh rock, as a benchmark.

Table 13-8: Comparison of results of transition material and fresh rock on flotation tests

Rougher comparative test	Feed grade (g/t)	Mass pull (%)	Recovery (%)	Concentrate 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.

13.5.3 Oxide Material

The deposit is characterized at the surface by an extensively weathered mineralization zone constrained from the surface to depths of 5 m to 30 m. It is a sensu stricto oxide horizon, with most of the sulphide minerals (including nickel sulphides) altered to limonitic/saprolitic oxide phases.

The oxide horizon does, however, account for approximately 10% of the stated mineral resources by tonnes. Bravo reviewed and investigated the amenability of recovering PGMs from

the oxide mineralization. Sighter flotation tests, pre-desliming, and gravity concentration were performed, but poor results were achieved.

13.5.3.1 Cyanide Leaching

The 4E grade of the sample ranged from 1.88 g/t to 3.24 g/t. Table 13-9 summarizes the results.

Table 13-9: 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%

Source: Bravo, 2023.

13.5.3.2 Carbon Loading and Ashing Tests

Loaded carbon was ashed, resulting in a mass of 0.24 g, representing a 98.8% reduction. The resultant final product grade was calculated on a mass balance (Table 13-10).

Table 13-10: Average grades – 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 achievable over 100 g/t.

13.6 Bravo 2024/2025 Program – Fresh Rock

13.6.1 Sampling

A global bulk composite of 947 samples from the North sector of the Luanga deposit, totalling approximately 3.6 t, was taken from diamond drill core samples, constituting a sample with a grade of 2.4 g/t 4E PGM and 0.19% Ni.

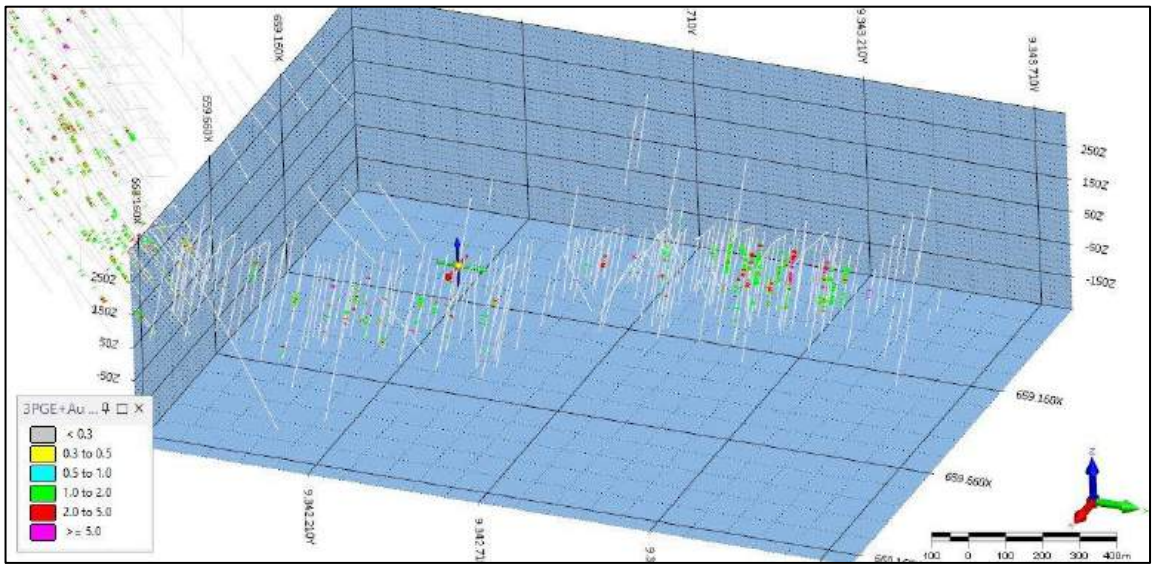


Figure 13-3: Distribution of samples collected for the global bulk sample constitution in the northern region

Source: Bravo, 2025.

Similar global composites were prepared for the Central and Southwest sectors of the deposit. The Central and Southwest sector composite was used in conjunction with the North sector composite for detailed comminution test work, while only the north composite has been used in further mineral processing tests to date. It is envisioned that a similar scope of work will be applied to the central and southwest sector global composites subsequently.

13.6.2 Comminution

Metso Brazil conducted detailed comminution tests on the North, Central, and Southwest sectors. A 200 kg sub-sample from each composite was taken and sent to its laboratory in Belo Horizonte, Brazil. 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, encompassing:

- Bond Abrasion Index (AI).
- Macon AI.
- Macon crushability.
- Bond Work Index (BWI).
- Bulk density.
- Specific gravity.
- Volumetric capacity index.
- Strength index.
- Product flakiness index.

Table 13-11 summarizes the results.

Table 13-11: Comminution test results

Technical parameter	Unit	North zone	Central zone	Southwest zone
Bond Ball Mill Work Index (Wi)	kWh/t	14.98	18.91	14.43
	Class	Medium-hard	Hard	Medium-hard
Bond Rod Mill Work Index (Wi)	kWh/t	16.4	19.3	16.5
	Class	Hard	Very hard	Hard
Macon Crushability	%	32.4	29.1	37.7
	Class	Medium	Medium	Medium
Bond Abrasion Index (Ai)	-	0.070	0.091	0.044
	Class	Low	Low	Low
Macon Abrasion Index (Ai)	g/t	228	502	148
	Class	Low	Low	Low
SAG Specific Energy	kWh/t	10.56	11.75	9.44
Specific Gravity	t/m³	2.95	3.03	2.91
Bulk Density	t/m³	1.68	1.73	1.71
Volumetric Capacity Index	%	83.13	77.89	75.46
	Class	Minimum	Minimum	Minimum
Strength Index (R)	%	190.64	248.31	161.97
	Class	Minimum	Minimum	Minimum
Flakiness Index	%	1.69	14.16	14.16
	Class	Cubical	Cubical	Cubical

Source: Bravo, 2025.

The North and Southwest sectors exhibit material which has a low abrasion index, moderate crushability and medium to hard ball-milling indices. The Central sector exhibits higher material hardness, with higher abrasion, crushability and milling indices. All three sectors exhibit very similar SAG Circuit Specific Energy values.

13.6.3 Flotation

Bravo conducted several flotation optimization programs, including testing at CETEM, an independent verification and optimization program at Base Metal Laboratories (BML) in Canada, and a series of mini-plant flotation tests at CETEM. In 2026, BML carried out the flotation tests for Jameson technology.

13.6.3.1 Parameter Optimization Program – CETEM

Rougher kinetic tests were conducted to define the impact of conditioning time, cell energy input, pH and pulp density on performance. Some conditions and flotation behaviour are listed below:

- Baseline conditions were P_{80} 28 µm and CMC addition 4000 g/t.
- Highly alkaline conditions adversely affected performance, while acidifying the naturally alkaline pulp to 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 compared with a 20% test.

13.6.4 Base Metallurgical Laboratories (Intertek)

Base Metal Laboratories in Kamloops, Canada, conducted test work to validate the results, principally on the north sector global composite. Initial exploratory tests were also conducted on the Central sector global composite.

13.6.4.1 Sample Characterization

Table 13-12 shows a summary of the chemical analysis data.

Table 13-12: Average feed samples before the tests

Sample	Assays - percent or g/t						
	Pd	Pt	Au	3E	Ni	Cu	S
North zone composite	1.45	1.09	0.08	2.61	0.21	0.049	1.28
Central zone composite	1.73	0.56	0.05	2.35	0.25	0.045	0.64

Source: Bravo, 2025.

13.6.4.2 Metallurgical Tests

Metallurgical tests were conducted on the samples to evaluate current processing options and optimize operational conditions and the flowsheet. Development testing focused on the North sector composite and investigated various parameters. Two LCTs 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.

13.6.4.3 Rougher Flotation Test Work

A primary grind series of tests and some alternative flowsheet configurations, including multistage grinding (mill float – MF2), were tested. Talc was present and would need to be controlled. Table 13-13 and Table 13-14 summarize the flotation test conditions and results.

Table 13-13: Rougher test work parameters

Composite	Test	General description	Primary Grind (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	800 g/t Calgon	30	650	328	24
	T25	Fine/Coarse Circuit	46/43	770	106	31

Composite	Test	General description	Primary Grind (PG)	g/t		Time
			µm K80	PE26	PAX	Min
Central zone	T28	Baseline	33	475	88	25
	T31	Increase PE 26	33	825	88	25
	T32	Effect of PG	27	725	88	25

Notes:
1. PG: primary grind.
Source: Bravo, 2025.

Table 13-14: Average concentrate grades of rougher test work

Composite	Test	Mass (%)	Assay – percent or g/t							Distribution – percent					
			Pd	Pt	Au	3E	Ni	Cu	S	Pd	Pt	Au	3E	Ni	Cu	S
North zone	T01	36.3	3.2	2.1	0.2	5.5	0.4	0.1	1.0	80.0	73.0	84.0	77.0	60.0	95.0	29.0
	T02	14.6	5.8	5.0	0.2	11.1	0.9	0.2	5.6	57.0	59.0	39.0	57.0	59.0	64.0	63.0
	T03	34.2	2.7	1.4	0.7	4.8	0.3	0.1	1.2	72.0	61.0	97.0	71.0	56.0	93.0	32.0
	T04	22.6	5.0	3.4	0.2	8.7	0.6	0.2	1.9	77.0	76.0	77.0	77.0	65.0	94.0	34.0
	T05	12.2	7.7	7.8	0.5	15.9	1.2	0.4	3.5	73.0	77.0	62.0	74.0	66.0	94.0	33.0
	T06	37.1	3.1	2.3	0.2	5.6	0.3	0.1	0.8	74.0	74.0	77.0	74.0	54.0	92.0	22.0
	T08	16.7	5.5	5.0	0.3	10.8	1.0	0.3	6.4	80.0	81.0	72.0	80.0	76.0	95.0	82.0
	T10	19.4	5.0	4.7	0.2	10.0	0.8	0.2	3.4	77.0	81.0	72.0	79.0	70.0	95.0	53.0
	T12	18.0	8.3	5.4	0.3	14.0	0.9	0.3	3.5	87.0	81.0	71.0	84.0	71.0	92.0	51.0
	T13	15.3	8.4	5.3	0.3	14.0	1.0	0.3	3.4	90.0	75.0	62.0	83.0	70.0	91.0	40.0
	T14	17.3	6.7	4.1	0.3	11.1	0.8	0.3	3.4	78.0	70.0	48.0	73.0	64.0	89.0	45.0
	T21	29.9	4.9	3.1	0.1	8.1	0.5	0.2	1.9	86.0	85.0	76.0	86.0	70.0	95.0	46.0
	T22	11.9	11.7	8.4	0.4	20.6	1.2	0.4	4.7	84.0	80.0	75.0	82.0	70.0	94.0	43.0
	T23	15.6	8.6	6.5	0.4	15.4	0.9	0.3	3.7	85.0	83.0	77.0	84.0	75.0	92.0	47.0
	T25	14.6	10.0	5.6	1.2	16.7	1.0	0.3	5.4	85.0	78.0	92.0	83.0	70.0	95.0	56.0
Central zone	T28	16.5	8.3	3.1	0.8	12.2	1.1	0.3	2.1	74.0	67.0	94.0	73.0	66.0	91.0	49.0
	T31	9.6	13.1	5.3	1.0	19.4	2.1	0.4	4.0	78.0	76.0	91.0	78.0	71.0	93.0	63.0
	T32	13.6	9.3	4.1	0.8	14.1	1.4	0.3	2.6	80.0	76.0	93.0	79.0	68.0	90.0	55.0

Source: Bravo, 2025.

PGM 3E rougher recovery ranged from 57% to 86%, with mass recoveries ranging from 10% to 37%. Results showed that production of a flash concentrate from rougher 1 is feasible.

Particle size is a significant factor in recovery. The highest recoveries were generally achieved at 30 µm K80 or finer. Figure 13-4 shows the rougher tailings grade as a function of particle size and indicates a tendency for the tailings grade to decrease with finer grinding.

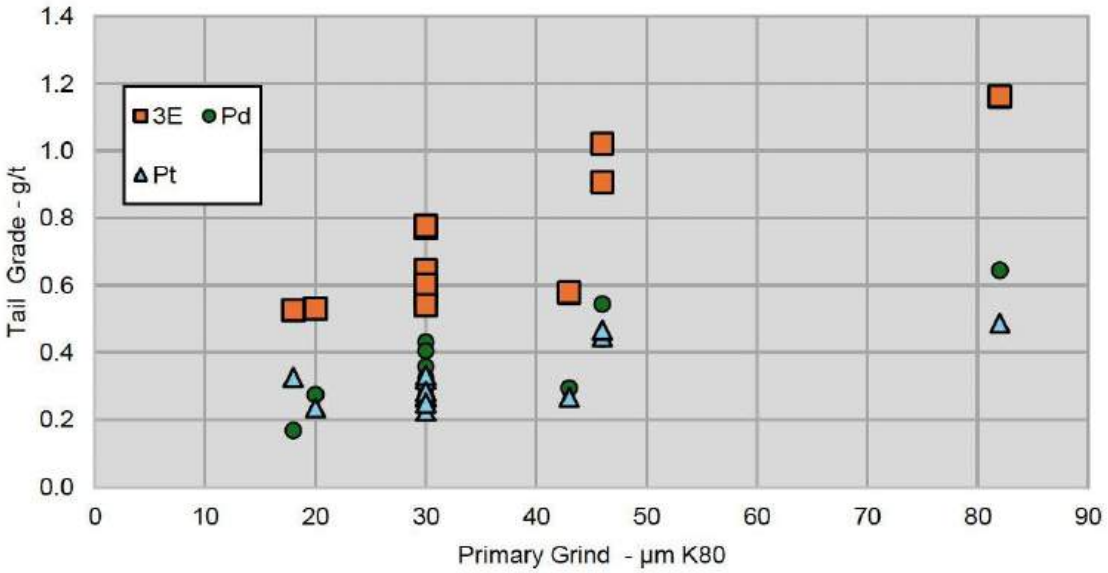


Figure 13-4: Grades of the rougher tailings versus the primary grind sizing for Pd, Pt, and 3E

Source: Bravo, 2025.

13.6.4.4 Cleaner Test Work

Cleaner tests were conducted to evaluate concentrate quality and recovery in the open circuit and to guide the LCTs. The following points are of note:

- For the North sector, 3E recovery to final concentrate ranged from 46% to 71%, and grade ranged from 70 g/t to 682 g/t.
- Nickel performance was variable, with recoveries ranging from 5% to 63%.
- Two cleaner tests were carried out on the central sector composite. In both tests, 63% of 3E was recovered to the concentrate, with grades ranging from 154 g/t to 198 g/t.
- Some results suggest a correlation with depressant PE26 dosing, indicating other interactions beyond just talc depression.
- Higher density resulted in higher mass recovery due to fine-particle entrainment. The increased talc entrainment could be controlled by increasing PE26 addition. A trade-off evaluating the differences in low-density versus higher reagent levels, among other parameters, should be considered.
- Regrinding produced significant increases in concentrate grade but lower recovery. Given the saleable grades without regrinding, it is not required for the tested samples.
- PGMs and nickel show high initial kinetics, which is favourable for circuit optimization.

13.6.4.5 Locked Cycle Tests

Two LCTs with north sector and one with central sector composites were conducted. Table 13-15 shows the results as a mass and metallurgical balance.

Table 13-15: Locked-cycle tests balance

LCT results cycle D + E																
Test/ composite/ flowsheet	Product	Mass	Assay – percent or g/t							Distribution – percent						
		%	Pd	Pt	Au	3E	Ni	Cu	S	Pd	Pt	Au	3E	Ni	Cu	S
T20: north zone HG/LG flowsheet	**Cycles D + E**															
	Feed	100	1.64	1.2	0.07	2.9	0.21	0.05	1.2	100	100	100	100	100	100	100
	Comb. con	1.8	71.7	47	2.5	121	6.9	2.7	19.1	81	74	69	78	62	94	29
	HG con	0.6	190.8	115.3	6.89	313	13.1	7.7	30.4	72	61	65	67	39	92	15
	LG con	1.2	11.4	12.8	0.24	24.4	3.7	0.1	13.4	8	13	4	10	22	2	14
	LG cinr tis	7.8	0.7	0.7	0.03	1.47	0.16	0.003	1.4	3	5	3	4	6	0	9
	Ro TIs	90.4	0.29	0.3	0.02	0.59	0.07	0.003	0.8	16	21	27	18	32	5	62
T30: north zone simplified flowsheet	**Cycles D + E**															
	Feed	100	1.4	1.1	0.05	2.54	0.2	0.05	1.2	100	100	100	100	100	100	100
	Bulk con	1	86.3	65.4	2.74	154	3.39	3.39	29.1	79	76	78	78	62	95	31
	CInr scav tis	12	0.54	0.54	0.01	1.1	0.004	0.004	1.4	5	6	3	5	11	1	15
	Ro TIs	86.7	0.27	0.22	0.01	0.5	0.002	0.002	0.7	16	18	19	17	27	4	54
T30: central simplified flowsheet	**Cycles D + E**															
	Feed	100	1.62	0.64	0.1	2.36	0.29	0.04	0.7	100	100	100	100	100	100	100
	Bulk con	0.8	137.2	54.3	10.49	202	15.95	4.22	29.4	67	67	86	68	44	83	34
	CInr scav tis	13.4	1.98	0.68	0.04	2.69	0.55	0.021	1.2	16	14	6	15	25	7	23
	Ro TIs	85.8	85.8	0.14	0.01	0.46	0.1	0.005	0.3	16	19	9	17	30	10	43

Source: Bravo, 2025.

13.6.5 Mini-plant Program

A mini-plant test program was run with the following aims:

- Produce larger quantities of concentrate to evaluate potential downstream processing opportunities.
- Further evaluate parameters determined by the bench-scale program.
- Evaluate the detailed concentrate chemistry and mineralogy.

The sample used in this program was collected from large-diameter diamond cores (ZW) at holes FM004, FM005, FM006 and FM007, drilled approximately 20 years ago. The drill cores were sampled and composited to achieve a target grade of 2 g/t 4E PGM.

CIT SENAI Laboratory in Belo Horizonte, Brazil, and Frank Rezende Consulting conducted a concentrate production program using the mini pilot plant. Six-grade profile bulk samples were prepared, totalling 2.5 t of material for processing test work.

The circuit consisted of three mechanical flotation cells, where the first produced a high-grade concentrate. The remaining 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 ground to a P_{90} of 38 µm, while general rougher cell reagent conditions were maintained in accordance with the Bravo standard conditions.

Resultant concentrates were composited for analysis of trace and deleterious elements, which were found to be below smelter thresholds, including As, Sb, Bi, Cd, Pb, Te, Zn, F, Hg, and Cl. Table 13-16 summarizes the results.

Table 13-16: Typical smelter thresholds 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
Cr_2O_3	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.

A remaining sample was sent to CETEM in Rio de Janeiro to conduct complementary test work, including the cleaner stage.

The circuit encompasses grinding followed by a 7-stage rougher, 2-stage cleaner and a single-stage recleaner and scavenger, at a P_{80} of 28 μm.

Recovery results were broadly in line with previous test work on high-talc samples, where the concentrate grade reached 36 g/t 4E and 3.7% Ni. Table 13-17 summarizes the results. The lower concentrate grade was attributed to excessive entrainment of gangue minerals.

Table 13-17: Metal recoveries in Phase 2

Element	Recovery
Pd	73%
Pt	77%
Rh	39%
Au	87%
Ni	59%

Source: Bravo, 2025.

13.6.6 Oxide Material Leaching

The Testwork Process Development laboratory performed metallurgical studies on oxide samples collected as a bulk sample of approximately 2 t from drill hole MDH902, located at the northern edge of the Central sector.

The sample was prepared, and an aliquot was sent to SGS-Geosol in Vespasiano, Brazil, for chemical head analysis. Table 13-18 summarizes the results.

Table 13-18: Head grade analysis results – leaching Phase 2

Head grade	Au (g/t)	Pd (g/t)	Pt (g/t)	Rh (g/t)	4EPGM + Au (g/t)
Split sample average	0.03	1.32	1.58	0.28	3.20

Source: Bravo, 2025.

Results of PGM + Au extractions varied in a broad range (Table 13-19).

Table 13-19: Parameters and recoveries of Phase 2

Condition	Best Elemental Recoveries			
	Pd	Pt	Au	Rh
Exploratory	82%	5%	74%	2%
NaCN concentration	78%	7%	89%	5%
2-stage pH	74%	-	65%	10%
Temperature	55%	4%	73%	-
Contact time	84%	-	65%	-
Size-by-size	71% 78% 67%	3% 3% 7%	84% 65% 73%	1% 2% 15%
Additives	83%	15%	79%	5%
Blend leach	75%	8%	87%	95%
Supplementary Standardized	66%	21%	90%	25%

Source: Bravo, 2025.

13.6.6.1 Size-by-size Analysis

A granulochemical characterization was carried out on the oxide sample used in the leaching program to better understand the deportment of metals by size fraction within the soil oxide domain. Table 13-20 shows the distribution.

Table 13-20: Size-by-size analysis results

Size	Mass (%)		Au			Pd		
	Simple	Acum.	(g/t)	Dist (%)	Acum. dist (%)	(g/t)	Dist (%)	Acum. dist (%)
6.3 mm	21.9%	21.9%	0.04	13.4%	13.4%	6.52	28.1%	28.1%
1.0 mm	23.1%	45.0%	0.04	16.1%	29.5%	10.23	46.5%	74.6%
0.300 mm	6.8%	51.9%	0.05	5.0%	34.5%	4.78	6.4%	81.0%
0.150 mm	3.1%	55.0%	0.04	2.0%	36.5%	5.57	3.4%	84.4%
0.075 mm	3.3%	58.3%	0.03	1.6%	38.1%	3.08	2.0%	86.4%
0.045 mm	4.6%	62.9%	0.25	18.8%	56.9%	2.19	2.0%	88.3%
0.038 mm	1.4%	64.3%	0.35	8.1%	65.0%	2.04	0.6%	88.9%
< 0.038 mm	35.7%	100%	0.06	35.0%	100%	1.58	11.1%	100%
Total	100%	-	0.06	100%	-	5.09	100%	-
Size	**Mass (%)**		**Pt**			**Rh**		
	Simple	**Acum.**	**(g/t)**	**Dist (%)**	**Acum. dist (%)**	**(g/t)**	**Dist (%)**	**Acum. dist (%)**
6.3 mm	21.9%	21.9%	6.13	25.5%	25.5%	1.67	36.67%	36.67%
1.0 mm	23.1%	45.0%	10.79	47.3%	72.7%	2.03	46.83%	83.5%
0.300 mm	6.8%	51.9%	4.46	5.8%	78.5%	0.38	2.61%	86.1%
0.150 mm	3.1%	55.0%	5.9	3.5%	82.0%	0.59	1.84%	88.0%
0.075 mm	3.3%	58.3%	4.24	2.6%	84.6%	0.39	1.27%	89.2%
0.045 mm	4.6%	62.9%	0.52	0.5%	85.1%	0.37	1.70%	90.9%
0.038 mm	1.4%	64.3%	0.14	0.0%	85.1%	0.34	0.47%	91.4%
< 0.038 mm	35.7%	100%	2.2	14.9%	100%	0.24	8.61%	100%
Total	100%	-	5.28	100%	-	1.00	100%	-

Source: Bravo, 2025.

13.6.7 Locked-cycle Tests

BML conducted LCTs on samples collected from core drills and advanced test work on dilution, simulating the Jameson cell flotation circuit. The results confirmed the production of a high-grade concentrate, reaching values up to 202 g/t of PGM in the Central zone.

Figure 13-5 shows a comparison of previous mechanical flotation tests against Jameson tests. It is notable that there is a gain in recovery for Au and Ni, with a significant improvement for PGM 3E.

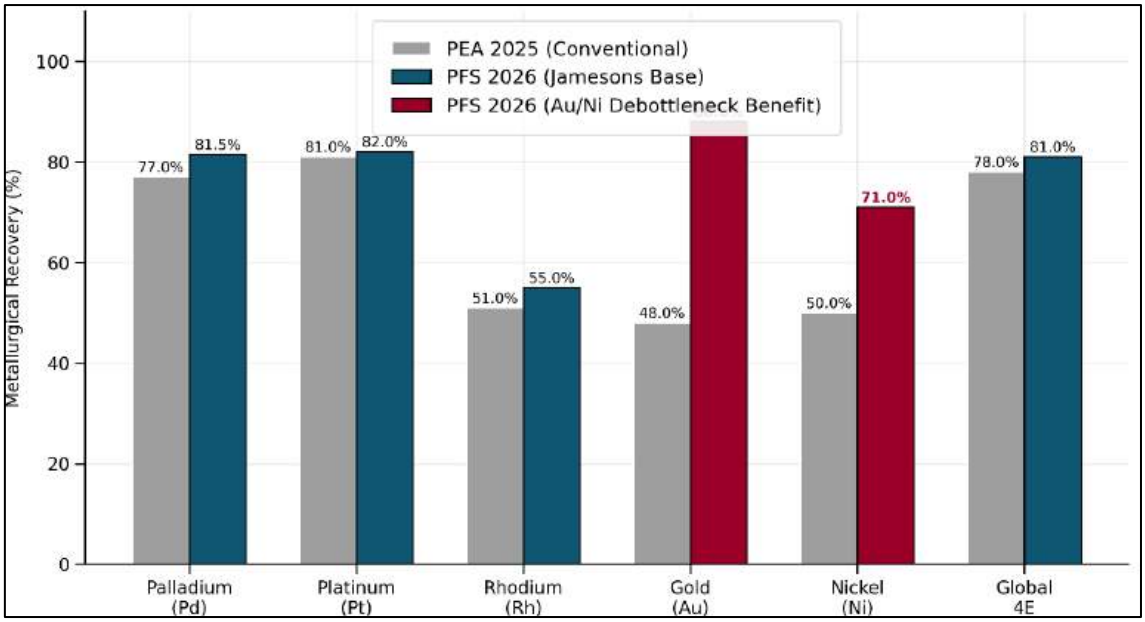


Figure 13-5: Comparison of recovery – mechanical cell versus Jameson cell, by element

Source: Bravo, 2026, adapted by GE21, 2026.

13.6.8 Recoveries and Assumptions

Based on the results of the test work, the QP for the report is of the opinion that the metallurgical test results are suitable for use in estimating mineral resources and mineral reserves. The following conclusions and assumptions are reasonable from this test work:

13.6.8.1 Palladium Recovery

Palladium is the highest-value metal from a metallurgical perspective at Luanga and exhibits favourable flotation kinetics. Pd achieved in laboratory LCTs conducted at the BML consistently demonstrated high Pd recoveries of 81.5%.

The Jameson cell provides an important contribution to improved flotation kinetics, given that palladium at Luanga occurs predominantly as fine-grained tellurides and disseminated alloys. The high-density downcomer mechanism promotes the recovery of these ultrafine particles, which may otherwise be subject to lower recovery due to entrainment and reduced flotation efficiency in conventional mechanical flotation cells.

Figure 13-6 highlights, in reddish-brown, palladium recovery as a function of PGM feed grade in Jameson cell test work.

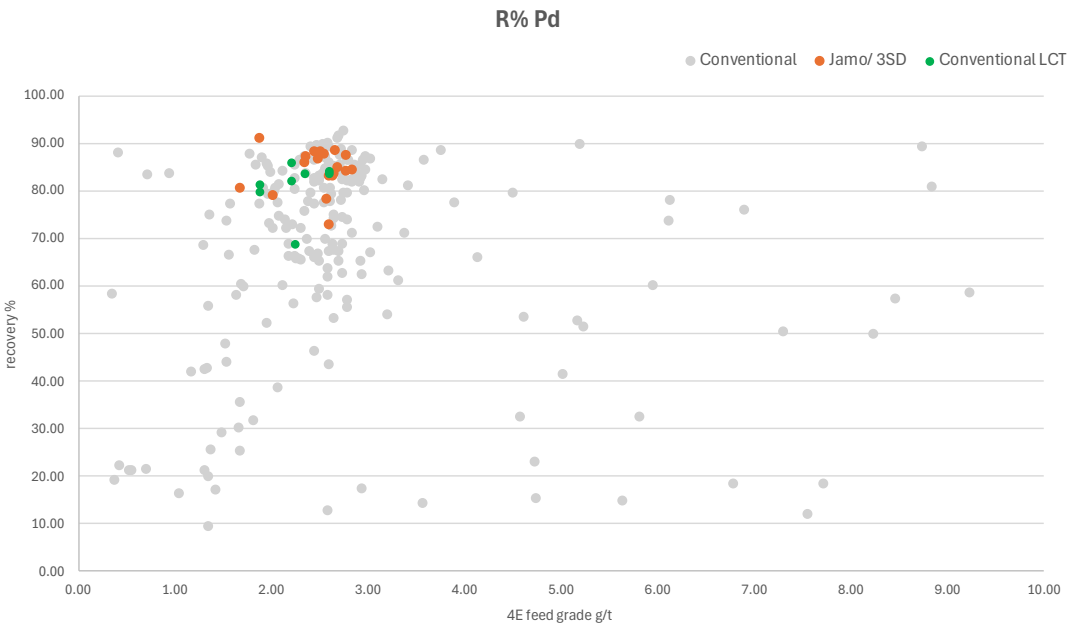


Figure 13-6: Pd recovery in Jameson cell as a function of PGM feed grade

Source: Bravo, 2026.

13.6.8.2 Platinum Recovery

Platinum demonstrated robust floatability under the proposed route. Combined with the use of a dispersant and a depressant, this provides kinetic stability against pyroxene-associated talc. Based on the test work, a recovery of 80% is feasible.

Figure 13-7 highlights, in reddish-brown, platinum recovery as a function of PGM feed grade in Jameson cell test work.

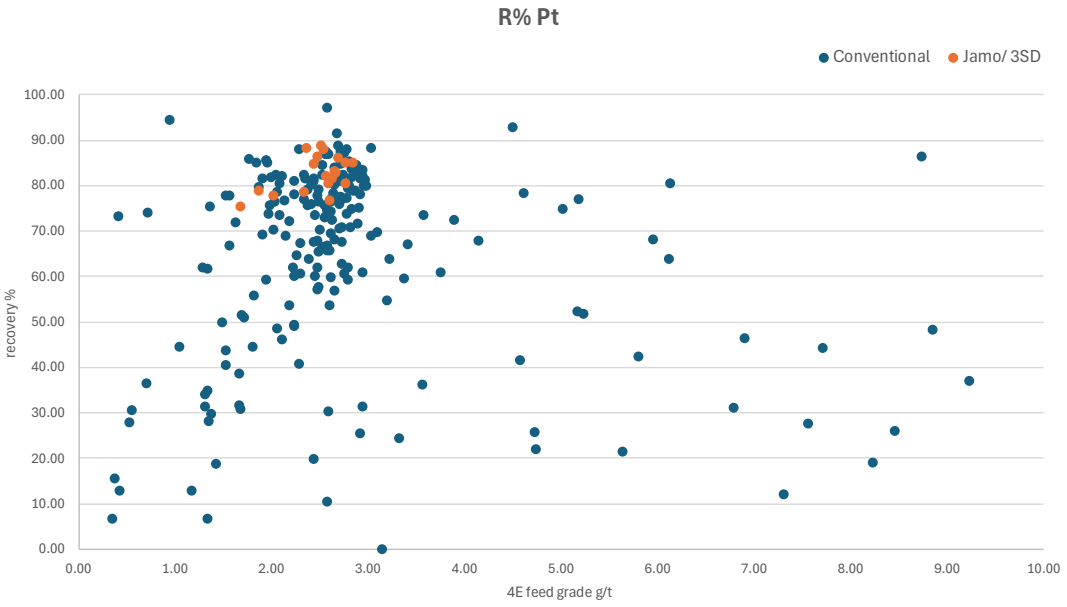


Figure 13-7: Pt recovery in Jameson cell as a function of PGM feed grade

Source: Bravo, 2026.

13.6.8.3 Rhodium Recovery

Rhodium recovery achieved 55% in the consolidated tests, as validated by the close correlation with the flotation kinetics of Pt and Pd.

Figure 13-8 highlights, in reddish-brown, rhodium recovery calculated as a function of PGM feed grade and its correlation with Pt-Pd in Jameson cell test work.

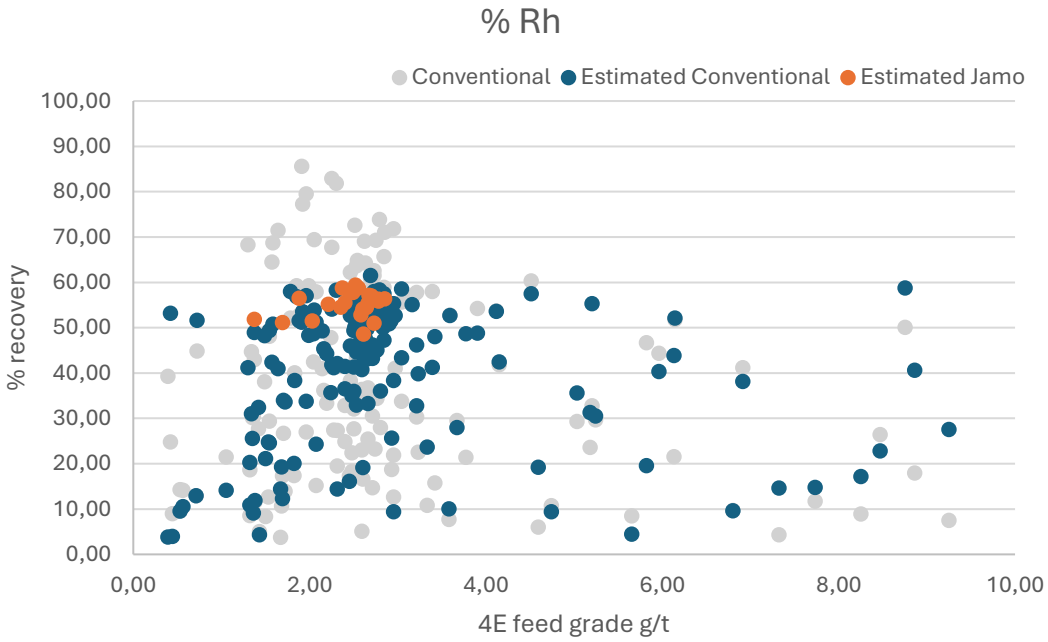


Figure 13-8: Rh recovery in Jameson cell as a function of PGM feed grade

Source: Bravo, 2026.

Figure 13-9 shows a significant correlation between Rh and Pt/Pd. Although different mineralogical domains seem to be involved, the response of high Pt/Pd and high Rh, and vice versa, is visible.

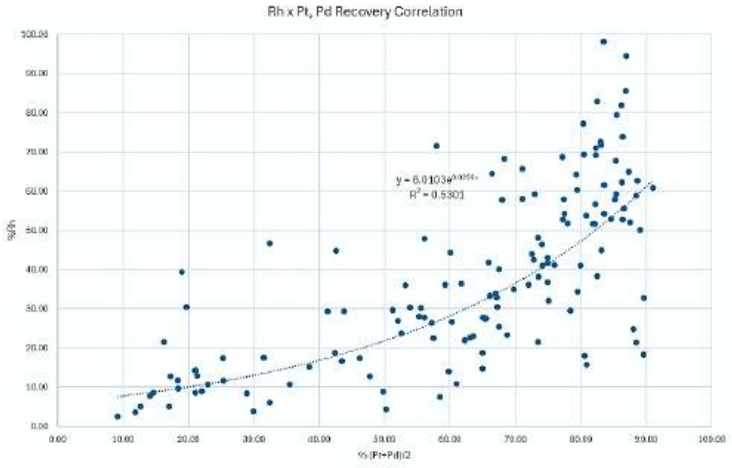


Figure 13-9: Rh x Pd/Pt recoveries

Source: Bravo, 2026.

13.6.8.4 Gold Recovery

Gold recovery reached values as high as 90%, based on BML locked-cycle test work. For this study, a recovery of 88% was assumed due to its relative variability in the deposit.

Figure 13-10 highlights, in reddish-brown, gold recovery as a function of feed grade in Jameson cell test work.

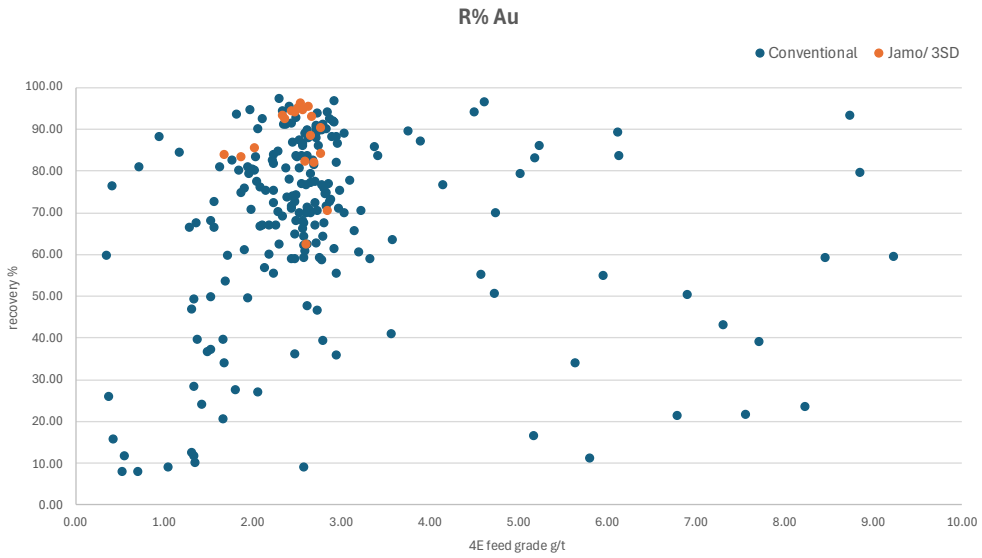


Figure 13-10: Au recovery in Jameson cell as a function of PGM feed grade

Source: Bravo, 2026.

13.6.8.5 Nickel Recovery

Nickel recovery achieved 70.6% in the Jameson cell test work, with active rejection of talc-bearing gangue and final concentrate grades of 5 - 8% Ni.

Figure 13-11 highlights, in dark green, low nickel variability, suggesting stable recovery.

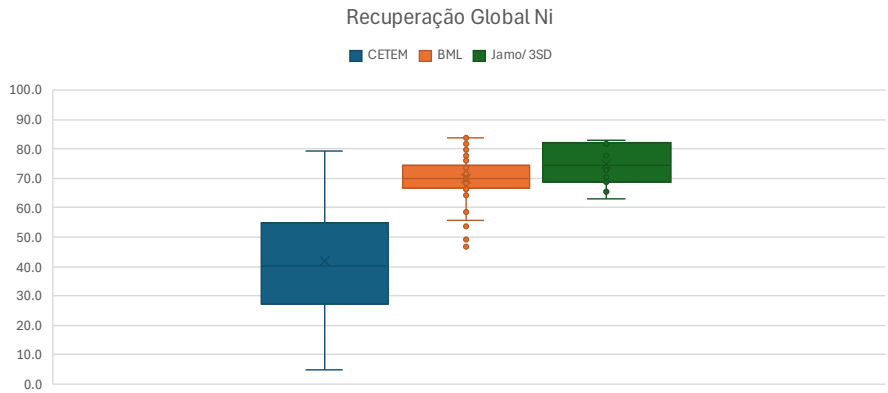


Figure 13-11: Ni recovery variability

Source: Bravo, 2026.

13.6.9 Tailings and Concentrate Thickening and Filtration Tests

13.6.9.1 Fresh Rock Tailings and Concentrate

Westech Brasil conducted vacuum sedimentation and filtration tests on fresh rock tailings and concentrate samples from the Luanga Project, collected from the mini-plant. The results indicated a sedimentation unit area of 0.45 m^2/tpd for tailings and 0.67 m^2/tpd for concentrate samples. The filtration rates obtained were 0.44 t/h m^2 and 0.70 t/h m^2 for tailings and concentrate samples, respectively.

13.6.10 Pyrometallurgy

A 68 g/t concentrate sample was sent to Arxo Metals in South Africa for preliminary roasting and smelting tests. The 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 concentrations were below detection limits for all precious metals, suggesting +99% recovery. Slag Ni was below the detection limit, suggesting high base metal recoveries, and low alloy entrapment in slag.

14 MINERAL RESOURCE ESTIMATES

GE21 carried out the 3D geological modelling, statistical and geostatistical studies, and grade estimate for the Luanga Project, assessing a set of factors, including the amount and spacing of available data, interpreted mineralization controls, mineralization style, and the quality of data used. Note that the MRE used in this PFS is the same as the 2025 MRE, and has an effective date of February 18, 2025. Drilling subsequent to that date has not been incorporated into an updated MRE as drilling is still ongoing. Once the 2026 drill program is completed and all results received and validated, an updated MRE will be prepared.

Geological modelling and estimation for the 2025 MRE 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 included assays from drilling up to February 18, 2025, being the effective date of the 2025 MRE, and was visually validated by considering relationships 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 DDH core samples from both Vale and Bravo campaigns, available on the MRE Effective Date. Table 14-1 summarizes the drilling database used in this Project stage. The map in Figure 14-1 shows the spatial distribution of the holes used.

Table 14-1: Luanga Project drill holes summary used in the 2025 MRE

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 contains geochemical results for the variables: Pd (ppm), Pt (ppm), Rh (ppm), Au (ppm), and Ni (ppm). Apart from these 5 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 assay results are quantitative for the variables presented, separated by sampling method and campaign, and are quantified in Table 14-2.

Table 14-2: Luanga Project assays summary for the 2025 MRE

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	99,034	98,231

Source: GE21, 2025.

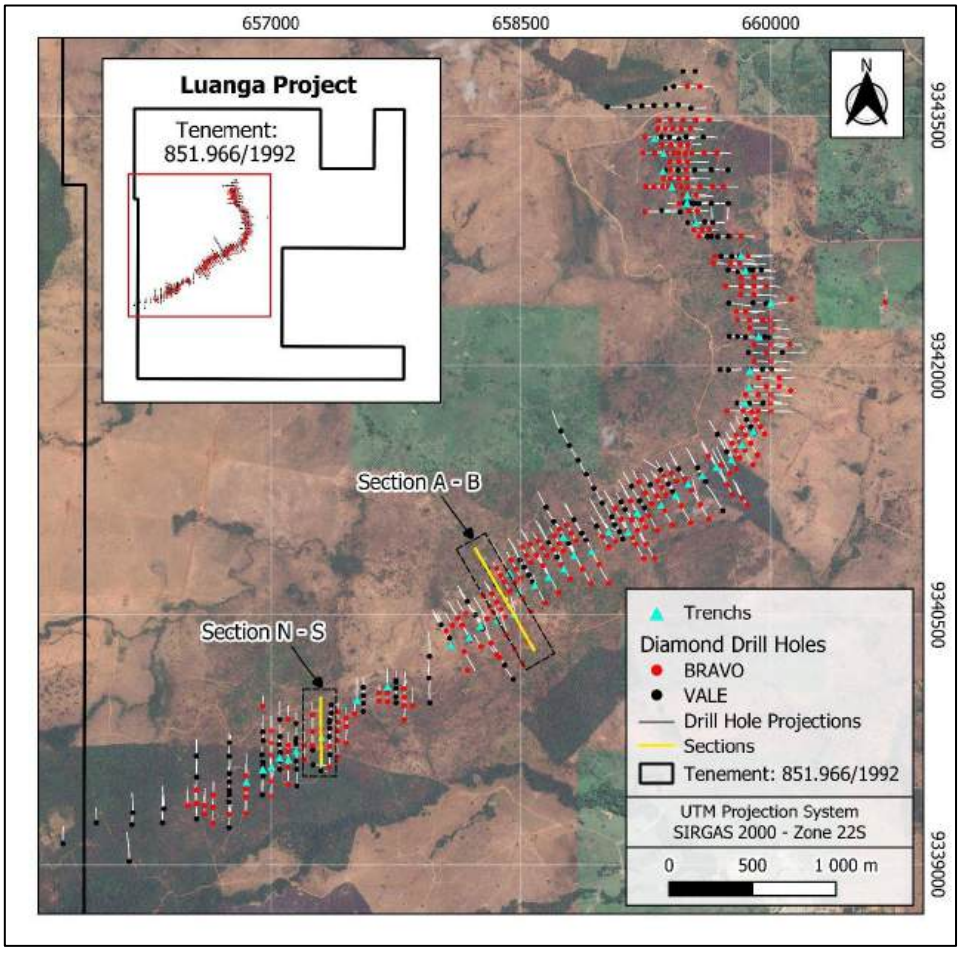


Figure 14-1: Map presenting the MRE database and the location of geologic models' sections

Source: GE21, 2025.

14.2 Geological Modelling

For the Luanga Project, four 3D models were generated. Each model spatializes different information about mineralization and 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 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, using continuity of PGM + Au grade. Two types of iso-grade surfaces were generated for the Luanga Project:

- Low Grade (LG): 3PGMs + Au > 0.3 ppm
- High Grade (HG): 3PGMs + Au > 1.0 ppm

The mineralization model was split into 9 regions, generating 18 grade shell groups (9 LG + 9 HG). These groups comprise 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

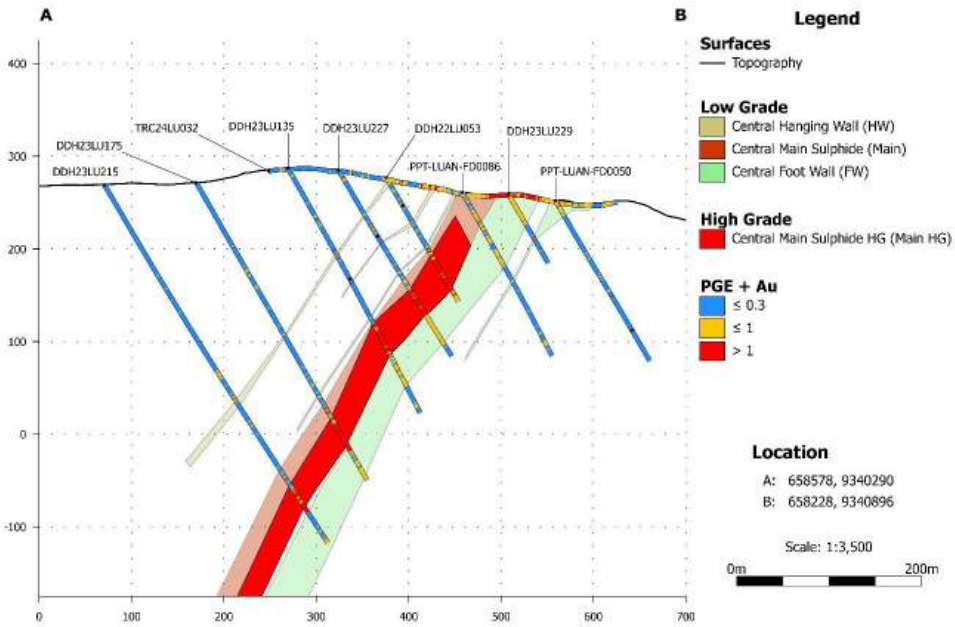


Figure 14-2: Grade shell model – section A-B – view of the central area

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

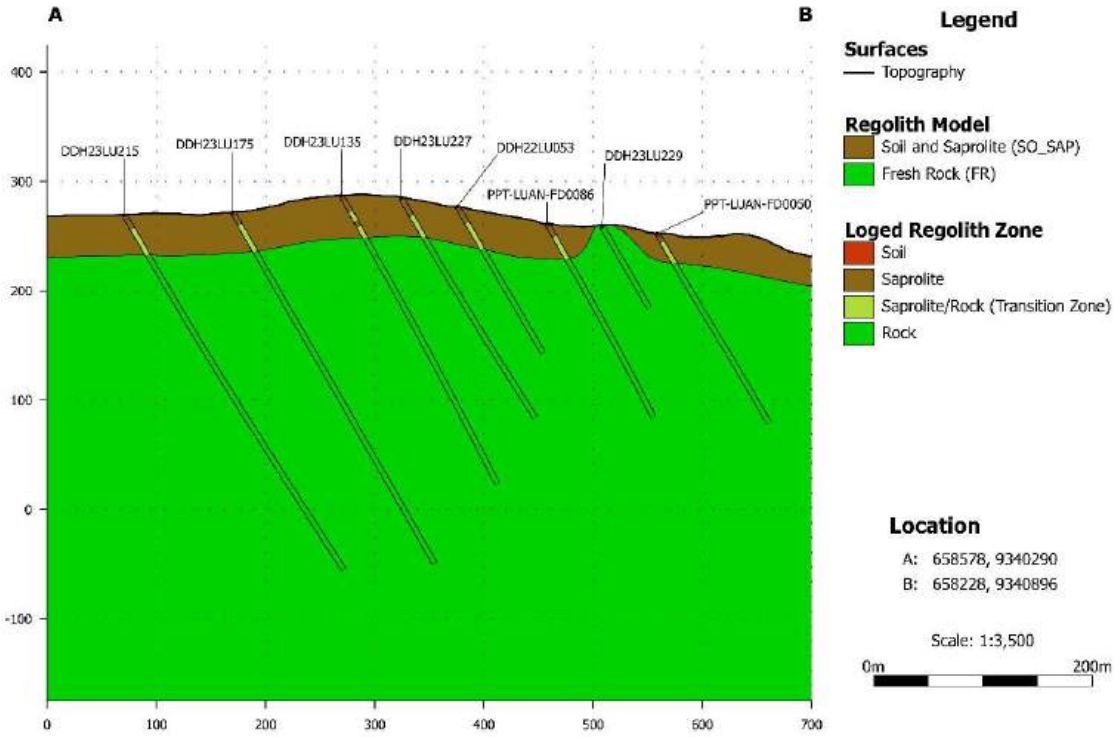


Figure 14-3: Regolith model – section A-B – view of the central area

Source: GE21, 2025.

14.2.3 Estimation Domains

After constructing 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), which were grouped into main zones (central, southwest, north N and south S) with respect to the stationary domain (HG and LG). These zones were estimated separately (8 domains in total).

The upper zone of the regolith model, Soil and Saprolite (SO_SAP), in 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 methods depending on 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 (HG and LG), presented in Table 14-3, were estimated individually.

Table 14-3: Summary of estimation domains and relationships between weathering model, stationary domain and estimation method

Domain Number	Weathering Model	Position	Stationary Domain	Estimation Method
1	Oxidized	Soil + Sap north	Soil + Sap north	OK
2		Soil + Sap central	Soil + Sap central	
3		Soil + Sap southwest	Soil + Sap southwest	
4	Fresh	central	LG	Turning Bands Simulation
5			HG	
6		southwest	LG	
7			HG	
8		north S	LG	
9			HG	
10		north N	LG	
11			HG	

Source: GE21, 2025.

14.2.4 Metallurgical Recovery Model

The metallurgical recovery 3D model was constructed to define different recovery factors for zones with different mineralogical and geochemical setups. Three domains were individualized (Figure 14-4):

- High Talc Material (High Talc)
- Oxidized Material (Ox)
- Fresh Material (Fresh)

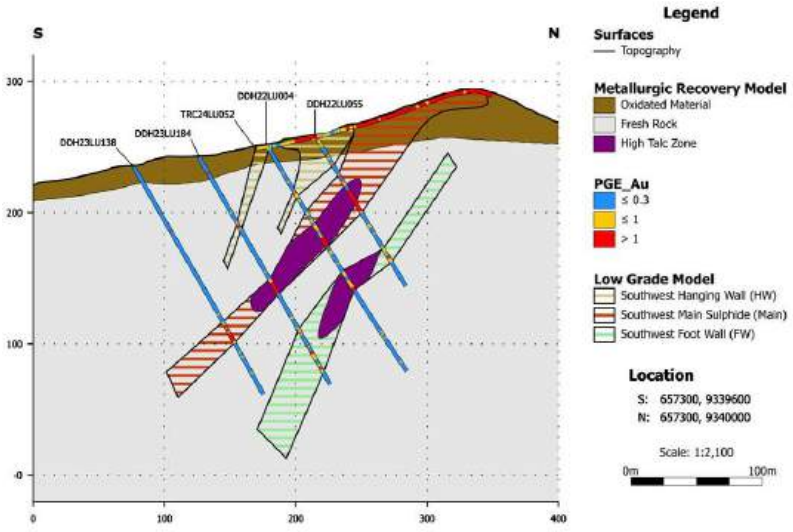


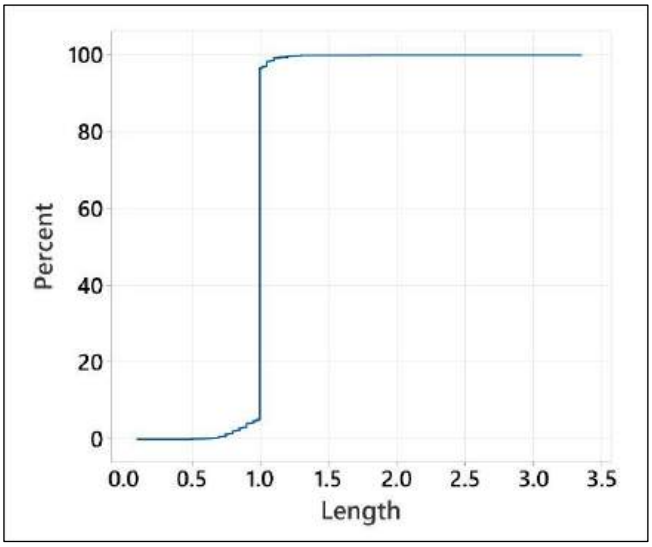
Figure 14-4: Metallurgical recovery model and low-grade shells – section N-S – view of the southwest area

Source: GE21, 2025.

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 m. GE21 performed the regularization of samples inside the modelled domains. The chosen composite size is 1 m. This composite was used for complementary statistical and geostatistical studies (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

Different analytical and sampling methodologies could represent different levels of support, since they exhibit different 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.

A 50 m boundary from the trench data and DDH 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 is shown in Figure 14-6.

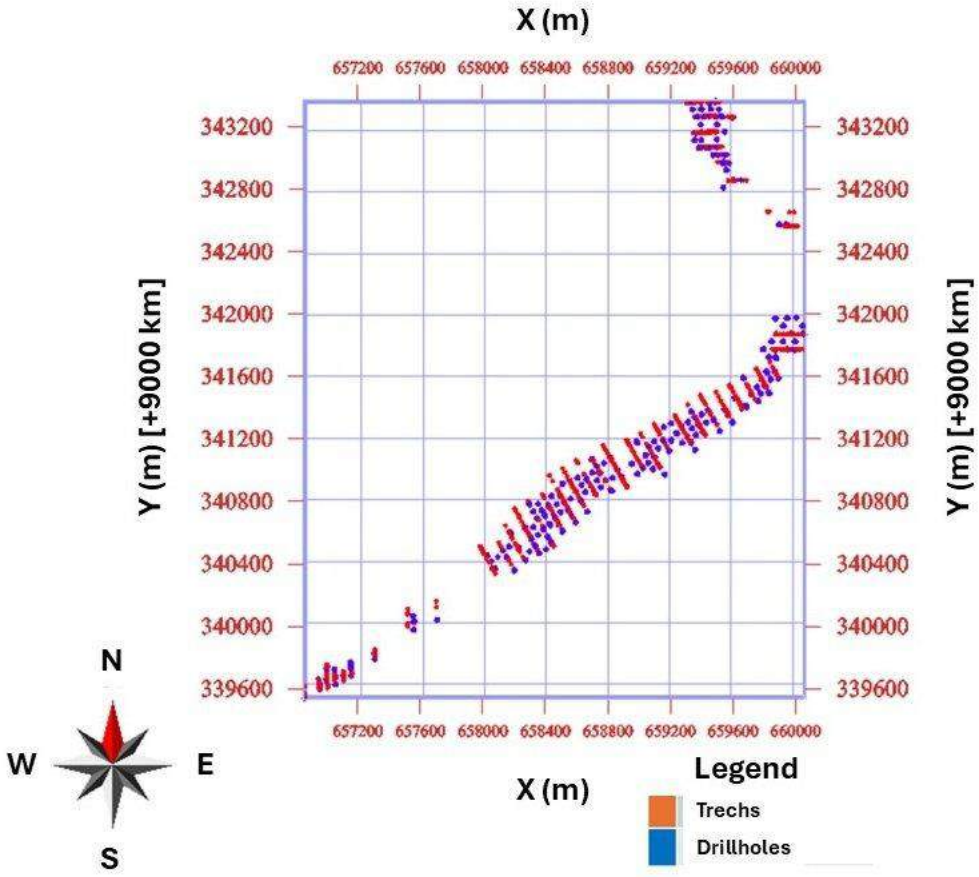


Figure 14-6: Dataset used for the transformation of trench data and DDH

Source: GE21, 2025.

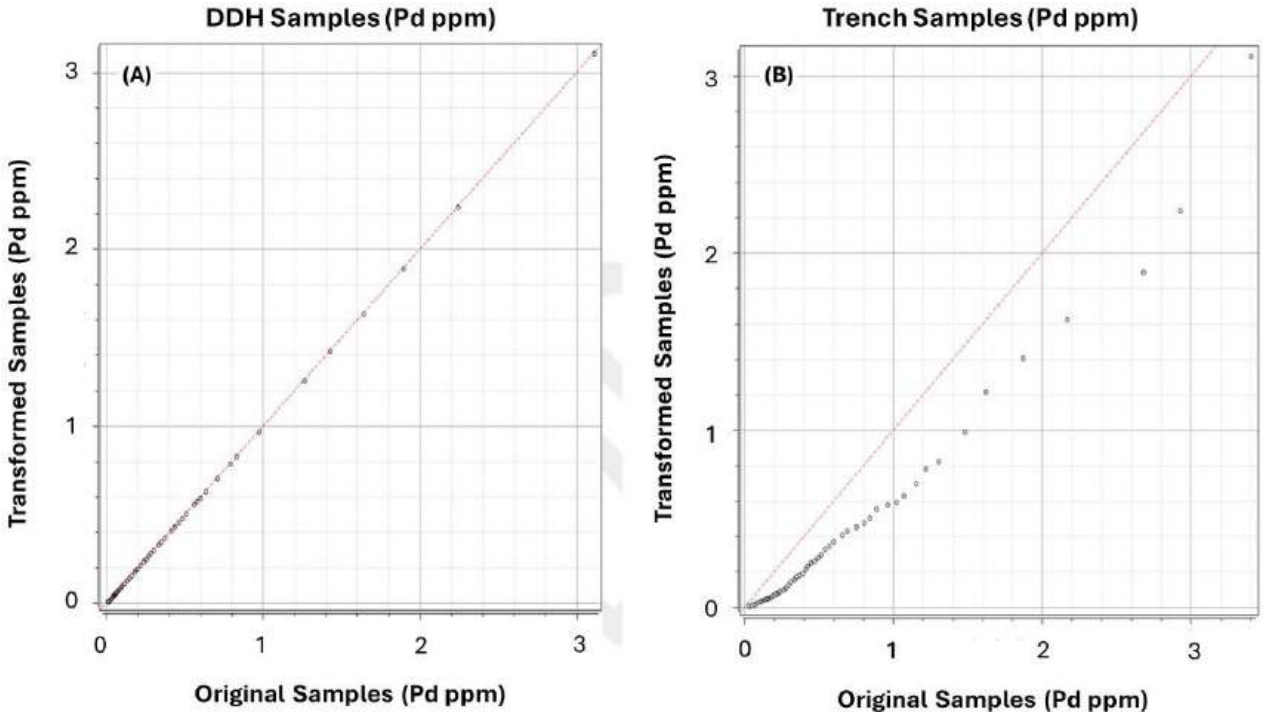


Figure 14-7: QQ plot of transformed and raw Pd values (trench data and DDH data)

Legend: A) QQ-plot of DDH raw and transformed sample; B) QQ-plot of trench raw and transformed samples.
Source: GE21, 2025.

Table 14-4 shows the statistics for the transformed and original composite data.

Table 14-4: Statistics for the 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)

The EDA was performed for Pd (ppm), Pt (ppm), Rh (ppm), Au (ppm) and Ni (ppm) variables, using the sample composites and considering the estimation. Table 14-5 shows the basic statistics for these variables.

Figure 14-8 to Figure 14-12 show the box plots for the Pd (ppm), Pt(ppm), Rh (ppm), Au (ppm) and Ni (ppm) variables.

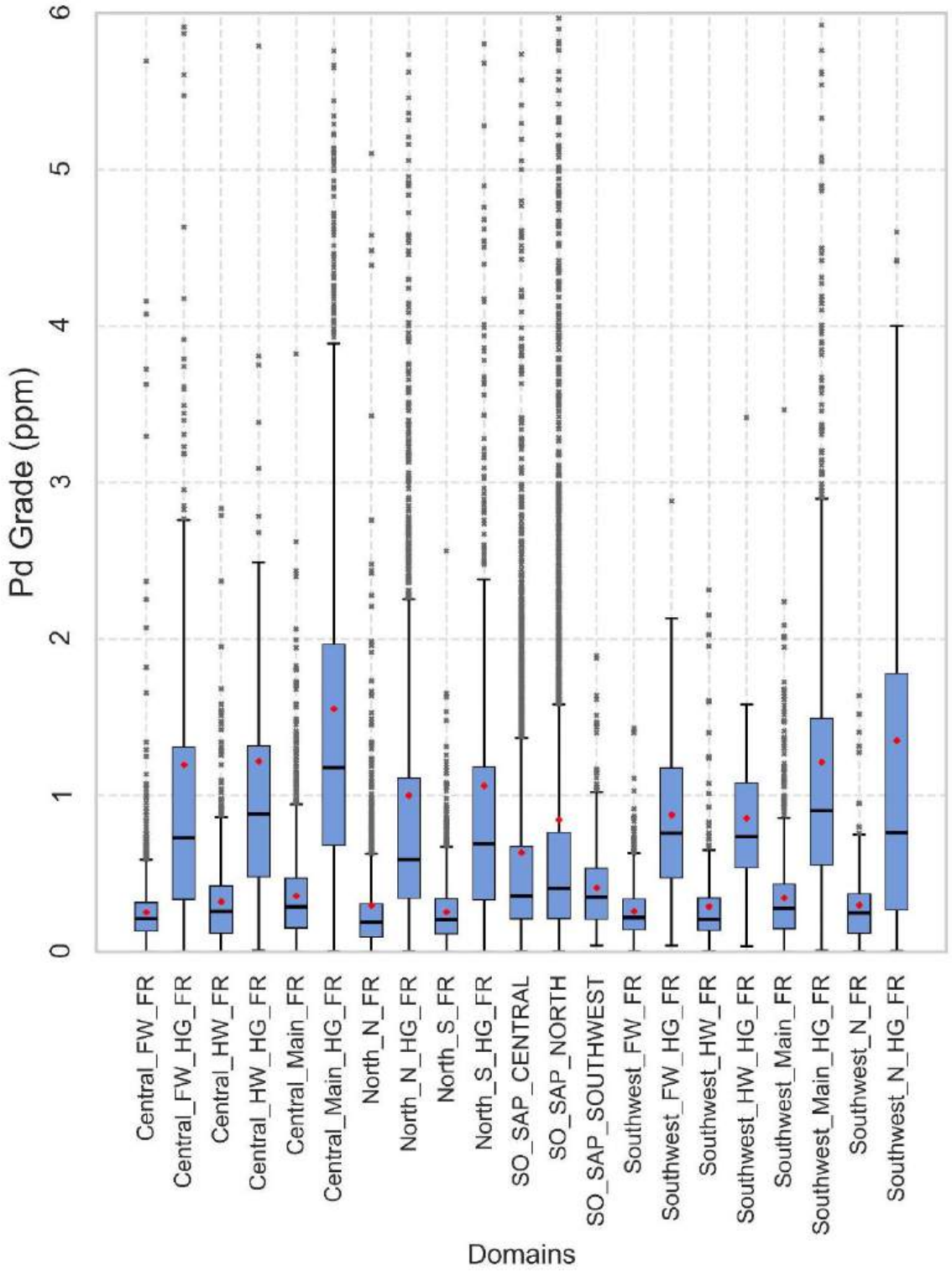


Figure 14-8: Pd ppm box plot chart by domains

Source: GE21, 2025.

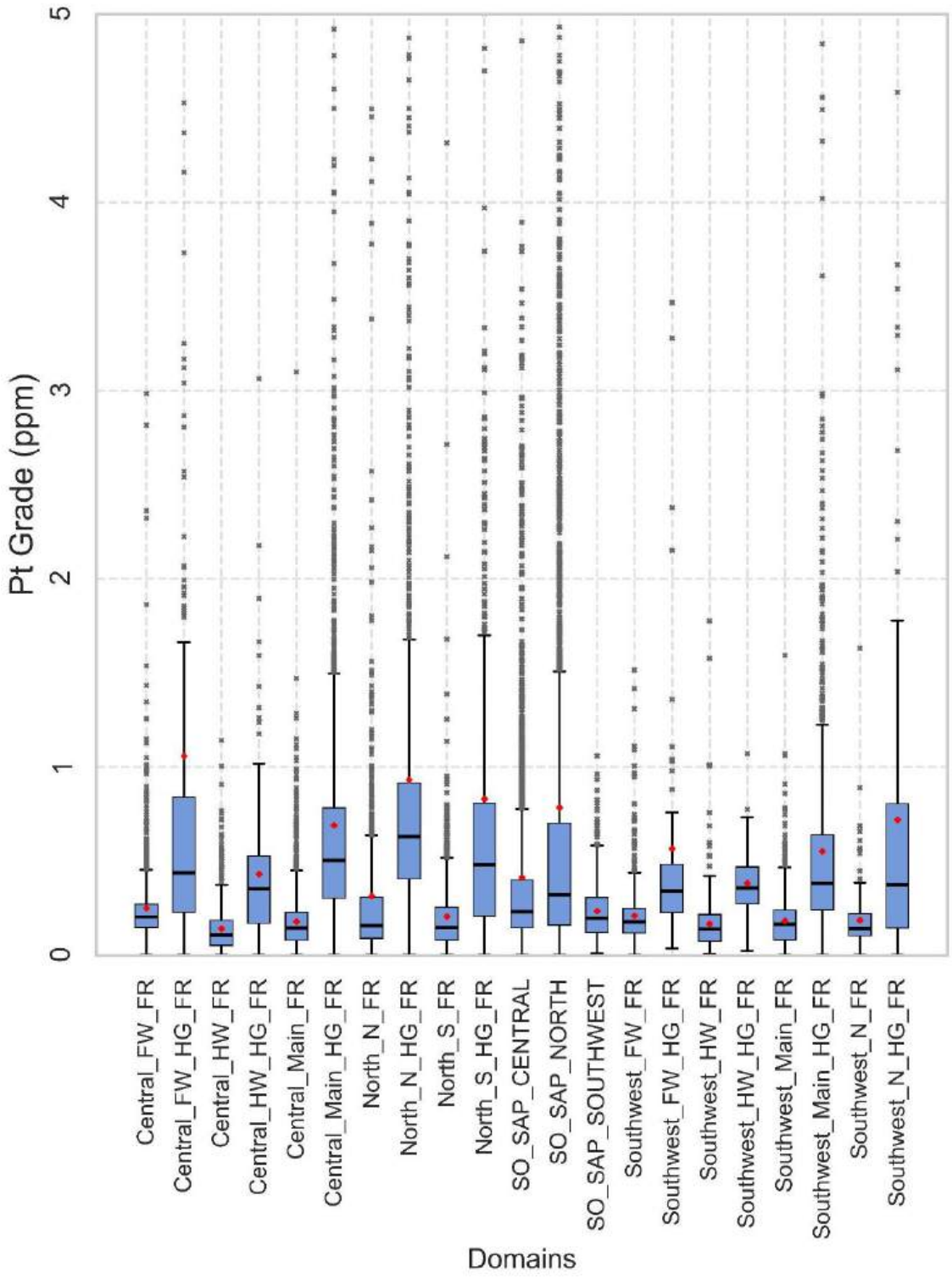


Figure 14-9: Pt ppm box plot chart by domains

Source: GE21, 2025.

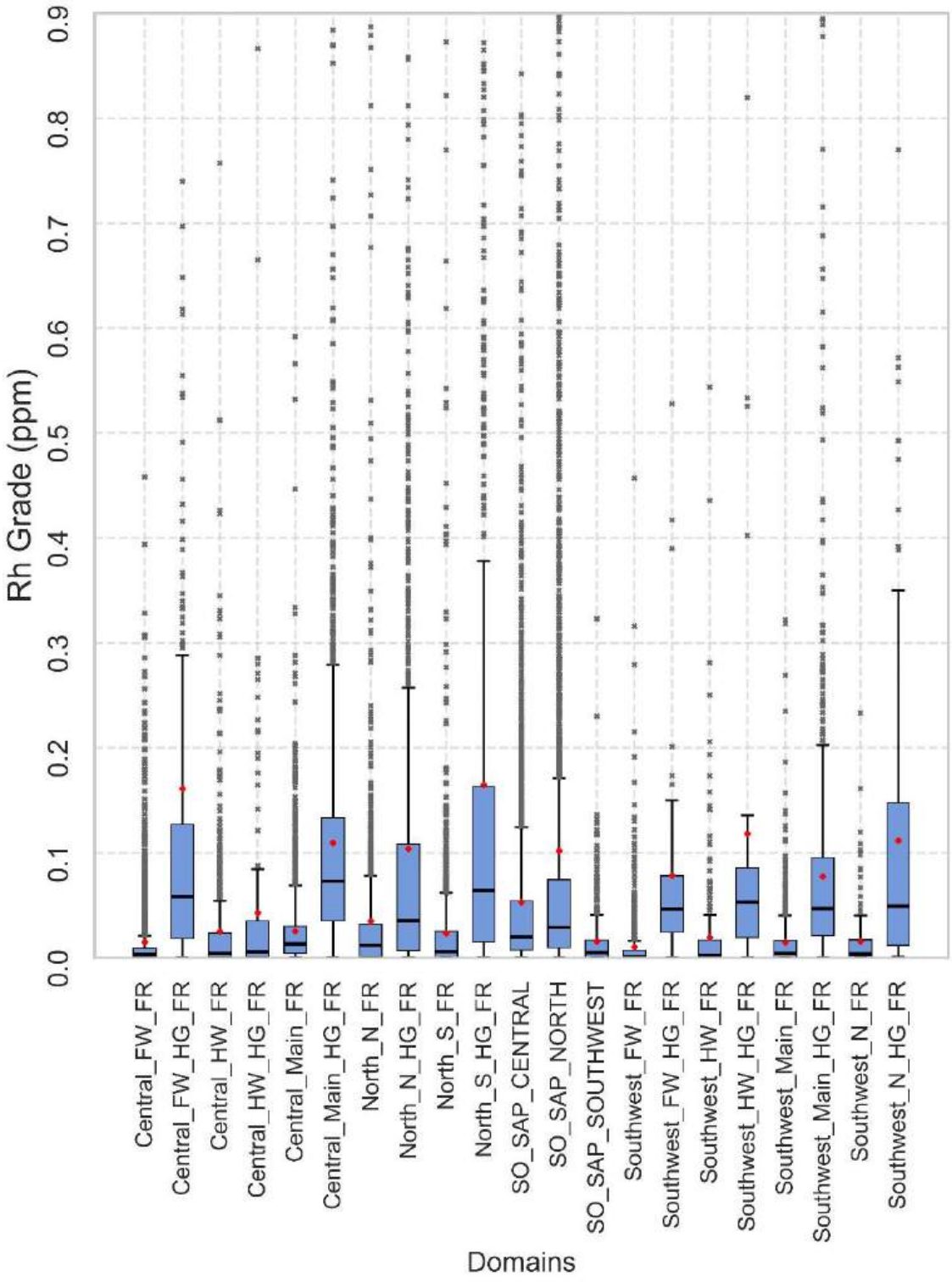


Figure 14-10: Rh ppm box plot chart by domains

Source: GE21, 2025.

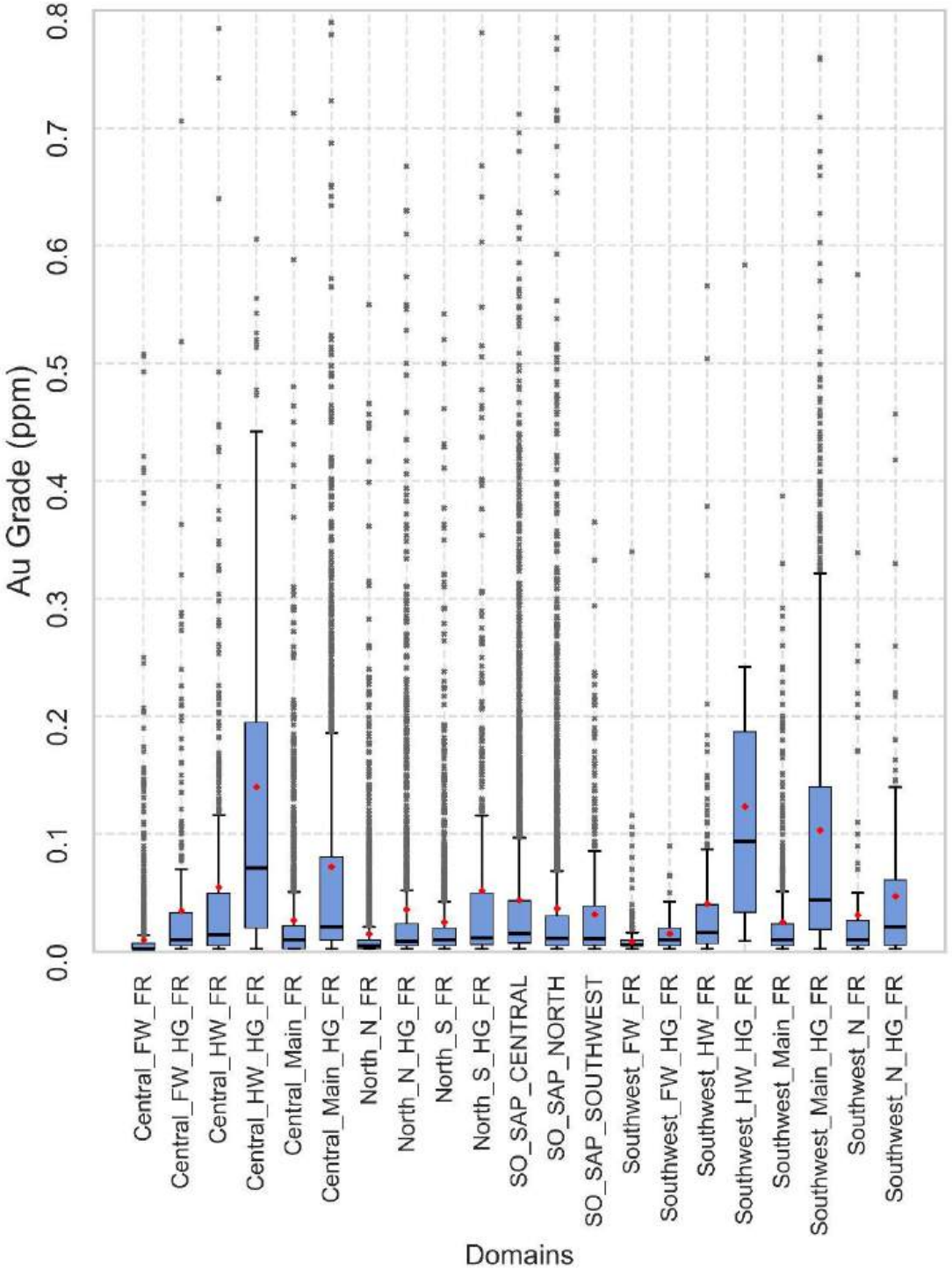


Figure 14-11: Au ppm box plot chart by domains

Source: GE21, 2025.

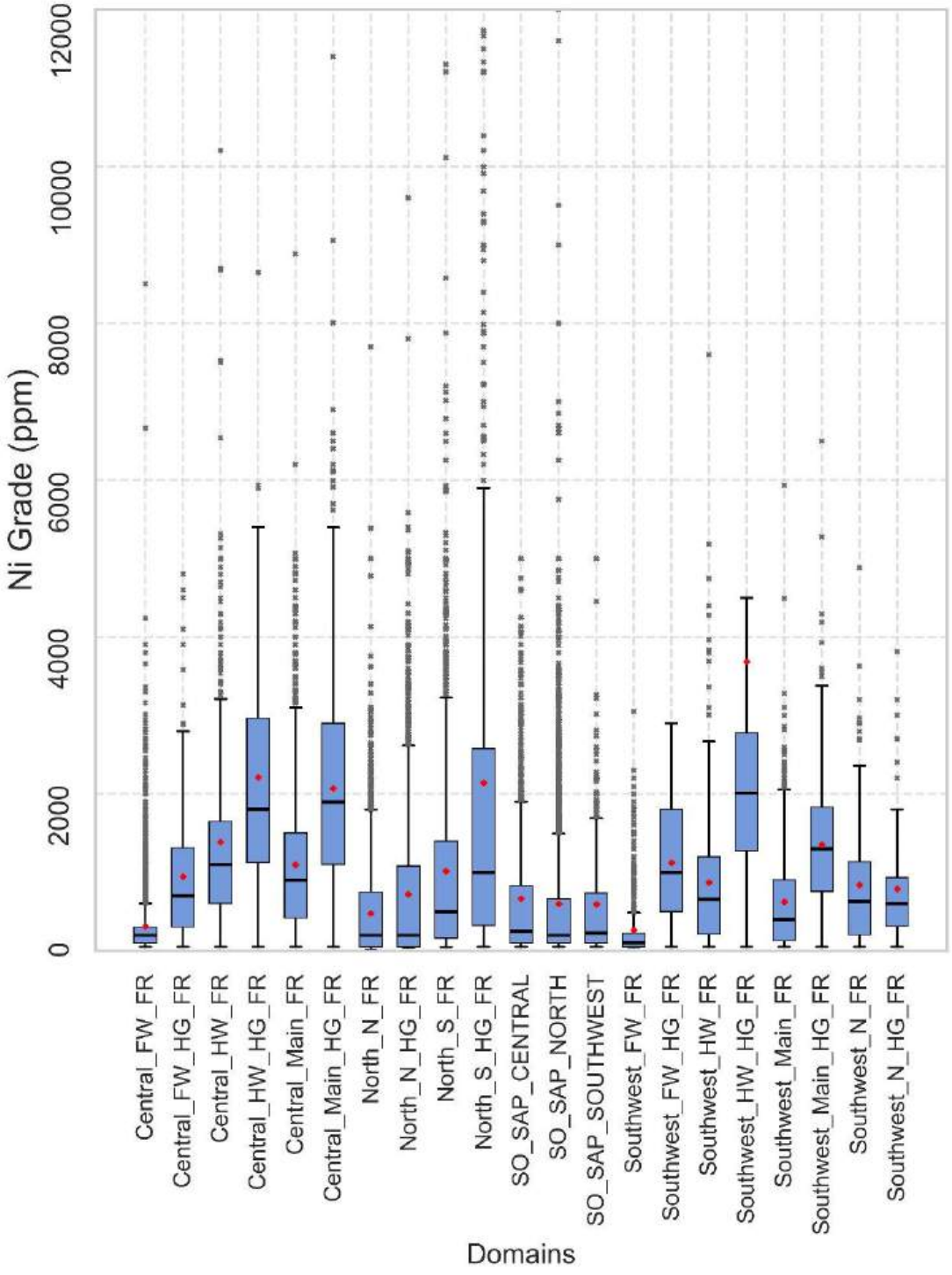


Figure 14-12: Ni ppm box plot chart by domains

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
	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	1,776.3	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

Variable	Domain	Length (m)	Mean	S.D.	C.V.	Var.	Min.	Q25	Q50	Q75	Max.
	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
	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

Variable	Domain	Length (m)	Mean	S.D.	C.V.	Var.	Min.	Q25	Q50	Q75	Max.
	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	206,575	50	100	200	300	8,503
	Central_FW_HG_FR	374.3	949	815	0.9	664,598	50	300	700	1,312	4,800
	Central_HW_FR	883.0	1,379	1,787	1.3	3,194,888	50	600	1,098	1,649	32,063
	Central_HW_HG_FR	162.6	2,221	1,914	0.9	3,663,130	50	1,124	1,800	3,000	18,767
	Central_Main_FR	1,895.0	1,094	942	0.9	888,094	50	417	900	1,500	17,410
	Central_Main_HG_FR	1,915.7	2,072	1,284	0.6	1,649,055	50	1,100	1,900	2,900	14,207
	North_N_FR	2,863.4	476	596	1.3	355,278	5	50	200	750	7,700
	North_N_HG_FR	1,737.5	722	1,030	1.4	1,061,372	40	50	200	1,088	9,600
	North_S_FR	1,776.3	1,018	1,346	1.3	1,811,107	48	161	500	1,400	17,544
	North_S_HG_FR	739.7	2,121	3,157	1.5	9,966,045	50	317	1,000	2,572	25,269
	SAP_SO_CENTRAL	1,777.3	612	886	1.4	785,780	50	100	213	725	5,000
	SAP_SO_NORTH	2,951.7	597	1,069	1.8	1,143,613	50	100	200	658	20,995
	SAP_SO_SOUTHWEST	465.8	686	940	1.4	883,853	50	100	250	886	5,000
	Southwest_FW_FR	771.6	263	388	1.5	150,880	46	50	101	225	3,051
	Southwest_FW_HG_FR	116.4	1,127	809	0.7	654,313	50	500	999	1,800	2,900
	Southwest_HW_FR	327.7	869	908	1.0	824,057	50	215	663	1,200	7,600

Variable	Domain	Length (m)	Mean	S.D.	C.V.	Var.	Min.	Q25	Q50	Q75	Max.
	Southwest_HW_HG_FR	28.5	3,784	5,567	1.5	30,989,448	52	1,302	2,020	3,402	25,018
	Southwest_Main_FR	938.5	624	647	1.0	418,231	50	126	399	905	5,930
	Southwest_Main_HG_FR	1,058.8	1,355	813	0.6	660,621	50	751	1,300	1,830	6,500
	Southwest_N_FR	171.3	839	812	1.0	659,308	50	203	649	1,108	4,886
	Southwest_N_HG_FR	135.6	791	658	0.8	433,066	50	315	600	997	3,818

Source: GE21, 2025.

14.4 Grade Estimation

14.4.1 Simulation Approach and Kriging Estimation

Geostatistical simulation and kriging are both spatial interpolation methods employed by geostatistics, but they have different purposes and methodologies. The kriging estimation produces a smooth Best Linear Unbiased Estimator (BLUE) value at unsampled locations, 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. The geostatistical simulations allow quantifying spatial uncertainty, risk assessment, and decision-making in Mineral Resource estimation.

The E-type post-processing consists of averaging the average of overall simulations over some support and is close to kriging estimation. These steps form the foundation of the mining planning and future activities in a 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 modelled variogram.
- The **histogram** (or Probability Distribution Function – PDF) represents the statistical distribution of values in the dataset. A valid conditional simulation preserves the data histogram.

If these conditions are satisfied, the simulations ensure statistical realism and spatial realism of the mineral deposit.

14.4.2 Simulation and Kriging Strategies

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

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			
	Rh	65	150	90	100	100	10	5	20	2x, 3x and infinite
	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.

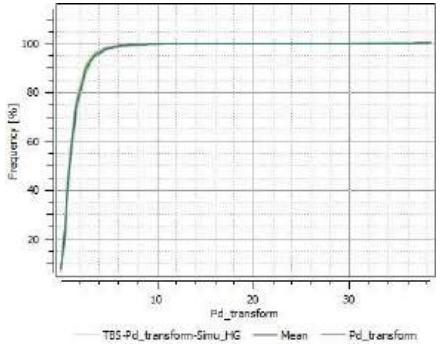
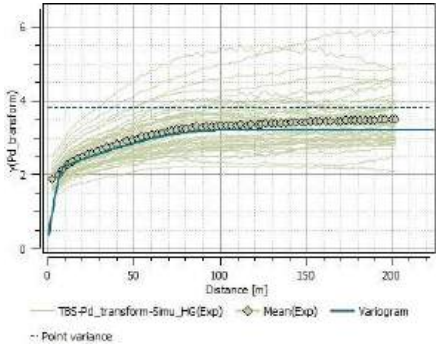
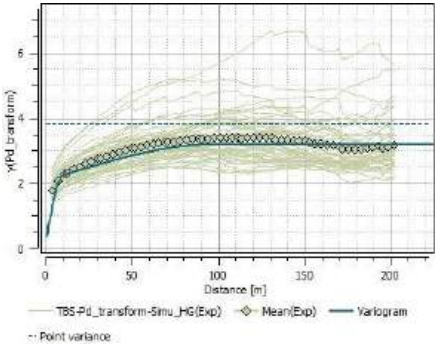
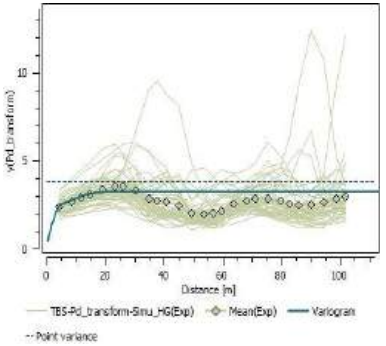
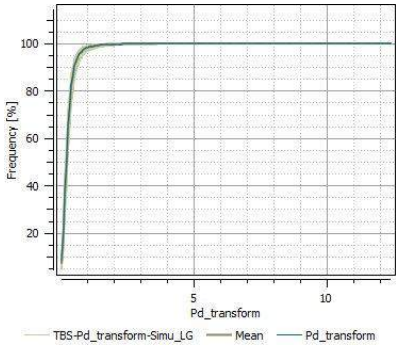
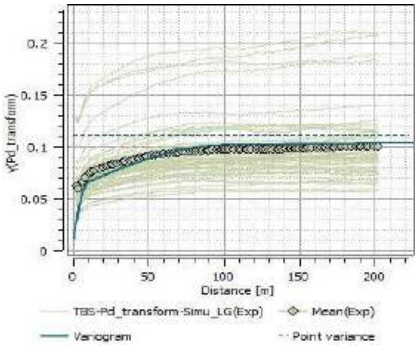
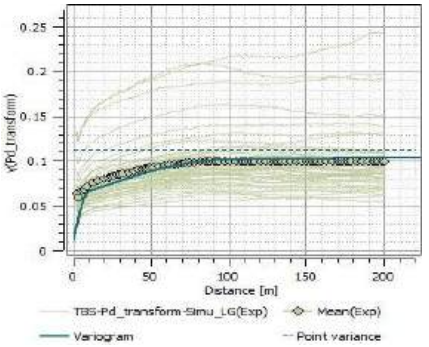
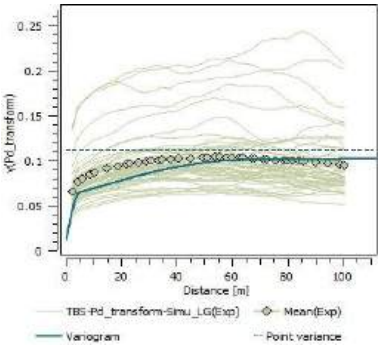
The OK strategy shown in Table 14-7 was used to estimate the variables in the fresh domain for comparison with E-type simulation values.

14.4.3 Variograms and Simulation Validation

Simulation was performed using Isatis.Neo Mining (version 2024.04.2), returning 50 realizations with 800 bands. Simulations are compared with raw variogram and histogram values. Table 14-8 shows an example of simulation validation in the central domain for the 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 the oxide domain.

Table 14-8: Example of simulation validation for Pd in the central area domains

Domain	Histogram	Major Axis	Intermediate Axis	Minimum Axis
Pd Central HG				
Pd Central LG				

Source: GE21, 2025.

Table 14-9: Variogram models for each element and domain in the 2025 MRE

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	∞*	∞*	∞*
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	-	-	-	-	-

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)
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 1000 m)																				

Source: GE21, 2025.

Table 14-10: Variogram models for oxide domain in the 2025 MRE

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 consider that a block 10 times discretized is sufficient to represent the point variability of the data. The simulation block has the support of 2.5 x 2.5 x 2.5 m. The parameters of the simulation block model are as shown in Table 14-11.

Table 14-11: Grid geometry of simulation block size

	X	Y	Z
Number of nodes	1,920	2,240	320
Mesh size	2.5 m	2.5 m	2.5 m
Grid origin (centre)	655,775.00 m	9,338,780.00 m	-241.49 m
Grid origin (corner)	655,773.75 m	9,338,778.75 m	-242.74 m

Source: GE21, 2025.

After simulation, the realizations are upscaled to a parental block model with the dimensions of 25 x 25 x 5 m according to Table 14-12. The value of the block is considered the average of all upscaled realizations.

Table 14-12: Grid geometry of parental block size 25 m x 25 m x 5 m

	X	Y	Z
Number of nodes	192	224	160
Mesh size	25 m	25 m	5 m
Grid origin (centre)	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 8 x 8 x 4 m 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 in the 2025 MRE

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/cm^3
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 Quadratic Distance (IQD) to estimate density samples in the fresh domain, using an 80 x 80 x 50 m 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 based on analysis of the in situ sampling dataset. The type averages for the block model attribution are shown in Table 14-14.

Table 14-14: Bravo's density values by all domains in the 2025 MRE

Density (g/cm^3)										
Name	Block Count	Mean	Standard Deviation	Coefficient of Variation	Variance	Minimum	Lower quartile	Median	Upper quartile	Maximum
Oxide Domain	1,253,506	1.46	0.01	0.01	0.00	1.46	1.46	1.46	1.46	2.96
Central FW HG	424,558	2.85	0.14	0.05	0.02	2.47	2.74	2.81	2.95	3.25
Central FW LG	820,882	2.89	0.13	0.05	0.02	2.46	2.82	2.88	2.97	3.43
Central HW HG	188,528	2.83	0.18	0.06	0.03	2.58	2.71	2.81	3.02	3.43
Central HW LG	267,722	2.82	0.13	0.05	0.02	2.47	2.73	2.80	2.88	3.44
Central Main HG	1,970,980	2.86	0.13	0.05	0.02	2.52	2.78	2.85	2.93	3.54
Central Main LG	293,494	2.94	0.17	0.06	0.03	2.58	2.80	2.90	3.10	3.42
North N HG	1,015,581	2.75	0.11	0.04	0.01	2.19	2.67	2.73	2.82	3.36
North N LG	370,692	2.77	0.10	0.04	0.01	2.56	2.69	2.74	2.88	2.97
North S HG	507,543	2.85	0.18	0.06	0.03	2.50	2.74	2.81	2.91	3.53
North S LG	303,666	2.92	0.13	0.04	0.02	2.59	2.81	2.92	3.04	3.23
Southwest FW HG	178,859	2.78	0.05	0.02	0.00	2.46	2.74	2.81	2.81	2.96
Southwest FW LG	312,514	2.84	0.12	0.04	0.02	2.56	2.71	2.93	2.94	2.99
Southwest HW HG	12,454	3.00	0.30	0.10	0.09	2.64	2.79	2.83	3.46	3.47
Southwest HW LG	102,013	2.80	0.12	0.04	0.01	2.58	2.72	2.78	2.90	3.20
Southwest Main HG	647,558	2.81	0.09	0.03	0.01	2.61	2.74	2.80	2.91	3.01
Southwest Main LG	127,113	2.79	0.12	0.04	0.01	2.53	2.69	2.76	2.90	3.13
Southwest N HG	49,404	2.74	0.05	0.02	0.00	2.67	2.71	2.72	2.76	2.86
Southwest N LG	40,546	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 Nearest Neighbour (NN) and ordinary kriging (OK) as comparisons to the Simulation (E-type) estimate.

NN-check plots (Figure 14-13 and Figure 14-16) show the results of the 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 acceptable limits.

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, OK, and the NN method in swath coordinate interval graphs along the X, Y, and Z axes. Figure 14-19 shows the swath plot validation results of the estimated Pd ppm.

Figure 14-17 and Figure 14-18 show the swath plot comparison of OK, simulation E-type and NN 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, as 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 E-type simulation estimates 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

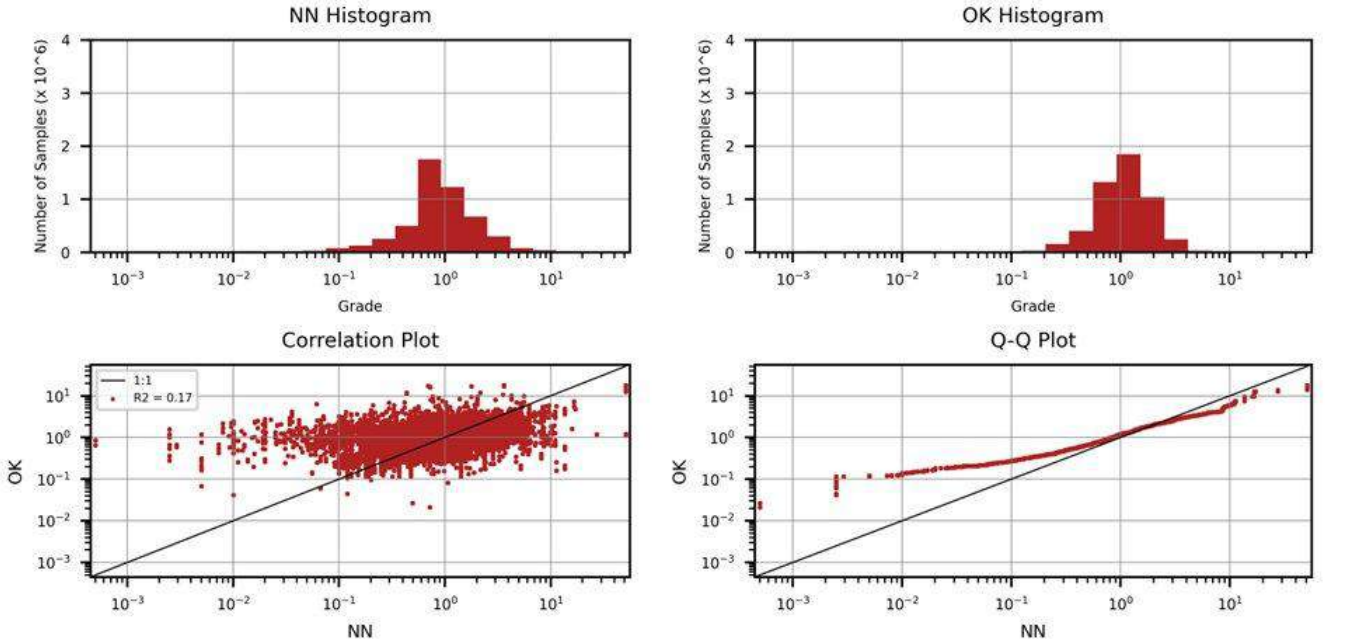


Figure 14-13: NN vs OK (HG)

Source: GE21, 2025.

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

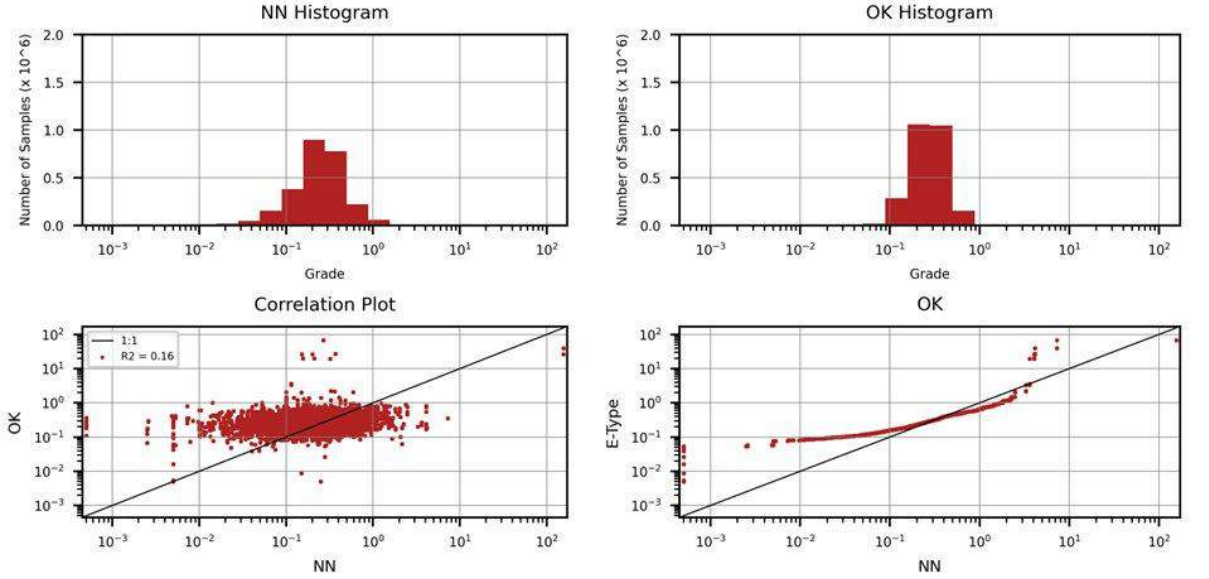


Figure 14-14: NN vs OK (LG)

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

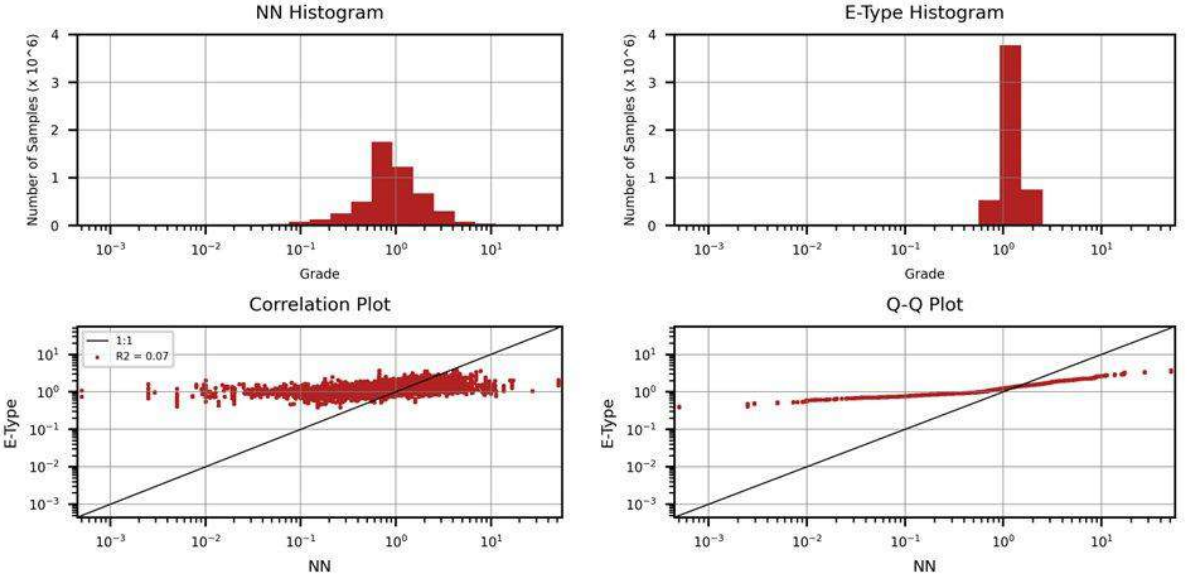


Figure 14-15: NN-check vs E-type (HG)

Source: GE21, 2025.

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

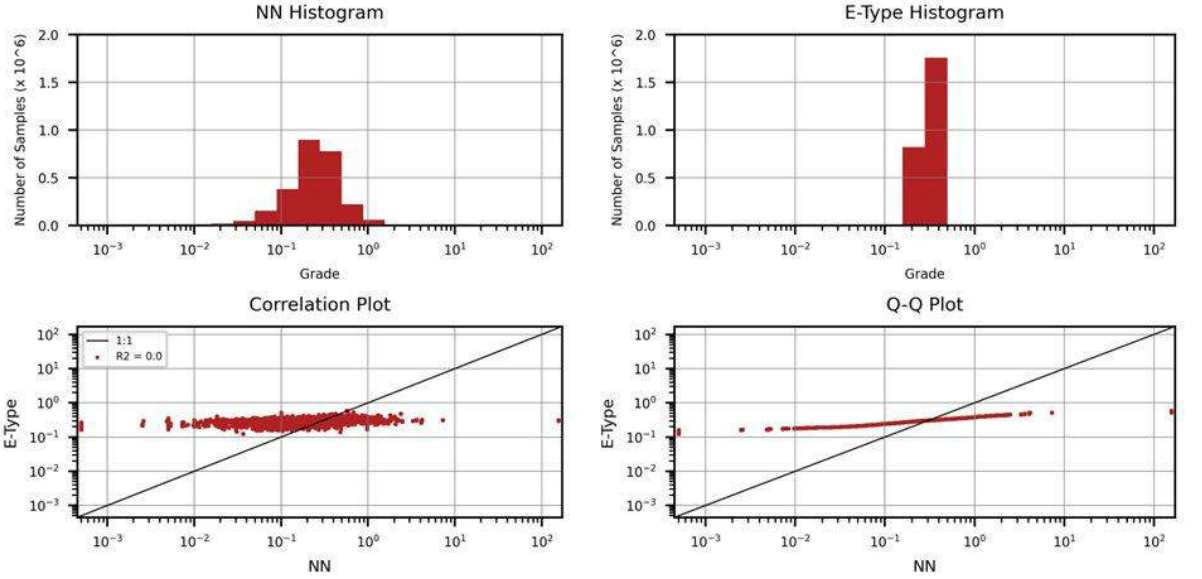


Figure 14-16: NN vs E-type (LG)

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

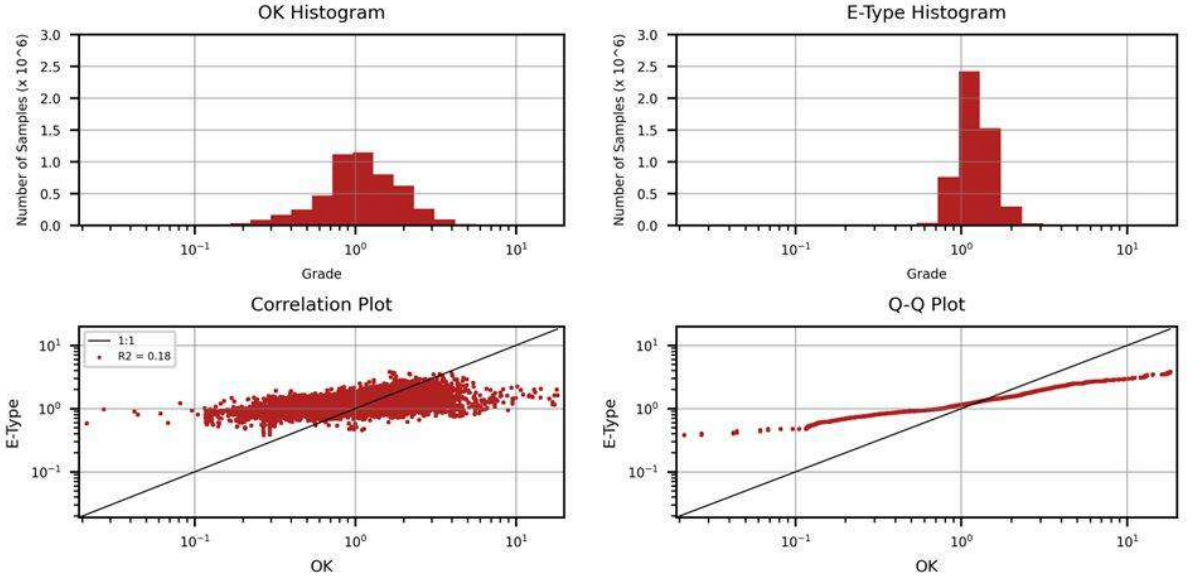


Figure 14-17: OK vs E-type (HG)

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

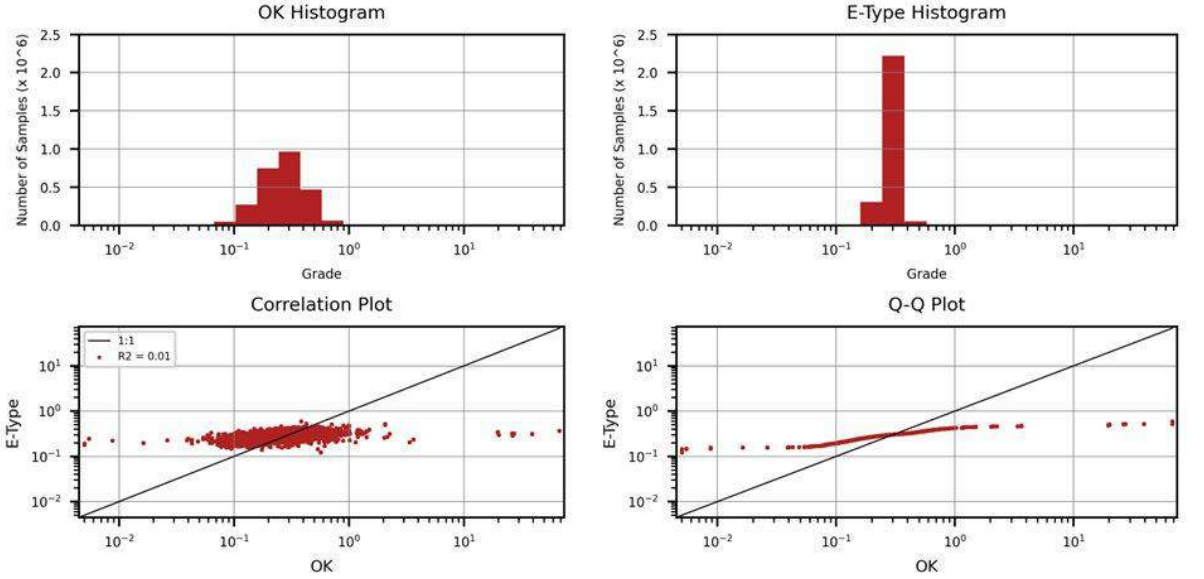


Figure 14-18: OK vs E-type (LG)

Source: GE21, 2025.

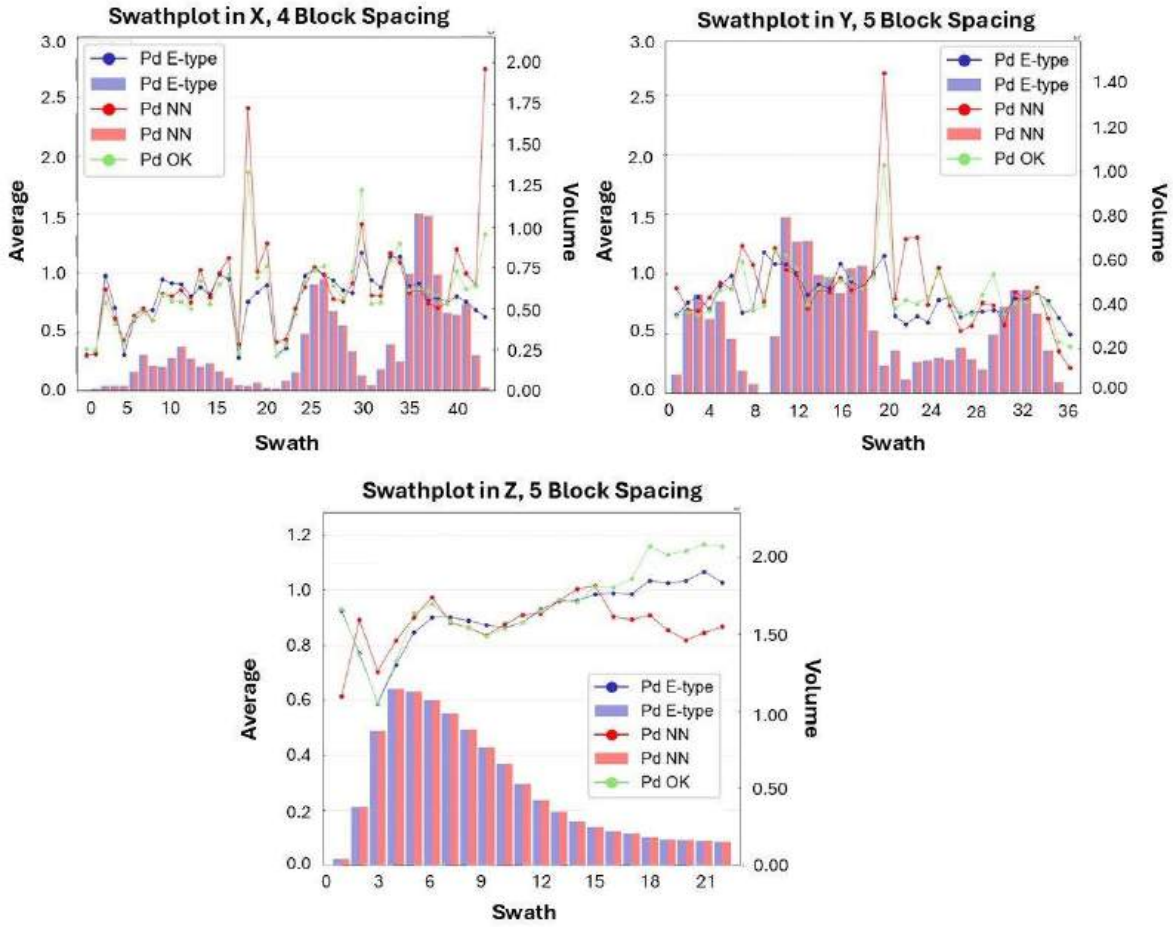


Figure 14-19: Swath plot E-type vs NN vs OK grades Pd (ppm)

Source: GE21, 2025.

14.6 Classification of Mineral Resources

The 2025 Mineral Resource was classified per CIM Standards (2014) and followed CIM Best Practices Guidelines (2019), utilizing geostatistical and classical methods, along with economically- and mining-appropriate parameters relevant to the deposit type.

The 2025 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 closely enough for geological and level continuity to be reasonably assumed.
- A "**Measured Mineral Resource**" is that part of a Mineral Resource for which quantity, grade 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 established by GE21 for the Measured, Indicated, and Inferred categories were determined using 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 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 45 x 45 m, with a minimum 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 75 x 75 m, 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 Prospects for Eventual Economic Extraction (RPEEE), which GE21 usually assesses using a mathematical model pit shell, which limits the blocks classified as a Mineral Resource generated from an economic and geometric function.

Table 14-15 shows a perspective view of the Luanga Project's 2025 MRE classification.

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	
	CoG (Whittle)	Fresh	-	-
		Weathered	-	-
Processing	Metallurgical Recovery - Weathered	Pd	81.0%	Mill
		Pt	23.0%	Mill
		Rh	54.0%	Mill
		Au	90.0%	Mill
		Ni	0.0%	Mill
	Metallurgical Recovery - Fresh	Pd	77.0%	Mill
		Pt	81.0%	Mill
		Rh	51.0%	Mill
		Au	48.0%	Mill
		Ni	50.0%	Mill
	Metallurgical 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	USD/t mined

Optimization Parameters - RPEEE				
Item				Unit
		Processing Cost	7.50	USD/t ROM
		Grade Control	1.00	
		Logistics	0.50	
		Rehabilitation	1.00	
		G&A	1.50	
	Fresh / High Talc	Mining Cost	3.00	USD/t mined
		Processing Cost	9.00	USD/t ROM
		Grade Control	1.00	
		Logistics	0.50	
		Rehabilitation	1.00	
		G&A	1.50	
Selling	Price	Pd	1,380	USD/oz
		Pt	1,100	USD/oz
		Rh	6,200	USD/oz
		Au	1,500	USD/oz
		Ni	7.10	USD/lb
	Royalties	All	2.0	%

Source: GE21, 2025.

Table 14-16: 2025 MRE statement at a cut-off of 0.5 g/t PdEq*

Resource	Classification	Domain	Mass Mt	Average Value Pd eq ppm	Pd g/t	Pt g/t	Au g/t	Rh g/t	Ni %	Material Content Pd eq koz	Pd koz	Pt koz	Au koz	Rh koz	Ni 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 Mr. Porfirio Cabaleiro Rodriguez, B.Sc Min. Eng., FAIG, FAusIMM, director of GE21, an independent QP under NI 43-101. The Effective Date of the MRE is February 18, 2025.
2. Mineral Resources are reported using the CIM Definition Standards (2014) and were estimated in accordance with the CIM Best Practices Guidelines (2019), 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. Chemical elements are estimated using different estimation methodologies according to the weathering model. OK 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 be classified 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 MRE 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.
     - Payable royalties of 2% (only considering Financial Compensation for Exploitation of Mineral Resources—CFEM—for Mineral 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: $F1 = \frac{(Pt_p * Pt_R)}{(Pd_p * Pd_R)} Pt_t$ $F2 = \frac{(Rh_p * Rh_R)}{(Pd_p * Pd_R)} Rh_t$ $F3 = \frac{(Au_p * Au_R)}{(Pd_p * Pd_R)} Au_t$ $F4 = \frac{(Ni_p * Ni_R)}{(Pd_p * Pd_R)} Ni_t$
      $_P$ = 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. 10 Mtpa:
   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.

Table 14-17: 2025 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
			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	—

					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
		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
			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	—

					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
			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
			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

Resource	Classification	Weathering	Min Styles	Mass	Average Value						Material Content					
					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
	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	—
			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**

Resource	Classification	Weathering	Min Styles	Mass	Average Value						Material Content					
					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
			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
			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 by Mr. Porfirio Cabaleiro Rodriguez, B.Sc Min. Eng., FAIG, FAusIMM, director of GE21, an independent QP under NI 43-101. The Effective Date of the MRE is February 18, 2025.
2. Mineral Resources are reported using the CIM Definition Standards (2014) and were estimated in accordance with the CIM Best Practices Guidelines (2019), 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. Chemical elements are estimated using different estimation methodologies according to the Weathering Model. OK was applied to the 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 MRE 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 Financial Compensation for Exploitation of Mineral Resources—CFEM—for Mineral Reserves, a complete set of royalties must be considered).
7. 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
8. 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. 10 Mtpa:
   - 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.
9. The current MRE supersedes and replaces the previous estimate (2023), which should no longer be relied upon.
10. 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.
11. Totals may not sum due to rounding.

Source: GE21, 2025.

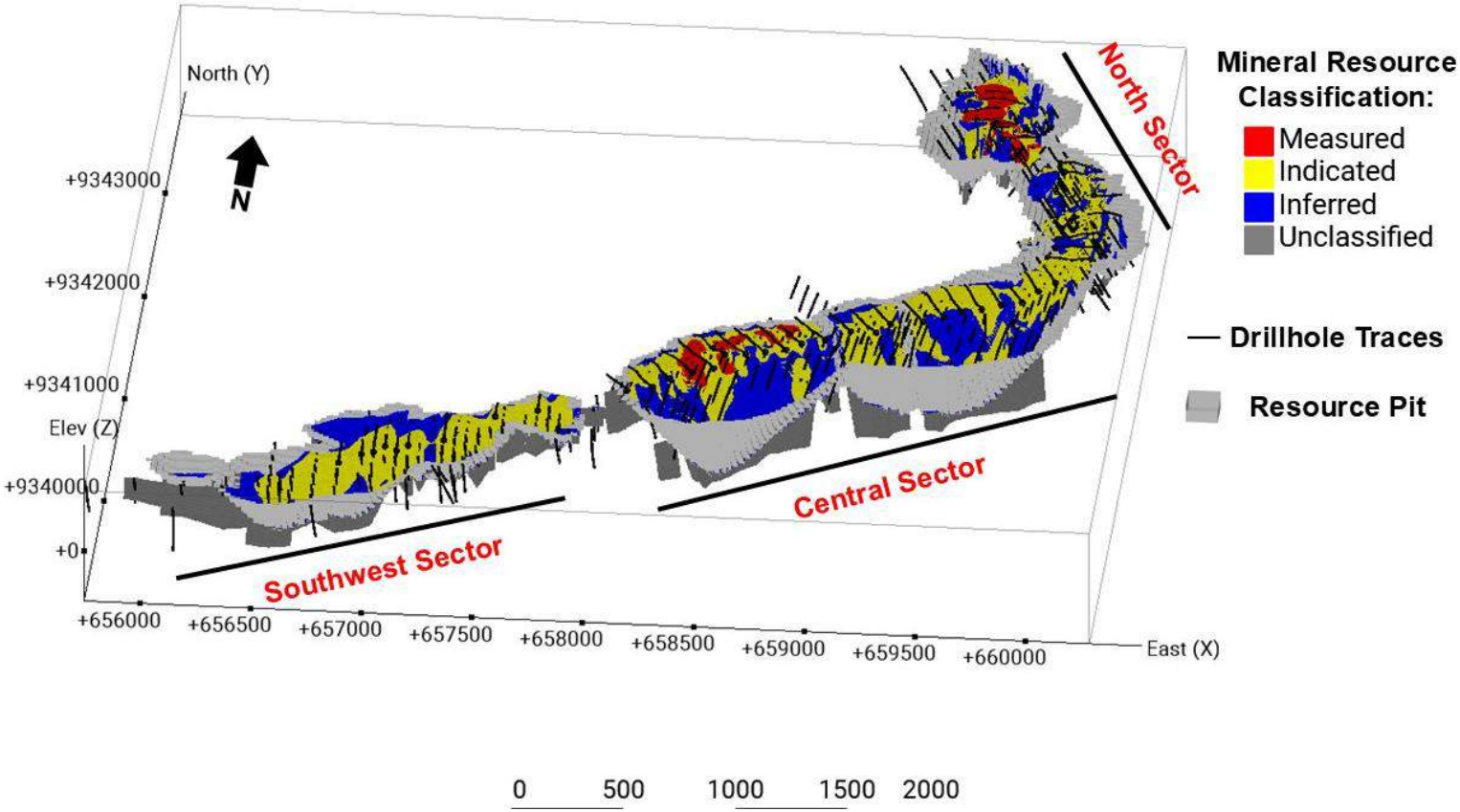


Figure 14-20: 2025 Mineral Resource estimate classification 3D view

Source: GE21, 2024.

14.7 QP Opinion

Based on the validation methods employed, the QP considers the estimate results obtained via conditional simulation acceptable and notes that no observable overall or local bias is present, 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 MRE. The QP further considers that the modifying factors used to estimate the mineral resource, as summarized in Table 14-16, are appropriate and support a determination reasonable prospects for eventual economic extraction, as required for a MRE under NI 43-101.

In the 2025 Technical Report, GE21 noted that if a further MRE upgrade is deemed desirable, GE21 recommended 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 2026 metallurgical test work programs.
- Mineral Resource estimation by the conditional simulation method, defining the SMU to determine recoverable Mineral Resource.

The first three recommendations have been, and are still being, advanced in 2025-2026 and, once completed, information generated from these work programs will be utilized in an updated MRE planned for 2027.

15 MINERAL RESERVE ESTIMATES

15.1 Introduction

Mineral Reserves represent the economically mineable portions of Measured and Indicated Mineral Resources. The Mineral Resource estimates for the PGM Luanga deposit were prepared by Mr. Porfirio Cabaleiro Rodriguez, B.Sc Min. Eng., FAIG, FAusIMM, a QP as defined under NI 43-101 regulations.

The conversion of Mineral Resources to Mineral Reserves requires downscaling to operational SMUs and applying realistic dilution and recovery assumptions to ensure that the Mineral Reserve estimates are technically consistent and aligned with mining industry best practices. The adopted SMU size was 5 m x 5 m x 5 m according to the operational and selectivity constraints. The E-type simulations estimated for 2025 Mineral Resources were based on the SMU volume, creating a Mineral Reserve block model. A simulation of possible dilutions is shown in Table 15-1, reporting Measured and Indicated Mineral Resources only.

Table 15-1: Mineral Reserve block model simulation using different block sizes

Model	Tonnage	PdEq	Metal content	Dilution
	(Mt)	(ppm)	(Moz PdEq)	(%)
Resource Model (2.5 x 2.5 x 2.5)	158.0	2.04	10.4	-
5x5x5	145.0	1.89	8.9	7.4%
10x10x5	141.0	1.84	8.3	9.8%
15x15x5	139.0	1.78	8.0	12.8%
20x20x5	132.8	1.74	7.4	14.8%

Source: GE21, 2026.

The conversion of 2025 Mineral Resources to the current Mineral Reserves estimate considered modifying factors such as metallurgical recoveries, mining dilution, ore loss, mining and processing costs, SG&A and royalty costs, logistics and refining costs, and projected prices for palladium, platinum, gold, rhodium and nickel.

The final pit design was guided by an optimization process using Dassault GEOVIA Whittle software. The design and mining plan presented in this Report are based on the Proven and Probable Mineral Reserves outlined in this section. The mine schedule for the deposit is detailed in Section 16.

15.2 Pit Optimization

To define the boundary and production sequence for the subsequent design, a pit optimization process was developed. This analysis uses geometric and economic criteria to determine the extent of the deposit that can be profitably mined and the corresponding cut-off grade.

15.2.1 Methodology

The pit optimization analysis was developed using the Dassault GEOVIA Whittle software. The optimizer uses the Lerchs-Grossman algorithm, modified for 3D application, to determine economic pit shells based on mining and processing cost inputs, revenue per block, and other parameters, as presented in Table 15-2. The costs were further detailed and developed throughout the PFS; therefore, those presented in Sections 21 and 22 may not fully correspond to the costs in this section.

Table 15-2: Pit optimization first pass parameters

Item			Unit	Value
Slope Angle	Weathered (oxidized)		°	33
	Fresh Rock - poor quality		°	48
	Fresh Rock - medium quality		°	51
	Fresh Rock - good quality		°	55
Mining	Block Size (minimum)		X-Y-Z (m)	5 x 5 x 5
	Block Size (SMU)		X-Y-Z (m)	5 x 5 x 5
	Density		g/cm³	Block Model
	Mining Recovery		%	95
	Mining Dilution		%	7.4
	Physical Constraints			ANM Mineral Rights Road (10 m buffer) Transmission Line (100 m buffer)
Processing	Metallurgic Recovery	Pd	Mill	77%
		Pt	Mill	81%
		Rh	Mill	51%
		Au	Mill	50%
		Ni	Mill	62%
Smelter Recoveries		Pd	%	99%
		Pt	%	99%
		Rh	%	95%
		Au	%	95%
		Ni	%	98%
Costs	Weathered	Mining Cost	US$/t mined	2.10
	Fresh	Mining Cost	US$/t mined	Waste=2.90 and ROM= 3.20
		Processing Cost	US$/t ROM	7.64
		Grade Control		1.00
		Rehabilitation		1.00
		G&A		5.00
		Thickening and filtration		4.00
		Smelting and Refining	US$/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	%	5.0

Source: GE21, 2026.

15.2.2 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 total revenue from product sales and total 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 ($ × tonne^{-1});
- MC = the cost of mining a tonne of waste ($ × tonne^{-1});
- PC = cost per tonne of mining the material as above cut-off mineralized material and processing it ($ × 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).

15.2.3 Optimization Results

A series 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 15-3 and Figure 15-1) was selected to guide the final pit design, as the best trade-off between added value and incremental stripping. Figure 15-2 shows the selected pit shell boundaries and Mineral Resources distribution.

Table 15-3: Pit optimization results

Revenue Factor	ROM	Waste	Total Mined	Strip Ratio	Pd	Pt	Rh	Au	Ni	NPV (1)
	(Mt)			(t/t)	(ppm)					(B USD)
30%	8.4	33.8	42.2	4.0	1.56	1.33	0.23	0.16	3,969	4.35
40%	20.9	109.1	130.0	5.2	1.41	1.12	0.19	0.14	3,166	3.72
50%	43.0	195.7	238.6	4.6	1.24	0.91	0.15	0.10	2,293	3.02
60%	57.5	272.5	330.0	4.7	1.16	0.84	0.13	0.09	2,099	2.80
70%	68.3	327.2	395.5	4.8	1.09	0.79	0.13	0.09	2,007	2.63
80%	76.9	365.2	442.2	4.8	1.03	0.75	0.12	0.08	1,918	2.49
90%	85.6	390.6	476.2	4.6	0.97	0.71	0.11	0.08	1,825	2.36
100%	96.7	428.2	525.0	4.5	0.94	0.69	0.11	0.07	1,776	2.24
110%	100.0	451.4	551.4	4.5	0.89	0.66	0.10	0.07	1,693	2.17
120%	105.8	471.4	577.2	4.5	0.86	0.64	0.10	0.07	1,635	2.10
130%	110.6	489.4	600.0	4.4	0.84	0.62	0.09	0.07	1,592	2.04
140%	116.5	514.1	630.7	4.4	0.81	0.61	0.09	0.06	1,541	1.98
150%	119.1	531.7	650.8	4.5	0.80	0.60	0.09	0.06	1,524	1.96
160%	121.3	568.1	689.4	4.7	0.80	0.60	0.09	0.06	1,510	1.95
170%	122.6	582.8	705.5	4.8	0.80	0.59	0.09	0.06	1,502	1.94
180%	123.1	588.7	711.8	4.8	0.80	0.59	0.09	0.06	1,499	1.93
190%	124.0	598.1	722.1	4.8	0.79	0.59	0.09	0.06	1,493	1.92
200%	124.4	604.6	729.0	4.9	0.79	0.59	0.09	0.06	1,490	1.92

Note: NPV figures on a comparative basis. No economic assessment was developed.
Source: GE21, 2026.

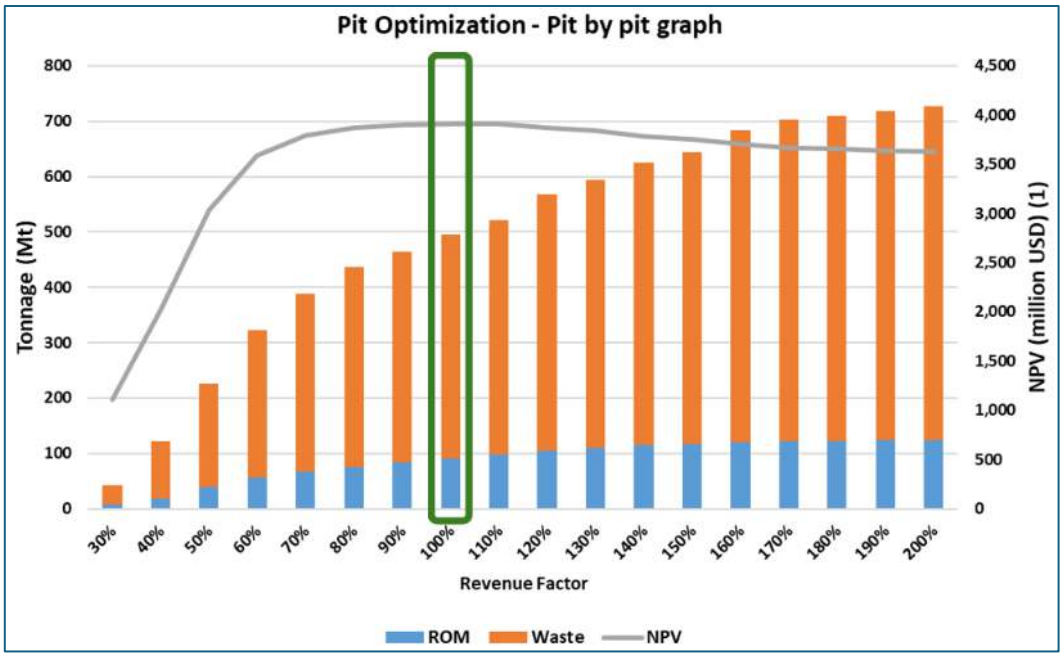


Figure 15-1: Pit-by-pit chart

Note: NPV figures on a comparative basis. No economic assessment was developed.
Source: GE21, 2026.

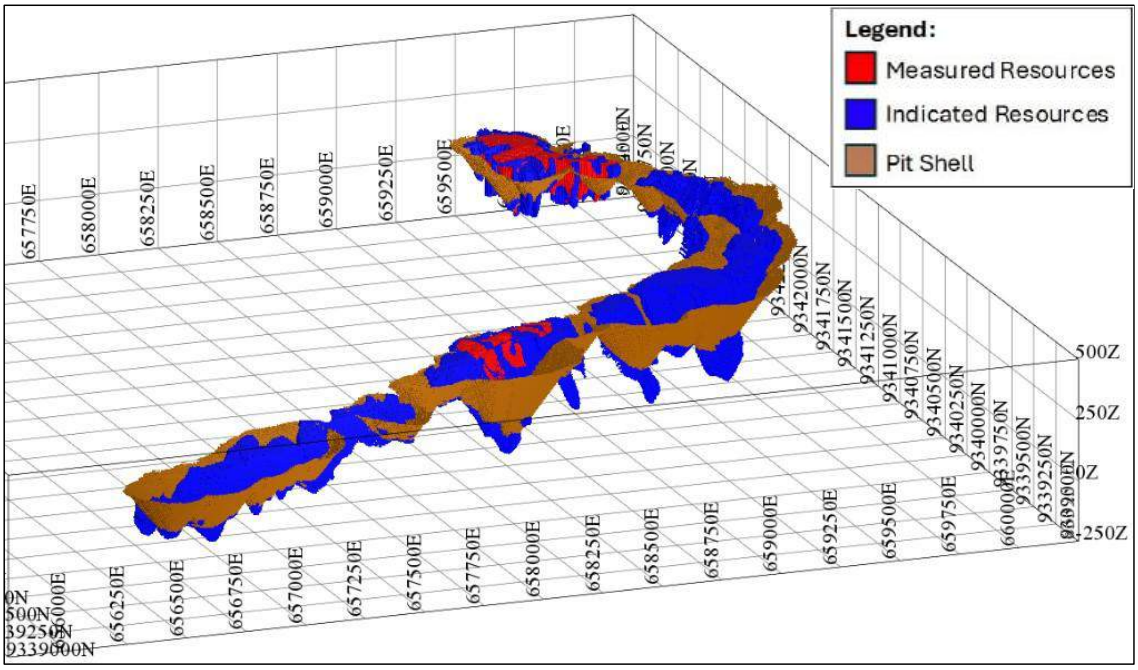


Figure 15-2: Selected pit shell boundaries

Source: GE21, 2026.

A Palladium Equivalent (PdEq) cut-off grade was estimated based on the First Pass Parameters presented in Table 15-2. The PdEq equation used is:

$$\text{PdEq} = \text{Pd (g/t)} + \text{F1} + \text{F2} + \text{F3} + \text{F4}$$

Where:

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

And:

$_P$ = Metal Price
$_R$ = Metallurgical Recovery

The estimated cut-off grade for the mine planning development is 0.87 g/t PdEq.

The selected pit shell resulted in 96.7 Mt of Run-of-Mine (ROM) material at 2.78 g/t PdEq at the estimated cut-off. The material below the cut-off was designated as Marginal in this PFS report because its economic potential depends on market conditions.

Table 15-4 presents the material within the pit shell, applying the cut-off of 0.87 g/t PdEq. These are in situ figures: mining dilution and recovery factors are not applied.

Table 15-4: Pit shell results using a cut-off of 0.87 g/t PdEq

Category	Mass	Pd	Pt	Rh	Au	Ni	PdEq
	(Mt)	(g/t)				(ppm)	(g/t)
Measured Resources	19.3	0.95	0.82	0.12	0.08	1,736	2.98
Indicated Resources	77.5	0.94	0.66	0.10	0.07	1,786	2.73
Measured + Indicated Resources	**96.7**	**0.94**	**0.69**	**0.11**	**0.07**	**1,776**	**2.78**
Inferred Resources	13.5	0.69	0.50	0.06	0.05	1,040	-
Low Grade	8.6						
Oxide	18.6						
Waste	387.5						
Strip Ratio	4.4						
Total Moved	525.0						

Note: * Cut-off grade of 0.87 g/t Pd_Eq applied
Source: GE21, 2026.

15.3 Final Pit Design

The next step in mining plan development is to design an operational pit design 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, as well as ramps to access the pit bottoms, resulting in a representation of the surface where the selected equipment would be able to operate efficiently and safely.

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.

The QP relies on the geometric and geotechnical parameters which were defined on the geotechnical model and structural analysis prepared by geologist Luis Navarro in the September 2025, report “Projeto Luanga – Bravo Mining – Modelo Geotécnico RMR”. The rock mass was characterized using the Rock Mass Rating (RMR) classification system, based on a database of 419 diamond drill holes, 84% of which include geomechanical logging, with near-surface soil/saprolite and transition-zone horizons modelled as separate weathering domains.

The model indicates a predominantly competent rock mass, with the pit walls expected to be excavated mainly in good- to fair-quality rock (RMR Classes II and III) and only localized occurrences of poor-quality material. The structural analysis identified a dominant steeply dipping NE-SW structural fabric, consistent with the orientation of the mineralized bodies, along with subordinate conjugate and low-angle sets.

The results of this study were used as the basis for the slope angles adopted in the open-pit design, which are considered appropriate for a PFS level of assessment. Additional geotechnical investigations and stability analyses are recommended to refine the slope design criteria in the next phase of study.

Table 15-5 presents the parameters used for the pit designs.

Table 15-5: Pit design geometric and geotechnical parameters

Parameter	Unit	Weathered (oxidized)	Fresh Rock		
			Poor quality	Medium quality	Good quality
Overall Slope	°	33	48	51	55
Bench Slope	°	46	72	77	83
Bench Height	m	10			
Berm Width	m	6			
Ramp Width	m	25			
Maximum Ramp Gradient	%	10			
Minimum Bottom Width	m	45			

Source: adapted from Navarro, 2025 (GE21, 2026).

Figure 15-3 presents the final pit design.

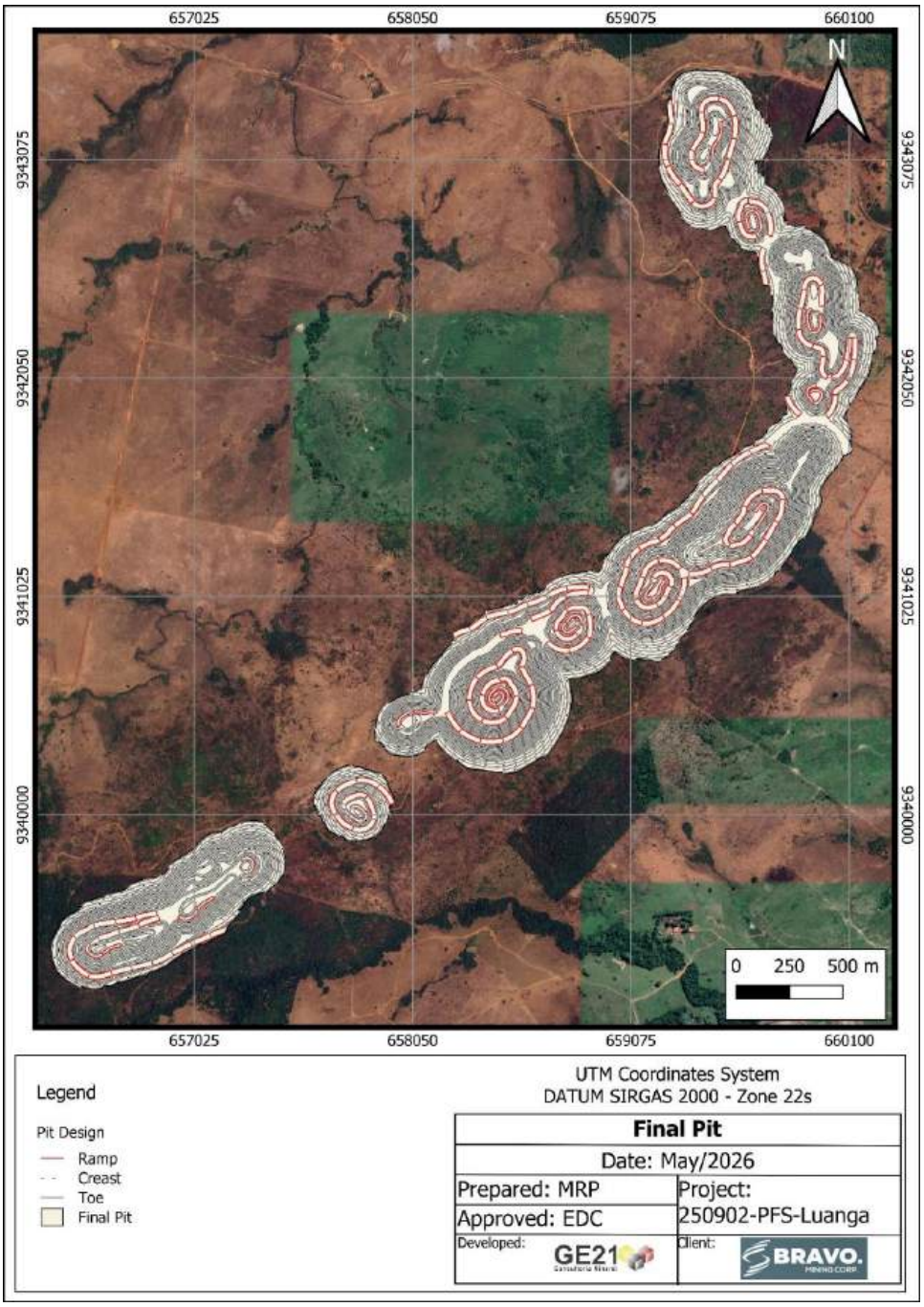


Figure 15-3: Final pit design

Source: GE21, 2026.

15.4 Mineral Reserve Statement

The Mineral Reserve Statement is presented in Table 15-6.

Table 15-6: Luanga Project – Mineral Reserve estimates

Category	Mass	Pd	Pt	Rh	Au	Ni	Pd Eq	Pd	Pt	Rh	Au	Ni	Pd Eq
	(Mt)	(ppm)						(koz)				(t)	(koz)
Proven Reserves	17.7	0.95	0.82	0.12	0.08	1,765	2.99	541	466	70	44	31,239	1,703
Probable Reserves	69.0	0.93	0.65	0.10	0.07	1,786	2.72	2,062	1,442	226	161	123,237	6,028
Total Reserves	**86.7**	**0.93**	**0.68**	**0.11**	**0.07**	**1,781**	**2.77**	**2,603**	**1,908**	**295**	**205**	**154,476**	**7,731**

Notes:
1. Mineral Reserves are reported in accordance with the CIM Standards and Best Practices Guidelines, as required by National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).
2. Mineral Reserves are the economic portion of Measured and Indicated Mineral Resources. No Inferred Resource was counted as Mineral Reserve.
3. Reserves are accounted at plant ROM pad.
4. The Mineral Reserve Estimates have been prepared by Mr. Porfirio Cabaleiro Rodriguez, B.Sc Min. Eng., FAIG, FAusIMM, director of GE21, an independent QP under NI 43-101.
5. The Effective Date of the Mineral Reserve Estimates is September 16, 2026.
6. Mineral Reserves block model using 5 m x 5 m x 5 m SMU.
7. Mining recovery factor of 95%. Mining dilution of 7.4% applied in the block model.
8. Defined cut-off grade of 0.87 g/t PdEq.
9. Pd_Eq formula: PdEq = Pd + Pt*1.11733 + Rh*2.60558 + Au*1.58174 + Ni*0.000394
10. The Mineral Reserve Estimates are reported within a designed pit, developed based on a selected pit shell generated by GEOVIA Whittle® software, using the following assumptions:
    - Metallurgical recovery of 77% Pd, 81% Pt, 51% Rh, 50% Au, 62% Ni to a saleable Ni-PGM concentrate.
    - Independent Geotechnical Testwork – Overall pit slopes of 33° in oxide, 48° in poor quality fresh rock, 51° in medium quality fresh rock, and 55° in good quality fresh rock.
    - The Mineral Reserves are reported on a dry density basis.
    - Smelting payability of 99% for Pd and Pt, 95% for Rh and Au, and 98% for Ni.
    - Payable royalties of 2%.
11. Costs:
    a. Mining costs: US$2.10/t oxide, US$2.90/t waste rock, and US$3.20/t ROM.
    b. Processing costs: US$7.64/t.
    c. Other costs: US$5.00/t processed for General & Administration, US$1.00/t processed for grade control, US$1.00/t processed for rehabilitation, US$4.00/t processed for thickening and filtration.
    d. Smelting and refining: US$251.00/t concentrate for smelting and refining, and US$80.00/t concentrate for smelter G&A.
12. Metal selling prices, based on Consensus Long-Term nominal prices: US$1,271.00/toz Pd, US$1,350.00/toz Pt, US$5,000.00/toz Rh, US$3,096.00/toz Au, and US$9.07/lb Ni.
13. The responsible QP is not aware of political, environmental, legal, title, taxation, socioeconomic, marketing, or other relevant risks that would materially affect the estimation of Mineral Reserves other than those discussed in this Technical Report.
14. Totals may not sum due to rounding.

Source: GE21, 2026.

The costs were taken from comparable projects in GE21's extensive database of mining operations in Brazil, which includes not only operating mines, but also recent actual costs from what could potentially be similarly sized operating mines in the Carajás region. Costs are considered at a throughput rate of 10 Mtpa.

16 MINING METHODS

This section presents the mining plan used in this PFS-level technical report in accordance with CIM NI 43-101. Open-pit mining was selected for mine production, with an approach that maximizes production while optimizing Project economics.

16.1 Introduction

Development of the Life-of-Mine (LOM) plan includes pit optimization, pit design, mine scheduling, and the application of economic and metallurgical modifying factors to Measured and Indicated Mineral Resources.

Pit optimization was performed using GEOVIA Whittle software, which established the pit boundary for the mining plan, applying reasonable modifying factors to the Mineral Resources estimate. A final pit design was developed based on operational and technical parameters, resulting in an estimated mine life of 10 years.

The Luanga deposit was projected to be mined by conventional open-pit mining methods for 9.6 years, with a plant feed rate of 6 Mt in the first year and 10 Mtpa from the second year until the end of LOM.

Vegetation and topsoil would be removed and stockpiled for use in future site reclamation. The ROM and waste would be mined with 10-metre-high benches, focusing on reducing mining dilution and increasing ROM recovery.

Drilling and blasting operations are expected for production purposes and are applicable exclusively to fresh rock mining, as oxidized material is expected to be “free-digging”. A staggered blast pattern arranged in an equilateral triangular configuration was selected for the blast design, as it provides optimal spatial distribution of explosive energy throughout the blast volume, promoting efficient and uniform rock fragmentation. Blasting parameters were defined and calibrated in accordance with this geometric configuration and the geomechanical characteristics of the fresh rock domain.

The ROM would be loaded into trucks and hauled to the processing plant. A temporary ROM stockpile was designed close to the plant to stock low-grade material along the LOM, which would be reclaimed by the end of the production period.

Tailings generated in the processing plant would be disposed of in a Tailings Storage Facility.

The mineralized oxidized material was not considered in the LOM in this PFS report. This material would be stockpiled in a different area for future use.

The waste, oxidized, and marginal materials would be loaded into trucks and deposited in separate dumps.

Drainage channels would be constructed along the mining area perimeters to collect rainwater and sediments, avoiding impacts on local terrain, vegetation and drainage. The channels would direct water to sumps located on the bottom of the pit, where it would be pumped for reuse.

16.2 Final Pit Design

The operational final pit was developed 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 would be able to operate efficiently and safely.

16.2.1 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 to define access and ramps to optimize average haulage distances.

16.2.2 Geometric and Geotechnical Parameters

The geometric and geotechnical parameters were defined based on the geotechnical model and structural analysis, as detailed in Section 15. Table 16-1 presents the parameters used for pit designs.

Table 16-1: Pit design geometric and geotechnical parameters

Parameter	Unit	Weathered (oxidized)	Fresh rock		
			Poor quality	Medium quality	Good quality
Overall slope	°	33	48	51	55
Bench slope	°	46	72	77	83
Bench height	m	10			
Berm width	m	6			
Ramp width	m	25			
Maximum ramp gradient	%	10			
Minimum bottom width	m	45			

Source: Adapted from Luis Navarro, 2025 (GE21, 2026).

16.2.3 Final Pit Design Results

Figure 16-1 presents the final pit design, and Table 16-2 lists the total material to be mined. An operational mining recovery of 95% was applied, based on the characteristics of the pit and material and on projects with similar operational features. A mining dilution factor was not applied, as dilution is already accounted for within the block model.

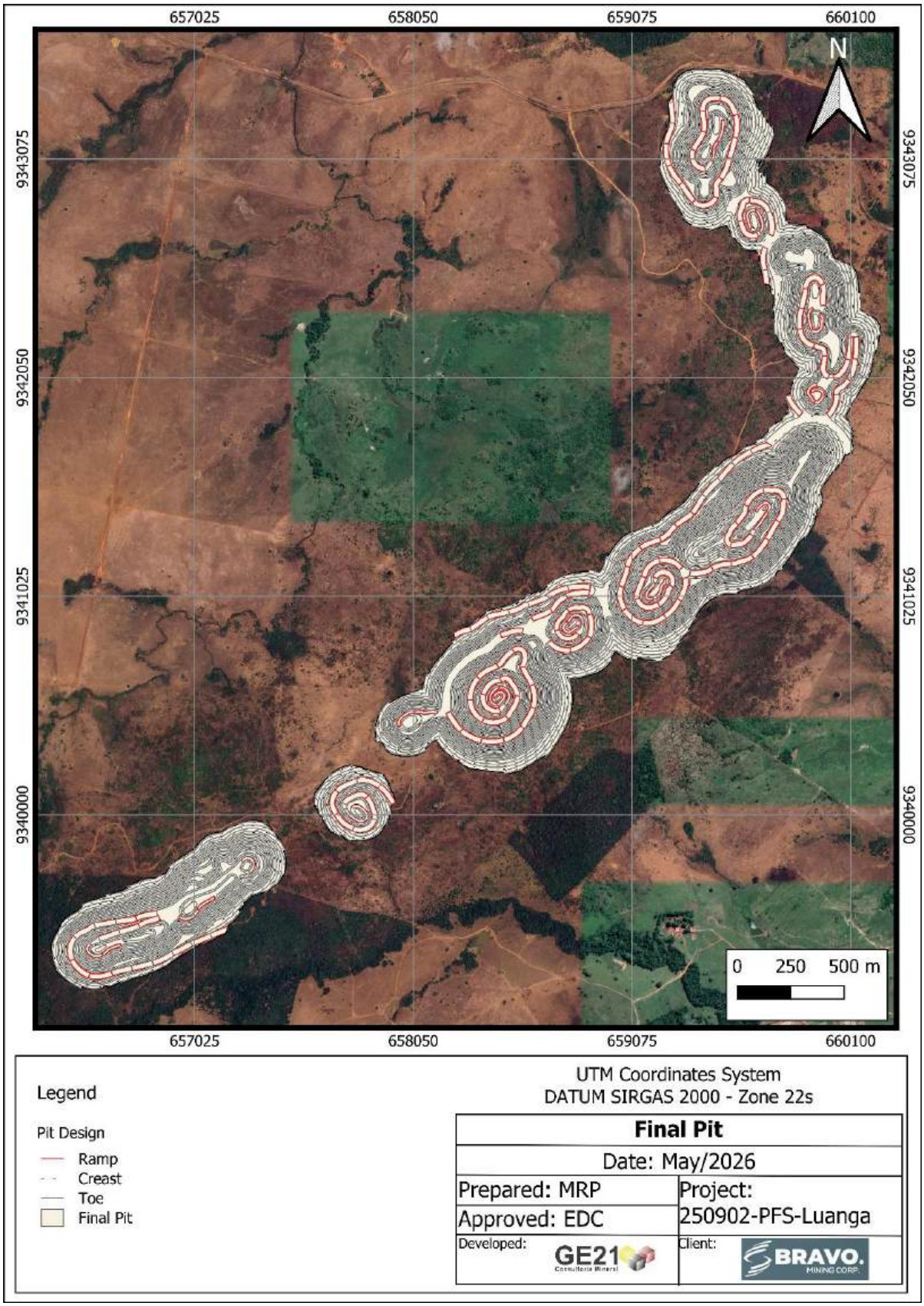


Figure 16-1: Final pit design

Source: GE21, 2026.

Table 16-2: Final pit results

Category	Mass	Pd	Pt	Rh	Au	Ni	Pd Eq
	(Mt)	(ppm)					
Proven Reserves	17.7	0.95	0.82	0.12	0.08	1,765	2.99
Probable Reserves	69.0	0.93	0.65	0.10	0.07	1,786	2.72
Total Reserves	86.7	0.93	0.68	0.11	0.07	1,781	2.77
Inferred Resources	17.4	0.67	0.49	0.06	0.05	1,033	-
Low grade	12.3	-	-	-	-	-	-
Oxide	18.9	-	-	-	-	-	-
Waste	527.4	-	-	-	-	-	-
Strip ratio	6.6	-	-	-	-	-	-
Total moved	662.6	-	-	-	-	-	-

Notes:
1. Estimated cut-off grade: 0.87 g/t Pd_Eq.
2. Mining dilution factor: 7% applied in the block model.
3. Mining recovery factor: 95%.
Source: GE21, 2026.

16.3 Mine Scheduling

Mine scheduling development focused on operational features of the mining operations and on concentrate production optimization. The production rate was defined as 10 Mtpa of ROM delivered to the processing plant.

The mining plan includes a one-year ramp-up period at 60% of 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 PdEq grade and lower strip ratio zones were prioritized at the beginning of LOM to optimize the Project's economics during the payback period.

The mining sequence was developed using the Dassault GEOVIA Whittle software pushback optimizer, and the corresponding end-of-period designs were done for the following periods:

- Year 1: Quarterly basis.
- Years 2 to 5: Semi-annual basis.
- Years 6 to 10: Annual basis.

Figure 16-2 to Figure 16-4 present the annual mine scheduling results.

Table 16-3, Table 16-4, and Figure 16-5 to Figure 16-8 present the detailed results. The end-of-period design corresponds to the final pit design (Figure 16-1).

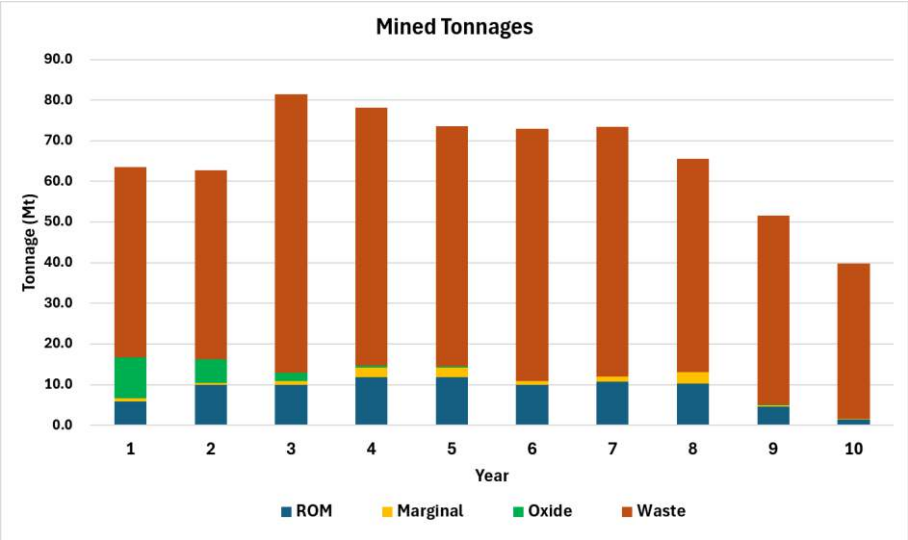


Figure 16-2: Mine scheduling results – mined tonnages graph

Source: GE21, 2026.

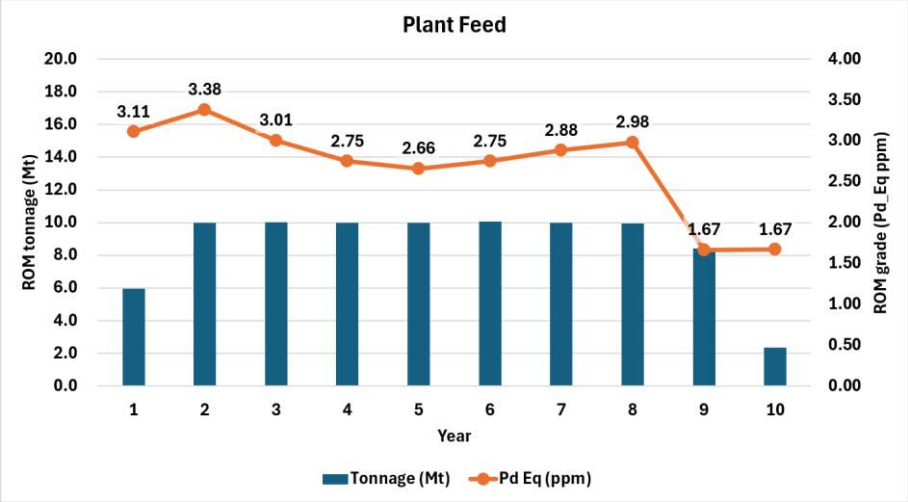


Figure 16-3: Mine scheduling results – plant feed graph

Source: GE21, 2026.

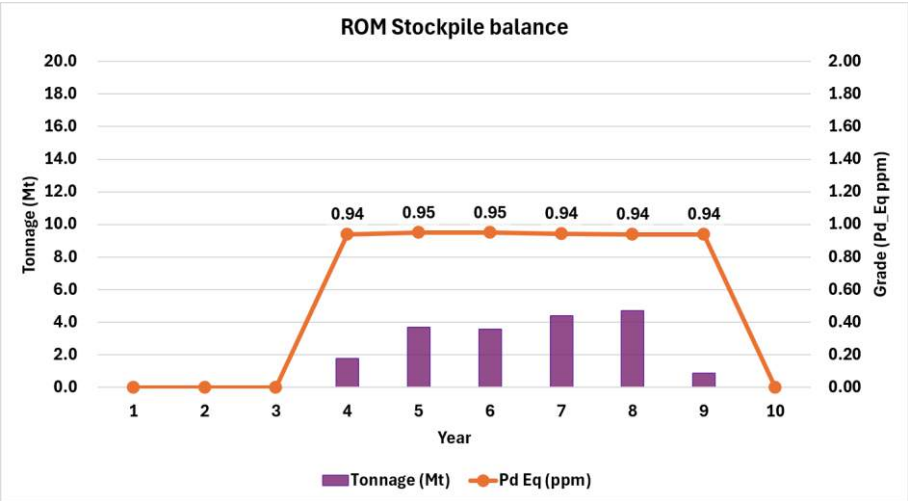


Figure 16-4: Mine scheduling results – ROM stockpile balance graph

Source: GE21, 2026.

Table 16-3: Mine scheduling – mining production

Year	Period	ROM							Marginal	Oxide	Waste	Total waste	Strip ratio	Total mined
		Tonnage	PdEq	Pd	Pt	Rh	Au	Ni						
		(Mt)	(g/t)						(Mt)				(t/t)	(Mt)
1	Q1	1.4	3.11	0.97	0.72	0.11	0.07	2,395	0.2	4.9	9.1	14.2	9.8	15.7
	Q2	1.4	3.13	0.97	0.77	0.12	0.07	2,226	0.1	0.7	13.9	14.6	10.4	16.0
	Q3	1.4	3.11	0.94	0.74	0.12	0.09	2,281	0.3	3.5	10.9	14.7	10.7	16.0
	Q4	1.7	3.10	0.97	0.86	0.10	0.08	2,008	0.1	1.1	13.1	14.2	8.2	15.9
	Subtotal	5.9	3.11	0.96	0.77	0.11	0.08	2,217	0.7	10.1	46.7	57.5	9.7	63.5
2	Sem1	5.0	3.41	1.06	0.83	0.14	0.11	2,335	0.3	3.9	21.9	26.1	5.2	31.1
	Sem2	5.0	3.35	1.15	0.77	0.12	0.11	2,220	0.2	1.9	24.4	26.5	5.3	31.5
	Subtotal	10.0	3.38	1.10	0.80	0.13	0.11	2,277	0.5	5.8	46.4	52.8	5.3	62.8
3	Sem1	5.0	3.09	1.05	0.71	0.12	0.10	2,027	0.4	0.7	34.5	35.7	7.1	40.7
	Sem2	5.0	2.91	1.04	0.73	0.11	0.07	1,702	0.4	1.3	34.0	35.7	7.1	40.7
	Subtotal	10.0	3.01	1.04	0.72	0.12	0.08	1,865	0.9	2.1	68.4	71.4	7.1	81.4
4	Sem1	5.8	2.50	0.83	0.66	0.10	0.05	1,553	1.3	0.2	32.1	33.6	5.8	39.5
	Sem2	5.9	2.45	0.88	0.70	0.09	0.06	1,185	1.0	0.2	31.5	32.7	5.5	38.7
	Subtotal	11.8	2.47	0.86	0.68	0.10	0.06	1,368	2.4	0.5	63.4	66.3	5.6	78.1
5	Sem1	6.1	2.31	0.87	0.61	0.08	0.05	1,227	1.8	0.3	29.1	31.3	5.1	37.3
	Sem2	5.8	2.47	0.86	0.63	0.10	0.06	1,458	0.5	0.0	29.8	30.3	5.2	36.1
	Subtotal	11.9	2.39	0.87	0.62	0.09	0.05	1,340	2.3	0.3	59.1	61.7	5.2	73.6
6	-	9.9	2.77	0.93	0.67	0.11	0.07	1,824	0.9	0.1	62.1	63.0	6.3	73.0
7	-	10.8	2.74	0.82	0.69	0.11	0.08	1,897	1.2	0.0	61.4	62.6	5.8	73.4
8	-	10.3	2.91	1.07	0.66	0.10	0.09	1,822	2.8	0.0	52.5	55.3	5.4	65.6
9	-	4.6	2.28	0.70	0.54	0.08	0.06	1,756	0.4	0.1	46.5	47.0	10.3	51.5
10	-	1.5	2.08	0.68	0.53	0.08	0.05	1,624	0.1	0.0	38.1	38.3	25.3	39.8
Total		86.7	2.77	0.93	0.68	0.11	0.07	1,781	12.3	18.9	544.7	575.9	6.6	662.6

Notes:
1. Estimated cut-off grade: 0.87 g/t Pd_Eq.
2. Mining dilution factor: 7% applied in the block model.
3. Mining recovery factor: 95%.
Source: GE21, 2026.

Table 16-4: Mine scheduling – ROM production

Year	Period	ROM – mine		ROM – stockpile						Plant feed		
		Tonnage	Pd Eq	added		removed		balance		Tonnage	PdEq	
		(Mt)	(g/t)	(Mt)	(g/t)	(Mt)	(g/t)	(Mt)	(g/t)	(Mt)	(g/t)	(koz)
1	Q1	1.4	3.11	-	-	-	-	-	-	1.4	3.11	145
	Q2	1.4	3.13	-	-	-	-	-	-	1.4	3.13	142
	Q3	1.4	3.11	-	-	-	-	-	-	1.4	3.11	137
	Q4	1.7	3.10	-	-	-	-	-	-	1.7	3.10	173
	Subtotal	5.9	3.11	0.0	0.00	0.0	0.00	0.0	0.00	5.9	3.11	595
2	Sem1	5.0	3.41	-	-	-	-	-	-	5.0	3.41	548
	Sem2	5.0	3.35	-	-	-	-	-	-	5.0	3.35	539
	Subtotal	10.0	3.38	0.0	0.00	0.0	0.00	0.0	0.00	10.0	3.38	1,086
3	Sem1	5.0	3.09	-	-	-	-	-	-	5.0	3.09	498
	Sem2	5.0	2.91	-	-	-	-	-	-	5.0	2.91	469
	Subtotal	10.0	3.01	0.0	0.00	0.0	0.00	0.0	0.00	10.0	3.01	969
4	Sem1	5.8	2.50	0.7	0.95	-	-	0.7	0.95	5.1	2.73	448
	Sem2	5.9	2.45	1.0	0.93	-	-	1.8	0.94	4.9	2.77	436
	Subtotal	11.8	2.47	1.8	0.94	0.0	0.00	1.8	0.94	10.0	2.75	883
5	Sem1	6.1	2.31	1.0	0.94	-	-	2.8	0.94	5.1	2.57	421
	Sem2	5.8	2.47	0.9	0.97	-	-	3.7	0.95	4.9	2.75	432
	Subtotal	11.9	2.39	1.9	0.96	0.0	0.00	3.7	0.95	10.0	2.66	853
6	-	9.9	2.77	-	-	-0.1	0.95	3.6	0.95	10.0	2.75	889
7	-	10.8	2.74	0.8	0.92	-	-	4.4	0.94	10.0	2.88	924
8	-	10.3	2.91	0.4	0.88	-0.1	0.94	4.7	0.94	10.0	2.98	954
9	-	4.6	2.28	-	-	-3.9	0.94	0.8	0.94	8.5	1.66	452

Year	Period	ROM – mine		ROM – stockpile						Plant feed		
		Tonnage	Pd Eq	added		removed		balance		Tonnage	PdEq	
		(Mt)	(g/t)	(Mt)	(g/t)	(Mt)	(g/t)	(Mt)	(g/t)	(Mt)	(g/t)	(koz)
10	-	1.5	2.08	-	-	-0.8	0.94	-	-	2.3	1.69	126
Total		86.7	2.77	4.9	0.94	-4.9	0.94	-	-	86.7	2.77	7,731

Notes:
1. Estimated cut-off grade: 0.87 g/t PdEq.
2. Mining dilution factor of 7% applied in the block model.
3. Mining recovery factor: 95%.
Source: GE21, 2026.

Table 16-5 presents the plant feed grouped by Mineral Reserves classification, and the contribution of each class within the plant feed.

Table 16-5: Mine scheduling – plant feed by Mineral Reserves classification

Year	Proven			Probable		
	Tonnage	PdEq	% of plant feed	Tonnage	PdEq	% of plant feed
	(Mt)	(g/t)		(Mt)	(g/t)	
1	2.6	3.11	44%	3.3	3.11	56%
2	3.4	4.18	34%	6.5	2.96	66%
3	2.5	4.11	25%	7.5	2.64	75%
4	2.9	2.66	29%	7.1	2.79	71%
5	1.9	2.51	20%	8.0	2.70	80%
6	1.0	2.69	10%	9.0	2.78	90%
7	0.6	2.41	6%	9.4	2.91	94%
8	0.9	2.06	9%	9.0	3.09	91%
9	1.9	2.51	22%	6.6	1.87	78%
10	0.01	1.87	0%	2.6	1.23	100%
Total	17.7	3.16	20%	69.0	2.70	80%

Source: GE21, 2026.

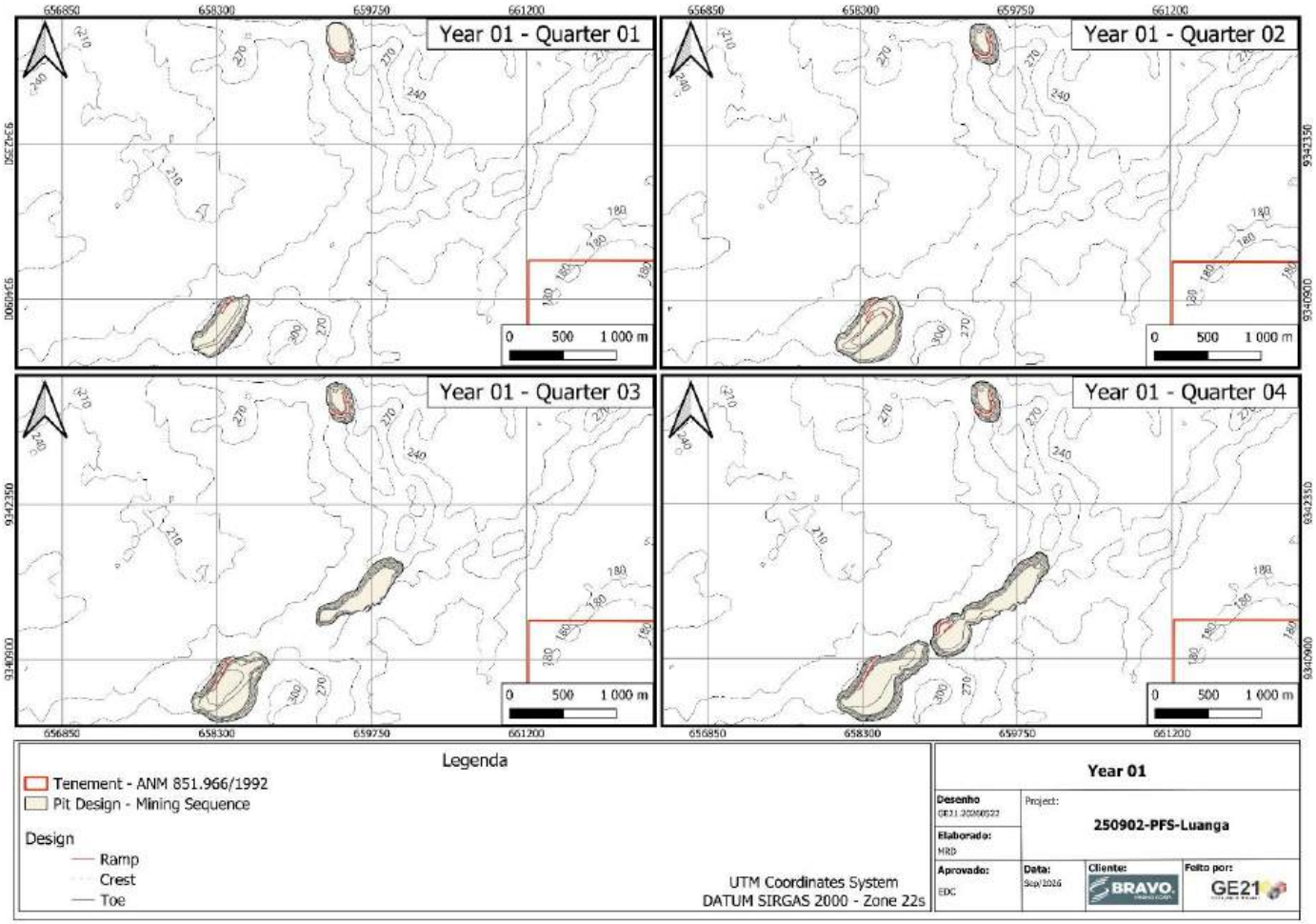


Figure 16-5: Mining schedule design – year 1

Source: GE21, 2026.

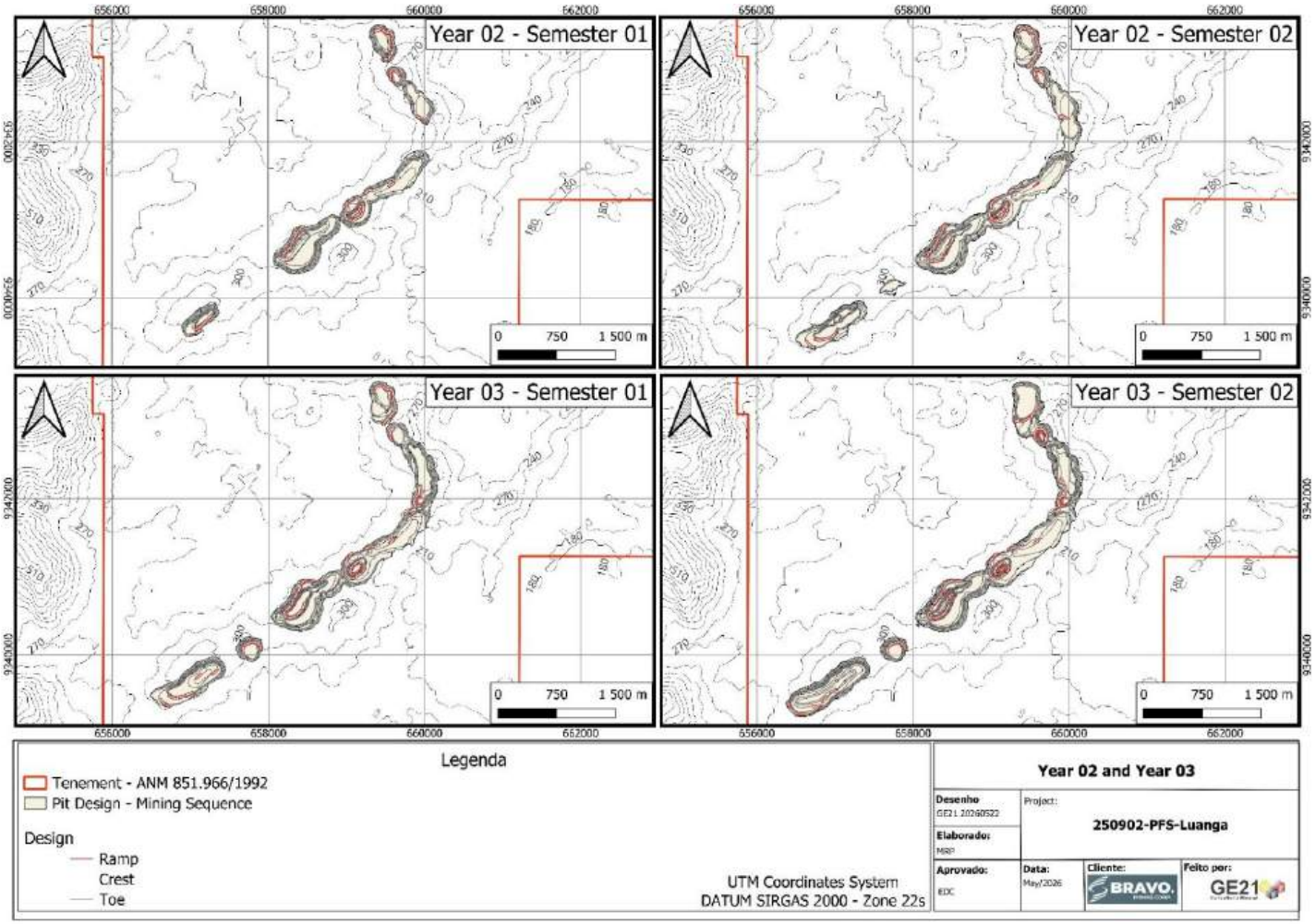


Figure 16-6: Mining schedule design – years 2 and 3

Source: GE21, 2026.

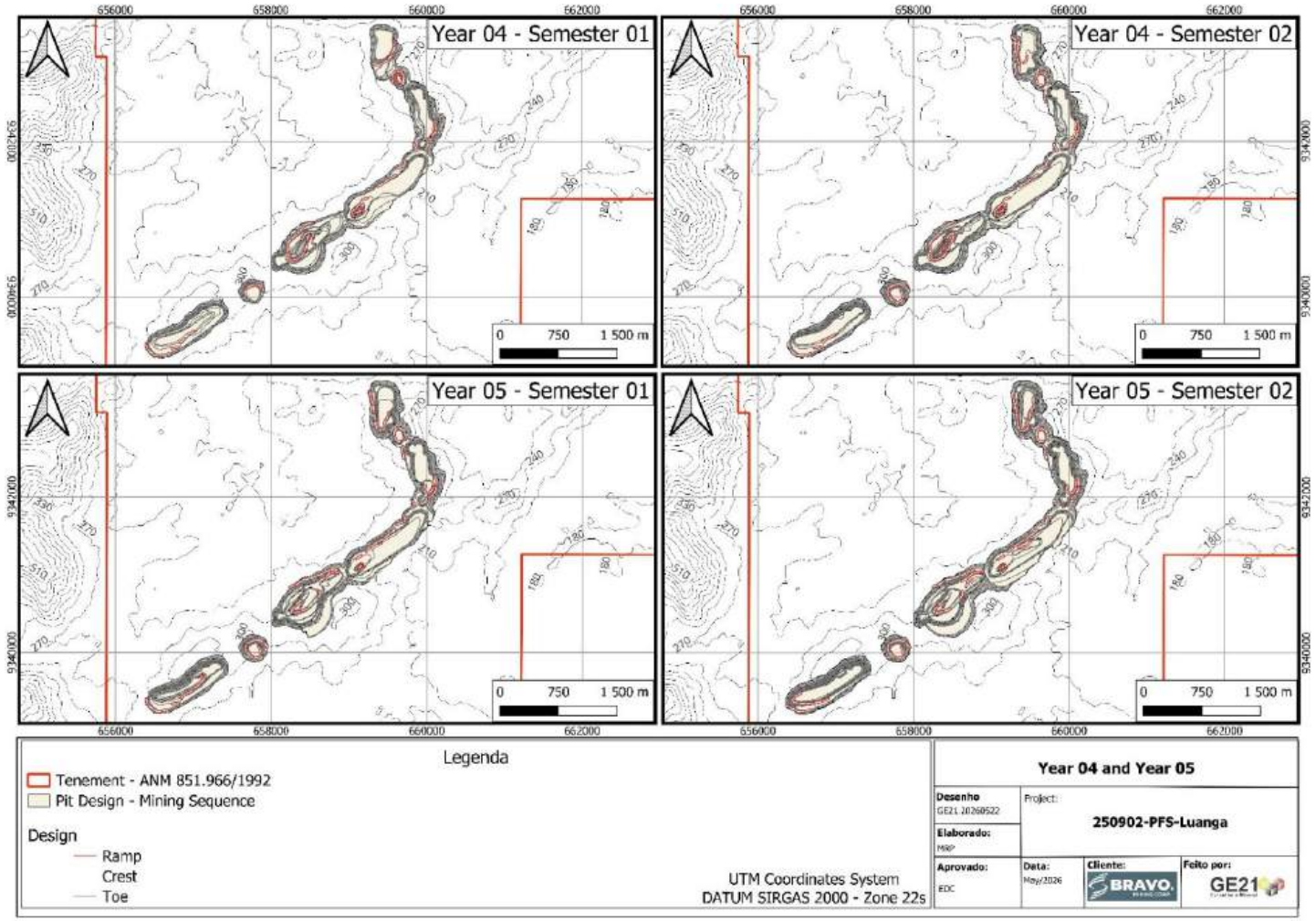


Figure 16-7: Mining schedule design – years 4 and 5

Source: GE21, 2026.

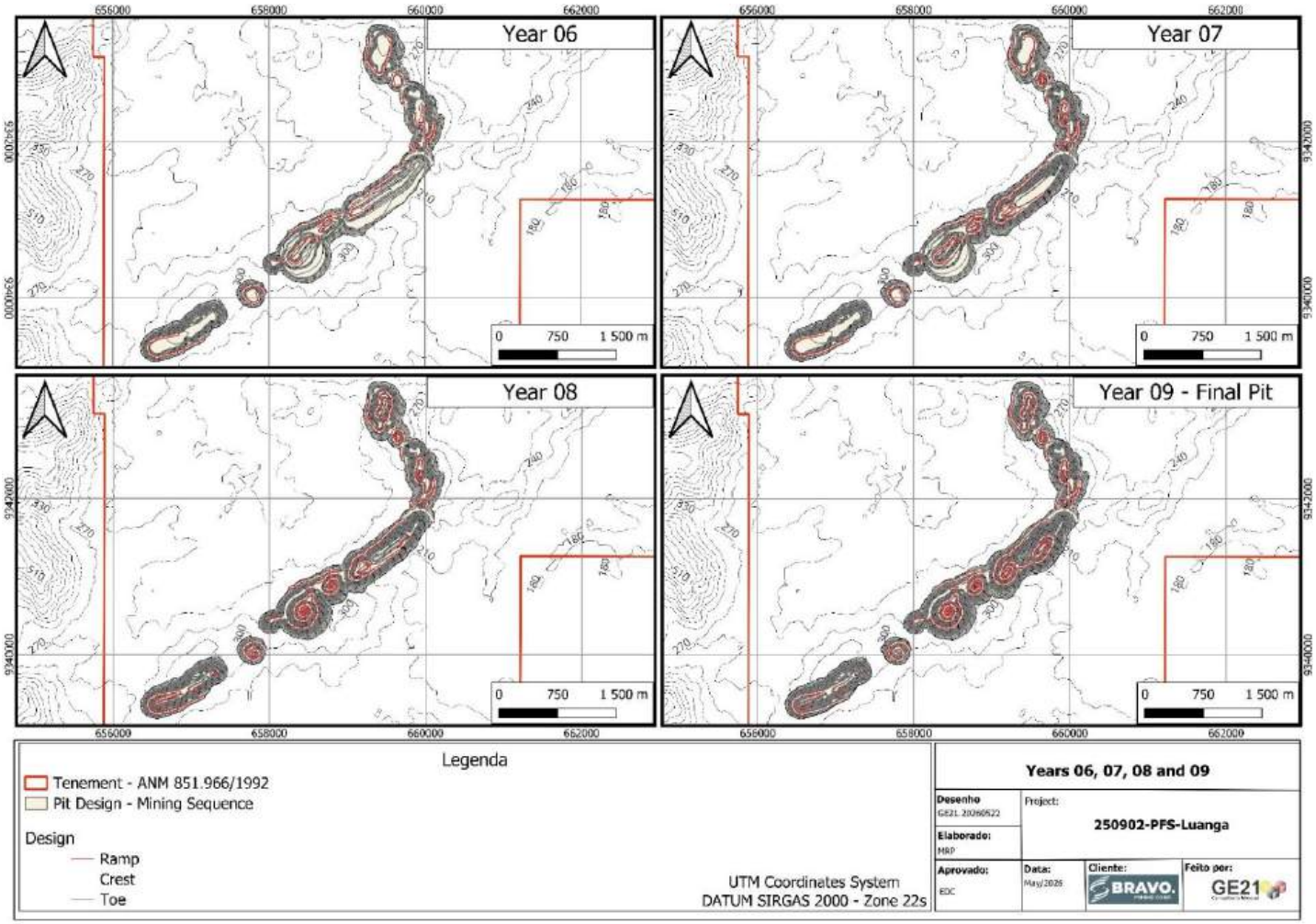


Figure 16-8: Mining schedule design – years 6, 7, 8 and 9

Source: GE21, 2026.

16.4 Waste Dump, Marginal, Oxide and ROM Stockpiles

The waste rock would be located in a dedicated waste dump, while marginal and oxide materials would be stored in separate stockpiles for potential future processing. A ROM stockpile is also considered to accumulate ore that is not directly discharged into the primary crusher feed hopper, in accordance with the mining schedule requirements.

The disposed volumes are estimated considering the factors presented in Table 16-6.

Table 16-6: Volume estimate factors

Material	Swelling factor	Compaction factor	Contingency
Waste	20%	5%	5%
Marginal	20%	5%	5%
Oxide	20%	5%	5%
ROM stockpile	20%	0%	5%

Source: GE21, 2026.

The conceptual designs were developed using assumed parameters based on the characteristics of the materials to be disposed of and on locational features (Table 16-7). These parameters must be confirmed after comprehensive geotechnical studies.

Table 16-7: Waste dump and stockpiles design parameters

Parameter	Unit	Value
Overall slope	°	21
Bench slope	°	29
Bench height	m	10
Minimum berm width	m	7.5
Ramp width	m	25
Ramp gradient	%	10

Source: GE21, 2026.

The waste dump and stockpile sequencing was developed according to the required volumes in the mine schedule and was designed with the following periods:

- Year 1 to 5: annual basis.
- Year 10: final designs (end of LOM).

Figure 16-9 to Figure 16-14 present the corresponding designs.

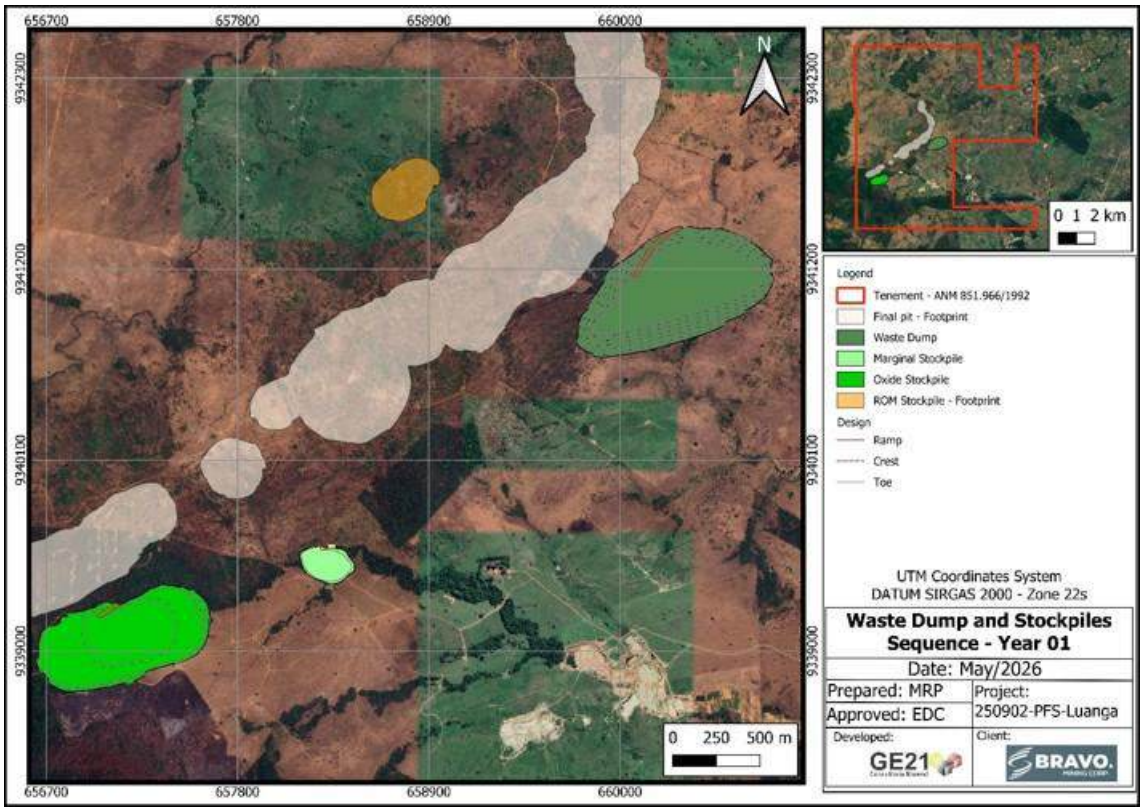


Figure 16-9: Waste dump and stockpiles sequence – year 1

Source: GE21, 2026.

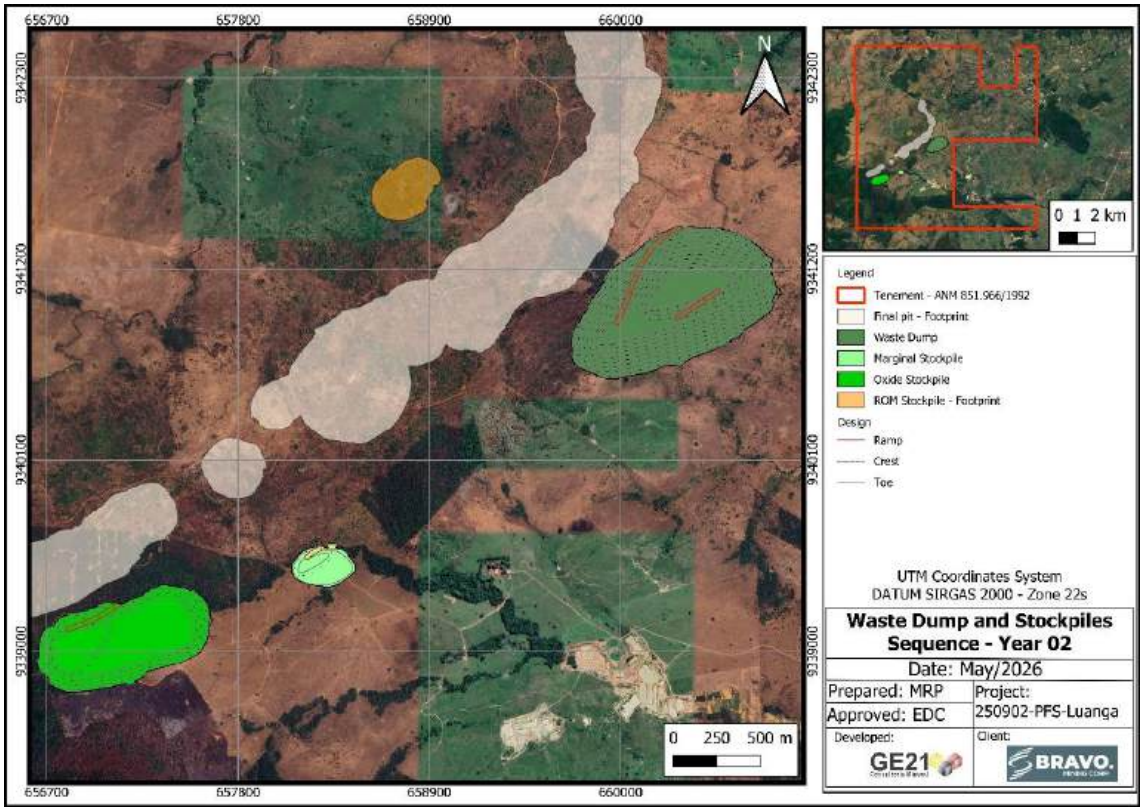


Figure 16-10: Waste dump and stockpiles sequence – year 2

Source: GE21, 2026.

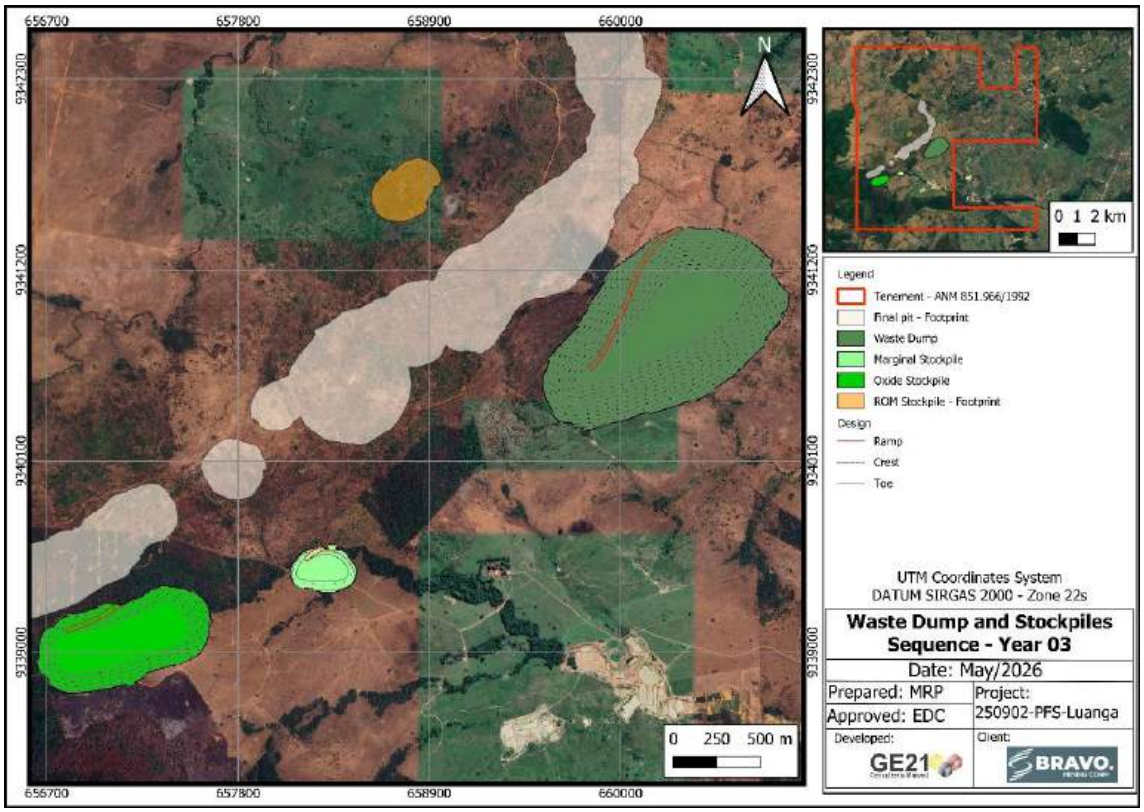


Figure 16-11: Waste dump and stockpiles sequence – year 3

Source: GE21, 2026.

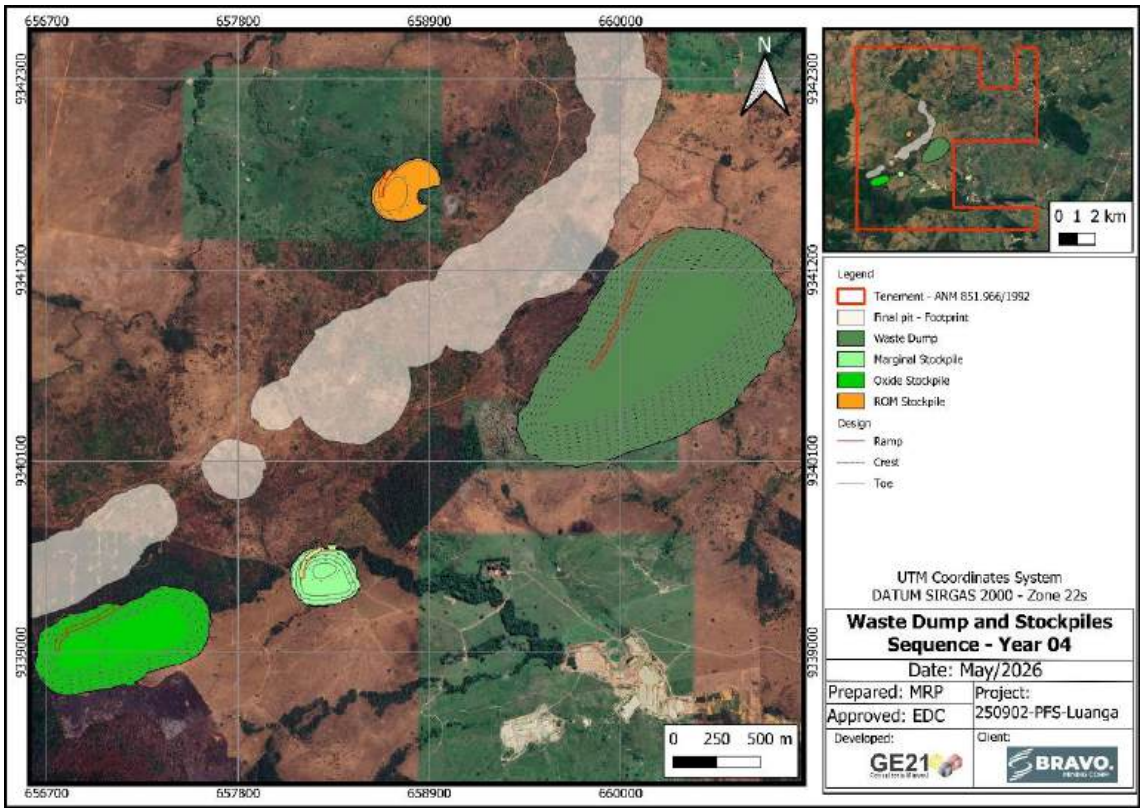


Figure 16-12: Waste dump and stockpiles sequence – year 4

Source: GE21, 2026.

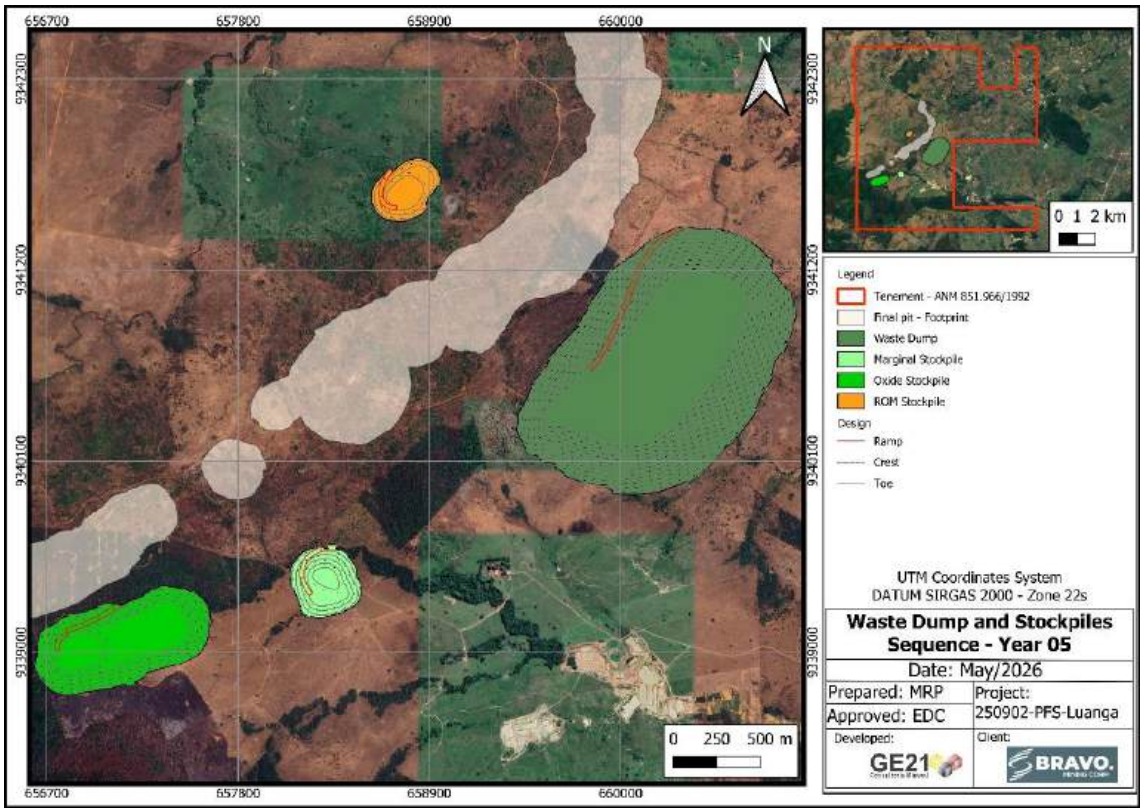


Figure 16-13: Waste dump and stockpiles sequence – year 5

Source: GE21, 2026.

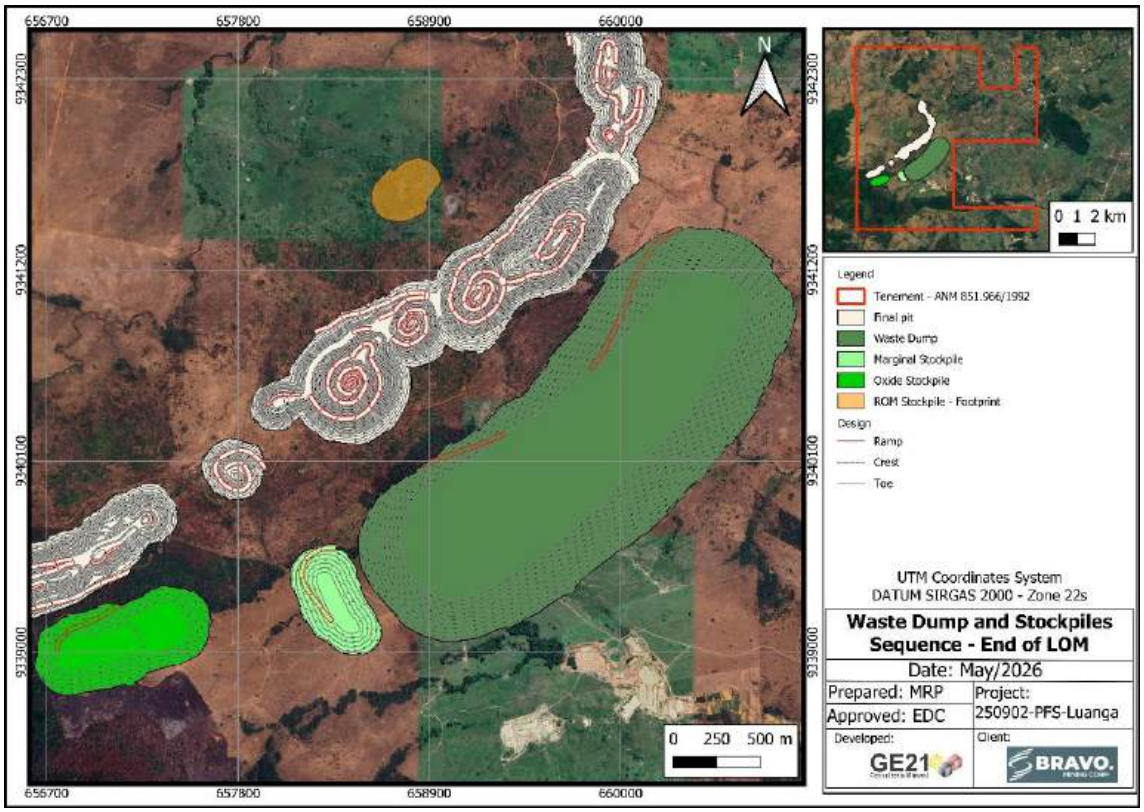


Figure 16-14: Waste dump and stockpiles sequence – year 10 (end of LOM)

Source: GE21, 2025.

16.5 Mining Equipment

Based on the annual production schedule for ROM and waste, along with the yearly average haulage distances, an outsourced mining fleet was estimated for the mining operations. For mining fleet sizing, ROM includes marginal ore, and waste includes oxide material.

Mining operations include:

- Drilling and blasting using explosives.
- Loading and haulage of ROM and waste.
- Discharge of ROM at the primary crusher feed hopper or to a temporary stockpile.
- Tipping of marginal ore, oxide material, and waste in separate dumps designated for each material type.
- Operation of mine infrastructure and supporting ancillary equipment.

ROM material would be mined, transported by haul trucks, and discharged directly into the crusher feed hopper. A portion of the ROM would be temporarily stored in a stockpile near the processing plant. Reclaiming of this material would be conducted using front end loaders and haul trucks, ensuring continuous and stable feed to the processing plant in the event of temporary production interruptions and for blending purposes.

Considering the deposit's characteristics, the required ROM and waste tonnages, and operational cost-efficiency aspects, the selected primary mining equipment are:

- Hydraulic excavators with a 70-tonne operating weight, assigned to ROM extraction.
- Hydraulic excavators with a 120-tonne operating weight, assigned to waste extraction.
- haul trucks with a 55-tonne payload would be used for the transportation of ROM.
- haul trucks with a 120-tonne payload would be used for the transportation of waste.

The support fleet was dimensioned in accordance with the primary equipment size and production targets, and includes:

- Front-end loaders for stockpile management, plant feed regularization, cleaning and conditioning of haul ramps and mining faces after blasting operations.
- Rotary and down-the-hole (DTH) drill rigs for production drilling.
- Bulldozers CAT D8 and D10 models for material spreading, dump shaping, and road maintenance.
- Motor graders and compactor rollers for haul road maintenance.
- Water trucks for dust suppression on roads and in working areas.
- Other ancillary equipment: excavator with rock breaker, backhoe, fuel and lubricant trucks, truck-mounted crane, lowboy truck, light towers, and light vehicles.

16.5.1 Drilling and Blasting

The drilling and blasting plan was developed using the GE21 internal database and benchmark parameters from projects with geomechanical conditions comparable to those at the

Luanga Project. It is estimated that 96% of both ROM and waste consist of hard rock and would require blasting. To ensure operational selectivity, open-pit benches were sized as below:

- ROM: 5-metre-height.
- Waste: 10-metre-height.

During early operations, drilling performance, rock hardness, and fragmentation would be monitored to allow adjustments to blast designs and patterns. Secondary rock breaking when required, would be carried out using either conventional explosives, hydraulic hammers mounted on excavators, or bulldozers to reduce oversize blocks.

Regarding explosive management, an on-site explosives mixing plant is planned. The required volume of explosives for same-day use would be mixed in this facility, reducing the need for storage and associated security costs. This is an increasingly adopted solution in Brazil that complies with regulatory and safety standards for explosives handling and storage in mining operations.

Drilling would be performed by Sandvik Pantera DP1510 or similar drill rigs, working with a hole diameter of 5.5". Figure 16-15 shows the selected equipment.



Figure 16-15: Sandvik Pantera DP1510 drill rig

Source: Sandvik, 2026.

16.5.2 Loading

Excavation and loading of ROM and waste rock would be performed using hydraulic excavators. The CAT 374 or similar, equipped with 4.6 m³ buckets, was selected for ROM loading, and the XCMG XE1350 or similar, equipped with 8 m³ buckets, for the waste. Figure 16-16 and Figure 16-17 show the selected equipment.



Figure 16-16: CAT 374 hydraulic excavator

Source: CAT, 2026.



Figure 16-17: XCMG XE1350 hydraulic excavator

Source: Earthmoving Equipment Magazine, 2026.

Due to the need for ore blending, a minimum of three mining fronts would be operated simultaneously. Additionally, for mine development purposes and to ensure the release of ore with the planned qualities, waste removal would also be carried out concurrently in other active fronts.

Backhoe-configured excavators allow excavation from the upper bench level, with haul trucks positioned on the lower bench. This setup significantly improves overall productivity, as the truck is typically positioned at an angle of approximately 30° to the excavator's longitudinal axis,

resulting in reduced swing time during excavation and loading operations. Operating from the upper bench also enhances the excavator operator's field of vision, which improves safety, load positioning, and productivity. Most importantly, it enables better selectivity during ore extraction.

16.5.3 Haulage

For mined material haulage, a Scania G500 10x4 haul truck or similar, with a payload capacity of 55 t, was selected for ROM transport, while an XCMG XDE130 or similar, with a 120-tonne payload capacity, was selected for waste. Figure 16-18 and Figure 16-19 show the selected equipment.



Figure 16-18: Scania G500 haul truck

Source: Estadão, 2026.



Figure 16-19: XCMG XDE130 haul truck

Source: XCMG, 2026.

Average haul distances were estimated based on mine sequencing between loading and dumping points. The average truck cycle time assumes a fixed 5.5 minutes for loading, turning, spotting, dumping, and operational interference delays. Average truck speeds were set at 25-30 km/h when loaded and 30–35 km/h when empty.

Table 16-8 presents truck productivity parameters.

Table 16-8: Truck productivity parameters

Parameters	Ore	Waste
Calendar hours	8,760	8,760
Availability	85%	85%
Utilization	82%	82%
Efficiency factor	95%	95%
Effective productive hours per year	5,800	5,800
Average Truck Payload (t)	55	120

Source: GE21, 2026.

16.5.4 ROM Reclaims from Temporary Stockpile

A portion of the ROM material would be temporarily stocked in the ROM stockpile, located close to the processing plant. Reclaiming of this material would be carried out using front end loaders and haul trucks, which would transport the ROM to the primary crusher hopper. This operation would be performed by one CAT 966 or similar wheel loader, and two Scania G500 10x4 or similar haul trucks, the same as those used for ROM haulage.

16.5.5 Mining Fleet Sizing

All equipment was sized to meet demand for both ROM and waste operations throughout the mine life. The selected units are commonly used and available in Brazil and are supported by established local distributors and service networks, ensuring efficient maintenance and operational reliability.

The mine fleet was sized based on production targets for ROM and waste, material characteristics, and hauling distances. For volume estimations, a moisture content of 5% and a swell factor of 20% (after compaction) were considered.

Fleet requirements were determined to meet peak production demands, particularly during the full-production phase. A detailed cost-benefit analysis is also recommended, as adding multiple waste dumps closer to the mining fronts is expected to reduce haulage distances for waste disposal and, consequently, minimize overall equipment requirements along the LOM.

Table 16-9 presents the selected mining equipment models and their corresponding technical specifications, while Table 16-10 summarizes the estimated fleet of main and ancillary equipment.

Table 16-9: Selected mining equipment

Equipment	Reference manufacturer	Model	Capacity	Units	Life (hours)
Main Equipment					
Excavator 70 t (ROM)	Caterpillar	CAT 374	4.6	m³	25,000
Excavator 120 t (waste)	XCMG	XE1350	8.0	m³	25,000
Truck 55-tonne 10x4 (ROM)	Scania	XT Heavy Tipper G500	55	t	60,000
Truck 120-tonne (waste)	XCMG	XDE130	120	t	60,000
Ancillary equipment					
Front-end loader	Caterpillar	CAT 966	5	m³	30,000
Drill rigs	Sandvik	Pantera DP1500	3.5-6	Ø in	25,000
Bulldozer – support	Caterpillar	CAT D8	340	HP	30,000
Bulldozer – heavy duty	Caterpillar	CAT D10	750	HP	30000
Motor grader	Caterpillar	CAT 16 M	320	HP	30,000
Excavator with rock breaker	Caterpillar	CAT 336 / CAT 180S	310	HP	15,000
Backhoe	JCB	JCB 3CX	92	HP	18,000
Compactor roller	Caterpillar	CS56B	156	HP	25,000
Fuel and lubricant truck	Scania	G 500 XT 8x4	30	t	60,000
Water truck	Scania	G 500 XT 8x4	40,000	L	60,000
Truck mounted crane	Scania	G 500 XT 8x4	30	t	60,000
Lowboy truck	Scania	G 500 XT 8x4	70	t	60,000
Light tower	Weber	WT 4000	4,000	W	15,000
Light vehicle	Toyota	Hilux 4x4	207	CV	25,000

Source: GE21, 2026.

Table 16-10: Mining equipment fleet estimates

Mining fleet	Years									
	1	2	3	4	5	6	7	8	9	10
Main equipment										
Excavator 70 t (ROM)	3	2	2	3	3	3	3	3	2	2
Excavator 120 t (waste)	6	6	7	6	6	6	6	5	5	4
Truck 55 ton 10x4 (ROM)	7	13	13	16	18	14	14	20	7	3
Truck 120-ton (waste)	40	37	51	43	47	42	40	45	33	28
Auxiliary equipment										
Front-end loader	2	2	2	2	2	2	2	2	3	3
Drill rigs	4	6	9	8	8	8	8	7	6	6
Bulldozer – D8	1	2	2	2	2	2	2	2	1	1
Bulldozer – D10	4	5	7	6	6	6	6	6	5	5
Motograder	3	4	4	4	4	4	4	4	3	3
Excavator with breaker/hammer	1	2	2	2	2	2	2	2	2	2
Back hoe	1	1	1	1	1	1	1	1	1	1
Compactor roller	1	1	1	1	1	1	1	1	1	1
Fuel and lubricant truck	4	5	6	6	6	5	5	6	4	4
Water truck	4	5	5	5	5	5	5	5	5	5
Truck-mounted crane	1	2	2	2	2	2	2	2	1	1
Lowboy truck (4 axles)	1	1	1	1	1	1	1	1	1	1
Light tower	11	15	15	15	15	15	15	15	13	13
Light vehicle	4	5	5	5	5	5	5	5	4	4
Total	98	114	135	128	134	124	122	132	97	87

Source: GE21, 2026.

16.5.6 Workforce

Bravo's own workforce would include staff dedicated to mine management and supervision (Table 16-11).

Table 16-11: Owner workforce estimates

Department	Position	Quantity
Management	General Manager	1
	Technical Services Manager	1
	Administrative Assistant	2
Technical services	Mining Engineer	2
	Geologist	2
	Geotechnical Engineer	1
Operations	Mine / Geology Coordinator	2
	Mine / Geology Technician	4
	Production Assistant	4
	Mine / Geology Supervisor	8
	Surveyor	3
	Survey Assistant	6
Engineering and Infrastructure	Engineering and Infrastructure Coordinator	1
	Infrastructure Supervisor	2
	Electrician	1
	Infrastructure Technician	4

Source: GE21, 2026.

Equipment operators would be selected depending on the type of equipment and would be managed by the outsourced company. Table 16-12 presents the required workforce.

Table 16-12: Contractor workforce estimates

Department	Position	1	2	3	4	5	6	7	8	9	10
Maintenance	Maintenance Coordinator	1	1	1	1	1	1	1	1	1	1
	Maintenance Supervisor	8	8	8	8	8	8	8	8	8	8
	Maintenance Controller	4	4	4	4	4	4	4	4	4	4
	Welding Technician	3	4	5	5	5	5	5	5	3	3
	Lubrication Technician	3	4	5	5	5	5	5	5	3	3
	Tire Technician	4	5	7	6	7	6	6	7	4	4
	Electrical Technician	5	6	7	8	8	8	8	7	5	4
	Mechanical Technician	18	22	27	30	31	30	29	28	18	15
Operation	Excavator Operators	46	56	60	60	60	60	60	56	46	42
	Loader Operators	10	10	10	10	10	10	10	10	10	10
	Truck Operators	152	184	248	214	235	208	198	227	159	132
	Ancillary Equipment Operators	105	118	145	136	136	131	131	127	105	92
	Total	359	422	527	487	510	476	465	485	366	318

Source: GE21, 2026.

16.5.7 Diesel Consumption

The diesel consumption was estimated based on hourly diesel consumption by equipment and the fleet's working hours. Table 16-13 shows the estimated diesel consumption by year.

Table 16-13: Mining fleet diesel consumption

	1	2	3	4	5	6	7	8	9	10
Main equipment										
Excavator 70 t (ROM)	881	881	881	1,322	1,322	1,322	1,322	1,322	881	881
Excavator 120 t (waste)	4,406	5,288	6,169	5,288	5,288	5,288	5,288	4,406	4,406	3,525
Truck 55-ton 10x4 (ROM)	1,410	4,583	4,583	5,640	6,345	4,935	4,935	7,050	2,468	1,058
Truck 120-ton (waste)	33,841	39,129	53,934	45,474	49,704	44,417	42,301	47,589	34,899	29,611
Auxiliary equipment										
Front-end loader	290	290	290	290	290	290	290	290	290	290
Drill rig	317	317	317	317	317	317	317	317	317	317
Bulldozer – support	169	339	339	339	339	339	339	339	169	169
Bulldozer – heavy duty	1,960	1,633	2,286	1,960	1,960	1,960	1,960	1,633	1,633	1,306
Motor grader	505	674	674	674	674	674	674	674	505	505
Excavator with rock breaker	380	380	380	380	380	380	380	380	380	190
Backhoe	31	31	31	31	31	31	31	31	31	31
Compactor roller	35	35	35	35	35	35	35	35	35	35
Fuel and lubricant truck	635	793	952	952	952	793	793	952	635	529
Water truck	925	1,157	1,157	1,157	1,157	1,157	1,157	1,157	1,157	771
Truck-mounted crane	66	132	132	132	132	132	132	132	66	55
Lowboy truck	73	73	73	73	73	73	73	73	73	73
Light tower	43	46	46	46	46	46	46	46	40	34
Light vehicle	62	78	78	78	78	78	78	78	62	62
Total litres per year x 1,000	**46,032**	**55,859**	**72,358**	**64,188**	**69,123**	**62,267**	**60,152**	**66,505**	**48,049**	**39,444**

Source: GE21, 2026.

16.6 Comments on Mining Methods

The selected open-pit mining method suits the dimensions of the orebodies. It limits dilution and secures a consistent supply to the process plant. The production fleet consists of 70-t excavators with 4.6 m³ buckets loading 55-t conventional haul trucks for ROM material, and 120-

t excavators with 8.0 m³ buckets loading 120-t off-road trucks. The auxiliary fleet comprises bulldozers, graders, drill rigs, water trucks, fuel trucks, compactors and front-end loaders.

The selected equipment sizes are standard across Brazilian mining operations. This supports the sourcing of equipment, spare parts, operators and maintenance personnel, and the workforce can be largely sourced from the surrounding communities.

The operation would follow a conventional drill, blast, load and haul cycle, with a minor portion of the total material movement as free-digging material.

The following recommendations are made for the next phase of study:

- Refine the mine schedule to optimize the extraction and deposition sequence of waste, marginal and oxide materials. The objective is to minimize average haul distances and the associated haulage costs.
- Carry out a trade-off study on wider haul roads and ramps that would allow larger-class loading and haulage equipment. The study should weigh the potential reduction in unit mining costs against the additional waste stripping and the effect on pit slope geometry.
- Assess dedicated haul roads for the different truck classes, separating the production fleet from auxiliary and light vehicle traffic. This would improve operational safety and haulage efficiency.
- Conduct a dedicated drill and blast study for the oxide and transition zones. It should define the free-dig/blast boundary, appropriate blast designs and powder factors, and expected fragmentation, supported by additional geotechnical and geomechanical characterization of these materials.

17 RECOVERY METHODS

17.1 General

The evolution of process studies based on the PEA Technical Report, filed with the System for Electronic Document Analysis and Retrieval+ (SEDAR+), has enabled process optimization and an increase in production capacity to 10 Mt per year. The processing route described in this Technical Report is intended for the processing of sulphide ore (fresh rock), with oxidized saprolite stored in an area designated in the master plan for future use.

This Technical Report assumes an accelerated ramp-up of the plant, reaching nominal capacity as early as the second year of operation to optimize return on investment.

The deposit is mineralogically complex: Platinum Group Metals (PGMs), gold, and nickel sulphides (predominantly pentlandite) are closely associated with a gangue of ferromagnesian silicates, notably talc. The hydrophobic nature of talc and its significant interference with the process required a comprehensive metallurgical test work.

17.2 Processing Circuit

The comminution circuit was designed to target an fine particle size distribution, in the range of 26 to 30 µm, which is necessary for the effective liberation of the valuable minerals. Variability tests demonstrated that the ore has moderate to high hardness, with the central zone exhibiting the greatest resistance to grinding. Table 17-1 summarizes the main parameters obtained in the tests.

Table 17-1: Comminution parameters considered by area

Technical parameter	Unit	North zone	Central zone	Southwest zone
Bond Ball Mill Work Index (Wi)	kWh/t	14.98	18.91	14.43
	Class	Medium-hard	Hard	Medium-hard
Bond Rod Mill Work Index (Wi)	kWh/t	16.4	19.3	16.5
	Class	Hard	Very hard	Hard
Macon crushability	%	32.4	29.1	37.7
	Class	Medium	Medium	Medium
Bond Abrasion Index (Ai)	-	0.070	0.091	0.044
	Class	Low	Low	Low
Macon Abrasion Index (Ai)	g/t	228	502	148
	Class	Low	Low	Low
SAG specific energy	kWh/t	10.56	11.75	9.44
Specific gravity	t/m³	2.95	3.03	2.91
Bulk density	t/m³	1.68	1.73	1.71
Volumetric capacity index	%	83.13	77.89	75.46
	Class	Minimum	Minimum	Minimum
Strength index (R)	%	190.64	248.31	161.97
	Class	Minimum	Minimum	Minimum
Flakiness index	%	1.69	14.16	14.16
	Class	Cubical	Cubical	Cubical

Source: Bravo and Weir, 2026.

Table 17-2 presents the process criteria adopted for estimating and sizing comminution equipment.

Table 17-2: Process criteria – crushing and grinding

Criteria	Unit	Value
Annual capacity	Mtpa	10
Operational availability of the crushing circuit	%	70
Operational availability of the HPGR and grinding circuit	%	85.7
Nominal feed rate to the crushing plant	t/h	1,631
Nominal feed rate for HPGR and grinding	t/h	1,332
Ball mill Bond index (BWi)	kWh/t	19
Axb		32
Specific capacity of HPGR	ts/h·m³	250
Specific energy consumption of HPGR	kWh/t	1.50
Mass recovery in flotation	%	1.7
PGM 3E metallurgical recovery	%	80.0
Palladium (Pd)	%	81.5
Platinum (Pt)	%	80.0
Rhodium (Rh)	%	55.0
Gold (Au)	%	88.0
Nickel (Ni)	%	70.6
3E PGM grade in concentrate	Gpt	100
Ni grade in concentrate	%	8.44

Source: Weir Technology, 2026.

17.3 Summary Description of the Circuit

The Run of Mine (ROM) is fed into a primary crushing unit, consisting of a Metso 60' × 90 gyratory crusher or similar, rated at 1,631 t/h. The crushed material is fed to two Weir Enduron double-deck screens, measuring 3.05 m × 6.10 m with screen appuratures of 80 mm and 50 mm. The undersize from the second deck is transferred to a buffer pile, while the oversize feeds two Weir Enduron EP900 cone crushers operating in parallel. The secondary crushing product returns to the screens in a closed circuit.

The ore is reclaimed from the stockpile by apron feeders and fed to a Weir Enduron RPM24 220/200 High Pressure Grinding Rolls (HPGR) via a conveyor belt. The HPGR output is classified on a double-deck Weir Enduron vibrating screen, measuring 3.7 m × 8.5 m. Oversize material is returned to the HPGR in a closed circuit. The undersize feeds a bank of 10 Weir 650CVD13 hydrocyclones, of which 8 are in operation, and the cluster underflow is fed to an 18 MW ball mill. The mill product returns to the cyclone feed box in a closed circuit.

The cluster's overflow is pumped to a new cluster with 6 Weir 500CVD10 cyclones, 4 of which are in operation. The underflow feeds a vertical stirred mill, whose product is combined with the overflow from that cluster in an open circuit and feeds the thickener in the flotation circuit.

Figure 17-1 shows the flow diagram of the comminution circuit, including the primary and secondary crushing stages. Figure 17-2 presents the HPGR circuit and vertical grinding.

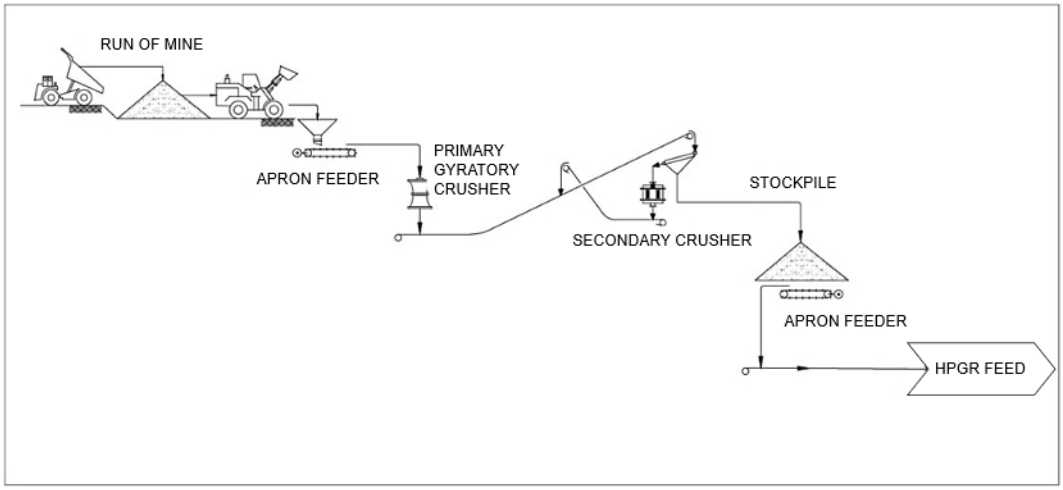


Figure 17-1: Flow diagram of the crushing circuit

Source: Onix, 2026.

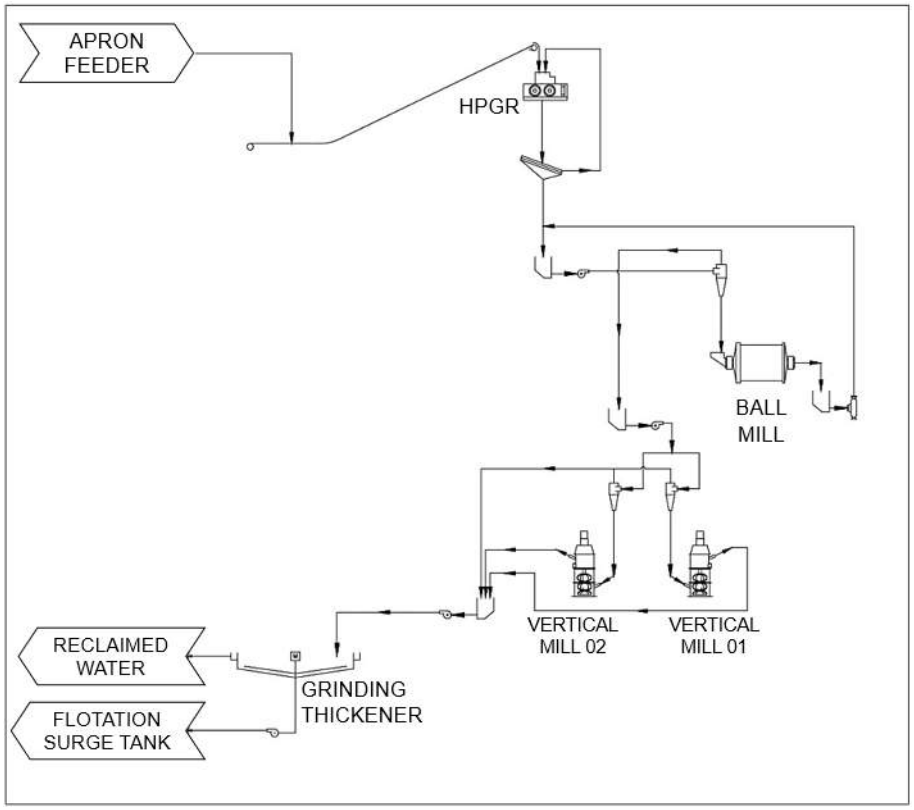


Figure 17-2: Flow diagram of the grinding circuit

Source: Onix, 2026.

The flotation circuit is fed by the thickener underflow. It consists of Jameson cells, configured as one rougher stage, two scavenger stages operating in series, and one cleaner stage designed to re-process the concentrate produced in the scavenger cells.

The concentrate from the rougher stage is combined with that from the cleaner stage, forming the final concentrate of the processing plant. The tailings from the scavenger stage constitute the final tailings of the circuit, while the tailings from the cleaner stage are returned to

the first scavenger cell (Figure 17-3). In all stages, a portion of the tailings is recirculated through the cell itself, which improves the performance of the operation.

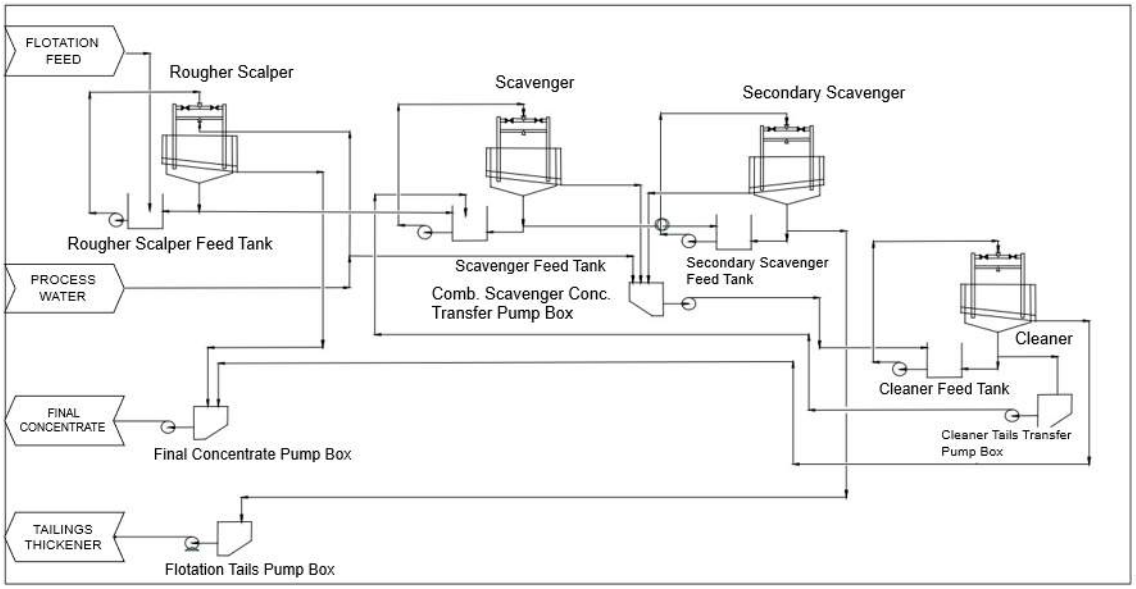


Figure 17-3: Flow diagram of the flotation circuit

Source: Glencore, 2026.

The produced concentrate is transferred to a concentrate thickener of 30 m diameter. The underflow is pumped to an agitated storage tank and then to the concentrate filtration unit. Figure 17-4 shows the designed circuit. The filter cake is stored undercover, while the filtrate returns to the reclaimed process water tank.

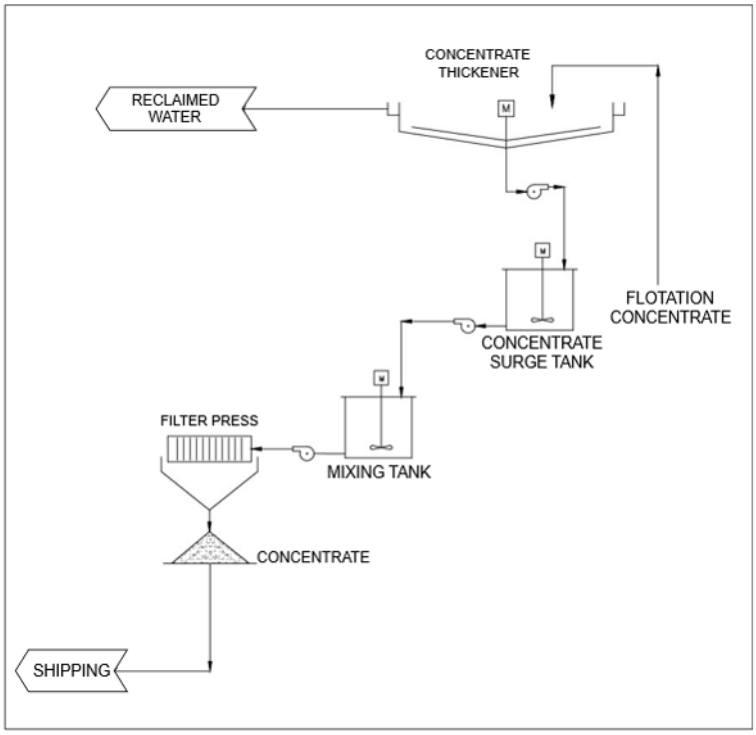


Figure 17-4: Flow diagram for final concentrate treatment

Source: Onix, 2026.

The tailings from the secondary scavenger stage are pumped to the treatment unit, constituting the circuit's final tailings. The underflow is pumped to the disposal dam, while the overflow proceeds to the reclaimed process water tank.

Figure 17-5 shows the circuit designed for tailings treatment.

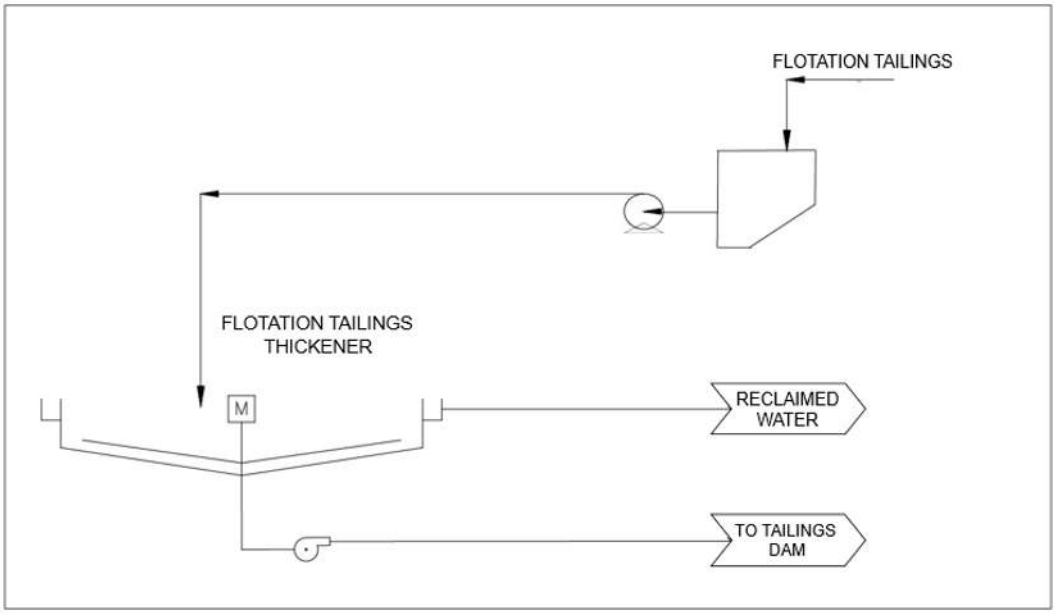


Figure 17-5: Flow diagram for tailings treatment

Source: Onix, 2026.

17.4 Mass Balance

Tests conducted at Base Metal Laboratories (BML) in Canada indicated concentrate grades exceeding 100 g/t for 3E (Pt, Pd, Rh) and Ni grades of circa 8%, with estimated overall recoveries of 80% and 70%, respectively.

Table 17-3 shows the expected mass balance for the industrial operation, considering the laboratory partitioning results and the expected average grades in the plant feed.

Table 17-3: Flotation mass balance

	Rougher			Scavenger			Cleaner			Final Concentrate
	Feed	Concent.	Tailings	Feed	Concent.	Tailings	Feed	Concent.	Tailings	
Mass (t/h)	1,332	5.1	1,326.9	1,962.1	166.8	1,310.7	361.9	16.2	150.6	21.3
Overall Mass Partition	100	0.4	99.6	147.3	12.5	98.4	27.2	1.2	11.3	1.6
Stage Mass Partition	100	0.4	99.6	100	8.5	66.8	100	4.5	41.6	-
∑E (g/t)	2.00	203.67	1.23	1.12	7.57	0.41	4.09	67.48	1.12	100.0
Ni (%)	0.19	11.04	0.15	0.13	0.8	0.06	0.41	7.61	0.07	8.4
PGM Dist.	100	38.9	61.1	82.6	47.4	20.0	55.6	41.1	6.3	80.0
Ni Dist.	100	22.0	78.0	101.2	52.5	29.4	57.6	48.5	3.9	70.6

Source: Adapted from Glencore Technology's by GE21, 2026.

17.5 Equipment List

Table 17-4 shows a list of the plant's main equipment. The sizing data was obtained from the suppliers of the main equipment (Weir, Glencore), and the remaining equipment was based on similar projects. Additional tests would be conducted by Bravo to verify and validate the sizing.

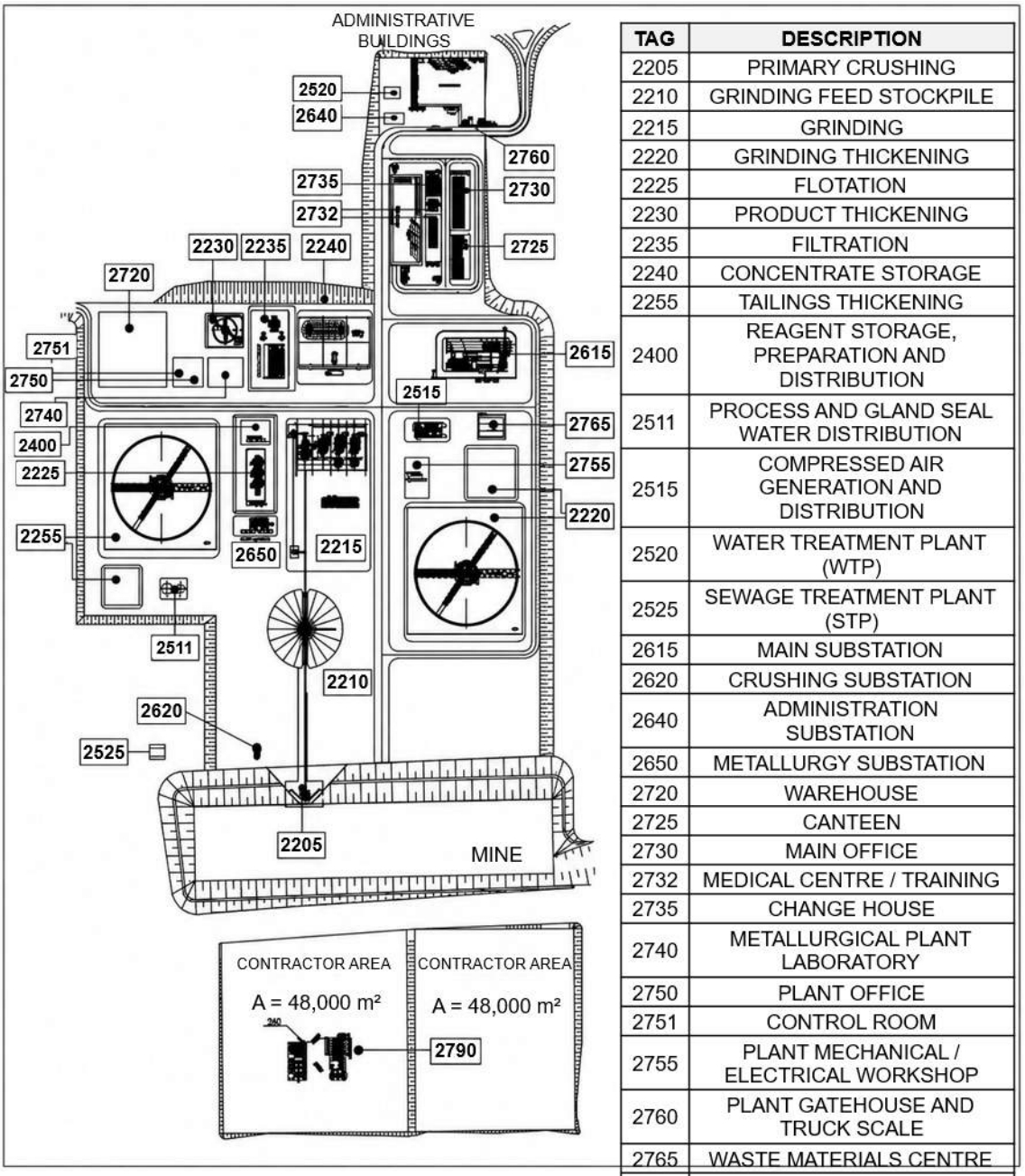
Table 17-4: List of major equipment at the processing plant

Area	Equipment	Qt	Specification	Installed power (kW)	Area	Equipment	Qt	Specification	Installed power (kW)
Crushing	Gyratory crusher	1	1,631 tph	600	Grinding thickener circuit	Grinding Thickener	1	125 m diameter	55
	Apron feeder	1	96" × 15 m	75		Thickener Underflow Pump	2	3,060 m³/h @ 30 m TDH	748
	Primary crusher conveyor	1	48" × 153 m	225		Vertical Sump Pump	1	25 m³/h @ 15 m TDH	4
	Vibrating screen	2	ENDURON® Elite, double deck, 3.05 × 6.10 m; 80 mm and 50 mm openings	45	Jameson flotation circuit	Jameson flotation package	1	Complete package	3,256
	Cone crusher	2	ENDURON® EP900, 45-mm CSS	630		Concentrate transfer pump	2	111.3 m³/h @ 16 m TDH	23
Coarse ore stockpile and reclaim	Apron feeder	3	60" × 7 m	25		Tailings transfer pump	2	3,598 m³/h @ 28 m TDH	598
	HPGR mill feed conveyor	1	48" × 172 m	150		Vertical sump pump	1	25 m³/h @ 15 m TDH	4
	Tramp metal magnet	1	For 48" conveyor	20	Tailings thickener circuit	Tailings thickener	1	125 m diameter	55
	Metal detector	1	For 48" conveyor	1		Tailings thickener underflow pump	2	1,833 m³/h @ 51 m TDH	440
HPGR circuit	Vibrating screen	2	ENDURON® Elite, double deck, 3.70 × 8.50 m; 12 mm and 6 mm openings	132		Vertical sump pump	1	25 m³/h @ 15 m TDH	4
-	-	-	-	-	Concentrate thickener circuit	Concentrate thickener	1	30 m diameter	55
Primary ball milling	Ball mill	1	26 ft × 44 ft	17,800		Concentrate thickener underflow pump	2	66 m³/h @ 13 m TDH	15
	Vertical sump pump	1	25 m³/h @ 15 m TDH	4		Vertical sump pump	1	25 m³/h @ 15 m TDH	4
	Hydrocyclone	10	650CVD13	-	-	-	-	-	-
Stirred milling circuit	Stirred mill	2	Optmil-50	8,500	-	-	-	-	-
	Vertical sump pump	1	25 m³/h @ 15 m TDH	4	-	-	-	-	-
	Hydrocyclone	8	500CVD10	-	-	-	-	-	-

Source: Onix, 2026.

17.6 Plant Footprint Plan

Figure 17-6 shows the planned facilities and structures. The footprint would occupy approximately 568,000 m², and the administrative buildings would occupy an additional 40,000 m².



TAG	DESCRIPTION
2205	PRIMARY CRUSHING
2210	GRINDING FEED STOCKPILE
2215	GRINDING
2220	GRINDING THICKENING
2225	FLOTATION
2230	PRODUCT THICKENING
2235	FILTRATION
2240	CONCENTRATE STORAGE
2255	TAILINGS THICKENING
2400	REAGENT STORAGE, PREPARATION AND DISTRIBUTION
2511	PROCESS AND GLAND SEAL WATER DISTRIBUTION
2515	COMPRESSED AIR GENERATION AND DISTRIBUTION
2520	WATER TREATMENT PLANT (WTP)
2525	SEWAGE TREATMENT PLANT (STP)
2615	MAIN SUBSTATION
2620	CRUSHING SUBSTATION
2640	ADMINISTRATION SUBSTATION
2650	METALLURGY SUBSTATION
2720	WAREHOUSE
2725	CANTEEN
2730	MAIN OFFICE
2732	MEDICAL CENTRE / TRAINING
2735	CHANGE HOUSE
2740	METALLURGICAL PLANT LABORATORY
2750	PLANT OFFICE
2751	CONTROL ROOM
2755	PLANT MECHANICAL / ELECTRICAL WORKSHOP
2760	PLANT GATEHOUSE AND TRUCK SCALE
2765	WASTE MATERIALS CENTRE
2790	THIRD-PARTY / CONTRACTOR WORKSHOP

Figure 17-6: Master plan

Source: Onix, 2026.

17.7 Overall Water Balance

The water balance assumes the flotation plant would be operating at a steady state of 10 Mtpa starting in the second year, which requires a more robust pumping and storage infrastructure from the start-up phase onward. Table 17-5 shows the inputs and outputs of the water balance.

Table 17-5: Summary of water system inputs and outputs

	Water stream	Design flow rate (m³/h)	Source/Destination
Inputs (supply)	Reclaimed water (thickeners)	3,420	Overflow from tailings and concentrate thickeners
	Reclaimed water (TSF)	827	Floating pump barge at the tailings storage facility
	Fresh water (main reservoir)	400	Internal water storage reservoir
	Rio Sereno pipeline (contingency)	136	Activated only during extreme drought conditions
Outputs (consumption and losses)	Process water distribution	3,420	Grinding and Jameson flotation circuits
	Potable and administrative water (WTP)	10	Domestic consumption and support facilities
	Water losses and retention (tailings/concentrate)	485	Moisture retained in tailings and concentrate, plus evaporation losses
	Other uses (gland sealing and firefighting)	Included	Pump sealing water and emergency reserve supply

Source: GeoHydroTech Engenharia, 2023; GE21, 2026; CAPEX Report N. BRV10-PFS-RL-0000-OX-000-002, Rev.1.

17.7.1 Water Collection and Storage Infrastructure

Table 17-6 presents the water collection and storage structures.

Table 17-6: Technical specifications of water supply infrastructure

Parameter	Unit	Value/Specification
Storage capacity	m³	2,700,000
Maximum embankment height	m	< 25.0
Rio Sereno pipeline length	m	4,880
Pipeline diameter	-	HDPE PN 10
Contingency water supply capacity	m³/h	136.0
Reservoir surface area (normal operating level)	km²	0.95

Source: GeoHydroTech Engenharia, 2023; GE21, 2026.

17.7.2 Tailings and Process Water Recovery

High-capacity thickeners are planned, with diameters of 125 m for tailings and 30 m for concentrate, to maximize the recovery of clarified water for the plant.

17.7.3 Utility Systems and Specialized Water Systems

The water collection and distribution system also includes a potable water supply with a nominal capacity of 30 m³/h, supplied by tube wells or the main reservoir.

It also provides gland seal water for pumps and for the fire-suppression system, which consists of a dedicated 50 m³ reservoir equipped with a set of motor pumps (electric + diesel + jockey) rated at 40 m³/h.

17.7.4 Monitoring, Automation, and Control

The water management system is automated and includes ultrasonic level sensors in the main reservoir, the Tailings Storage Facility, and the process tanks. It also includes *online* monitoring of turbidity and pH, as well as piezometers and surface markers to ensure the stability of the structures, in accordance with the Brazilian National Dam Safety Policy (Política Nacional de Segurança de Barragens – PNSB).

17.7.5 Conclusions on the Water System

The water management system was designed based on a minimum recovery rate of 60%. For the DFS phase, additional studies will be conducted and may include:

- Geotechnical drilling with in-situ permeability tests to confirm the values of the parameters adopted for infiltration, among others.
- Long-term hydrological models using Monte Carlo simulation, based on 50-year historical data series, to validate the resilience of the 2.7 Mm³ reservoir in the face of multi-year drought cycles.
- Evaluation of the use of paste thickeners to reduce the volume of water sent to the dam.

17.8 Energy Supply

17.8.1 Project Philosophy and Strategic Context

The estimated nominal demand is 58 MW, with plans to connect to a 230 kV transmission line on the national grid, to be negotiated with the Brazilian National Electric Energy Agency (*Agência Nacional de Energia Elétrica* – ANEEL).

The sizing of the connection and the substation allows for growth and future loads, subject to the consolidation of the power balance for the HPGR-B-S route.

Power supply would be provided through a direct connection to the Brazilian National Interconnected System (*Sistema Interligado Nacional* – SIN).

17.8.2 Process Load and Demand Balance

Table 17-7 shows the preliminary electrical demand estimates among the plant's main load centres.

Table 17-7: Distribution of demand by loads and process

Process area	Connected load (kW)	Estimated annual energy consumption (MWh/a)
Crushing and stockpile, HPGR	5,436	32,876
Coarse grinding (ball mill)	17,107	133,022
Optmil-50 vertical mills	16,105	125,235
Jameson flotation	3,365	27,981
Thickening and Filtration	2,236.1	20,450
Utilities and water management	767	5,713
Infrastructure, support buildings, and administrative facilities	1,124	4,334
Reserve for increased load	3,000	-
Total	**49,145**	**349,521**

Source: Onix Engenharia, 2026.

The reserve accounts for the anticipated load resulting from possible changes to the comminution route.

17.8.3 Options for Connection to the National Integrated System

Three options for connecting to the SIN were evaluated, considering the criteria of overall cost, electrical losses, and regulatory complexity. The selected option involves connecting to the Carajás Substation at 230 kV via a dedicated transmission line approximately 34 km long, with the capacity to meet the Project's long-term demand.

17.9 Chemical Laboratory

The laboratory has a projected area of approximately 1,000 m², with the capacity to process about 500 samples per day, and is located near the grinding and flotation areas.

Figure 17-7 shows the planned layout of the laboratory.

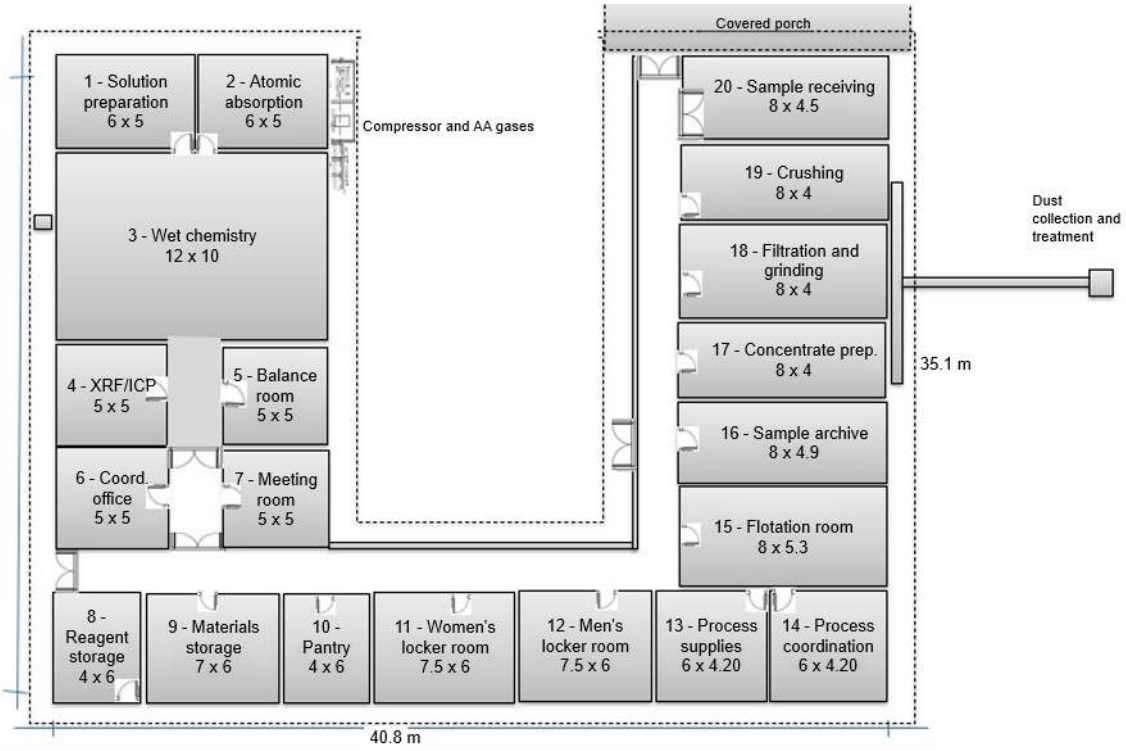


Figure 17-7: Planned layout for the laboratory

Source: Onix Engenharia, 2026.

17.10 Process Laboratory (Metallurgy)

Designed to support both geometallurgy and the industrial plant, the laboratory would feature the following facilities:

- Comminution testing.
- Jameson L150 flotation mini-plant.

17.11 Downstream Processing – Barcarena Smelter

17.11.1 Basis and Level of Study

The bulk sulphide concentrate produced by the Luanga concentrator, described in the preceding sections, is treated at a dedicated smelter in Barcarena, Pará State, within the Vila do Conde Export Processing Zone (ZPE). The smelter site lies approximately 500 km north of the mine by road and 8.5 km from the port loading berths. The concentrate would be trucked from the mine, and the smelter products would either be exported through Vila do Conde or sold in the domestic market.

The smelter is part of the integrated project. Its recoveries, products, capital cost and operating cost are carried in the economic analysis presented in Section 22. The process plant, infrastructure and utilities engineering support capital and operating cost estimates to Association for the Advancement of Cost Engineering (AACE) Class 4 accuracy, which is the same level of definition as the concentrator described above and is suitable for a PFS.

The design summarized in this section is taken from the Updated Smelter Study for the Luanga Project (Jones and Müller, 2026). The plant layout is at a conceptual level. A ground geotechnical investigation of the smelter site and a groundwater study to confirm sustainable yields have not been carried out, and continuous converting has not been piloted.

17.11.2 Process Selection

Bravo reviewed flash smelting, Ausmelt, alternating current (AC) electric arc furnace smelting, direct current (DC) reductive electric arc furnace smelting and hydrometallurgical options against criteria of technological appropriateness, technology risk, environmental management and cost. Oxidative fluidized bed roasting followed by reductive smelting in a DC electric arc furnace was selected.

The selection rests on sulphur behaviour and the properties of the furnace. Removing sulphur in a roaster ahead of the furnace produces a calcine that smelts to an iron-rich alloy whose liquidus temperature is close to that of the slag, rather than the low-melting, highly superheated matte produced by conventional smelting. Containment risk falls accordingly, since it is the superheated matte that penetrates refractories and causes the furnace failures recorded in the platinum industry. Sulphur leaves the process as a steady, high-strength SO_2 stream from a single unit, which supports a smaller, less costly acid plant than the intermittent, dilute gas produced by converting. It also limits fugitive sulphur emissions at the furnace and the converter.

The DC arc furnace accommodates wide variations in concentrate chemistry, including the elevated magnesia content of the Luanga concentrate and any chromium reporting to it, which reduces the burden on the concentrator to control concentrate quality. The furnace is sealed, so carbon monoxide, sulphur dioxide and dust are contained, and the circuit requires fewer gas collection and cleaning points. A single central electrode gives an even temperature distribution,

avoiding the hot spots that shorten refractory life. Electrode consumption is lower because there are fewer, smaller electrodes that are not immersed in the slag.

17.11.3 Supporting Test Work

The route is supported by pilot- and demonstration-scale test work carried out over more than a decade on various South African PGM-bearing feed materials, much of it at Mintek under the direct supervision of one of the authors of the smelter study, along with confirmatory bench-scale work on Luanga concentrate.

- In 1998, approximately 30 t of PGM-bearing sulphide concentrate was roasted in a fluidized bed at about 1,000 °C, reducing sulphur from 4.55% to 0.13%. It was then smelted in a pilot-scale DC arc furnace at an average temperature of 1,650 °C. Average precious metal loss to slag over the campaign was 2.9 g/t, with values as low as 0.3 g/t, at a specific energy requirement of 650 kWh/t of calcine.
- Between April 2004 and August 2008, more than 37,000 t of PGM-containing feed material was dried and smelted in a 1.5 MW DC arc furnace over four and a half years of continuous toll treatment. Across the 36,952 t processed, the weighted average feed grade was 51 g/t 4E, and the weighted average slag grade was 1.1 g/t, with results below the 0.28 g/t detection limit frequently reported. Gross energy consumption fell from 1.22 MWh/t in the first campaign to 0.83 MWh/t in the fifth; electrode consumption averaged 1.5 kg per MWh, and the furnace ran for more than 40 months without a major incident.
- Two later campaigns in a 3 MW DC arc furnace treated material with a sulphur content high enough to produce a reductive matte rather than an alloy. The superheated matte attacked the sidewall refractories, causing a containment failure in March 2009. In the second campaign, approximately a quarter of the precious metal fed was locked up in the furnace hearth. The contrast between four and a half years of incident-free alloy smelting and six months of high-sulphur operation is the direct basis for retaining the roasting step.
- Pilot roasting test work was carried out by Technip in the United States and by Outotec in Germany, and 300 t of flotation concentrate was roasted at Germiston in South Africa. Total sulphur was reduced from about 7% to below 1%, and to as low as 0.3% in the cyclone product.
- Bravo completed exploratory bench-scale test work on Luanga concentrate at Arxo Metals in 2024. A 68 g/t 4E sample was roasted at 1,000 °C for 120 minutes, reducing sulphur to 1.23% against a target of below 2%. It was then smelted in a 35 kVA DC furnace to produce a nickel-iron alloy grading 716 g/t 4E, with precious metals and nickel in the slag below detection limits. Mass accountability was poor because of the small scale and limited sample mass, so the calculated recoveries are not relied upon. The key results are the alloy grade and the barren slag, which together confirm that Luanga concentrate is amenable to reductive smelting in a DC arc furnace.

17.11.4 Feed Specification and Design Criteria

The design concentrate composition is taken from locked-cycle and mini-plant test work on Luanga ore, completed in Canada and Brazil, and is the concentrate produced by the

concentrator described (Table 17-8). It is reported on an as-received basis and includes the contained moisture.

Table 17-8: Design concentrate composition (as-received basis)

Component	Value (%)	Component	Value
SiO_2	31.2	Cr_2O_3	0.12%
S	17.3	Pt	41 g/t
FeO	16.0	Pd	80 g/t
MgO	11.9	Rh	1 g/t
Ni	7.7	Au	14 g/t
Cu	3.7	As	176 g/t
CaO	1.79	Cl	179 g/t
Al_2O_3	1.58	F	85 g/t
Co	0.43	Moisture	8.0%
Fe_2O_3	0.16	-	-

Note: Precious metal grades total 136 g/t on a Pt, Pd, Rh and Au basis, of which 122 g/t is Pt, Pd and Rh.
Source: Bravo, 2026.

Iron in the concentrate is present predominantly as FeO. This matters to the flowsheet because the process collects the valuable metals in an iron alloy, and iron is reduced from the oxide in the feed. At 16% FeO, the concentrate carries approximately 3.3 t/h of iron, which is sufficient for alloy production without a supplementary iron source (Table 17-9).

Table 17-9: Smelter design criteria

Parameter	Value	Unit
Annual concentrate throughput, wet	200,000	t/a
Plant availability	85	%
Operating hours	7,446	h/a
Design feed rate, wet	26.9	t/h
Design feed rate, dry	24.7	t/h
Dryer product moisture	1	%
Roaster type	Bubbling fluidized bed	
Roaster operating temperature	1,000	°C
Sulphur removal in roasting	98	%
Furnace type	DC electric arc, single electrode	
Furnace nominal rating	20	MW
Furnace power intensity	250	kW/m²
Furnace hearth diameter	9.5	m
Furnace shell diameter	10	m
Slag tapping temperature	1,650	°C
Alloy tapping temperature	1,600	°C
Converter type	Top-blown, batch	
Degree of iron removal in converting	97	%
Atomization water pressure	100	MPa
Atomized product particle size	10-150	µm
Acid plant SO_2 conversion efficiency	91	%

Source: Bravo, 2026.

17.11.5 Process Description

Concentrate is received at the smelter as filter cake, then sampled and assayed, and homogenized in a blending facility to deliver a consistent feed chemistry to the roaster. Oversize material is milled, where necessary, to ensure reactivity in the roaster. The blended concentrate

is dried from 8% to approximately 1% moisture in a rotary dryer fired on natural gas, evaporating 1.9 t/h of water. Dryer off-gas passes through a baghouse, and the collected dust returns to the roaster feed system. Figure 17-8 illustrates this process.

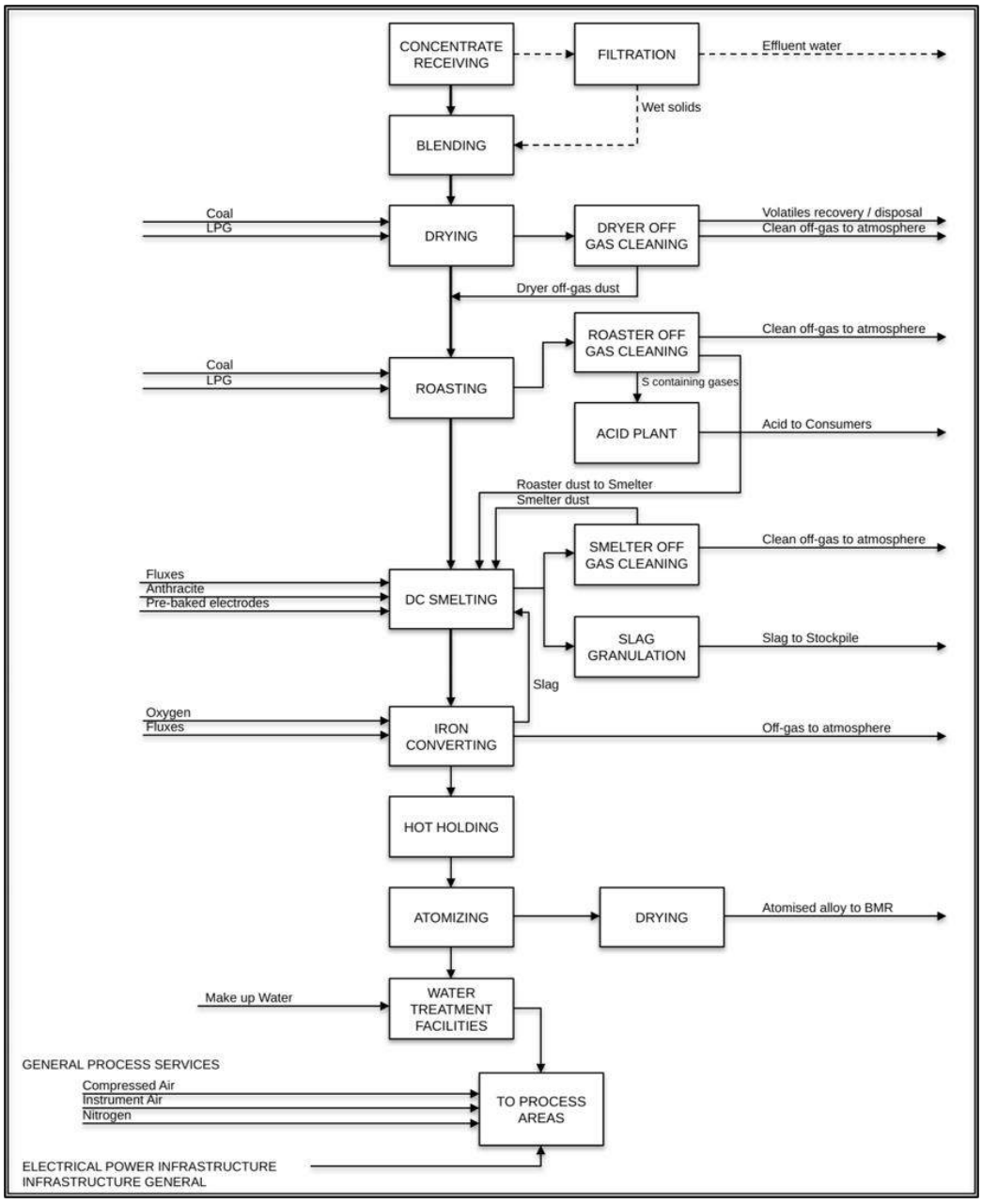


Figure 17-8: Process block flow diagram

Source: Bravo, 2026.

Dried concentrate at 24.9 t/h reports to a bubbling fluidized-bed roaster operating at 1,000 °C under oxidizing conditions. Sulphides are oxidized, and sulphur is released as SO_2, with sulphur in the solids decreasing from 18.7% to 0.37%, a removal of approximately 98%. Arsenic largely volatilizes and reports to the off-gas, where it is removed together with chloride and fluoride in wet gas cleaning. Sulphur oxidation is strongly exothermic, and the roaster operates autogenously at the design feed sulphur grade, with a small energy surplus removed by cooling rather than requiring supplementary fuel. The roaster off-gas passes through a cyclone and a hot

electrostatic precipitator, and the dust is recycled to the roaster feeder. Calcine leaves the roaster at 20.1 t/h.

Calcine is charged hot to a DC electric arc furnace together with anthracite as a reductant and magnesia as a flux. The furnace operates with a single central graphite electrode as the cathode and a conductive hearth anode, at a slag tapping temperature of 1,650 °C and an alloy tapping temperature of 1,600 °C. Carbon addition reduces iron oxide to metallic iron, which collects the precious metals, nickel, copper and cobalt in an alloy phase that separates from the slag by density. The degree of reduction, controlled by the carbon addition, sets both the recovery of the valuable metals and the mass of alloy produced, and is the primary operating lever for the furnace. At a nominal 20 MW rating and a power intensity of 250 kW/m², the furnace has a hearth diameter of 9.5 m and a shell diameter of 10.0 m. Slag and alloy are tapped through separate tap holes, with slag tapped more frequently and alloy less often. Furnace off-gas, predominantly carbon monoxide, is cooled in a radiant duct and cleaned in a hot electrostatic precipitator and a baghouse. The recovered dust, which carries precious metals and nickel, is recycled to the furnace.

Furnace alloy is transferred by ladle to a top-blown converter, where oxygen is blown through the melt to oxidize iron in the presence of a silica flux. A 97% iron removal is assumed, producing a fayalitic slag and a converter alloy enriched in nickel, copper, cobalt and precious metals. Converter slag remains in contact with high-grade alloy and is therefore stockpiled separately from furnace slag and re-treated, either by recycling to the furnace or by milling and flotation, to recover entrained metal.

The converter alloy is held molten in an induction furnace and water-atomized at 100 MPa into a fine powder with a target particle size of 10 to 150 µm. The powder is dewatered, dried to below 1% moisture, screened, and packaged. The fine particle size is selected to give rapid leaching kinetics, allowing a smaller reactor volume and lower stirring power in the downstream hydrometallurgical circuit. The atomized alloy is leached in sulphuric acid to dissolve nickel, copper, cobalt, and residual iron, leaving an enriched precious metal residue that reports to precious metal refining. Base metals are recovered from the leach liquor.

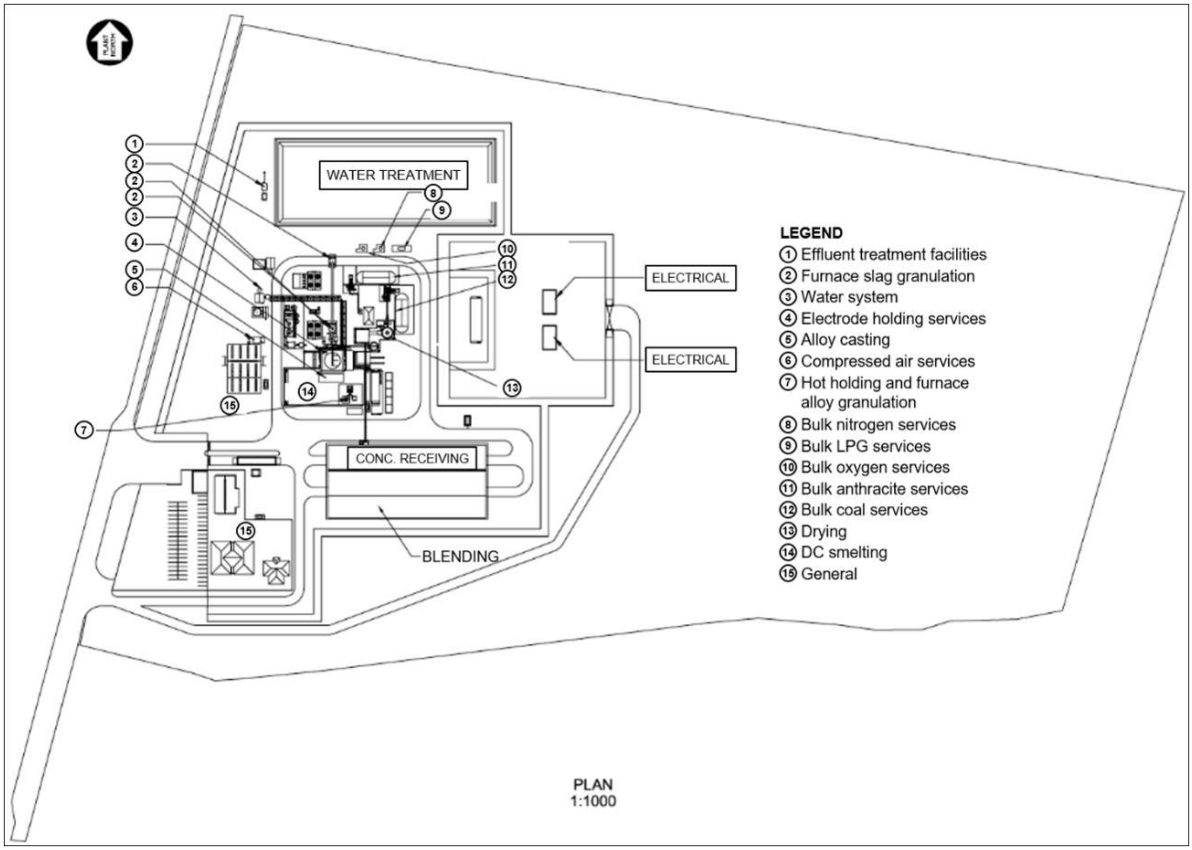


Figure 17-9: Smelter plot plan

Legend: The layout is at a conceptual level and will be optimized in later studies.
Source: Bravo, 2026.

Roaster off-gas is quenched, scrubbed, and passed to a double-contact sulphuric acid plant, which converts SO_2 to 98% sulphuric acid. Tail gas is scrubbed before discharge. Granulated furnace slag is a vitrified, non-leachable product intended for sale in the regional cement and road construction markets, with an on-site repository providing storage to mitigate off-take interruptions.

17.11.6 Metal Deportment and Recovery

Deportment of the valuable metals into the alloy is a function of the extent to which iron oxide in the feed is reduced to the metallic state. The relationship is described by the recovery equation of Jones, Geldenhuys and Reynolds (2009), in which the recovery R of an element is related to the recovery of iron R(Fe) through a coefficient Kγ characteristic of that element:

$$R = (K\gamma \times R(Fe)) / (1 – (1 – K\gamma) \times R(Fe))$$

The furnace is designed to operate at 30% recovery of total iron into the alloy. This operating point balances the recovery of payable metals against the mass of alloy produced, since every additional tonne of iron reduced adds to the alloy that must subsequently be converted and refined. The recoveries adopted for the design are set at or slightly below the values predicted by the recovery equation at that iron recovery (Table 17-10).

Table 17-10: Deportment to furnace alloy

Element	Ky	Predicted recovery (%)	Adopted recovery (%)	Basis
Fe			30	Set point
Pt, Pd, Rh, and Au	184	98.8	98	Recovery equation
Ni	28	92.3	90	Recovery equation
Co	9	79.4	80	Recovery equation
Cu		-	80	Assumed
Cr	0.025	1.1	1	Recovery equation
Si		-	0.5	Assumed

Note: The recovery equation of Jones, Geldenhuys and Reynolds (2009) was applied.
Source: Bravo, 2026.

In these deportments, the furnace slag carries approximately 4 g/t of precious metals, together with 1.18% nickel and 1.14% copper, which represent the balance of base metals not reduced at the selected operating point. Chromium reports almost entirely to the slag as an oxide, a behaviour that allows the route to accept high-chromium feeds without the spinel accretion encountered in conventional matte smelting. Converting is taken to lose no precious metal to the converter slag before that slag is re-treated.

17.11.7 Pyrometallurgical Modelling

The design mass and energy balance was developed using a purpose-built thermochemical process model, which is the source of the stream compositions, energy requirements and equipment sizing reported in this section.

The model is constructed on a species basis rather than an elemental basis. Sulphide speciation in the feed is resolved by first allocating sulphur to nickel, cobalt and copper sulphides, and then apportioning the residual sulphur and non-oxide iron between pyrrhotite and pyrite according to the iron-to-sulphur ratio, using the Fe/S stoichiometric limits of 1.742 for FeS and 0.871 for FeS_2. Each unit operation is then solved as a separate mass balance with its own inlet and outlet temperatures.

The energy balance is thermodynamic rather than empirical. Every species carries a standard enthalpy of formation and a four-term heat capacity polynomial, held in a thermodynamic database within the model. The enthalpy of each stream is evaluated at its actual temperature relative to the pure elements in their standard states at 25 °C. The process energy requirement for each unit is the difference between the total enthalpy of its outlet streams and that of its inlet streams, to which a separately stated vessel heat loss is added. This treatment captures the exothermic contributions from sulphur oxidation in the roaster and from iron oxidation in the converter, and it is the basis on which the roaster is shown to operate autogenously and the converter to sit close to thermal balance.

Deportment in the furnace is calculated using the recovery equation described in Section 17.11.6 rather than fixed recovery factors, so that recoveries respond consistently to changes in the degree of reduction. The model sizes the furnace from the calculated power requirement and a specified power intensity, and it carries supporting calculations for slag density from partial molar

volumes and for the FeO-MgO-SiO_2 phase relations relevant to the produced slag composition. Chlorine, fluorine and arsenic are currently excluded from the mass balance.

The model is a working design tool rather than a validated commercial simulator, and it has not been benchmarked against operating plant data for this feed. Its outputs are consistent with the Mintek campaign data cited in Section 17.11.3 and with the confirmatory Luanga test work, and are appropriate for a Class 4 level of study. Confirmation against larger-scale test work on Luanga concentrate is recommended before advancing the design.

17.11.8 Mass and Energy Balance

Table 17-11 summarizes the overall balance at design throughput. The furnace dominates energy demand, while the roaster carries a small energy surplus.

Table 17-11: Overall mass and energy balance at design throughput

Stage	Mass in (t/h)	Mass out (t/h)	Process energy (MW)	Total including losses (MW)
Drying	26.9	26.9	1.93	2.43
Roasting	29.2	29.2	-0.55	-0.05
Smelting	22.0	22.1	14.70	17.70
Converting	4.5	4.5	Thermally balanced	Thermally balanced
Acid plant	14.0	14.0	-	-

Notes:
1. Drying energy is supplied as natural gas, and smelting energy is electrical.
2. A negative value denotes a net energy surplus.
Source: Bravo, 2026.

Drying evaporates 1.9 t/h of water at a thermal efficiency of 79.5%, consuming approximately 243 Nm^3/h of natural gas. Roasting produces 20.1 t/h of calcine and 25.6 t/h of off-gas carrying 9.1 t/h of SO_2 at approximately 36% by mass, a strength well suited to acid production. Smelting 20.1 t/h of calcine with 0.4 t/h of anthracite and 1.5 t/h of magnesia flux produces 3.9 t/h of alloy and 17.5 t/h of slag, with a specific energy requirement of 0.731 MWh/t of calcine and an electrical thermal efficiency of 83%. Converting 3.9 t/h of furnace alloy with 0.6 t/h of silica flux produces 2.9 t/h of converter alloy and 1.9 t/h of converter slag. The acid plant converts the 9.1 t/h SO_2 stream to 12.7 t/h of 98% sulphuric acid. The estimated product composition is presented in Table 17-12.

Table 17-12: Calculated product compositions (mass %)

Component	Furnace alloy	Furnace slag	Converter alloy	Converter slag
Ni	47.25	1.18	64.60	-
Cu	20.18	1.14	27.59	-
Fe	25.69	-	1.05	-
Co	2.35	0.13	3.21	-
S	2.50	-	3.41	-
C	1.44	-	-	-
Si	0.50	-	-	-
Pt, Pd, Rh, and Au	0.091	0.0004	0.124	-
SiO_2	-	47.84	-	33.71
MgO	-	26.86	-	-
FeO	-	17.37	-	66.29
CaO	-	2.76	-	-
Al_2O_3	-	2.53	-	-

Component	Furnace alloy	Furnace slag	Converter alloy	Converter slag
Cr_2O_3	-	0.18	-	-
Rate (t/h)	3.94	17.49	2.88	1.90

Note: Furnace alloy is tapped at 1,600 °C and furnace slag at 1,650 °C.
Source: Bravo, 2026.

17.11.9 Products

The saleable products of the smelter are an atomized nickel-copper-cobalt alloy carrying the precious metals, sulphuric acid, and granulated furnace slag. Table 17-13 shows the annual production at design throughput.

Table 17-13: Annual production at design throughput

Product or contained metal	Quantity	Unit
Atomized converter alloy	21,455	t/a
Contained Pt, Pd and Rh	769,000	oz/a
Contained Au	88,000	oz/a
Contained Ni	13,860	t/a
Contained Cu	5,920	t/a
Contained Co	688	t/a
Sulphuric acid (98%)	94,400	t/a
Granulated furnace slag	130,200	t/a
Converter slag for re-treatment	14,200	t/a

Note: Contained metal is the quantity reported to the atomized alloy, before downstream refining losses, which are addressed separately.
Source: Bravo, 2026.

Copper and cobalt are reported in Table 17-13 as process outputs of the smelter design. Neither is estimated or reported in the Mineral Resource estimate disclosed in this Technical Report, and neither is assigned a value in the economic analysis. The quantities shown follow from the design feed composition and the deportments given in Table 17-10. They do not constitute an estimate of the contained metal in the deposit.

The atomized alloy grades approximately 65% nickel, 28% copper, and 3% cobalt, with a combined platinum, palladium, rhodium, and gold content of approximately 1,240 g/t. It is suitable as feed for base metal leaching and precious metal refining. Furnace slag, a vitrified silicate with a residual precious metal content of approximately 4 g/t, is produced at a rate of approximately 10,850 t per month.

17.11.10 Energy, Water and Consumables

Electrical supply is taken from the Vila do Conde substation, approximately 1.5 km from the site, via a 138 kV transmission line and a 40 MVA transformer. Grid power in the region is predominantly hydroelectric, and the smelter is expected to operate on a substantially renewable supply. Natural gas is available from the liquefied natural gas infrastructure established at Barcarena. Water is drawn from a cooling-water reservoir fed by groundwater and returned process water, with the Pará River, approximately 4 km from the site, as an alternative source.

The energy, water and principal consumables are presented in Table 17-14.

Table 17-14: Energy, water and principal consumables

Item	Requirement	Unit
Installed electrical load	30.05	MW
Average electrical demand	27.20	MW
Annual electrical consumption	202.6	GWh/a
Raw water make-up	220	m³/h
Circulating water duty	720	m³/h
Natural gas	2.5	million Nm³/a
Oxygen, converting, tapping and lancing	1.6	million Nm³/a
Anthracite reductant	2,980	t/a
Magnesia flux	11,170	t/a
Silica flux	4,470	t/a
Lime for gas scrubbing	4,000	t/a
Graphite electrodes	1.3-1.5	kg/t feed

Source: Bravo, 2026.

Oxygen for roasting and for the acid plant is supplied as air. Purchased oxygen is required only for converting, tapping, and lancing.

17.11.11 Key Performance Criteria

Table 17-15 shows the performance criteria adopted as the design basis for the smelter, and carried into the economic analysis.

Table 17-15: Design performance criteria

Criterion	Design basis
Recovery of Pt, Pd, Rh and Au to alloy	98%
Recovery of Ni to alloy	90%
Recovery of Cu to alloy	80%
Recovery of Co to alloy	80%
Sulphur removal in roasting	98%
Sulphur capture as sulphuric acid	91% of SO_2 to acid
Residual precious metals in furnace slag	Approximately 4 g/t
Plant availability	85%

Source: Bravo, 2026.

These recoveries apply only to the smelter and are in addition to the concentrator recoveries stated in the preceding sections. Overall project recovery from ore to atomized alloy is the product of the two and is presented in Section 22, together with the assumptions applied to downstream refining.

18 PROJECT INFRASTRUCTURE

The Project occupies an area of approximately 5,000 ha, comprising at least the following infrastructure and on-site facilities:

- Open Pit
- Site access and haul roads
- Waste Dump
- Marginal Stockpile
- Oxide Stockpile
- Temporary ROM Stockpile
- Sediment ponds
- Power Supply System
- Civil and Maintenance Facilities
    - Truck shop
    - Plant workshop
    - Warehouse
    - Change House
- Tailings Storage Facility
- Water Storage Facility
- Administrative Buildings
    - Main Office
    - Cafeteria
    - Training Center
    - Control Room
    - Fire Protection System
    - Effluent Treatment
- Medical Clinic and Fire Brigade
- Communication System
- Laboratory
- Explosives Magazine

Figure 18-1 shows the Project's master plan.

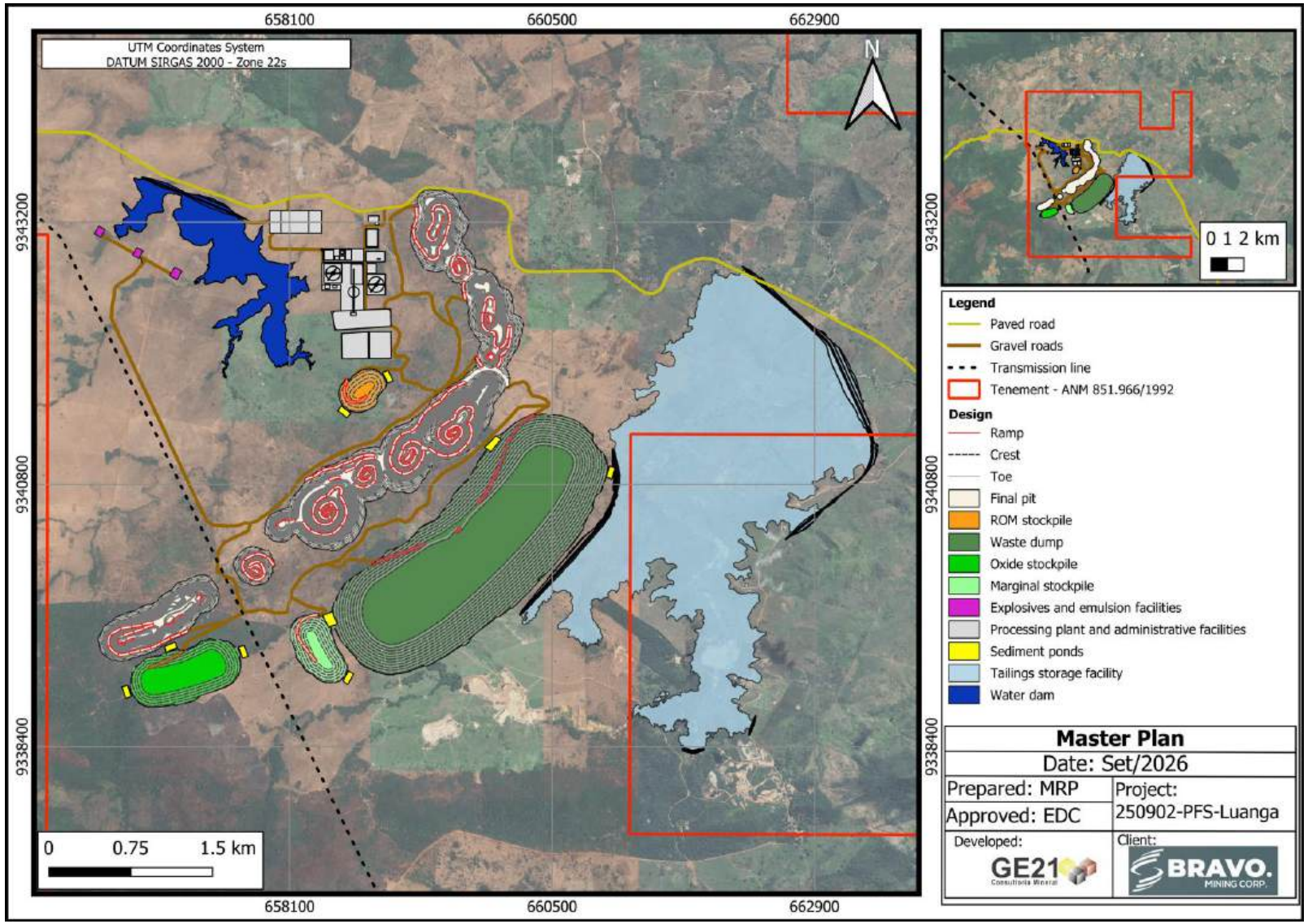


Figure 18-1: Master plan

Source: GE21, 2026.

18.1 Roads

A contractor would be hired to construct internal access roads and improve traffic conditions on the existing unpaved road section (i.e., widen, straighten, improve drainage, lay and compact a rock base formation suitable for heavy traffic, and add vertical signage). The construction and maintenance of the site access roads would follow a detailed designed plan that would consider minimum width, curve radius, and ramps required for safe traffic. A total of 24.4 km of internal access roads would be developed over the mine life, comprising both upgrade to existing roads and the construction of new sections.

The mine access transport profiles were estimated to optimize haulage distances between mining areas, the waste dump, marginal and oxide stockpiles, the temporary ROM stockpile, and the processing plant. The centerlines were estimated with a maximum gradient of 10%.

The road designs would be developed in the next phases of the project, in compliance with operational safety and equipment technical standards, following the recommended criteria below:

- Minimum curve radius: 30 m.
- Road width:
  - One-way traffic: 2 times the width of the largest vehicle.
  - Two-way traffic: 3 times the width of the largest vehicle.
- Safety windrows: minimum height equal to half the diameter of the largest haul truck's tire.
- Drainage channels at the toe of the benches along the accesses to collect and convey surface runoff and protect the road platform.
- Cross slope: a 2% transverse gradient across the running surface to promote surface drainage and direct runoff toward the drainage channels, minimizing water ponding on the road platform.
- Water trucks operating continuously, 24 hours per day, for dust suppression on active haul roads, ensuring safe and environmentally responsible operations.
- Road construction and maintenance are supported by bulldozers, motor graders, small hydraulic excavators, water trucks, and compactor rollers.

Figure 18-2 presents a typical haul road cross-section.

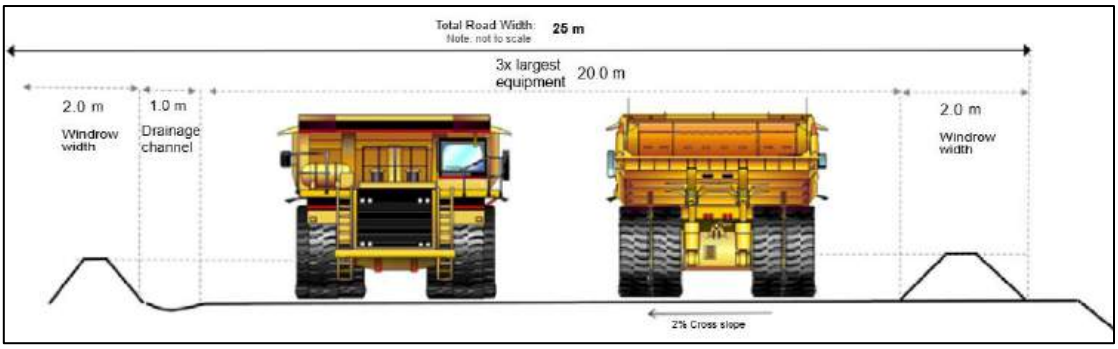


Figure 18-2: Typical haul road cross-section

Source: GMS, 2021 (adapted by GE21, 2026).

All signage and traffic control measures must follow Brazilian traffic legislation and Mining Regulatory Standards (NRM). Emphasis will be placed on ensuring adequate visibility, clear signage, and proper delineation of operational routes to promote the safety of personnel and equipment.

18.2 Power Supply

The electrical system planned for the Project consists of:

- Main Substation – receiving a 230 kV supply from the grid operated under the coordination of the National Electric System Operator (ONS), the entity responsible for coordinating and controlling the operation of power transmission facilities in the National Interconnected System (SIN) and for planning the operation of the country's isolated systems, under the oversight and regulation of the National Electric Energy Agency (ANEEL).
- A double radial circuit with two 60 MVA, 230/13.8 kV oil-immersed transformers, each operating at 50% load, with the 13.8 kV bus-tie circuit breaker normally open.
- Should one of the 60 MVA transformers fail, the bus-tie circuit breaker will be closed, and the remaining transformer would take 100% of the load until the failed unit is restored.

Three options for connection to the SIN were evaluated, considering overall cost, electrical losses and regulatory complexity. The selected option is a 230 kV connection to the Carajás Substation via a dedicated transmission line of approximately 34 km, with capacity to meet the Project's long-term demand.

The main substation would be built adjacent to the process plant and designed to the following technical specifications:

- Rated capacity of 60 MVA.
- Step-down from 230 kV to 13.8 kV for internal plant distribution.
- Distribution from the motor control centers (MCCs) to the consumption points via overhead lines and cable galleries, equipped with protection, control and power factor correction systems (target 0.92).
- Diesel generator set to supply critical loads in an emergency.

18.3 Civil and Maintenance Facilities

18.3.1 Truck shop

An electromechanical workshop is planned to service the fleet of trucks and equipment used in the pit and for ore haulage to the process plant. The building would have a steel structure with metal roof sheeting, side cladding in masonry and metal sheeting, and a reinforced concrete floor.

The service bays would be aligned side by side and served by an overhead crane. In addition to the maintenance bays for machinery and equipment, the facility would include areas for lubrication, washing, tire service, and covered, protected areas for the storage of used oil and lubricants.

The workshop would also include areas for tire storage, component welding, electromechanical sub-assembly rooms, a tool room, and rooms for operating personnel. The entire complex would be surrounded by drainage channels and oil–water separator boxes.

18.3.2 Plant workshop

The plant workshop would be fully roofed and enclosed on the sides with masonry. It would support the maintenance of plant equipment and would be fitted with machining, cutting, welding and painting equipment, among others, as well as equipment for horizontal handling and lifting of parts and materials.

The plant workshop layout is presented in Figure 18-3.

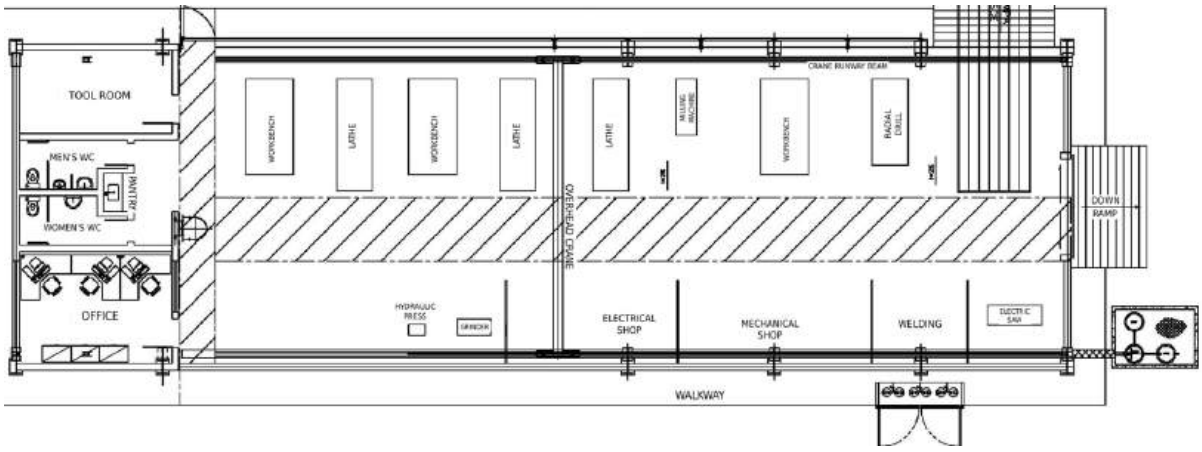


Figure 18-3: Plant workshop layout

Source: Onix, 2026.

18.3.3 Warehouse

The warehouse would serve the entire industrial plant. It would consist of a steel-structure shed with side enclosure in exposed concrete block masonry with tooled joints and a latex paint finish on the exterior and interior. The shed would be fitted with shelving for the storage of spare parts, equipment and materials. The roof would be trapezoidal metal sheeting on exposed steel

purlin framing, without a ceiling. Window and door frames would be galvanized steel with an enamel paint finish and clear glass.

Thresholds and window sills in precast concrete. Foundations on reinforced concrete shallow spread footings. An uncovered area adjacent to the enclosed section is also planned for the storage of larger items, enclosed by metal fencing and gates. Transport, loading and unloading equipment would have easy access to the warehouse premises.

18.3.4 Change House

The Project includes a 276 m² building in masonry with a concrete structure, located at the process plant. The layout includes a men's change room with 220 lockers and sanitary facilities for 60 people per shift, and a women's change room with 90 lockers and sanitary facilities for 30 people per shift. The change house is intended to serve the industrial plant personnel.

Walls in exposed concrete block masonry with tooled joints and a latex paint finish on the exterior. Interior walls tiled up to the ceiling. Interior floors in heavy-traffic ceramic tile and external walkways in float-finished concrete. Roofing in trapezoidal metal sheeting on steel purlin framing, without a concrete slab, with a PVC ceiling only.

Restrooms with granite countertops, terrazzo partitions and aluminum door frames (h = 210 cm). Internal toilet doors in marine-grade MDF. Window and door frames in galvanized steel with an enamel paint finish, obscured glass in the restrooms and clear glass elsewhere. Foundations on shallow spread footings, with reinforced concrete columns and beams.

The change house, sized to serve all Industrial Plant, Maintenance and Mine personnel, would be built in an easily accessible location close to the gatehouse.

18.4 Mine Drainage and Dewatering

The open-pit perimeters and their upper benches would have drainage channels to direct water to the collection system. Runoff and groundwater will be retained in sumps and pumped out of the pit. The main pumping system would be installed at the bottom of the pit to operate during periods of high rainfall or high water percolation. When possible, water would be diverted to sumps and pumped for reuse. Excess water would be discharged to the natural drainage system once it meets the applicable environmental discharge standards.

The detailed drainage and dewatering designs would be developed in the next phases and properly sized based on hydrogeological and hydrological studies. Fine-sedimentation sumps would be constructed within the mine area to capture fine materials generated during mining activities, particularly from drainage and run-off. They would be strategically located to follow the natural topography, reducing earthworks and environmental disturbance. The sumps would contribute to erosion control, water management, and regulatory compliance, and their integrity would be monitored through standard controls.

18.5 Waste Dump, Marginal, Oxide and ROM Stockpiles

The waste dump and marginal and oxide stockpiles are dimensioned to receive materials from the mining areas throughout the Life of Mine (LOM). A temporary ROM stockpile is also considered to stock ROM during the operational phase and would be reclaimed in the last year of operation.

A 5% contingency factor is considered in volume requirement estimates to account for uncertainties surrounding material features and operational issues.

Drainage channels would be constructed along the perimeters of those facilities to collect rainwater and carried sediments, reducing their impact on local vegetation and terrain. The channels would direct the water to settling ponds located at the bottom of the slope. When possible, the water would be pumped for reuse.

Section 16 of this Report presents the geotechnical and design parameters. The corresponding designs are presented in Figure 18-4 to Figure 18-7.

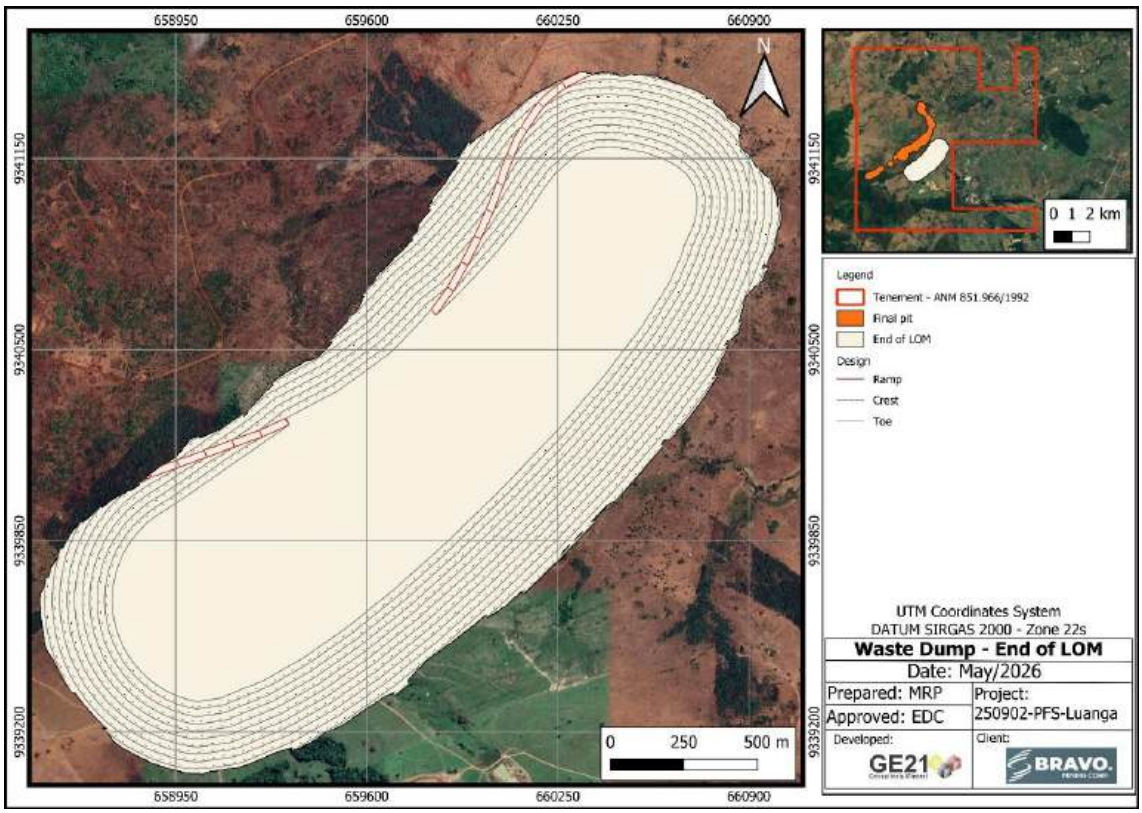


Figure 18-4: Waste dump design

Source: GE21, 2026.

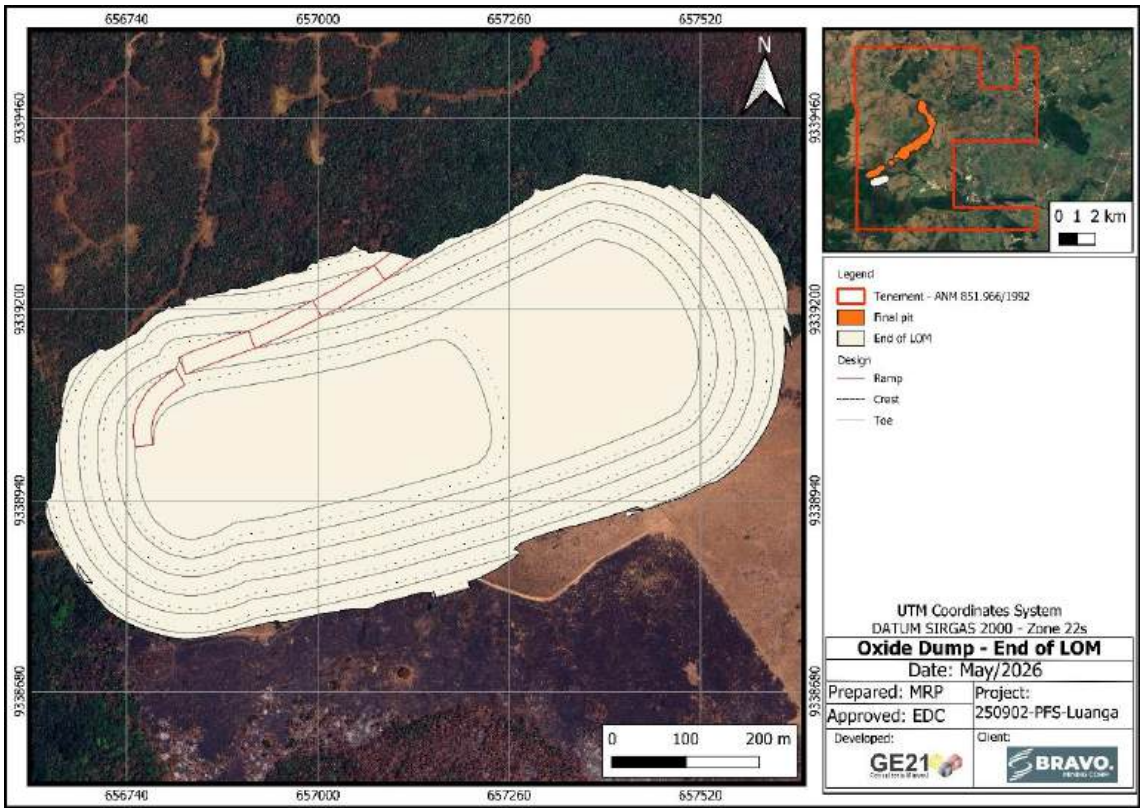


Figure 18-5: Oxide stockpile design

Source: GE21, 2026.

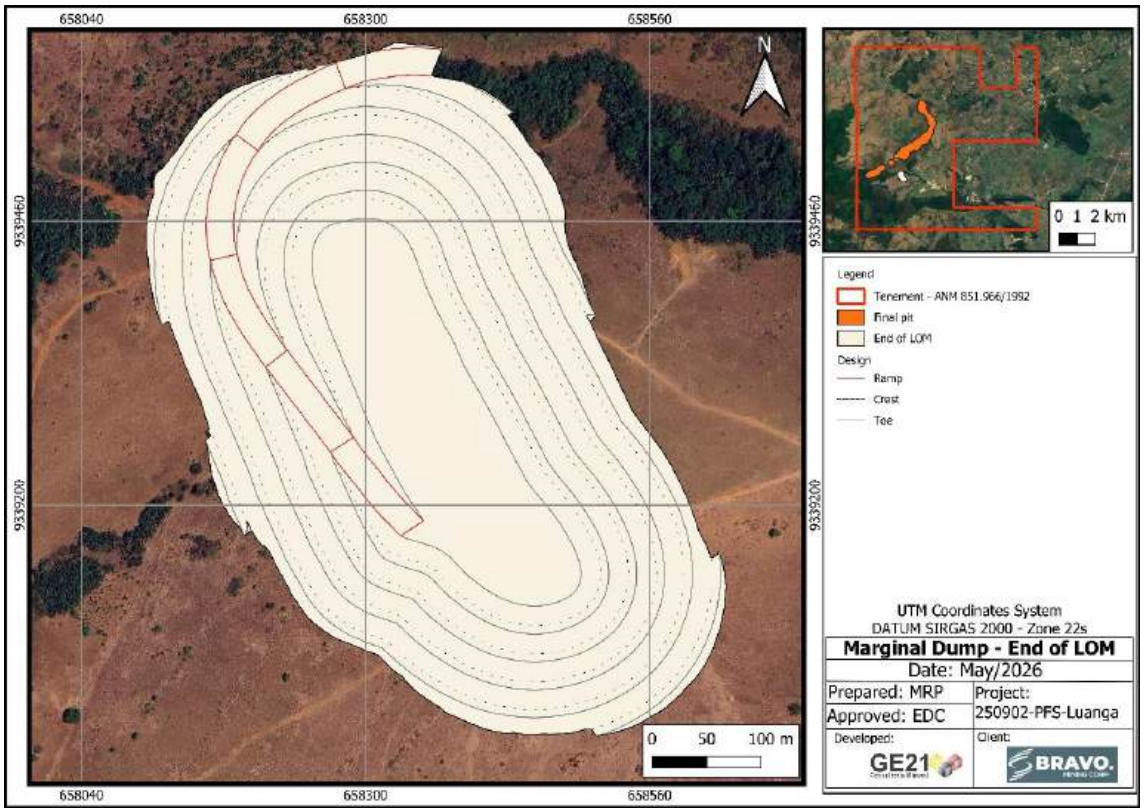


Figure 18-6: Marginal stockpile design

Source: GE21, 2026.

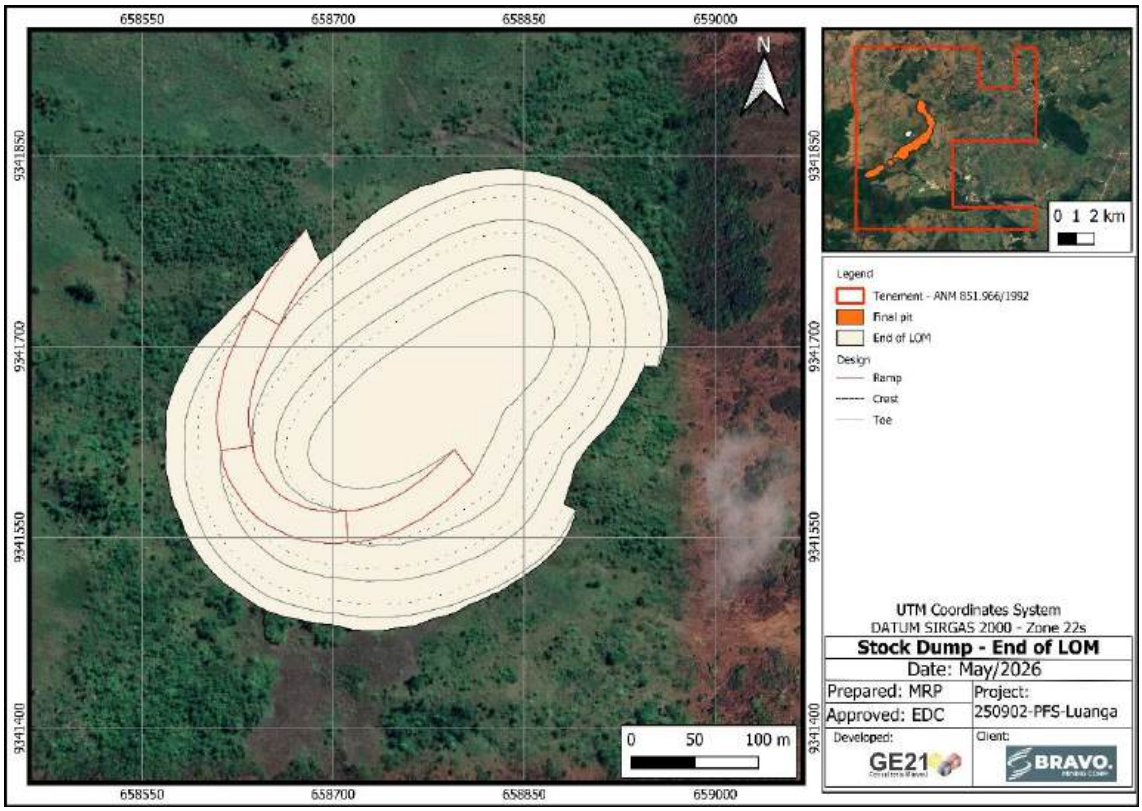


Figure 18-7: ROM stockpile design

Source: GE21, 2026.

18.6 Tailings and Water Storage Facilities

The tailings and water storage systems have been developed by ALB Engenharia. The work comprised the preliminary sizing of the main structures, preliminary quantity estimates, water and mass balances, and the construction schedule. The system consists of three structures:

- Tailings Storage Facility (TSF) B8: main embankment, constructed in four stages.
- B8A Embankment: auxiliary containment structure between the B8 impoundment and the waste rock dump, with no impoundment of its own.
- B1 Water Storage Dam: independent non-contact water reservoir that regulates the integrated site water balance.

The concept considered siting, geotechnical, hydrological, environmental, operational, constructability and economic aspects, with structural safety as the overriding criterion. ANM requirements, applicable standards of the Brazilian Association of Technical Standards (ABNT), and the International Commission on Large Dams (ICOLD) and the Global Industry Standard on Tailings Management (GISTM) good practice were also taken into account.

18.6.1 Basis of Design and Assumptions

The estimates for the three structures are based on a LIDAR topographic survey, from which the elevation–area–capacity curves and earthworks estimates were derived, and on

existing hydrological data (GHT, 2024). The hydrological and operating assumptions are presented in Table 18-1.

Table 18-1: Hydrological and operating assumptions

Parameter	Value
Plant operating hours	8,060 h/year (92% availability)
Mean annual precipitation	1,476 mm
Mean annual evaporation	1,841 mm
Runoff coefficient	0.255 (land catchment); 1.0 (pond)
Seepage	Assumed negligible

Source: ALB, 2026.

The hydrological data and runoff coefficient are preliminary and should be confirmed in subsequent engineering phases.

18.6.2 Design Criteria

All three structures are conceived as compacted earthfill embankments, based on the following conceptual design criteria:

- Downstream construction methodology.
- Geometry: 2.5H:1V slopes, 6.0 m wide berms at 10.0 m vertical intervals and 8.0 m wide crests, with safety windrows.
- Foundation: average excavation depth of 2.5 m, based on regional experience, in the absence of site-specific geotechnical investigation.
- Internal drainage: 0.80 m wide sand chimney filter, extended up to the pond water level; gravel blanket drain enclosed by two 0.20 m sand filter layers; flow measurement box to monitor seepage flow and water quality.
- Compaction: 98% of Standard Proctor maximum dry density, with moisture content within ±2% of optimum.
- Surface drainage and erosion protection: sod cover on slopes and berms, half-round concrete channels, collection boxes and HDPE downchutes.
- Spillway: uncontrolled concrete overflow spillway (5.0 m × 2.0 m, 10 m long) discharging into a rock-cut chute of variable length for each structure. A preliminary check for a 1,170 mm 30-day rainfall event indicated a flow depth of approximately 0.57 m. A freeboard of 2.0 m was adopted.
- Instrumentation: piezometers, water level indicators, survey benchmarks and displacement monuments, to be detailed in subsequent engineering phases.

No stability analyses were carried out at this stage. A minimum factor of safety of 1.5 for long-term drained conditions, in accordance with ABNT NBR 13028, was adopted as a design assumption. This assumption is to be verified by specific analyses supported by geotechnical investigation.

18.6.3 Tailings Storage Facility

Eight alternative sites were assessed for tailings storage, as summarised in Table 18-2.

Table 18-2: Tailings storage alternatives

Alternative	Estimated Volume	Status
B2 to B5	8 to 25 Mm³	Discarded: would require multiple structures and interfere with downstream areas
B6	64 Mm³	Discarded: interference with a downstream manganese project
B7	23 Mm³	Discarded: interference with a manganese project and a watercourse
B8	**62 Mm³ (≈ 95 Mt)**	**Selected; located southeast of the plant**
B9A / B9B	14 / 8.8 Mm³ (≈ 22 / 13 Mt)	Assessed; located north of the plant

Source: ALB, 2026.

Alternative B8, shown in Figure 18-8 and Figure 18-9, was selected because it provides capacity for the life-of-mine tailings production in a single structure and has the lowest estimated unit costs.

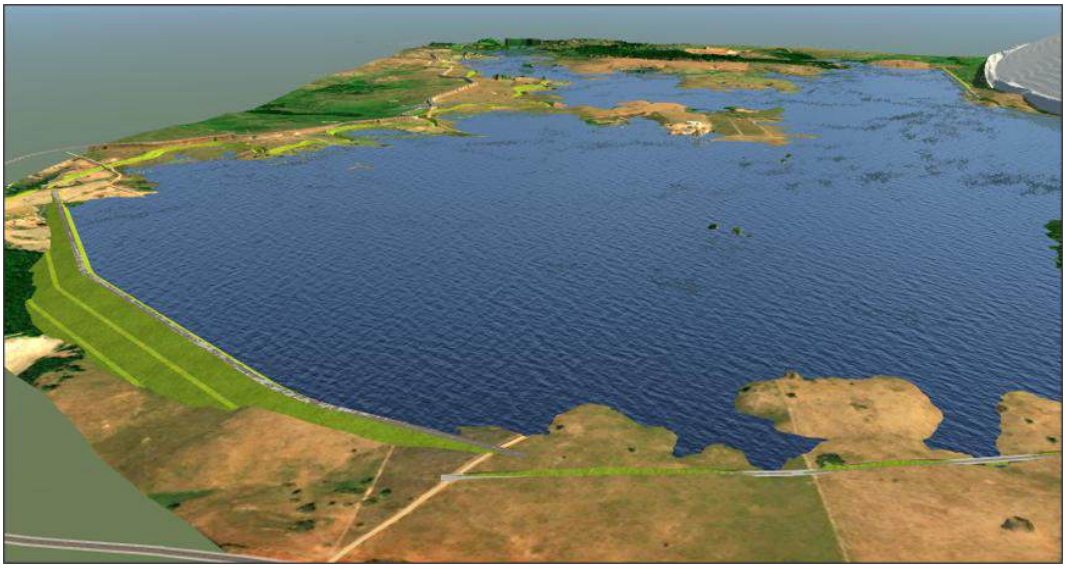


Figure 18-8: 3D view of the B8 TSF

Source: ALB, 2026.

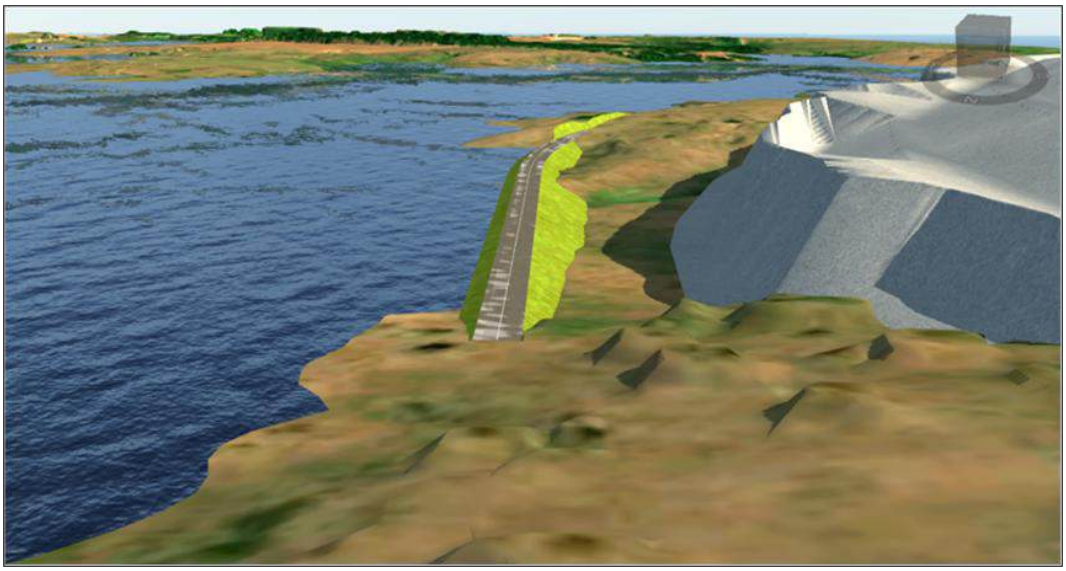


Figure 18-9: 3D view of the B8A embankment

Source: ALB, 2026.

The tailings parameters were provided by process engineering and are summarised in Table 18-3.

Table 18-3: Tailings production and deposition assumptions

Parameter	Value
Tailings production	95 Mt
Slurry flow rate	2,624 t/h (50% solids), equivalent to 1,751 m³/h
Solids specific gravity	2.99 t/m³
Void ratio (assumed)	1.0
Estimated settled tailings volume	≈ 7.07 Mm³/year

Source: ALB, 2026.

The B8 and B8A embankments would be raised concurrently in four stages using the downstream method, with matching crest elevations. The B8A embankment, with an estimated fill volume of approximately 0.7 Mm³, was designed to the same criteria as a conventional dam and serves to protect the toe of the waste rock dump from the B8 impoundment. A service life of 2 years was adopted for each intermediate stage and 4 years for the final stage.

The TSF stage geometry and quantities summary is presented in Table 18-4.

Table 18-4: TSF geometry and estimated quantities

Structure	Crest (m)	Base / Pond Bottom (m)	Height (m)	Crest Length (m)	Compacted Fill (m³)
B8 – Stage 1	164.5	146.0	18.5	840	381,427
B8 – Stage 2	169.5	164.5	5.0	1,040	295,896
B8 – Stage 3	173.0	169.5	3.5	2,760	364,467
B8 – Stage 4	176.0	173.0	3.0	3,300	743,774
B8 – Total	176.0	146.0	30.0	3,300	1,785,564
B8A – Stage 1	164.5	161.0	3.5	1,000	141,609
B8A – Stage 2	169.5	164.5	5.0	1,450	119,010
B8A – Stage 3	173.0	169.5	3.5	1,760	146,589
B8A – Stage 4	176.0	173.0	3.0	1,820	292,261
B8A – Total	176.0	161.0	15.0	1,820	699,470

Source: ALB, 2026.

The estimated cumulative storage capacity of B8 is approximately 22.6 Mt at Stage 1, 46.1 Mt at Stage 2, 68.9 Mt at Stage 3 and 92.9 Mt at Stage 4.

Stage 1 of B8/B8A would be built in Year 0, during project construction. Subsequent raises would be built in Years 2, 4 and 6, each preceded by its respective engineering, and would enter operation in the year following construction. Stage 4 would operate from Year 7 to Year 10.

18.6.4 Water Storage Dam

Alternative B1, presented in Figure 18-10, was retained from previous studies. Its purpose is to store water for the project, acting as the regulating element of the integrated site water balance. The reservoir is independent of the process plant and receives no effluent from the plant or the TSF.

The estimated annual regulated yield of B1 is approximately 7.3 Mm³, equivalent to approximately 898 m³/h.

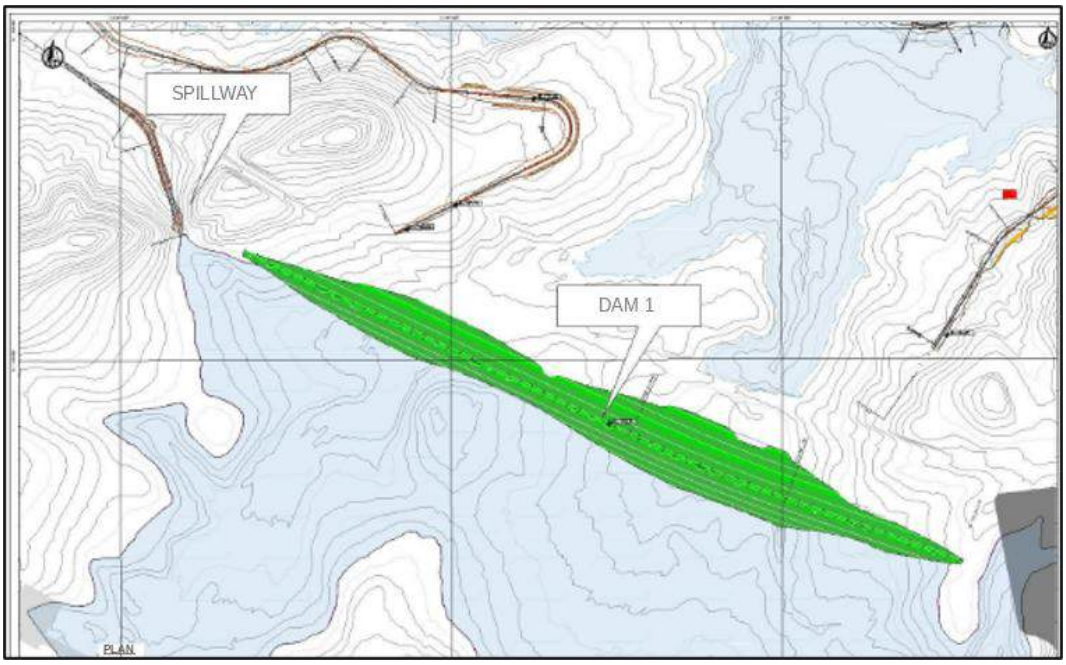


Figure 18-10: B1 water storage dam

Source: ALB, 2026.

B1 estimated geometry is presented in Table 18-5.

Table 18-5: B1 estimated geometry

Structure	Crest (m)	Pond Bottom (m)	Height (m)	Crest Length (m)	Compacted Fill (m³)
B1 – Water	190.0	167.0	23.0	1,000	To be detailed

Source: ALB, 2026.

B1 would be built in a single stage in Year 0, prior to the start of operations. Spillway overflows are non-contact water that may be discharged to the environment, subject to the applicable permits.

18.6.5 Integrated Water Balance

Water balances were estimated for each B8 stage, accounting for slurry water, direct precipitation on the impoundment, runoff from the contributing catchment, and evaporation losses. The operating principle is to keep the volume of free water in the tailings impoundment to a minimum. Pumping prioritises reclaim water from B8, with B1 supplying the balance of plant demand. Catchment runoff into B8 is controlled by a perimeter diversion channel.

For Stage 1, the recoverable free water volume in B8 was estimated at approximately 7.9 Mm³/year, against a plant water demand of approximately 10.6 Mm³/year. The resulting deficit, of approximately 2.7 Mm³/year, is supplied by B1.

The make-up water required from B1 remains below its estimated regulated yield in all stages. The preliminary balances indicate that plant demand can be met solely by rainfall runoff from the structures' catchments, with no abstraction from external water sources. The design storm, void ratio and runoff coefficient are to be refined in subsequent engineering phases.

18.6.6 Regulatory Framework

The structures are subject to the National Dam Safety Policy (Federal Law No. 12,334/2010, as amended by Law No. 14,066/2020), National Water Resources Council (CNRH) Resolution No. 241/2024, and the environmental licensing requirements of the Pará State Secretariat for Environment and Sustainability (SEMAS/PA), which is responsible for environmental licensing and control.

- TSF (B8 and B8A): dam safety oversight lies with the National Mining Agency (ANM). ANM Resolution No. 95/2022 is currently in force; ANM Resolution No. 220/2025 will take effect on 2 August 2027, repealing ANM Resolution No. 95/2022. ABNT NBR 13028:2024 also applies. The downstream raising method is consistent with the legal prohibition of upstream-raised tailings dams.
- Water storage dam (B1): as a water storage dam, safety oversight lies, under Law No. 12,334/2010, with the authority that granted the water use rights. Construction is subject to the corresponding water use grant (outorga), which must also cover the discharge of excess water.

18.6.7 Geotechnical Risks

A preliminary geotechnical risk matrix was prepared for the structures, identifying the following potential failure modes: foundation failure, slope instability, internal erosion (piping), uplift pressure, differential settlement, seepage through the foundation and abutments, overtopping, construction quality control deficiencies, and discontinuities in the foundation rock mass. For the TSF, static or seismic liquefaction of the tailings is also included. For each risk, the matrix identifies warning signs, associated investigations and instrumentation, and preventive measures.

The main gap at this stage is the absence of geotechnical investigation at the structure sites. This campaign should include drilling, characterisation and strength testing of borrow materials, permeability testing, and testing of the tailings, to support stability, seepage and liquefaction analyses in the next engineering phase.

18.7 Water Management

To meet the potable water demand, a Water Supply System (WSS) would be installed, comprising abstraction, treatment, storage and distribution. The main supply source would be water wells to be drilled within the Project area, complemented by storage systems to ensure a reliable supply.

Based on an estimated peak workforce of approximately 1,200 workers, including direct employees and contractors, and an average consumption of 150 liters per person per day, the maximum potable water demand was estimated at approximately 180 m³/day. The treatment system would be sized to fully meet this demand, allowing for operational safety factors and possible seasonal variations in consumption.

The Water Treatment Plant (WTP) would be designed to produce water in compliance with the potability standards set by current health legislation, including the guidelines of the Brazilian Ministry of Health. After treatment, the water would be stored in appropriate reservoirs and distributed to the operational, administrative and support areas of the Project.

The system would include periodic water quality monitoring, covering physical, chemical and microbiological parameters, to ensure water remains fit for human consumption throughout the life of the Project. Operating and maintenance procedures would also be established to ensure continuity of supply and the sanitary safety of users.

18.8 Administrative Buildings

An area close to the main gate and to the Project facilities would be designated for the administrative buildings.

18.8.1 Main Office

The main office conceptual design comprises a 754 m² building in masonry with a concrete structure. The layout includes 70 workstations, a reception area, individual meeting rooms and manager offices, restrooms and a pantry, as presented in Figure 18-11.

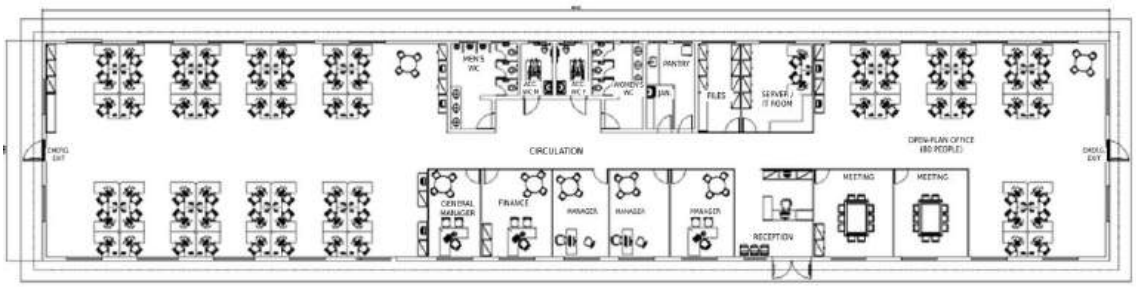


Figure 18-11: Main office layout

Source: Onix, 2026.

The building would house the offices for administrative, technical support and management activities directly linked to the production areas. It would consist of a single block, plus a dedicated space for telecommunications and IT equipment.

Walls in exposed concrete block masonry with tooled joints and a latex paint finish on the exterior and interior, except for the restrooms and pantry, which would be tiled up to the ceiling. Interior floors in heavy-traffic ceramic tile and external walkways in float-finished concrete. Roofing in trapezoidal metal sheeting on steel purlin framing, without a concrete slab, with a PVC ceiling only.

Restrooms with granite countertops, terrazzo partitions and aluminum door frames (h = 210 cm). Internal toilet doors in marine-grade MDF. Window and door frames in galvanized steel with an enamel paint finish, obscured glass in the restrooms and clear glass elsewhere. Thresholds and window sills in granite. Partitions in MDF and tempered glass. Foundations on shallow spread footings, with reinforced concrete columns and beams.

18.8.2 Cafeteria

A kitchen and central restaurant would be built in a location easily accessible to all employees. The restaurant kitchen would serve all shift and administrative personnel as well as occasional visitors (suppliers, visitors, temporary service providers).

The layout includes a dining hall for 120 people, a full kitchen, a nutritionist's office, cold rooms, storerooms, change rooms for kitchen staff, restrooms for dining hall users and verandas for external circulation.

The dining hall would accommodate all shift and administrative personnel as well as occasional visitors. Walls in exposed concrete block masonry with tooled joints and a latex paint finish on the exterior. Interior walls tiled up to the ceiling. Kitchen floor in extruded ceramic tile and remaining areas in heavy-traffic ceramic tile. Cold rooms with thermal insulation on floor, walls and ceiling. Verandas and circulation areas in smooth cement flooring and walkways in float-finished concrete.

Roofing in trapezoidal metal sheeting on steel purlin framing, without a concrete slab, with a PVC ceiling only, over the dining hall, change rooms, restrooms and verandas. The kitchen and cold rooms would have a roof over a concrete slab and a PVC ceiling.

Change rooms and restrooms with granite countertops, terrazzo partitions and aluminum door frames (h = 210 cm). Internal toilet doors in marine-grade MDF. Kitchen countertops in stainless steel. Partitions in MDF and tempered glass, where applicable.

Window and door frames in galvanized steel with an enamel paint finish, obscured glass in the change rooms and restrooms and clear glass elsewhere. Cold rooms would be fitted with purpose-built cold storage doors. Thresholds and window sills in granite. Foundations on shallow spread footings, with reinforced concrete columns and beams.

18.8.3 Training Center

The layout includes a room with capacity for 40 people for operational and administrative staff training and for work meetings.

Walls in exposed concrete block masonry with tooled joints and a latex paint finish on the exterior and interior, except for the restrooms and pantry, which would be tiled up to the ceiling. Interior floors in heavy-traffic ceramic tile and external walkways in float-finished concrete.

Roofing in trapezoidal metal sheeting on steel purlin framing, without a concrete slab, with a PVC ceiling only.

Window and door frames in galvanized steel with an enamel paint finish and clear glass. Thresholds and window sills in granite. Partitions in MDF and tempered glass. Foundations on shallow spread footings, with reinforced concrete columns and beams.

18.8.4 Control Room

The control room would have walls in exposed concrete block masonry with tooled joints and a latex paint finish on the exterior and interior, except for the restrooms and certain rooms (as required by applicable standards), which would be tiled up to the ceiling. Interior floors in heavy-traffic ceramic tile and exterior areas in float-finished concrete.

Roofing in trapezoidal metal sheeting on steel purlin framing, with a concrete slab over the rooms and restrooms.

18.8.5 Fire Protection System

The fire water tank would be sized to provide a minimum autonomy of 30 minutes supplying two hydrants simultaneously, as required by Brazilian standard NBR 13714, allowing time for other response measures to be taken. The fire water network would extend over the entire industrial area, the administrative buildings and the explosives magazine area.

The fire water tank would be fitted with a level switch interlocked directly with the control circuit of its feed pump, ensuring automatic start-up of the pump and maintenance of a minimum operating level.

18.8.6 Effluent Treatment

The effluent treatment system comprises the infrastructure, equipment and services intended to collect and treat the wastewater generated, thereby preventing the spread of disease and the contamination of water bodies following discharge to the environment.

The following treatment systems would be installed:

- Sanitary Sewage;
- Industrial Effluents;
- Solid Waste.

18.9 Medical Clinic and Fire Brigade

The layout includes a clinic with individual rooms for reception, treatment and rest, restrooms and an ambulance shelter. It also includes rooms for fire brigade members, a PPE room and a shelter for the fire brigade vehicle. The clinic would be equipped to provide first aid and to perform the medical examinations required by labor legislation. Serious cases would be transferred by ambulance to hospitals in nearby towns.

Walls in exposed concrete block masonry with tooled joints and a latex paint finish on the exterior and interior, except for the restrooms and certain rooms (as required by applicable standards), which would be tiled up to the ceiling. Interior floors in heavy-traffic ceramic tile and exterior areas in float-finished concrete.

Roofing in trapezoidal metal sheeting on steel purlin framing, without a concrete slab, with a PVC ceiling only in the rooms and restrooms, and exposed roofing over the vehicle shelters. Window and door frames in galvanized steel with an enamel paint finish, obscured glass in the restrooms and clear glass elsewhere.

Restrooms with granite countertops, terrazzo partitions and aluminum door frames (h = 210 cm). Internal toilet doors in marine-grade MDF.

The medical clinic and fire brigade loyout is presented in Figure 18-12.

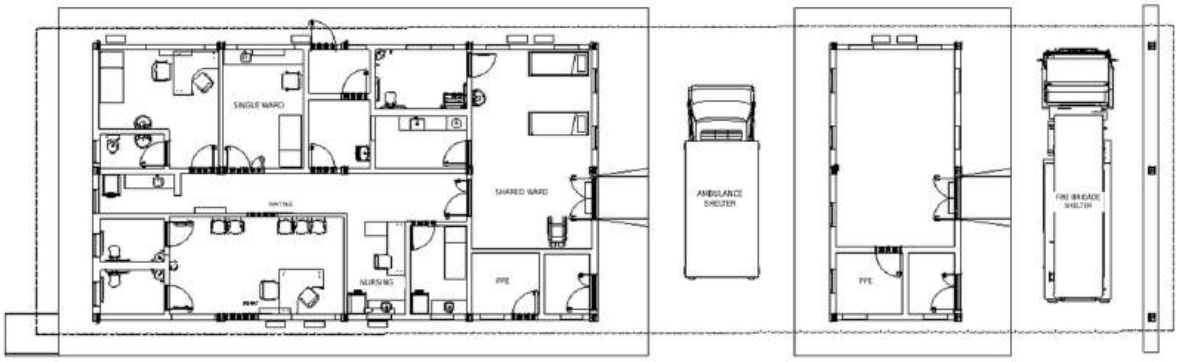


Figure 18-12: Medical and fire brigade layout

Source: Onix, 2026.

18.10 Communication System

The communication system would comprise:

- Data Network Subsystem;
- Telephony Subsystem;
- Emergency Telephony;
- CCTV – Operational Security;
- Access Control;
- Communication network for electrical equipment, such as circuit breakers and intelligent relays;
- Communication network for automation equipment;
- Equipment and personnel monitoring;

System integration would be via redundant fiber-optic cable.

Communication and control hardware would be distributed, and the network would be expanded as development progresses. Radios would be hardwired into mobile equipment. Handheld radios would 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 would be supported by a hybrid network architecture (i.e., optical fibre for the backbone and copper twisted-pair cable for the local network). Single-mode fibre-optic cable would be used for low-loss and high-bandwidth optical systems. This type of cable is capable of continuous

operation with good performance in environments with high levels of suspended dust, moisture, and noise.

The mining operation would be supported by dedicated software platforms covering fleet management and equipment dispatch, drill-and-blast planning, grade control and ore reconciliation, maintenance management, and environmental monitoring and reporting. Standard enterprise collaboration tools would ensure secure communication and document control between site personnel, remote technical teams, and corporate management.

18.11 Laboratory

A laboratory would be established to support operational activities, metallurgical control, geometallurgy, process development and continuous optimization of the process plant. The laboratory would act as a technical support unit for monitoring the performance of the plant circuit and evaluating the characteristics of the different geometallurgical domains of the deposit.

Activities would include process mineralogy and characterization of ore samples, particle size analyses, sample preparation, bench-scale metallurgical testwork, monitoring of ore response to processing and evaluation of opportunities for operational improvement. The laboratory would also support metallurgical reconciliation campaigns, quality control and validation of process parameters.

The facility would be fitted with appropriate equipment and instrumentation to perform the analytical and experimental work required to monitor operations and support the technological development of the Project, in compliance with applicable safety, environmental and quality control requirements.

The results generated would be used to inform operational decision-making, improve the performance of the processing circuit and support geometallurgical control programs over the life of the Project.

18.12 Explosives Magazine

Due to the gentle slope of the local terrain, the magazines would be located in areas confined by artificial barricades made of mechanically compacted fill. The fill barricades would have an average height of 3.00 m, an average crest width of 3.50 m and side slopes of 1.5H:1.0V.

Walls in exposed concrete block masonry with tooled joints and a latex paint finish on the exterior and interior. Interior floors and external walkways would be in smooth float-finished concrete to facilitate cleaning and avoid friction.

Roofing in trapezoidal metal sheeting on steel purlin framing, without a concrete slab. The magazine access door would be a double-leaf hardwood door.

To provide adequate ventilation inside the magazines, aluminum louvers would be installed at suitable heights to promote circulation and continuous renewal of the internal air.

Thresholds and window sills in granite. Foundations on shallow spread footings, with reinforced concrete columns and beams.

As a physical security and access control measure for the magazines, a perimeter fence would be built around the buildings, consisting of 12-gauge galvanized wire mesh with a 2" opening, fixed to 2.00 m high concrete posts and topped with a double galvanized concertina razor-wire coil, 450 mm in diameter, with 54 loops and 30 mm blades.

The fence gates would be built with galvanized tubular frames and fitted with combination locks.

The layout includes three emulsion tanks, each with a storage capacity of 30 t, installed on reinforced concrete bases. The emulsion loading and unloading lanes would be built in waterproofed concrete rated for heavy vehicles, with masonry side curbs and speed bumps at the trailer access points to delimit the area and contain any oil leaks from trailers and/or trucks.

Collection and drainage channels would be built around the lanes to direct stormwater to an oil–water separator (OWS).

The compressor would be installed on a float-finished concrete base, and a firewall in exposed concrete and containment curbs would be built.

19 MARKET STUDIES AND CONTRACTS

This section presents the marketing context, contractual framework and market analysis supporting the mineral processing, product marketability and revenue assumptions used in the PFS.

19.1 Introduction and Basis of Section

Bravo acquired the Luanga Project from Vale S.A. The Project hosts a Neo-Archean layered mafic-ultramafic intrusion prospective for platinum (Pt), palladium (Pd), rhodium (Rh), gold (Au) and nickel (Ni), with a recently identified iron-oxide-copper-gold (IOCG)-style copper-gold discovery on the same land package. Luanga is one of only a small number of PGM-bearing development projects globally that is not located in South Africa, Zimbabwe or Russia—the three countries that together account for the substantial majority of current global primary PGM supply—and is, at the time of writing, the only advanced-stage PGM+Au+Ni project of scale in Brazil.

In November 2025, the Brazilian government's export processing zone council (CZPE) approved the creation of an Export Processing Zone (ZPE) at Barcarena, Pará, with a Bravo subsidiary, Bravo Metals Ltda., named as the anchor industrial project; the ZPE was formally established by presidential decree in January 2026 and Bravo Metals received operating authorisation (Resolution CZPE/MDIC No. 128) in August 2026 to manufacture nickel-PGM mattes and alloys, including platinum and palladium-bearing intermediate products, at Barcarena. Separately, Luanga has been recognised by the Brazilian federal government as a Strategic Minerals Project under Brazil's critical minerals policy framework. These developments materially inform the marketing strategy discussed in this section, in that they open a credible in-country downstream processing pathway that does not exist for any other undeveloped PGM project of comparable scale.

19.2 Products and Marketing Strategy

19.2.1 Product Description

At PFS level, the Project is expected to produce a PGM-Au-Ni sulphide flotation concentrate as its primary mine-site product, containing payable platinum, palladium, rhodium, gold, and nickel with the potential for as yet unquantified copper recovery, consistent with the mineralogy of the Neo-Archean intrusive complex. As with comparable Bushveld- and Sudbury-style deposits, minor quantities of ruthenium, iridium, osmium and cobalt are expected to report to concentrate but are not typically treated as payable metals under reported concentrate grades and resulting standard industry terms (Section 19.6.1). Cobalt has also been recovered in concentrates produced from Luanga metallurgical test work, but is also unquantified in the MRE.

Bravo's intention, reflected in the Barcarena ZPE authorisation obtained in 2026, considers downstream processing of Luanga concentrate into a nickel-PGM matte or alloy

product within Brazil, rather than exporting flotation concentrate exclusively as a bulk commodity. A matte or alloy product is considerably more compact per unit of contained value than flotation concentrate, is more readily transportable to third-party precious-metal refineries internationally, and—depending on final flowsheet and payability terms negotiated with a toll refiner—can capture a larger share of the recoverable metal value domestically. The PFS is evaluating downstream processing as the base case; the base case product and point of sale used in the PFS economic model should be cross-referenced against Section 22 (Economic Analysis) of the PFS.

19.2.2 Downstream Processing Strategy — the Barcarena Option

Historically, PGM-bearing concentrates produced outside South Africa have faced a structural marketing constraint: smelting and precious-metal refining capacity for this mineralogical suite is concentrated almost entirely in South Africa (toll smelters serving the Bushveld Complex), with secondary capacity in Russia, Zimbabwe and at a small number of specialist European refineries that accept high-grade PGM residues. A project located in Brazil would, under a concentrate-export model, incur substantial freight costs and be exposed to the availability, allocation policies and commercial terms of a limited number of offshore counterparties—dynamics discussed in general terms in Section 19.6.1.

The Barcarena ZPE initiative is intended to address this constraint by establishing smelting/converting capacity for nickel-PGM mattes and alloys within Brazil, adjacent to Vale's and Hydro's existing industrial infrastructure at the Barcarena export complex in Pará (including deep-water port access, established bulk-handling infrastructure, an exisiting natural gas terminal, and access to hydroelectric power). Under the ZPE regime, qualifying industrial activity benefits from a long-term (20-year) tax and customs regime, though separate environmental licensing and construction permitting is required and has not yet been obtained. If developed, a Barcarena facility would convert Luanga concentrate into a PGM-Ni +/-Cu matte or alloy for export to third-party precious-metal refineries (most plausibly in South Africa or Europe) for final metal separation and refining, retaining base-metal (nickel, +/- copper) value and a portion of smelting margin within Brazil.

Domestic alloy production at Barcarena followed by offshore toll refining, are evaluated as part of the PFS study. Concentrate sales in the early years of production, with latterly smelter development is evaluated as an alternative scenario.

19.2.3 Transport and Logistics

The Project is located in the Carajás Mineral Province of Pará State, approximately 40 km east-northeast of Parauapebas and roughly 600 km south of the port city of Belém, within a region that hosts substantial existing heavy-haul and port infrastructure developed to support the Carajás iron ore system, including the Estrada de Ferro Carajás (EFC) railway and export terminals at the mouth of the Amazon. Any concentrate product would be trucked from site to a seaport (Belém/Barcarena or the Ponta da Madeira terminal at São Luís) for export, or delivered directly to a Barcarena processing facility should the domestic downstream processing.

19.3 Global Platinum Group Metals and Nickel Market Overview

Palladium, platinum and rhodium markets are shaped by a combination of fundamental mine and recycled supply, automotive and industrial demand, investment flows (exchange-traded products, bars and coin), and the drawdown or build of above-ground stocks; nickel is additionally shaped by stainless steel demand (its largest end use by volume) and by the more recent, and currently secondary, influence of battery and electric-vehicle demand. This section summarises the current (2026) supply-demand position for each of the four metals relevant to Luanga's product suite, drawing on the most recently published data from Johnson Matthey, the World Platinum Investment Council (WPIC), the International Nickel Study Group (INSG) and other reputable market data providers, as cited.

19.3.1 Palladium

Palladium demand is dominated by autocatalysts for gasoline (spark-ignition) light-duty vehicles—accounting for approximately 78-82% of total demand in 2024-2026—where it is used alongside rhodium in three-way catalytic converters; hybrid-electric vehicles, which still carry a combustion engine and catalyst, continue to require palladium loadings, whereas battery-electric vehicles do not. Industrial demand (notably electronics, dental and chemical uses, at approximately 14-15% of demand) and investment (bar, coin and exchange-traded product holdings) are secondary but volatile components of the demand base; jewellery demand is negligible for palladium, unlike for platinum. On the supply side, Russia—via Nornickel's Norilsk operations, where palladium is recovered as a co-product of nickel-copper sulphide mining—has historically supplied approximately 40-46% of mined output, with South Africa (recovered as a co-product alongside platinum and rhodium from the same Merensky and UG2 ore by Sibanye-Stillwater, Impala Platinum and Valterra Platinum) contributing a further 32-36%, and North America (Sudbury and Stillwater-type deposits) and Zimbabwe making up the balance. Because palladium is essentially always a co- or by-product of platinum or nickel mining, its primary supply is structurally insensitive to its own price and instead follows capital and operating decisions taken on the basis of the host mine's primary metal.

Johnson Matthey's 2026 PGM Market Report identifies palladium as having recorded a deficit in every year since 2012, with the deficit widening from 218 koz in 2024 to 416 koz in 2025, before an expected swing to a modest 214 koz surplus in 2026 as gasoline-vehicle production softens, autocatalyst recycling recovers strongly on the back of high metal prices, and Russian mine output declines to its lowest level in more than two decades. Johnson Matthey's full supply-demand balance is summarised in Table 19-1 and Figure 19-1.

Table 19-1: Global palladium supply-demand balance, 2024-2026F

Palladium (koz)	2024	2025	2026F
Mine supply	6,928	6,589	6,032
Recycling supply	2,921	3,135	3,427
Total supply	9,849	9,724	9,459
Automotive demand	8,329	8,227	7,829

Palladium (koz)	2024	2025	2026F
Jewellery demand	85	83	82
Industrial demand	1,424	1,448	1,449
Investment demand	229	382	-115
Total demand	10,067	10,140	9,245
Market balance	-218	-416	214

Source: Johnson Matthey, PGM Market Report 2026

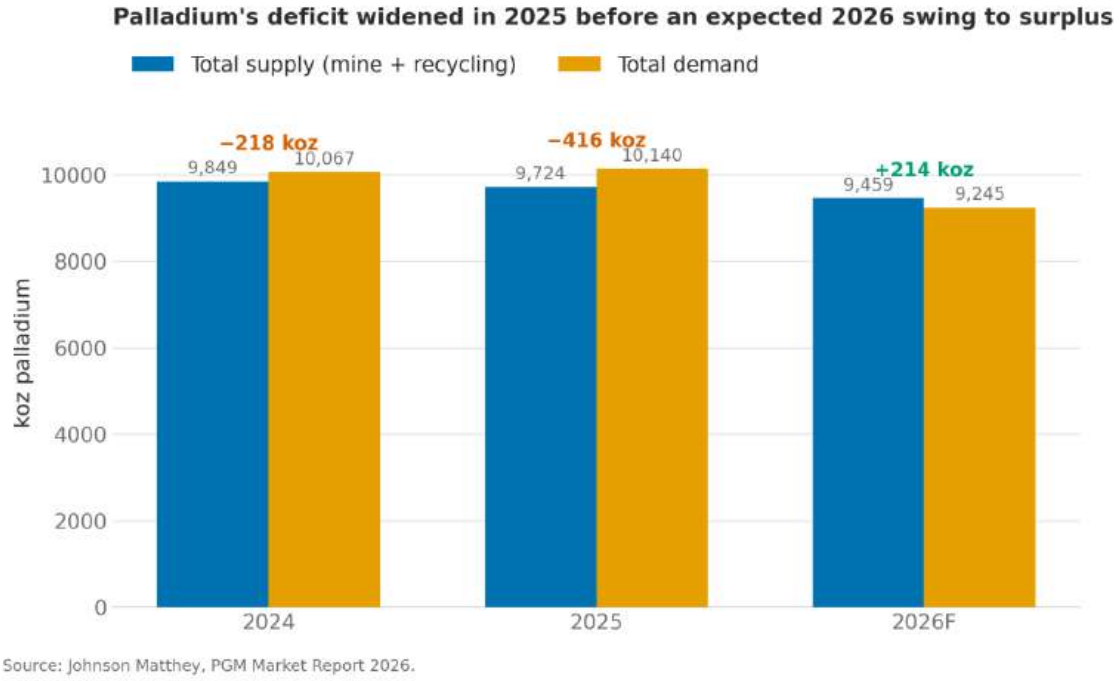


Figure 19-1: Global palladium supply and demand, 2024-2026F, showing the widening deficit through 2025 and the forecast swing to surplus in 2026

Source: Johnson Matthey, PGM Market Report 2026.

Reported palladium prices have also been highly volatile through 2025-2026: having risen to highs above US$1,800/oz during 2025 and ended that year at around US$1,600/oz, palladium opened 2026 near US$1,692/oz, then spiked to a 2026 high of approximately US$2,196/oz in late January—driven by many of the same stock-tightness and investment dynamics affecting platinum over the same period—before falling by roughly 30% into March and touching a nine-month low of approximately US$1,177/oz by 30 June as PGM exchange-traded product holdings were liquidated (PGM ETF holdings fell approximately 11.5% between March and May alone) and US dollar strength weighed on the wider precious metals complex. As at September 4, 2026, spot palladium traded at approximately US$1,430/oz, up approximately 27% year-on-year (Figure 19-2). As with platinum, sell-side forecasts for the remainder of 2026 diverge sharply: Bank of America's June 2026 outlook anticipated palladium reaching approximately US$2,200/oz by year-end, while JPMorgan's July 2026 outlook was materially more conservative, at approximately US$1,350/oz by year-end 2026 and US$1,300/oz through 2027. This wide dispersion reinforces the case for a consensus-based price deck rather than reliance on any single house view (Section 19.5).

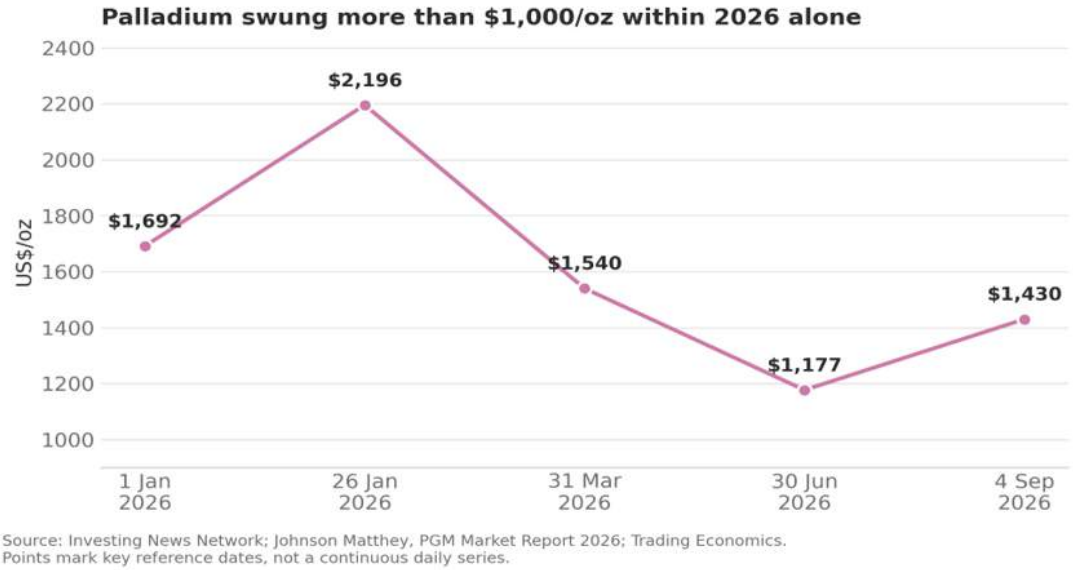


Figure 19-2: Palladium spot price at key reference dates through 2026, illustrating the metal's exceptional intra-year volatility

Source: Investing News Network; Johnson Matthey, PGM Market Report 2026; Trading Economics.

19.3.2 Platinum

Platinum's demand base spans autocatalysts (predominantly diesel light-duty and heavy-duty vehicles, and increasingly hydrogen fuel-cell catalysts), jewellery (approximately one-quarter to one-third of demand, dominated by Chinese and Indian consumption), industrial applications (glass, chemical, petroleum, electrical and medical uses) and investment (bars, coin and exchange-traded products). On the supply side, mined production is heavily concentrated in South Africa (roughly 70-75% of mine supply), with Zimbabwe, Russia and North America contributing the balance; autocatalyst, jewellery and industrial recycling supplement mine supply.

According to the World Platinum Investment Council's Platinum Quarterly (Q1 2026 release), the platinum market recorded its third consecutive annual deficit in 2025 and is forecast to record a fourth consecutive deficit in 2026, albeit a narrower one as elevated prices curb jewellery and net investment demand. The WPIC's full supply-demand balance is summarised in Table 19-2, Figure 19-3 and Figure 19-4.

Table 19-2: Global platinum supply-demand balance, 2023-2026F

Platinum (koz)	2023	2024	2025	2026F
Mine supply	5,620	5,787	5,561	5,551
Recycling supply	1,515	1,536	1,679	1,826
Total supply	7,135	7,323	7,240	7,377
Automotive demand	3,204	3,108	3,031	2,959
Jewellery demand	1,849	2,008	2,214	1,958
Industrial demand	2,491	2,526	2,049	2,238
Investment demand	388	713	1,136	519
Total demand	7,933	8,355	8,431	7,674
Market balance	-799	-1,033	-1,191	-297
Above-ground stocks (year-end)	4,268	3,235	2,044	1,747

Source: World Platinum Investment Council, Platinum Quarterly Q1 2026.

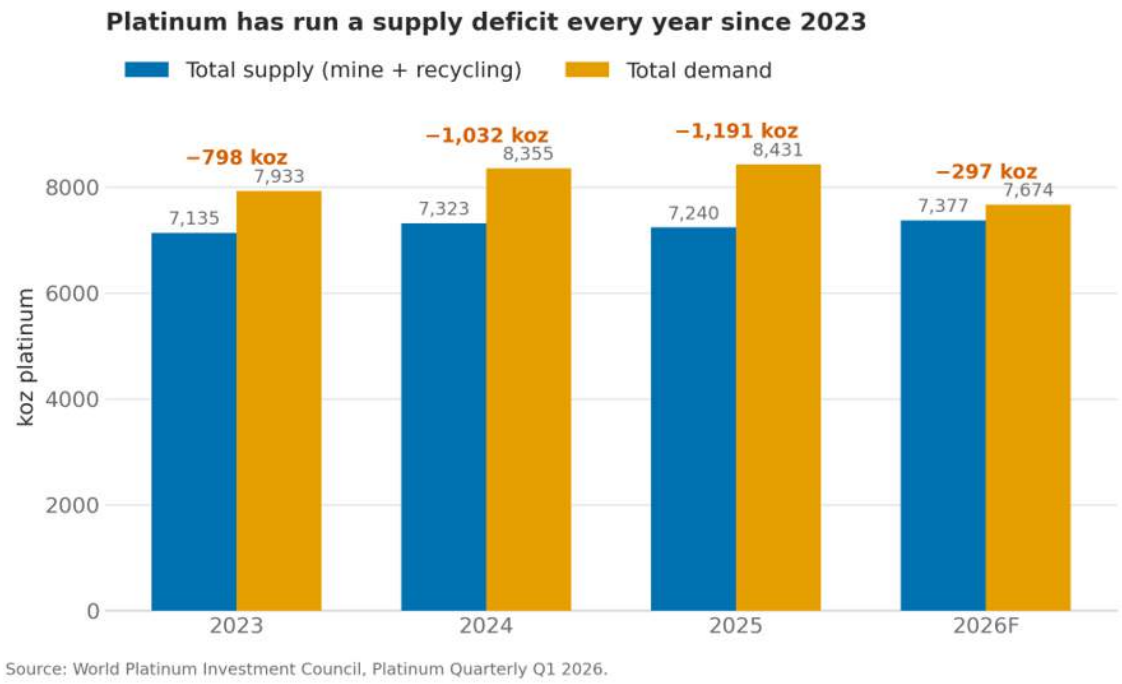


Figure 19-3: Global platinum supply and demand, 2023-2026F, showing the annual market deficit

Source: World Platinum Investment Council, Platinum Quarterly Q1 2026.

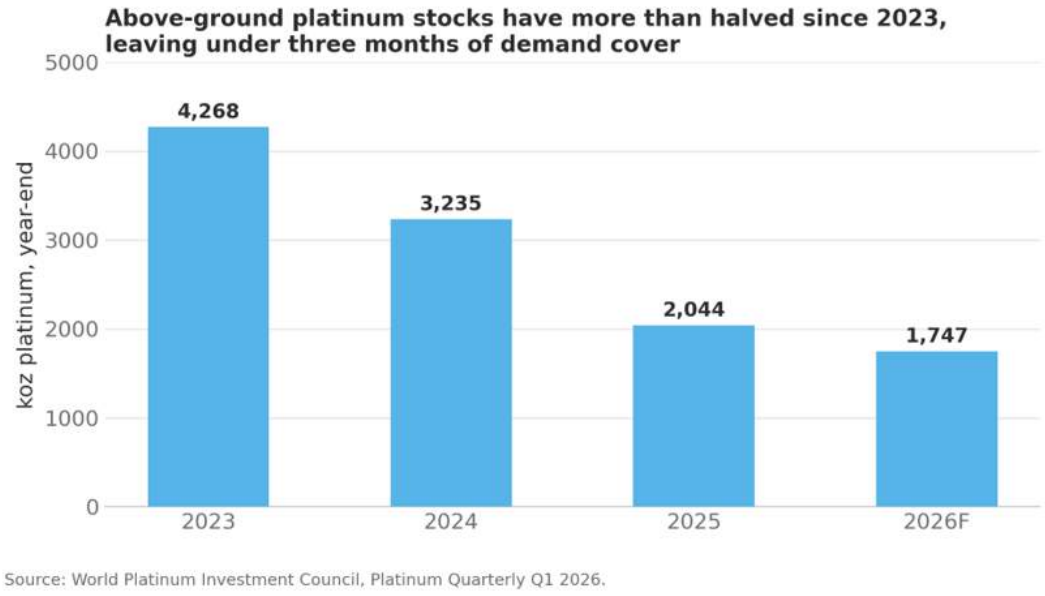


Figure 19-4: Above-ground platinum stocks, 2023-2026F. Cumulative deficits have drawn down inventory buffers to under three months of global demand by the end of 2026

Source: World Platinum Investment Council, Platinum Quarterly Q1 2026.

Reported platinum prices have been volatile through 2026: having opened the year near US$2,137/oz, the price spiked to an all-time intraday high of approximately US$2,924/oz in late January 2026 on the back of shrinking above-ground stocks and strong investment inflows, before retracing to a seven-month low near US$1,566/oz by end-June as higher-for-longer interest rate expectations and US dollar strength weighed on non-yielding assets. As at 3 September 2026, spot platinum traded at approximately US$1,830/oz, up approximately 33% year-on-year. Sell-side forecasts for the balance of 2026 and into 2027 vary considerably by house: Bank of

America's June 2026 outlook anticipated an average of approximately US$3,000/oz by Q4 2026, while JPMorgan's July 2026 outlook was more conservative, at approximately US$1,800/oz by year-end 2026 and US$1,950/oz through 2027. This dispersion illustrates the importance of using a consensus-based price deck, rather than any single house view, for PFS-level economic evaluation (Section 19.5).

19.3.3 Rhodium

Rhodium is the smallest and most illiquid of the three metals by traded volume, used almost exclusively in gasoline autocatalysts for nitrogen oxide (NOx) reduction, where increasingly stringent real-driving-emissions standards (particularly in China and Europe) have driven higher average loadings per vehicle. Supply is essentially a by-product of platinum and palladium mining in South Africa, Russia and Zimbabwe, with no primary rhodium mines; this makes rhodium supply structurally inelastic to its own price.

Johnson Matthey estimates the rhodium market recorded a deficit of 50 koz in 2025 (up from 9 koz in 2024), narrowing to a small forecast surplus of approximately 15 koz in 2026 as lower mine shipments are offset by a rebound in autocatalyst scrap recycling to a four-year high. Rhodium prices remain extremely volatile given the market's small absolute size (annual production measured in the hundreds of thousands of ounces): rhodium reached an all-time high above US$29,800/oz in March 2021, subsequently fell to the low thousands, and had recovered to approximately US$9,400/oz by 4 September 2026, up approximately 31% year-on-year.

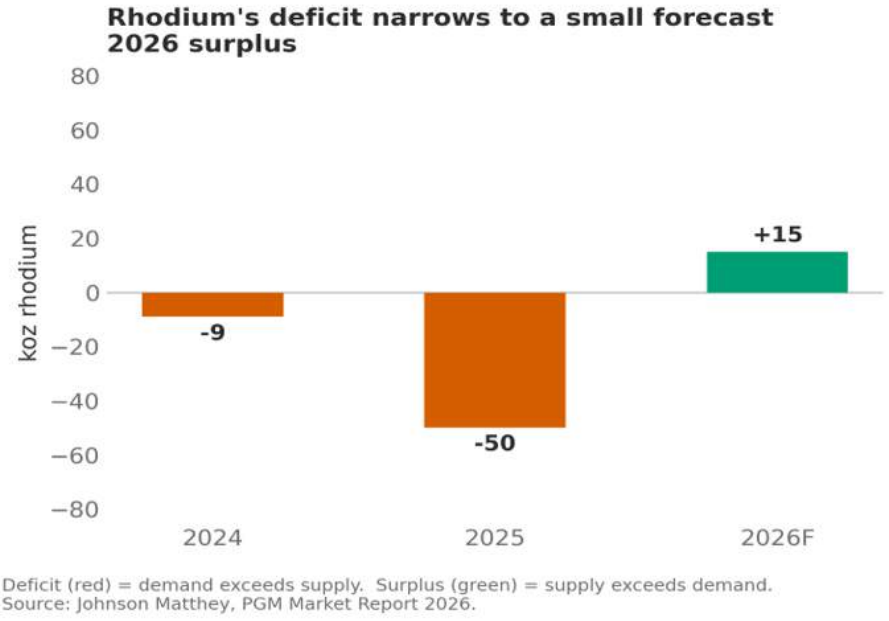


Figure 19-5: Rhodium market balance, 2024-2026F, moving from persistent deficit toward a small forecast 2026 surplus

Source: Johnson Matthey, PGM Market Report 2026.

19.3.4 Nickel

Nickel demand is led by stainless steel production (over 60% of primary demand), with the balance split between alloy/specialty steel, plating, and battery/EV applications (chiefly nickel-manganese-cobalt, NMC, cathode chemistries). Unlike the PGMs, the nickel market has been in structural surplus since at least 2023, driven overwhelmingly by rapid growth in Indonesian nickel pig iron (NPI) and high-pressure acid leach (HPAL) supply; Indonesia now accounts for approximately 60% of global mined nickel output.

The International Nickel Study Group's most recent published estimates put the 2025 global primary nickel surplus at approximately 209 kt, widening to approximately 261 kt in 2026 on continued Indonesian capacity growth, notwithstanding periodic government-driven mining licence suspensions and environmental enforcement actions in Indonesia that have introduced short-term supply volatility (Table 19-3). LME-tracked nickel exchange inventories are at their highest level in more than four years. On the demand side, growth in battery/EV nickel consumption has been slower than earlier industry projections, as the global cathode mix has shifted toward lithium-iron-phosphate (LFP) chemistries that contain no nickel; NMC's share of the global battery market is estimated to have fallen from approximately 25% in 2024 to approximately 18% in 2025.

Reflecting this surplus, nickel prices have been comparatively range-bound relative to the PGMs: LME nickel traded at approximately US$16,760/t (approximately US$7.60/lb) on 4 September 2026, up a comparatively modest 9.7% year-on-year, with consensus forecasts clustering around US$15,000-18,000/t through the balance of 2026, subject to downside risk from elevated inventories and upside risk from any sustained Indonesian supply disruption.

Table 19-3: Global primary nickel supply-demand balance, 2023-2026F

Nickel (kt)	2023	2024	2025F	2026F
Primary production	3,417	3,526	3,810	4,090
Primary usage	3,195	3,347	3,600	3,820
Market balance (surplus)	+223	+179	+209	+261

Source: International Nickel Study Group (INSG); 2025F/2026F per INSG's October 2025 update.

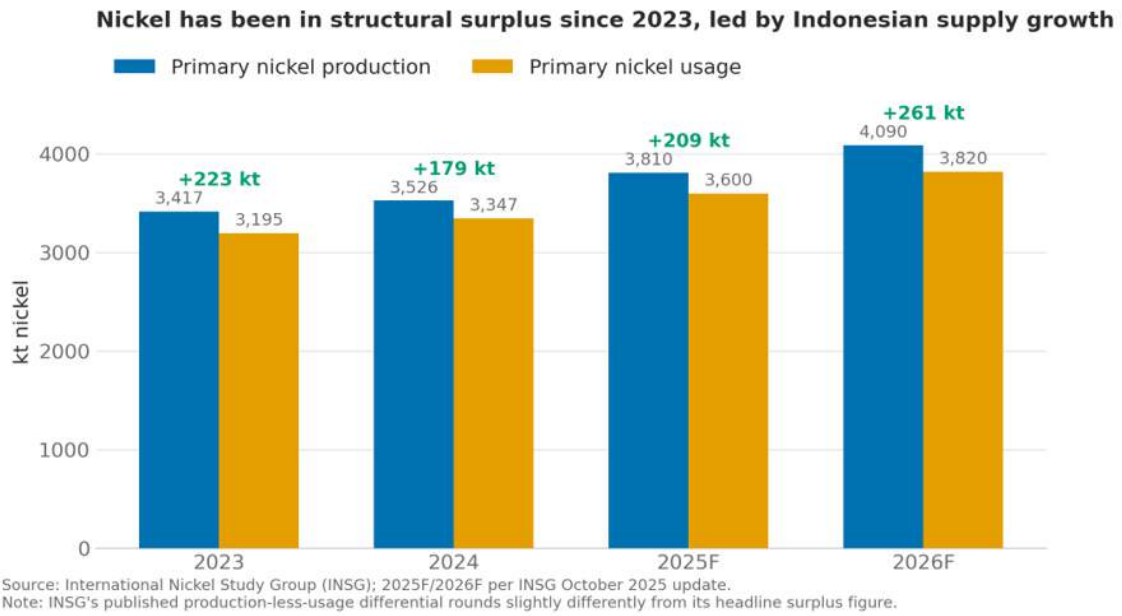


Figure 19-6: Global primary nickel supply-demand balance, 2023-2026F, showing the persistent structural surplus

Source: International Nickel Study Group (INSG); 2025F/2026F per INSG's October 2025 update. INSG's published production-less-usage differential rounds slightly differently from its headline surplus figure.

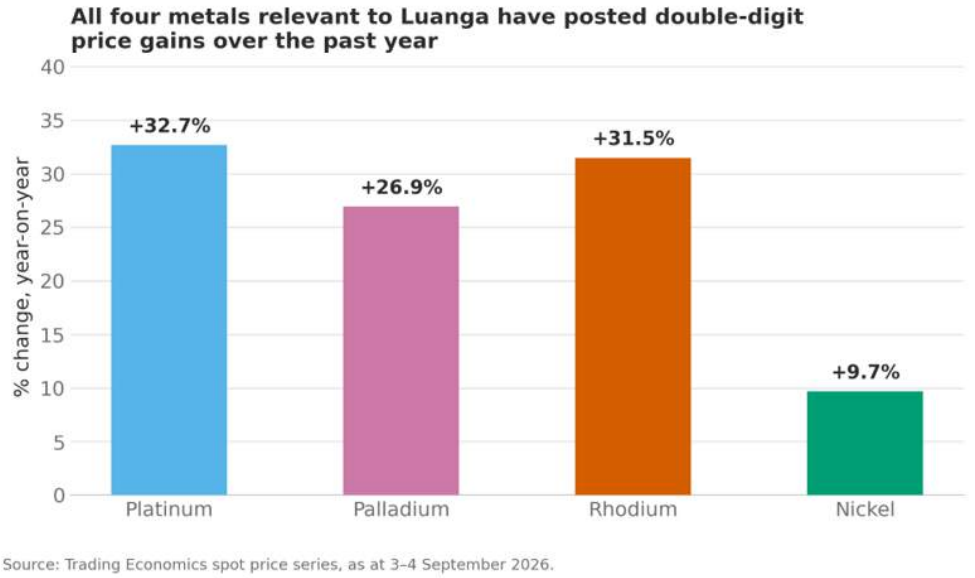


Figure 19-7: Twelve-month price performance of the four metals relevant to Luanga's product suite, as at early September 2026. Platinum, palladium and rhodium have all posted substantial gains on tightening PGM fundamentals, while nickel's gain has been comparatively muted, consistent with its ongoing market surplus

Source: Trading Economics spot price series, 3-4 September 2026.

19.4 Supply-Side Structural Considerations and Luanga's Positioning

The structural backdrop described in Section 19.3 is, for platinum, palladium and rhodium, one of persistent supply concentration risk. South Africa alone hosts on the order of 63 million kilograms of the roughly 76 million kilogram global PGM reserve base, and its traditional Bushveld operations continue to face high and rising costs, deepening mines, electricity and water

constraints, and—following Anglo American's demerger of its platinum division as the newly-named Valterra Platinum in mid-2025—a period of corporate restructuring across the sector's major producers (alongside Sibanye-Stillwater, Impala Platinum and Northam Platinum). Commentators tracking the sector note that only a handful of listed PGM development projects exist globally with the potential to bring on meaningful new supply, and that near-term producer capital allocation has favoured shareholder returns over new mine investment even as PGM prices have risen sharply through 2026. Russian PGM (and palladium in particular) output, concentrated in Nornickel's Norilsk operations, is expected to decline further in 2026 as the operation processes lower-grade ore, while Zimbabwean output remains constrained by limited local smelting capacity and periodic government policy discussions regarding the export of unbeneficiated PGM-bearing material.

Against this backdrop, Luanga's Brazilian location is, from a portfolio and counterparty-diversification perspective, a distinguishing feature: it represents a potential new, geopolitically distinct source of PGM (and nickel) supply outside the traditional South Africa-Russia-Zimbabwe axis, at a time when replacement supply is broadly recognised across the market commentary reviewed above as necessary to sustain global PGM demand. Brazil's own status as an established, large-scale nickel and base-metal producing jurisdiction (via Vale's existing Brazilian nickel operations) and the government's formal recognition of Luanga as a Strategic Minerals Project support the Project's development case, but do not, on their own, guarantee downstream processing or offtake terms; these remain to be negotiated (Section 19.6.2).

19.5 Price Assumptions

Consistent with standard PFS-level practice the Luanga PFS economic analysis adopts metal price assumptions derived from a consensus of contemporary analyst and bank forecasts, sense-checked against long-term historical real prices, rather than short-term spot prices, which have been unusually volatile through 2026 (Section 19.3). The adopted price deck is presented in Table 19-4.

Table 19-4: Commodity prices assumptions for the economic evaluation of the Luanga Project pre-feasibility study

Commodity	Luanga PFS Price Deck (US$) LOM Average
Palladium (US$/Oz)	1,245
Platinum (US$/Oz)	1,700
Rhodium (US$/Oz)	8,000
Gold (US$/Oz)	3,500
Nickel (US$/tonne)	17,000

Source: Bravo, 2026.

19.5.1 Sulphuric Acid

The elemental sulphur and sulphuric acid pricing environment in Brazil is highly dislocated from other regions of the world. This is principally driven by (a) Brazil being the largest crop agricultural economy in the world with a large fertiliser demand (and thus sulphur demand), and

(b) having very limited local supply of sulphur. The latter is due to the low volumes of oil refining occurring domestically and where it occurs, the applied technology does not facilitate sulphur capture as a by-product.

As a result, more than 70% of Brazil's industrial sulphur demand is imported. The market is further impacted by the highly restricted number of ports that can handle sulphur and/or sulphuric acid, and the large distances that material needs to be transported to the interior and north of the country.

Quarterly sulphuric acid benchmark pricing provided by the CRU Group was used to determine the sulphuric acid price assumption for the study. The benchmark pricing is based on delivery to the port of Rio Grande do Sul and a three-year trailing average until prior to the commencement of the Iran War was applied. The Iran War has caused significant volatility in the sulphur market globally and has been excluded from the analysis on the basis that it would introduce upside bias into the price assumption.

Sulphuric acid prices are subsequently analysed considering the CRU benchmark prices and then adjusted for delivery to the appropriate region in Brazil: in the case of this study being northern Pará state. The resulting price assumption for the economic evaluation of the Luanga Project is $650 per tonne CIF Barcarena.

As reference, prices for sulphuric acid have nominally been trading between $1200 and $1400 per tonne through the first half of 2026 in northern Pará state.

19.6 Contracts and Commercial Arrangements

19.6.1 Typical PGM/Nickel Concentrate and Matte Contract Terms

Commercial arrangements for the sale of PGM-and-nickel-bearing concentrates and intermediate products (mattes, alloys and high-grade residues) are typically negotiated bilaterally and vary considerably in structure, but share a common set of commercial building blocks that a marketing strategy for Luanga would need to address:

- Metal accountability (payability): the percentage of contained metal value paid to the seller, which reflects the buyer's downstream recovery, refining cost and cost of capital. Payable percentages for platinum, palladium, gold and rhodium in established markets typically range up to approximately 85%, and for nickel and copper typically up to approximately 75%, though actual terms are commercially negotiated and vary by counterparty, product form and contract duration; minor metals (ruthenium, iridium, osmium) and cobalt are frequently recovered but not paid.
- Treatment and refining charges: a charge, usually expressed per dry tonne of concentrate treated (for smelting/converting/acid-plant costs) and/or per unit of refined metal (for base- and precious-metal refining), that is deducted from the value of accountable metal.
- Penalties: charges applied for deleterious or problematic elements (e.g., arsenic, bismuth, antimony, selenium, tellurium, lead, zinc, chromite and excess moisture), with defined rejection limits above which material may not be accepted.

- Pricing references and the quotational (“pipeline”) period: base metals are typically priced with reference to the London Metal Exchange; platinum and palladium with reference to the twice-daily London Bullion Market Association (LBMA) platinum and palladium price; and rhodium, ruthenium and iridium, being highly illiquid, with reference to fabricator or dealer quotations. A quotational period (commonly one to several months after delivery, and historically as long as approximately nine months for rhodium given its longer processing pipeline) determines the pricing date(s) used, with provisional payments and final settlement adjustments typical of the industry.
- Contract duration, renewal and other terms: length of contract, renewal conditions, delivery terms, assay and dispute-resolution procedures, and—where relevant—advance payment mechanisms against accountable metal value net of treatment and refining charges.

These terms apply, with variations, to both flotation-concentrate sale/toll-treatment arrangements and to the sale of higher-value-added intermediate products such as furnace or converter mattes; matte and alloy products, because they are more compact and higher-grade, typically command more favourable base-metal terms but a narrower and more specialised counterparty base for final precious-metal refining.

19.6.2 Status of Luanga Marketing Arrangements

As at the effective date of this PFS, Bravo has not announced any executed offtake, toll-treatment or concentrate/matte purchase agreement for Luanga production. This is consistent with standard industry practice at the Pre-Feasibility Study stage: firm commercial commitments from smelters, refiners or trading houses are not typically sought or offered until a project has advanced through Definitive Feasibility Study, permitting and financing, and potential counterparties are generally unwilling to allocate scarce smelting or refining capacity years in advance of first production. Bravo's principal near-term commercial focus, reflected in the Barcarena ZPE authorisation obtained in August 2026, has been on securing the regulatory and fiscal framework that would allow a domestic downstream processing option to be developed alongside, or instead of, a conventional concentrate-export marketing strategy.

In developing a marketing strategy through Definitive Feasibility Study, Bravo will need to evaluate and approach both (a) established South African and international toll smelters and PGM/nickel trading houses with the technical capability to treat a Ni-Cu-PGM concentrate of Luanga's expected mineralogy, and (b) engineering, construction and technology partners capable of delivering smelting/converting capacity at Barcarena.

For the purposes of the economic evaluation, alloy toll treatment commercial terms are based on the disclosed terms between a major South Africa smelter and European refiner. This amounts to c. $25/4E ounce refinery charge. A further $400 per tonne nickel refinery charge was also assumed, resulting in a total toll refinery charge of $1,037.85 per alloy tonne.

19.6.3 Government Agreements and Strategic Minerals Status

The Project benefits from two Brazilian government frameworks material to its marketing and development strategy:

- Strategic Minerals Project designation: the federal government has identified Luanga as a Strategic Minerals Project under Brazil's critical minerals policy, which is intended to support permitting prioritisation and policy alignment for projects producing metals considered critical to the domestic and global energy transition and industrial supply chains.
- Barcarena Export Processing Zone authorisation: Bravo Metals Ltda. was approved by the CZPE as the anchor industrial project for a new Export Processing Zone at Barcarena, Pará, formally established by presidential decree in January 2026, with operating authorisation for nickel-PGM matte/alloy manufacturing granted in August 2026 (Resolution CZPE/MDIC No. 128). The ZPE regime provides for a long-term (20-year) preferential tax and customs treatment for qualifying export-oriented industrial activity; it does not itself constitute environmental or construction permitting, which remain separate, outstanding regulatory requirements should the downstream processing option be advanced.

No mineral offtake agreements, smelter contracts or metal sales agreements have been executed as at the effective date of this section. Land tenure, surface rights and other non-marketing agreements applicable to the Project are addressed elsewhere in this Report (see Section 4, Property Description and Location, and Section 20, Environmental Studies, Permitting and Social or Community Impact).

19.7 Qualified Person's Statement on Marketing and Contracts

The QP responsible for this Section has reviewed the market data, product marketability, and contractual status described herein, and confirms that the results support the assumptions in this Report. The QP has relied on the market data sources cited for global PGM and nickel supply-demand and pricing information, and on information provided by Bravo Mining Corp. regarding the Barcarena ZPE authorisation, Strategic Minerals Project designation, and the status of marketing discussions and agreements, and has not independently verified this Company-sourced information beyond reasonable due diligence. The QP is not aware of any information that would materially affect the marketability of the Project's anticipated products, other than the general market and counterparty-concentration risks described in Sections 19.3 and 19.4, which are common to the PGM industry as a whole and are not specific to Luanga.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACTS

This environmental feasibility analysis pertains to the mining project known as the Luanga Project, owned by Bravo, and is based on verifying the compliance of the activity's environmental aspects and impacts with applicable environmental legislation, as well as on an analysis of the history of the environmental permitting process.

This assessment is conducted within the framework of the Brazilian environmental permitting system, which is mandatory for mining activities, as provided for in Federal Law No. 6,938/1981, regulated by Federal Decree No. 99,274/1990. These instruments establish and implement the Brazilian National Environmental Policy (*Política Nacional do Meio Ambiente* – PNMA). In this case, the State of Pará has jurisdiction over the licensing process through the State Secretariat for the Environment and Sustainability (*Secretaria de Estado de Meio Ambiente e Sustentabilidade* – SEMAS/PA), as established by Complementary Law No. 140/2011.

Under the terms of the Brazilian National Environment Council (*Conselho Nacional do Meio Ambiente* – CONAMA) Resolution No. 237/1997, the environmental permitting process generally follows a three-phase model: Preliminary Licence (*Licença Prévia* – LP), Installation Licence (*Licença de Instalação* – LI), and Operating Licence (*Licença de Operação* – LO). The LP certifies the project's environmental feasibility in terms of design and location, establishing the guidelines and conditions to be observed in subsequent phases.

On June 13, 2024, Bravo filed an application for a LP for the Luanga Project with SEMAS/PA, accompanied by the respective Environmental Impact Study and Environmental Impact Report (*Estudo de Impacto Ambiental e Relatório de Impacto Ambiental* – EIA/RIMA). These were prepared in accordance with CONAMA Resolution No. 01/1986, a fundamental technical tool to support decision-making regarding the project's environmental feasibility in terms of location.

The Luanga Project is located in the rural portion of Serra Leste, east of the Serra dos Carajás, in the municipalities of Curionópolis and Marabá, PA. The Directly Affected Area (*Área Diretamente Afetada* – ADA) is located entirely within Curionópolis, within the Carajás Mineral Province, with a total area of approximately 7,810.02 ha, and is linked to ANM mining tenement No. 851,966/1992. The project involves the extraction and processing of metallic minerals (PGMs, gold, and nickel) using open-pit mining methods. At this time, the plan is to process the sulphide material and store the oxidized material for future use.

In 2024, the RIMA presented the Project with an estimated LOM of 10 years**,** a projected average extraction of 8.2 Mt of ROM, and an estimated production of approximately 100 kt of concentrates, based on two distinct geological types (saprolitic ore and sulphide ore), which required the design of specific processing plants for each type of material.

The Project's implementation is economically significant in the regional context, with the potential to generate jobs, increase tax revenue, and boost the mineral production chain.

In the course of the environmental analysis (RIMA, 2024), it was found that the project is entirely within the Amazon Biome. The area has already undergone historical processes of anthropogenic impacts, primarily associated with livestock ranching and small-scale mining, and features fragmented forest remnants and reduced ecological connectivity. In this context, the main environmental constraints include the need to manage vegetation removal and landscape alterations, potential interference with the local water regime (including drainage diversion and lowering of the lowering of the water table), the generation of particulate matter and noise, as well as impacts on fauna and flora resulting from the construction of mining structures.

According to the RIMA (2024), the impact assessment concluded that there is no impact on rural settlements administered by the Brazilian National Institute of Colonization and Agrarian Reform (*Instituto Nacional de Colonização e Reforma Agrária* – INCRA). The Barreiro Cocal settlement was cited only as a reference location in the surrounding area during the field survey conducted in the ADA.

It should also be noted that there is no direct overlap between the ADA and Conservation Units (*Unidades de Conservação* – UCs), which are located only in the surrounding region, at distances of up to approximately 50 km. Nor were any overlaps identified with Indigenous lands, Quilombola territories, INCRA rural settlements, military areas, or other legally protected areas with restricted use. This reinforces the project's locational feasibility from the perspective of the main instruments of land-use planning and environmental protection.

In accordance with the principles of transparency and public participation set forth in environmental legislation, a public hearing was held on December 12, 2024, in the municipality of Curionópolis, PA. The participation of authorities, technical agencies, and local communities strengthened the participatory process inherent in the environmental permitting of projects with significant impact.

Additionally, on February 5, 2025, Bravo formalized the Environmental Offset Agreement (*Termo de Concordância de Compensação Ambiental* – TCCA) based on the Environmental Offset Calculation System (*Sistema de Cálculo de Compensação Ambiental* – SCA), with the Environmental Offset (*Compensação Ambiental* – CA) value determined at R$ 29,343,989.84 (equivalent to 0.7903% of Capex), in accordance with the compensation mechanisms provided for in current legislation.

Following the appropriate technical analysis and deliberation by the competent environmental agency, SEMAS/PA issued LP No. 2062/2025 on March 3, 2025. Thus, the environmental feasibility of the Luanga Project was certified, valid through February 26, 2030, and a set of environmental conditions to be met by the developer was established.

The developer demonstrated compliance with the environmental conditions within the established deadlines, meeting the 30-day obligation by April 1, 2025, and the 180-day obligation

by September 4, 2025, thereby demonstrating adherence to the licensing agency's requirements. Subsequently, on January 6, 2026, a formal notification was issued regarding the modification of the tailings disposal system, highlighting ongoing technical adjustments throughout the project's development.

Regarding land regularization and relations with landowners, the project has formalized contracts with owners of the areas within the project's boundary polygon, and lease agreements are expected to be refined during the next stages of the environmental permitting process.

The project includes a Conceptual Mine Closure Plan (*Plano Conceitual de Fechamento de Mina* – PCFM), prepared by Bravo in 2025. The plan calls for a progressive, planned closure, including dismantling operational structures, geometric reconfiguration of slopes in mined areas, physical and chemical stabilization of decommissioned structures, landscape restoration through revegetation, and rehabilitation of degraded areas. It also includes the definition of monitoring measures, post-closure maintenance and control, as well as risk assessment, and proposals for future uses of the area, taking into account safety aspects, environmental rehabilitation, and the potential for reintegration into the local territory and socioeconomic environment.

Based on the foregoing, and considering the history of the permitting process, the submission and approval of the EIA/RIMA, the holding of a public hearing, the absence of direct interference in environmentally sensitive areas subject to stricter legal restrictions, and the issuance of Permit No. 2062/2025 by SEMAS/PA, it is concluded that the environmental and locational feasibility of the Luanga Project has been demonstrated in accordance with applicable legislation.

During the course of the studies, the Project underwent an engineering review following the granting of the LP. The design changes resulting from this review included a new concept for mining, processing, and tailings disposal planning, which now calls for a tailings dam, as well as a new mine closure plan, as presented throughout this Technical Report.

It is worth noting that the new processing design, by eliminating the oxidized ore processing plant, results in positive environmental impacts, such as reduced consumption of water and process reagents, lower tailings generation, and a reduction in the areas of intervention associated with the respective processing and disposal facilities. Additionally, the engineering review included an update to the Project's energy demand to approximately 86 MW, based on the electrical studies conducted by Grid Energia (GRID) that supported it.

Since these changes alter the design and configuration evaluated during the licensing process that supported LP No. 2062/2025, they must be formally submitted to SEMAS/PA.

It should be emphasized that part of the environmental and impact assessments presented in this Section are based on the scenario originally licensed under the EIA/RIMA, prior to the engineering modifications incorporated into the project. The environmental implications of these changes will be the subject of a specific assessment during subsequent licensing stages with SEMAS/PA.

Regarding mine closure costs, according to the Economic Utilization Plan (*Plano de Aproveitamento Econômico* – PAE) (Bravo, 2025), this activity is planned to be implemented during the last 5 years of operation and the 5 years following the cessation of operations. The estimated cost is US$ 22.0 M for pre-closure and closure (years 5 through 10) and US$ 3.6 M for post-closure, totalling US$ 25.6 M. Regardless of any modifications that may arise in relation to the project's new approach, the mine closure cost was deemed valid.

Finally, it should be noted that the Project's continuation is contingent upon full compliance with the conditions established in the LP and obtaining subsequent licences (LI and LO), which are expected to evaluate, in greater detail, the detailed design plans, environmental control systems, and the planned environmental mitigation, monitoring, and compensation programs.

20.1 QP Opinion – Ms. Branca Horta de Almeida Abrantes

Ms. Abrantes conducted a comprehensive analysis of the information presented in Section 20, which addresses the socio-environmental situation, environmental permitting, environmental and territorial characterization, environmental management, social and community relations, and mine closure-related aspects of the Luanga Project.

The QP conducted a technical analysis of the information submitted by the developer and a review of the applicable regulatory framework, based on Brazilian federal environmental legislation and the laws of the State of Pará. The analysis included a review of the timeline of environmental permitting, the status of LP No. 2062/2025 and compliance with its conditions, water use permits, other environmental authorizations, and the project's compliance with SEMAS/PA and other competent agencies. The environmental information supporting the project was also evaluated, such as the EIA/RIMA, the Mine Closure Plan, and other environmental and socio-environmental studies submitted, covering the characterization of the physical, biotic, and socioeconomic environments, as well as the assessment of environmental impacts and the respective mitigation measures.

The QP also evaluated, based on the information presented in Section 20, the environmental management systems planned for the project, including the management of liquid effluents, solid waste, and air emissions, water resource management, and systems for the disposal of waste rock and tailings. The assessment also covered environmental monitoring programs, the restoration of degraded areas, revegetation, and environmental control of operations.

In addition, Ms. Abrantes evaluated information regarding the community relations and socio-environmental development programs planned for communities within the area of influence, covering initiatives for community communication and participation, environmental education, training, prioritization of local labour, and support for the municipality's economic diversification and regional socioeconomic development. The assessment also considered the PCFM,

environmental rehabilitation strategies, long-term obligations, and estimates of the Luanga Project's closure costs.

Given the progress of the conceptual design and its subsequent stages of maturity, a need was identified to adjust the originally adopted assumptions. Due to changes in the Project's design and configuration evaluated under the current permit, these modifications must be formally submitted to SEMAS/PA. Based on the content evaluated to date, no environmental, social, or permitting factors have been identified that, individually or collectively, could materially affect the project developer's ability to develop and extract the Mineral Resources and Mineral Reserves, in accordance with NI 43-101, subject to compliance with legal requirements. Ms. Abrantes is of the opinion that the content analyzed and the verification work performed are adequate for the purposes of this Report.

21 CAPITAL AND OPERATING COSTS

21.1 Basis of Estimate

This Section presents the capital and operating cost estimates for the Base Case described in Section 16 to Section 18 of this Report: a conventional open pit mine and flotation concentrator at the Luanga site, and a Bravo-owned pyrometallurgical smelter within the Barcarena Export Processing Zone (ZPE), approximately 600 km from the mine, producing a saleable PGM-nickel alloy and sulphuric acid. The mine site, concentrator and smelter are constructed concurrently. Alternative development cases evaluated with the PFS financial model are described in Section 24.

The mine site and concentrator capital cost estimate was prepared by GE21 with the support of Onix Engenharia (Onix), which was responsible for the estimates of the process plant, site civil works, electrical infrastructure, water dam and Tailings Storage Facility (TSF). GE21 estimated the mine-related capital (pit, waste dump and stockpile preparation, mine accesses and contractor mobilization). Closure costs were escalated from the closure estimate in Bravo's Economic Development Plan (PAE, Plano de Aproveitamento Econômico).

Prices for the major mechanical equipment were obtained from budget quotations from equipment suppliers. The remaining equipment, civil works and electrical items were priced based on recent proposals and the Onix database, and the dam construction unit rates were obtained from recent quotations from two experienced contractors. Mine-related unit rates are based on the GE21 database, and recent references from projects in the Carajás region.

The Barcarena ZPE smelter capital and operating cost estimates were prepared by Bravo on the flowsheet, mass and energy balance and design parameters developed by Dr. Rodney Jones, independent pyrometallurgical consultant. Equipment was priced from an itemised equipment list on a Chinese original equipment manufacturer (OEM) basis, with a Western OEM basis carried as a high-cost alternative that is not used in the Base Case. The QP responsible for the smelter cost estimates is Dr. Rodney Jones.

All estimates are expressed in US dollars (US$) in real terms at 2026 price levels, with no allowance for escalation. Costs estimated in Brazilian reais (R$) were converted at R$ 5.50 : US$ 1.00, and cost references from earlier dates were escalated to the estimate base date. The estimates are classified as AACE International Class 4, with an expected accuracy range of −20% to +30%, and each line carries an embedded contingency of 15%.

21.2 Capital Costs

21.2.1 CAPEX Summary

The pre-production (growth) capital cost of the Base Case is estimated at US$ 784.9 M, comprising US$ 603.6 M for the mine site and concentrator and US$ 181.3 M for the Barcarena ZPE smelter. Sustaining capital over the life-of-mine (LOM) is estimated at US$ 98.2 M, of which

US$ 83.9 M relates to the mine site and US$ 14.3 M to the smelter. Mine closure and rehabilitation is estimated at US$ 25.6 M, giving a total LOM capital cost of US$ 908.7 M (Table 21-1). All capital numbers include an embedded 15% contingency.

Growth capital is incurred over a 24-month construction period in 2028 and 2029 (Year −1 and Year 0), ahead of potential first plant feed in January 2030 (Year 1). Mine-site sustaining capital comprises the progressive preparation of the pit, waste rock storage facility (WRSF) and stockpile areas, the TSF raises, the expansion of the mine access roads and contractor mobilization costs during operation. Smelter sustaining capital comprises a single major refractory campaign in the seventh year of smelter operation.

Table 21-1: Capital cost estimate summary

CAPEX Item	US$ M
Pre-Production Mine Site and Plant	
Mining and pre-production	19.7
Process plant, mechanical equipment	280.6
Civil works, buildings and earthworks	51.6
Electrical, substation and distribution	65.3
Tailings storage facility (starter stage)	14.5
Water storage dam	15.2
Indirect costs (freight, imports, first fill)	30.2
Owner's costs	47.7
EPCM	71.6
Financial costs (insurance and taxes)	7.2
Subtotal – mine site and plant growth capital	603.6
Pre-Production Smelter	
Barcarena ZPE smelter – growth capital	181.3
Total Pre-Production CAPEX	784.9
Sustaining Capital (LOM)	
Mine site and plant sustaining capital	83.9
Smelter sustaining capital	14.3
Total Sustaining Capital	98.2
Closure	
Mine closure and rehabilitation (LOM)	25.6
Total CAPEX (LOM)	908.7

Note: all values include an embedded 15% contingency. Totals may not add due to rounding.
Source: GE21, Onix and Bravo, 2026; PFS financial model.

Table 21-2 sets out the mine site capital by area as estimated by GE21 and Onix, split between initial and sustaining capital. The sustaining capital in the mining and TSF areas accounts for all of the US$ 83.9 M of mine-site sustaining capital.

Table 21-2: Mine site and plant capital cost by area

Mine Site CAPEX (US$ M)			
Area	Total	Initial	Sustaining
Mining and pre-production (incl. mine access roads)	53.6	19.7	33.9
Process plant – mechanical equipment	280.6	280.6	–
Civil works, buildings and earthworks	51.6	51.6	–
Electrical, substation and distribution	65.3	65.3	–
Tailings storage facility	64.5	14.5	50.0
Water storage dam	15.2	15.2	–

Mine Site CAPEX (US$ M)			
Indirect costs	30.2	30.2	–
Owner's costs	47.7	47.7	–
EPCM	71.6	71.6	–
Insurance, taxes, duties and fees	7.2	7.2	–
Total mine site and plant	687.5	603.6	83.9
Mine closure	25.6	–	25.6
Total including closure	713.1	603.6	109.5

Note: all values include an embedded 15% contingency. Totals may not add due to rounding.
Source: GE21 and Onix, 2026.

21.2.2 Mining and Pre-Production

Mining would be carried out by a contractor under the Owner's supervision. The mining fleet and its associated infrastructure (workshops, fuel farm and offices) are therefore provided by the contractor and recovered through the contractor rates, and the mine-related capital comprises the preparation of the areas to be mined and occupied by the WRSF and stockpiles, the hydrogeological and geotechnical studies, the mine access roads and the contractor mobilization (Table 21-3). The main unit rates and quantities are:

- Pit area preparation: vegetation clearing at US$ 25,800/ha and topsoil removal at US$ 39,300/ha, applied to 257 ha, progressively cleared in line with the mine schedule, and six dewatering pumping systems at US$ 172,000 each;
- Technical studies: hydrogeological (US$ 0.8 M) and geotechnical (US$ 1.7 M) studies for the pit;
- WRSF and stockpile base preparation: US$ 67,850/ha, including clearing, topsoil removal, underdrains, peripheral drainage channels and HDPE-lined sediment ponds, applied to 317 ha of WRSF and 75 ha of stockpile areas, plus US$ 0.23 M per structure for geotechnical investigations and acid rock drainage studies;
- Mine access roads: 24.4 km of new roads at US$ 155,000/km, built progressively according to the mine schedule (US$ 3.8 M, of which US$ 2.1 M initial);
- Contractor mobilization and demobilization: US$ 1.2 M and US$ 0.9 M, respectively, scaled from a reference operation.

Table 21-3: Mining and pre-production CAPEX

Mining CAPEX (US$ M)			
Description	Total	Initial	Sustaining
Pit area preparation and studies	20.3	7.8	12.4
WRSF base preparation	21.7	4.1	17.7
Stockpile base preparation	5.8	4.6	1.2
Mine access roads	3.8	2.1	1.7
Contractor mobilization / demobilization	2.1	1.2	0.9
Total	53.6	19.7	33.9

Note: values include 15% contingency. Totals may not add due to rounding.
Source: GE21, 2026.

21.2.3 Process Plant

Onix estimated the process plant capital cost for a nominal throughput of 10.0 Mtpa from the mechanical equipment list derived from the process flowsheet described in Section 17. Prices for the main equipment, which represents most of the plant cost, were obtained from supplier

quotations, notably the Weir comminution equipment package (US$ 27.7 M), the primary gyratory crusher (1,631 t/h; US$ 5.0 M), the 26' x 44' ball mill (17.8 MW; US$ 20.2 M), the Jameson flotation cell package (US$ 7.0 M) and the two 125 m diameter thickeners (US$ 4.8 M each). The remaining equipment was priced from recent proposals for equipment of similar size.

The installed cost was obtained by applying Lang factors of 1.5 to 3.0, depending on equipment type, to the equipment prices excluding taxes. The factors cover the equipment and all materials and labour required for its installation, including piping, concrete and steel structures, platework, power supply, automation and instrumentation. The total equipment cost of US$ 90.2 M gives an installed cost of US$ 244.0 M, or US$ 280.6 M including contingency (Table 21-4).

Table 21-4: Process plant CAPEX by area

Process Plant CAPEX (US$ M)	
Area	Installed Cost
Primary Crushing and Weir Comminution Package	119.3
Ore Stockpile and Reclaim	7.6
Grinding and Classification	88.9
Flotation	15.8
Thickening (Grinding, Tailings and Concentrate)	37.2
Concentrate Filtration	6.7
Reagents and Flocculant	1.6
Water, Compressed Air, Fire Protection and Other	3.5
Total	**280.6**

Note: values include 15% contingency.
Source: Onix, 2026.

21.2.4 Civil Works, Buildings and Earthworks

Onix estimated the civil works, support buildings and earthworks at US$ 51.6 M including contingency. The main items, including contingency, are:

- Earthworks for the industrial area platforms, including clearing, cut and fill compacted to 98% of Standard Proctor, and slope protection by hydroseeding: US$ 25.0 M;
- Reinforced concrete structures of the two 125 m diameter thickeners, including pump rooms and tunnels: US$ 16.9 M;
- Concrete process water tanks (45 m x 45 m and 60 m x 60 m) and the reclaim tunnel under the stockpile: US$ 4.6 M;
- Administrative and plant support buildings, with a total built area of 3,800 m², at an average unit cost of US$ 1,150/m² before contingency, and excluding furniture: US$ 5.0 M.

21.2.5 Electrical

The electrical CAPEX (US$ 65.3 M including contingency) is mainly related to the external power supply system, estimated from preliminary budgets based on cost per kilometre of transmission line, the fixed cost of the grid connection point and the Project internal substation. The Project substation was relocated 3.2 km northwest of its original position, closer to the most competitive grid connection point (GRID Energia, 2026). The main components, before contingency, are:

- 230 kV single-circuit transmission line, 34.2 km: US$ 12.9 M (US$ 376,000/km);
- 230 kV double-circuit line, 23.8 km: US$ 10.1 M (US$ 424,000/km);
- Sectioning substation (double busbar, four switches): US$ 13.9 M;
- 230 kV internal substation (main and transfer busbar): US$ 13.8 M;
- 230 kV connection bay: US$ 5.0 M.

The plant unit substations, installed in e-houses (crushing, grinding, metallurgy, administrative buildings, workshop, water intake, mine and explosive magazines), total US$ 1.2 M before contingency and were priced from the Onix database and recent proposals.

21.2.6 Tailings Storage Facility

The TSF comprises compacted earthfill dams (Dam 8 and Dam 8A) built in four stages: the starter stage, built before the start of production (US$ 14.5 M), and raises in Years 2, 4 and 6 (US$ 12.1 M, US$ 13.6 M and US$ 24.3 M, totalling US$ 50.0 M of sustaining capital). Onix estimated the cost by applying contractor unit rates to the quantities estimated for each stage. The main quantities and unit rates are:

- Vegetation clearing and grubbing: 464 ha at R$ 12.50/m² (US$ 2.27/m²);
- Borrow excavation, loading and hauling: 2.0 Mm³ at R$ 22.50/m³ (US$ 4.09/m³);
- Compacted embankment fill: 1.8 Mm³ at R$ 14.50/m³ (US$ 2.64/m³);
- Internal drainage (filter blanket and chimney drain): 52,000 m³ of sand and 22,000 m³ of gravel at R$ 380 to 410/m³ (US$ 69 to 75/m³).

The estimate also includes mobilization and site installation for each stage, spillway, surface and perimeter drainage, and geotechnical instrumentation. The total TSF cost is US$ 64.5 M, including contingency.

21.2.7 Water Dam

The water supply dam is a compacted earthfill dam built in a single stage before the start of production. Onix estimated the cost using the unit rates adopted for the TSF, applied to the main quantities of 92 ha of vegetation clearing, 0.91 Mm³ of borrow excavation and 0.81 Mm³ of compacted fill. The water dam cost is estimated at US$ 13.2 M before contingency, or US$ 15.2 M including contingency.

21.2.8 Indirect Costs

Onix estimated the indirect costs, Owner’s costs, EPCM and insurance as percentages of the corresponding direct costs (US$ 415.0 M before contingency, covering the process plant, site civil works, electrical, TSF and water dam), as presented in Table 21-5. Owner's costs include spare parts, light and maintenance vehicles, PPE, office, restaurant and infirmary equipment, and personnel recruitment. EPCM includes engineering, construction management, procurement, the Owner's team, commissioning and training.

Table 21-5: Indirect costs, owner's costs and EPCM

Indirect Cost Estimate (US$ M)		
Item	Basis	Total
Freight	5% of equipment cost	5.4
Import Costs	Taxes on imported equipment	18.5
First Fill, Lubricants and Technical Assistance	6% of mechanical equipment cost	6.2
Subtotal – Indirect Costs		**30.2**
Owner's Costs	10% of direct costs	47.7
EPCM	15% of direct costs	71.6
Insurance, Taxes, Duties and Fees	1.5% of direct costs	7.2
Total		**156.6**

Note: values include 15% contingency.
Source: Onix, 2026.

21.2.9 Barcarena ZPE Smelter

The smelter treats the full concentrate output of the concentrator. Its unit operations are concentrate reception and preparation, drying, fluidized bed roasting with a double-contact double-absorption (DCDA) sulphuric acid plant, DC electric arc furnace smelting, top-blown rotary converting, and atomization of the alloy to powder, supported by dust handling, utilities, power supply, water and effluent systems, reductant and flux handling, and buildings and civil infrastructure. The design basis is approximately 200 ktpa of concentrate, against a peak Base Case concentrate production of approximately 170 ktpa.

The direct installed cost (DIC) of each section was built from the landed price of each item on the equipment list, multiplied by an installation factor. Indirect costs were then applied as EPCM at 18% of DIC, Owner’s costs at 5% of DIC and a contingency of 15% of DIC, with an allowance of US$ 1.0 M for land, ZPE licensing and permitting. Without ZPE benefits the smelter growth capital is estimated at US$ 271.8 M.

Under the ZPE regime, import duty (II), Industrial Products Tax (IPI) and Social Integration and Security Tax (PIS/COFINS) are suspended on imported and domestically acquired equipment, and Interstate and Intermunicipal Tax (ICMS) is exempt under Convênio ICMS 99/98. The smelter study estimates the resulting saving at 33.3% of the imported equipment cost, or US$ 90.2 M on the smelter estimate excluding the land and permitting allowance. The financial model applies the same 33.3% relief to the whole smelter estimate, giving ZPE-basis growth capital of US$ 181.3 M (Table 21-6).

Table 21-6: Barcarena ZPE smelter growth capital

Smelter CAPEX (US$ M)		
Section	Without ZPE	With ZPE
Concentrate reception and preparation	5.4	3.5
Concentrate drying	4.6	3.0
Fluidized bed roasting	55.1	36.3
Sulphuric acid plant (DCDA)	21.6	14.4
DC electric arc furnace smelting	56.1	37.7
Converting (TBRC)	8.8	5.8
Alloy atomization	7.3	4.8
Dust handling	1.8	1.2
Utilities	5.0	3.3

Smelter CAPEX (US$ M)		
Power supply infrastructure	11.9	8.2
Water and effluent systems	5.6	3.8
Carbon, reductant and flux handling	2.4	1.6
Buildings and civil infrastructure	10.7	7.3
Direct installed cost	196.3	130.9
EPCM (18% of DIC)	35.3	23.6
Owner's costs (5% of DIC)	9.8	6.5
Contingency (15% of DIC)	29.4	19.6
Land, ZPE licensing and permitting	1.0	0.7
Total smelter growth capital	271.8	181.3

Note: Chinese OEM equipment basis. The with-ZPE column reflects the relief applied by section in the smelter study; the financial model applies the average relief of 33.3% to the land and permitting allowance as well. Totals may not add due to rounding.
Source: Bravo smelter capital estimate (BravoSmelter CAPEX v4), 2026; PFS financial model.

Smelter growth capital is spent over the same 24-month construction period as the mine site: US$ 95.2 M in 2028 and US$ 86.1 M in 2029.

21.2.10 Sustaining Capital

Mine-site sustaining capital of US$ 83.9 M was estimated by GE21 and Onix and is scheduled by year in Table 21-7. The largest items are the three TSF raises (US$ 50.0 M) and the progressive WRSF base preparation (US$ 17.7 M). No sustaining capital is required for the process plant beyond the maintenance supplies and services carried in the operating cost, and the contractor fleet carries no Owner's sustaining capital.

Smelter sustaining capital of US$ 14.3 M (US$ 17.0 M without ZPE benefits) covers a full reline of the DC electric arc furnace (US$ 8.3 M) and a major refractory campaign on the fluidized bed roaster (US$ 6.0 M), both scheduled in the seventh year of smelter operation (2036). The two campaigns were estimated bottom-up from refractory quantities and labour, with allowances for copper coolers, the bottom anode assembly, electrode column hardware, shell repair, cranage and dry-out. Intra-campaign hot repairs and routine refractory maintenance remain in operating cost.

21.2.11 Closure and Rehabilitation

Mine closure costs are based on the closure cost estimate in the Project PAE of R$ 103.7 M at December 2023 prices, escalated to July 2026 with a 15% contingency added, giving US$ 25.6 M. GE21 schedules closure and progressive rehabilitation between Year 5 and Year 14 (2034 to 2043). No salvage value is assumed. No decommissioning cost or salvage value is carried for the smelter as it is assumed it would continue to operate based on other sources of feedstock.

21.2.12 Capital Expenditure Schedule

Table 21-7 presents the capital expenditure by year for the Base Case.

Table 21-7: Capital expenditure schedule – Base Case

US$ M	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040–43	Total
	Y−1	Y0	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11–14	
Mining and pre-production	13.7	6.0	6.4	6.2	3.9	3.6	6.8	5.5	0.3	0.6	0.6	0.1	–	53.6
Process plant	168.4	112.3	–	–	–	–	–	–	–	–	–	–	–	280.6
Civil works, buildings and earthworks	31.0	20.6	–	–	–	–	–	–	–	–	–	–	–	51.6
Electrical	39.2	26.1	–	–	–	–	–	–	–	–	–	–	–	65.3
Tailings storage facility	8.7	5.8	–	12.1	–	13.6	–	24.3	–	–	–	–	–	64.5
Water storage dam	9.1	6.1	–	–	–	–	–	–	–	–	–	–	–	15.2
Indirects, Owner's costs, EPCM, insurance	94.0	62.7	–	–	–	–	–	–	–	–	–	–	–	156.6
Mine site and plant subtotal	364.1	239.5	6.4	18.3	3.9	17.2	6.8	29.8	0.3	0.6	0.6	0.1	–	687.5
ZPE smelter – growth	95.2	86.1	–	–	–	–	–	–	–	–	–	–	–	181.3
ZPE smelter – sustaining	–	–	–	–	–	–	–	–	14.3	–	–	–	–	14.3
Closure and rehabilitation	–	–	–	–	–	–	4.2	6.9	5.0	4.4	1.5	0.4	3.2	25.6
Total capital expenditure	459.2	325.6	6.4	18.3	3.9	17.2	11.1	36.7	19.6	5.0	2.1	0.5	3.2	908.7

Note: values include contingency; smelter values on the ZPE basis. Totals may not add due to rounding.
Source: GE21 CAPEX rev04; PFS financial model.

21.2.13 Exclusions

The capital cost estimate excludes the following items:

- Escalation beyond the 2026 estimate base date, any interest during construction and financing costs (the economic analysis is unlevered);
- Exchange rate variations from R$ 5.50 : US$ 1.00;
- Sunk costs, including exploration, studies and permitting incurred before the start of construction;
- Construction-period general and administrative costs of the mine-site and corporate entities, which are carried in the operating cost estimate from 2028 (Section 21.3.6);
- The mining fleet, which is provided by the mining contractor;
- Processing facilities for oxide mineralization, which is not processed in the Base Case;
- Working capital, which is calculated in the financial model (Section 22).

21.3 Operating Costs

21.3.1 OPEX Summary

Operating costs comprise mining, processing, smelting (including concentrate transport from the mine to the smelter), general and administrative (G&A) costs and irrecoverable indirect taxes. GE21 estimated the mining costs, Onix the processing costs and Bravo the smelting and G&A costs, at 2026 price levels. Costs in R$ were converted at R$ 5.50 : US$ 1.00.

Over the LOM of 86.7 Mt processed and 662.8 Mt of total material moved (dry basis), the total operating cost of the Base Case is estimated at US$ 4,124.2 M, or US$ 47.56/t processed, as summarized in Table 21-8. Alloy refining charges, alloy shipping, royalties and production levies are treated as deductions from revenue and are presented in Section 22.

Table 21-8: Operating costs estimate summary – Base Case

Description	US$ M (LOM)	Unit cost
Mining		
Ore mining	362.8	
Waste rock mining	2,039.3	
Marginal material mining	46.8	
Oxide / weathered material mining	67.3	
Grade control, geology and mine management	24.5	
Total mining cost	***2,540.7***	***US$ 3.83/t moved***
Processing		
Labour	55.1	
Power	124.0	
Reagents (flotation and others)	159.3	
Laboratory	20.8	
Media and liners	197.9	
Plant maintenance supplies	59.3	
Plant maintenance services	65.2	
Equipment rental	15.6	
Tailings and water dams	37.9	
Other costs (10% of total)	68.2	
Total processing cost	***803.3***	***US$ 9.26/t processed***
Smelting (Barcarena ZPE)		
Energy – electricity and natural gas	70.9	
Consumables – electrodes, reductant, flux, oxygen, refractories	34.7	
Labour	46.3	
Maintenance – materials and contracted services	27.3	
Smelter G&A – insurance, environmental, legal, IT, travel	11.4	
Concentrate logistics – mine to ZPE	50.2	
Total smelting cost	***240.8***	***US$ 164.20/t concentrate***
General and Administration		
G&A – mine site	316.8	
G&A – Bravo Mining corporate / head office	84.0	
G&A – Bravo Metals ZPE entity	35.0	
Community and social investment	9.3	
Exploration expensed during operations	30.0	
Total G&A	***475.1***	***US$ 5.48/t processed***
Other		
Irrecoverable taxes (PIS/COFINS and ICMS) on operating costs	64.3	US$ 0.74/t processed
Total Operating Cost (LOM)	**4,124.2**	**US$ 47.56/t processed**

Note: totals may not add due to rounding.
Source: GE21, Onix and Bravo, 2026; PFS financial model.

Table 21-9 presents the annual operating cost by category.

Table 21-9: Annual operating cost – Base Case

US$ M	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	Total
Mining	–	–	225.4	239.6	310.6	293.6	287.8	277.5	276.5	263.4	207.3	159.2	2,540.7
Processing	–	–	55.1	92.5	92.9	92.5	92.5	92.6	92.6	92.4	78.0	22.0	803.3
Smelting	–	–	16.5	27.7	27.9	27.7	27.7	27.8	27.8	27.7	23.4	6.6	240.8
G&A	32.5	32.4	39.6	40.2	40.1	39.9	39.8	39.9	40.0	40.0	39.7	39.1	475.1
Irrecoverable indirect taxes	–	–	4.4	7.4	7.4	7.4	7.4	7.4	7.4	7.4	6.2	1.8	64.3
Total operating cost	**32.5**	**32.4**	**341.0**	**407.5**	**478.9**	**461.1**	**455.2**	**445.3**	**444.3**	**430.8**	**354.6**	**228.7**	**4,124.2**

Note: the LOM total includes US$ 12.0 M of corporate G&A in 2026 and 2027, before the start of construction, which is excluded from the valuation in Section 22 as a sunk cost. Totals may not add due to rounding.

Source: PFS financial model.

21.3.2 Mining OPEX

Mining would be carried out by a contractor. The contractor unit rates cover drilling, blasting, loading, hauling and dumping of ore and waste, crusher feed and rehandling, grade control and ancillary services, and were based on rates currently practised in the Carajás region and on the GE21 database. The rates are gross contract prices with ISS, PIS/COFINS and ICMS embedded, and no further indirect tax is charged on them. The contractor rates are dry: diesel is estimated separately from GE21's fuel consumption and charged in addition, at R$ 6.20/litre (US$ 1.13/litre) delivered to site. The Owner's mine management team is a fixed cost of US$ 1.6 M per year, separate from the mine-site G&A.

The financial model applies the GE21 unit rates year by year to the mine schedule. Over the LOM schedule of 662.8 Mt of total material moved, including 86.7 Mt of ore, the total mining cost is US$ 2,540.7 M, equivalent to US$ 3.83/t moved or US$ 29.30/t processed. Waste and marginal material are charged at the waste rate. Oxide material, which is not processed in the Base Case, is mined and stockpiled. The GE21 LOM average unit rates are presented in Table 21-10; the main components are load and haul (US$ 1.91/t for ore and US$ 1.99/t for waste), drilling (US$ 0.41/t and US$ 0.26/t) and blasting (US$ 0.53/t and US$ 0.28/t). The annual cost ranges from US$ 3.56/t to US$ 4.04/t moved, driven by the ore and waste mix and the haul profile.

Table 21-10: Mining OPEX

Mining OPEX (LOM Average)		
Item	Unit	Cost
Ore (contractor)	US$/t ore	3.07
Waste (contractor)	US$/t waste	2.53
Ancillary Services (contractor)	US$/t moved	0.22
Contractor – Average	**US$/t moved**	**2.83**
Diesel	US$/t moved	0.98
Mine Management	US$/t moved	0.02
Total Mining OPEX	**US$/t moved**	**3.84**

Note: GE21 LOM averages on the GE21 schedule weighting. Applied year by year to the PFS mine schedule, the rates give US$ 3.83/t moved.
Source: GE21, 2026.

21.3.3 Processing OPEX

Onix estimated the processing operating costs for a throughput of 10.0 Mtpa, with the comminution circuit based on the Weir equipment package. All processing costs are applied per tonne processed (Table 21-11). The main assumptions are:

- Media and liners: steel and ceramic media for the Weir comminution equipment (US$ 1.06/t), ball mill grinding media at 0.5 kg/t and US$ 2.30/kg (US$ 1.15/t), and crusher and mill liners, based on recent quotations and the Onix database;
- Reagents: depressant (CMC) at 520 g/t, collector (PAX) at 185 g/t, frother (MIBC) at 85 g/t, promoter at 50 g/t and flocculant at 50 g/t, priced at current market costs (US$ 0.42/kg to US$ 2.55/kg);

- Power: consumed power of 49.1 MW, corresponding to 35.0 kWh/t, at US$ 0.041/kWh (US$ 14.3 M per year), sourced from renewable generation;
- Labour: 162 employees, including plant, laboratory, maintenance and administrative personnel, at a total cost of US$ 6.4 M per year;
- Maintenance: supplies estimated at 5% of the plant direct equipment cost (US$ 6.8 M per year) and services based on benchmarks of similar operations;
- Equipment rental: front-end loader, skid-steer loader, forklift, truck-mounted crane and crane, totalling US$ 1.8 M per year;
- Tailings and water dams: engineering and regulatory compliance, dam maintenance and maintenance of the mine and external pumping systems (US$ 4.4 M per year);
- Other costs: an allowance of 10% of the processing cost.

Table 21-11: Processing OPEX

Processing OPEX	
Item	US$/t processed
Media and Liners	2.28
Reagents	1.84
Power	1.43
Plant Maintenance – Services	0.75
Plant Maintenance – Supplies	0.68
Labour	0.64
Tailings and Water Dams	0.44
Laboratory	0.24
Equipment Rental	0.18
Other Costs	0.79
Total	**9.26**

Source: Onix, 2026.

21.3.4 Smelting

Bravo estimated the smelter operating cost bottom-up from the mass and energy balance, the equipment list and the staffing plan, at the design throughput of approximately 200 ktpa of concentrate, and the financial model applies it as a variable cost per tonne of concentrate treated. The Base Case smelts 1,466.7 kt of dry concentrate over the LOM, producing 146.7 kt of PGM-nickel alloy (an alloy yield of 10%) and 878.1 kt of sulphuric acid. The principal cost drivers are electricity for the DC arc furnace and the acid plant, natural gas for drying, graphite electrodes, reductant and flux, oxygen, refractories, labour and maintenance.

The ZPE regime relieves the indirect taxes embedded in the cost of energy (ICMS on electricity and natural gas), imported and domestic consumables and maintenance materials. Labour and bought-in services carry no relief. The resulting unit costs are presented in Table 21-12. Across all smelter cost lines, the ZPE regime reduces the smelting cost by US$ 41.20/t of concentrate, or US$ 60.4 M over the LOM.

Table 21-12: Smelter operating cost per tonne of concentrate

US$/t concentrate			
Item	Without ZPE	With ZPE	LOM (US$ M)
Energy – electricity and natural gas	60.00	48.35	70.9
Consumables – electrodes, reductant, flux, oxygen, refractories	31.61	23.62	34.7
Labour (including CLT employment charges)	31.59	31.59	46.3
Maintenance – materials and contracted services	26.71	18.63	27.3
Smelter G&A	7.74	7.74	11.4
Concentrate logistics – mine to ZPE	47.73	34.25	50.2
Total	205.38	164.18	240.8

Note: totals may not add due to rounding.
Source: Bravo smelter operating cost estimate (BravoSmelter OPEX v3), 2026; PFS financial model.

The smelter operating cost does not include a slag disposal cost. The smelter produces approximately 0.72 t of slag per tonne of concentrate, or about 1.06 Mt over the LOM. Slag is assumed to be disposable to the construction and agricultural industries established in the region.

21.3.5 Concentrate and Alloy Logistics

Filtered concentrate is hauled from the Luanga site to the smelter in the Barcarena ZPE at the Port of Vila do Conde, approximately 600 km by road and rail, at US$ 47.73/t of concentrate. Transport into a ZPE is zero-rated for the indirect taxes under Complementary Law 214/2025 and relieved of ICMS under Convênio ICMS 99/98, reducing the applied rate to US$ 34.25/t and the LOM cost to US$ 50.2 M, which is carried within the smelting cost.

The atomized alloy is shipped from Vila do Conde to a European refiner at US$ 359.20/t of alloy (ocean freight and port handling), with insurance at 0.15% of value. The 10-to-1 reduction in mass between concentrate and alloy reduces the outbound freight exposure accordingly. Alloy freight and insurance (US$ 49.7 M over the LOM) and refining charges are deducted from revenue in the economic analysis (Section 22).

21.3.6 General and Administration

G&A costs are carried as fixed monthly costs for each entity and accrue from the start of construction for the mine site and from first production for the ZPE entity:

- Mine-site G&A, US$ 2.2 M per month (US$ 26.4 M per year), covering site management, administration, human resources, health and safety, environment, community relations, security, camp and site services, from 2028 to 2039 (US$ 316.8 M);
- Bravo Mining corporate and head office costs attributable to the Project, US$ 0.5 M per month (US$ 6.0 M per year), from 2026 to 2039 (US$ 84.0 M, of which US$ 12.0 M falls before the start of construction and is treated as sunk in Section 22);
- Bravo Metals ZPE entity costs, US$ 0.29 M per month (US$ 3.5 M per year), from 2030 to 2039 (US$ 35.0 M);
- Community and social investment at 0.1% of revenue (US$ 9.3 M) and exploration expensed during operations at US$ 3.0 M per year (US$ 30.0 M).

Total G&A is US$ 475.1 M, or US$ 5.48/t processed. Mine-site and corporate G&A during the two construction years (US$ 64.9 M) is carried in operating cost and not in the growth capital.

21.3.7 Irrecoverable Indirect Taxes

The processing and G&A cost estimates are stated before indirect taxes. The financial model adds PIS/COFINS at 9.25% on processing and G&A inputs, ICMS at 20% on processing inputs and ISS at 2% on G&A services, and treats 100% of the PIS/COFINS and 60% of the ICMS as recoverable credits. The irrecoverable balance is US$ 64.3 M over the LOM (US$ 0.74/t processed). Mining contractor rates already include their indirect taxes, and smelter costs are estimated on a tax-inclusive landed basis with the ZPE relief applied, so no further indirect tax is charged on either.

22 ECONOMIC ANALYSIS

22.1 Introduction and Cautionary Statement

This Section presents the economic analysis of the Base Case: a vertically integrated operation comprising conventional open pit mining, a flotation concentrator at the Luanga site and a Bravo-owned pyrometallurgical smelter in the Barcarena Export Processing Zone (ZPE), producing a saleable PGM-nickel alloy and sulphuric acid, with the mine site, concentrator and smelter constructed concurrently. The production schedule is derived solely from the Proven and Probable Mineral Reserves set out in Section 15. Inferred Mineral Resources are not included in the Base Case mine plan or in the financial model.

The alternative development cases evaluated with the PFS financial model, being Alternate Case A (staged smelter development, prepared at pre-feasibility level), whereas Alternate Case B (5 Mtpa throughput) and the Opportunity Case (which includes Inferred Mineral Resources), are not part of the Base Case economic analysis. They are described in Section 24, where Alternate Case B and the Opportunity Case are identified as preliminary economic assessments.

The economic analysis contains forward-looking information, including the Mineral Reserve estimate, production schedule, metallurgical recoveries, metal prices, capital and operating costs, tax and fiscal incentives, and the timing of permitting and construction. Actual results may differ materially from those presented. The principal risks to the forecasts are discussed in Section 25.

22.2 Methodology

The economic analysis uses a discounted cash flow model of the after-tax, unlevered project free cash flow. The principal features of the model are:

- Cash flows are in constant (real) 2026 US dollars, with no escalation of prices or costs and no inflation;
- The valuation date is 1 January 2028, the start of construction (Year 0 for discounting). Cash flows are discounted at a real, after-tax rate of 8% using the mid-period convention;
- Study, permitting and corporate costs incurred before the valuation date (US$ 12.7 M in 2026 and 2027) are treated as sunk and excluded from the NPV, IRR and payback;
- The Project is assumed to be 100% equity funded. No debt, interest or financing costs are included;
- The model runs on quarterly periods for the first production year, semi-annual periods to 2034 and annual periods thereafter, matching the GE21 mine schedule;
- The model consolidates the mining and concentrating entity and the Bravo Metals smelting entity. Concentrate is transferred to the smelter at 100% of its net smelter value, so that the transfer price does not change the consolidated result;

- Working capital is calculated on 30 days of receivables and payables, 30 days of ore stockpile and concentrate at the mine, 15 days of concentrate in transit, 20 days of alloy inventory and 45 days of consumables and spares, and is released at the end of the operation;
- Payback is measured from the start of production (January 2030) and is reported at the resolution of the model periods.

22.3 Principal Assumptions – Base Case

22.3.1 Production Schedule

The Base Case mines 86.7 Mt of ore and 576.1 Mt of waste (662.8 Mt in total), at a strip ratio of 6.6:1) over a mine life of 10 years, with peak material movement of 81.4 Mt in Year 3. The concentrator ramps up to its nameplate throughput of 10 Mtpa by the end of Year 1 and processes an average of 8.67 Mtpa, with a peak of 10.03 Mt in Year 3. The LOM average head grade is 2.77 g/t PdEq. The concentrator produces 1,466.7 kt of dry concentrate grading approximately 100 g/t PGM+Au and 7% Ni, all of which is smelted at Barcarena to produce 146.7 kt of alloy and 878.1 kt of sulphuric acid.

Table 22-1 summarizes the payable production.

Table 22-1: Base Case production summary

Item	Unit	Value
Total ore mined and processed	Mt	86.7
Total waste mined	Mt	575.9
Strip ratio	t/t	6.64
Life of mine	years	10
Concentrate produced and smelted (dry)	kt	1,466.7
PGM-Ni alloy produced	kt	146.7
Sulphuric acid produced	kt	878.1
Payable palladium	koz	2,089
Payable platinum	koz	1,507
Payable rhodium	koz	161
Payable gold	koz	181
Total payable 4E	koz	3,938
Average annual payable 4E	koz/y	393.8
Payable nickel	t	98,247

Note: 4E = Pd + Pt + Rh + Au.
Source: PFS financial model.

22.3.2 Metallurgical Recoveries and Payability

Concentrator recoveries for fresh ore are 82% Pd, 80% Pt, 55% Rh, 88% Au and 71% Ni (Section 13), giving a mix-weighted 4E recovery of 80%. Material scheduled from the low-grade stockpiles is recovered at lower rates (64% Pd, 68% Pt, 40% Rh, 38% Au and 30% Ni). Smelter recoveries from concentrate to alloy are 98% for the 4E metals and 90% for Ni.

The alloy is assumed to be sold to a European refiner. All contained Pd, Pt, Rh, Au and Ni in the alloy is payable, and a toll treatment and refining charge of US$ 1,037.85/t of alloy is

deducted, based on current market terms and equating to approximately US$ 25/oz. Over the LOM, refining charges total US$ 152.2 M and alloy freight and insurance US$ 49.7 M.

22.3.3 Metal Prices and Exchange Rate

Metal prices are long-term real prices. The platinum, palladium, gold and nickel prices were developed with reference to the long-term real forecasts published by Consensus Economics Inc. in August 2026. No long-term real forecast is published for rhodium; its price was set after a review of peer economic studies and recent market behaviour. The sulphuric acid price of US$ 650/t reflects the historical average CRU benchmark price for Rio Grande do Sul before the onset of the Iran conflict, plus estimated freight and delivery to northern Pará. Copper is not credited. The price assumptions are presented in Table 22-2. On the LOM production weighting of the four metals, the prices equate to a 4E basket price of US$ 1,798/oz.

Table 22-2: Metal price and exchange rate assumptions

Commodity	Unit	LOM Price
Palladium	US$/oz	1,245
Platinum	US$/oz	1,700
Rhodium	US$/oz	8,000
Gold	US$/oz	3,500
Nickel	US$/t (US$/lb)	17,000 (7.71)
Sulphuric acid	US$/t	650
Exchange rate	R$: US$	5.50

Source: Consensus Economics (August 2026); Bravo, 2026.

22.3.4 Capital and Operating Costs

Capital and operating costs are as presented in Section 21: growth capital of US$ 784.9 M, sustaining capital of US$ 98.2 M, closure and rehabilitation of US$ 25.6 M and operating costs of US$ 4,124.2 M (US$ 47.56/t processed) over the LOM.

22.4 Taxes, Royalties and Fiscal Incentives

22.4.1 Royalties and Production Levies

The following royalties and levies are deducted from revenue. Over the LOM they total US$ 774.6 M:

- Financial Compensation for the Exploitation of Mineral Resources (CFEM) at 2.0% for PGMs and nickel and 1.5% for gold, levied on revenue net of freight and taxes (US$ 179.3 M);
- Private royalties to Vale (1.0%) and BNDES (2.0%) (US$ 273.6 M);
- Landowner royalty at 1.0% for PGMs and nickel and 0.75% for gold (US$ 89.7 M);
- Pará State mineral resources inspection fee (TFRM) at R$ 14.40/t of ore mined (US$ 227.3 M);
- ANM annual tenement fees of US$ 0.35 M per year (US$ 4.7 M).

Alloy and acid sales are exports and are immune from PIS/COFINS and ICMS on revenue.

22.4.2 Corporate Income Tax and SUDAM

Corporate income tax comprises (IRPJ) at 15% plus a 10% surtax on annual taxable income above R$ 240,000, and the social contribution on net profit (CSLL) at 9%, a combined statutory rate of 34%. Tax losses may be carried forward indefinitely, with utilization capped at 30% of taxable income in any year. Mine-site, plant and smelter assets are depreciated on a straight-line basis over five years and sustaining capital over 2.5 years, reflecting the accelerated depreciation available in the SUDAM area.

Companies in the Legal Amazon may apply for the SUDAM incentive, a 75% reduction of IRPJ (including the surtax) for an initial period of ten years of operation. The Base Case applies SUDAM to both the mining and the smelting entities from the first year of taxable profit and does not assume renewal of the incentive. The SUDAM benefit can only be confirmed once an application has been submitted and approved. Over the LOM the SUDAM benefit is US$ 628.3 M and the income tax charge after SUDAM is US$ 511.0 M, an effective rate of 15.25%.

22.4.3 ZPE Regime

Bravo Metals Ltda., a wholly owned subsidiary of Bravo, was authorized on 25 August 2026 by the National Council for Export Processing Zones (CZPE Resolution No. 128, published 1 September 2026) to establish its industrial project in the Barcarena ZPE for a period of 20 years. The ZPE regime (Law 11,508/2007 as amended by Law 14,184/2021) suspends import duty, IPI, PIS/COFINS, PIS/COFINS-Import and AFRMM on imports and domestic purchases of equipment, raw materials, intermediate products and services, and Convênio ICMS 99/98 provides an ICMS exemption, with equivalent treatment for IBS and CBS under Complementary Law 214/2025. The regime does not change income tax, which is addressed by SUDAM. In the model, the ZPE regime reduces smelter growth capital by US$ 90.2 M, smelter sustaining capital by US$ 2.7 M and smelter operating costs by US$ 41.20/t of concentrate (US$ 60.4 M). Compared with an otherwise identical case without ZPE benefits, the regime adds US$ 109.6 M to the after-tax NPV.

22.5 Results – Base Case

The Base Case returns an after-tax NPV at an 8% discount rate of US$ 1,448.7 M and an after-tax IRR of 35.1%, with a post-construction payback of 2.0 years.

Table 22-3 summarizes the results.

Table 22-3: Base Case economic results

Item	Unit	Value
Returns		
After-tax NPV at 8% (real)	US$ M	1,448.7

Item	Unit	Value
After-tax IRR (real)	%	35.1
Payback period, post-tax, from start of production	years	2.0
Peak funding requirement	US$ M	877
NPV / peak funding	x	1.65
Project free cash flow (LOM, undiscounted, from Year 0)	US$ M	2,815
Revenue and Cost		
Gross revenue	US$ M	9,322.9
of which alloy	US$ M	8,752.1
of which sulphuric acid	US$ M	570.8
Refining charges, alloy freight and insurance	US$ M	201.9
Royalties and production levies	US$ M	774.6
Net revenue	US$ M	8,346.4
Net revenue per payable 4E ounce	US$/oz	2,119.5
Total operating cost	US$ M	4,124.2
EBITDA (LOM, after closure provision)	US$ M	4,196.6
EBITDA margin	%	50.3
C1 cash cost, net of by-product credits	US$/oz 4E	478
All-in sustaining cost (AISC)	US$/oz 4E	706
Capital		
Growth capital	US$ M	784.9
Sustaining capital	US$ M	98.2
Closure and rehabilitation	US$ M	25.6
Total capital	US$ M	908.7
Taxation and Royalties		
CFEM, TFRM, ANM fees and private royalties	US$ M	774.6
Income tax (IRPJ and CSLL) after SUDAM	US$ M	511.0
SUDAM benefit realised	US$ M	628.3

Note: C1 cash cost and AISC are non-IFRS measures. C1 comprises total operating costs (mining, processing, smelting, G&A and irrecoverable indirect taxes) net of nickel and sulphuric acid revenue, per payable 4E ounce; AISC adds royalties and production levies, sustaining capital and closure. Refining charges and alloy freight are netted in revenue and are in neither measure. Totals may not add due to rounding.
Source: Bravo, 2026; PFS financial model.

The undiscounted LOM free cash flow from the valuation date is US$ 2,815.2 M. Peak funding of US$ 877 M is reached in 2030, and the cumulative free cash flow turns positive in 2032. The difference between peak funding and growth capital reflects construction-period G&A, initial working capital and the first year of ramp-up.

Table 22-4 and Figure 22-1 present the annual cash flow.

Table 22-4: Base Case annual cash flow

US$ M unless stated	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040–43	Total
Plant feed (Mt)	–	–	5.95	9.99	10.03	9.99	9.98	10.00	10.00	9.97	8.42	2.38	–	86.71
Payable 4E (koz)	–	–	299.2	538.3	494.6	427.1	412.3	437.9	429.8	467.1	345.0	86.5	–	3,937.9
Payable Ni (kt)	–	–	8.0	14.5	12.0	8.7	8.6	10.8	11.8	11.7	9.7	2.6	–	98.2
Gross revenue	–	–	718.5	1,295.5	1,158.9	978.3	935.8	1,033.6	1,054.4	1,094.7	838.5	214.6	–	9,322.9
Refining, alloy freight and insurance	–	–	(13.9)	(23.3)	(23.4)	(23.3)	(23.2)	(23.3)	(23.3)	(23.2)	(19.6)	(5.5)	–	(201.9)
Royalties and levies	(0.4)	(0.3)	(57.9)	(102.1)	(94.2)	(88.1)	(85.8)	(86.6)	(89.9)	(90.9)	(61.0)	(16.8)	–	(773.9)
Net revenue	(0.4)	(0.3)	646.7	1,170.2	1,041.4	867.0	826.8	923.8	941.3	980.6	757.9	192.2	–	8,347.1
Operating costs	(32.5)	(32.4)	(341.0)	(407.5)	(478.9)	(461.1)	(455.2)	(445.3)	(444.3)	(430.8)	(354.6)	(228.7)	–	(4,112.2)
Operating margin	(32.8)	(32.7)	305.7	762.7	562.5	405.9	371.6	478.5	496.9	549.8	403.3	(36.4)	–	4,234.9
Income tax paid	–	–	(15.4)	(81.3)	(64.0)	(38.7)	(31.4)	(57.8)	(72.0)	(82.4)	(62.9)	(5.0)	–	(511.0)
Working capital movement	(0.3)	–	(96.4)	(52.4)	(1.0)	16.5	4.9	(8.8)	(0.6)	(1.7)	28.2	68.1	43.5	–
Growth capital	(459.2)	(325.6)	–	–	–	–	–	–	–	–	–	–	–	(784.9)
Sustaining capital	–	–	(6.4)	(18.3)	(3.9)	(17.2)	(6.8)	(29.8)	(14.6)	(0.6)	(0.6)	(0.1)	–	(98.2)
Closure and rehabilitation	–	–	–	–	–	–	(4.2)	(6.9)	(5.0)	(4.4)	(1.5)	(0.4)	(3.2)	(25.6)
Free cash flow	(492.3)	(358.4)	187.5	610.7	493.6	366.5	334.0	375.2	404.7	460.6	366.5	26.1	40.3	2,815.2
Cumulative free cash flow	(492.3)	(850.7)	(663.1)	(52.4)	441.1	807.6	1,141.7	1,516.9	1,921.6	2,382.3	2,748.8	2,774.9	2,815.2	2,815.2
Discounted free cash flow (8%)	(473.7)	(320.5)	152.5	465.8	349.2	240.2	202.6	210.7	210.4	221.7	163.4	10.8	15.6	1,448.7

Note: excludes US$ 12.7 M of sunk costs in 2026 and 2027. Brackets denote outflows. Totals may not add due to rounding.
Source: PFS financial model.

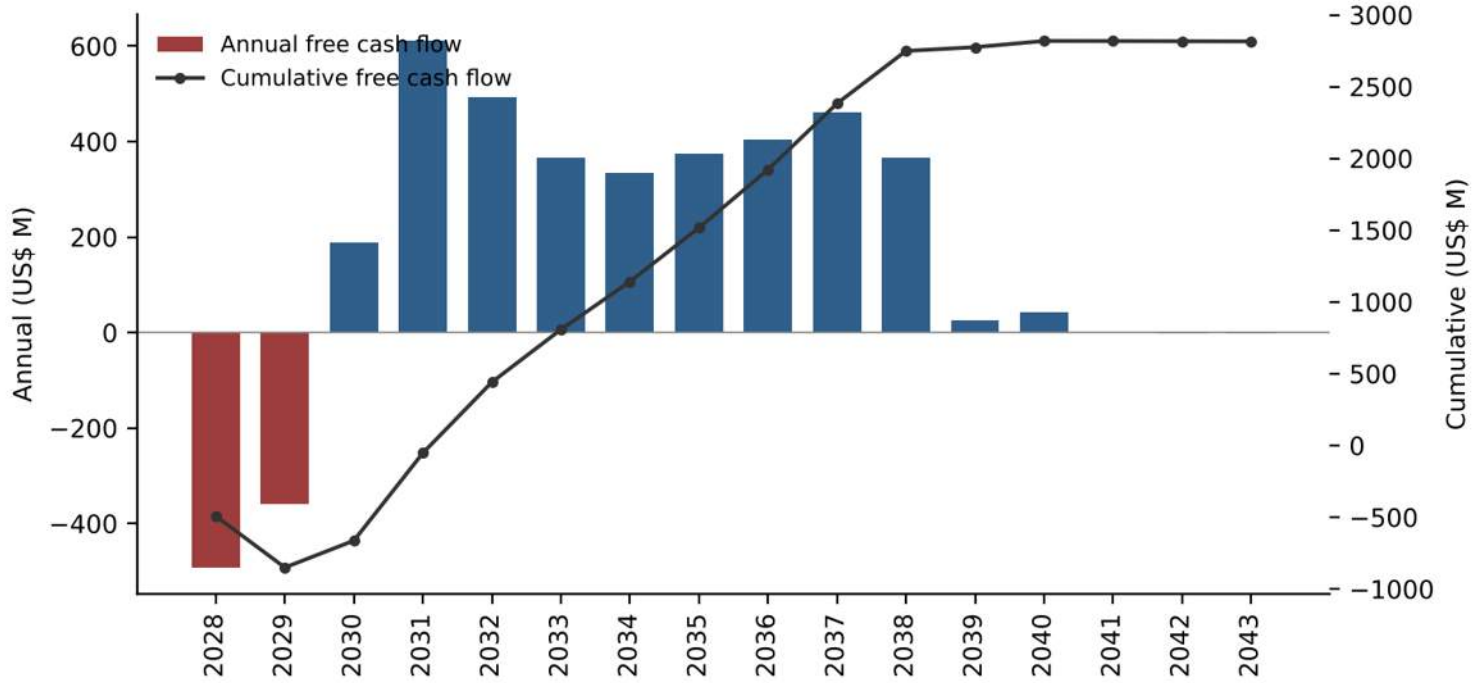


Figure 22-1: Base Case annual and cumulative free cash flow

Source: Bravo, 2026; PFS financial model.

22.6 Sensitivity Analysis – Base Case

The sensitivity of the after-tax NPV and IRR to metal prices, total operating cost, smelter operating cost and capital cost was tested by re-running the full financial model with one driver varied at a time between −30% and +30%, all other inputs held at the Base Case. The metal price sensitivity varies all metal and acid prices together. The total operating cost sensitivity varies mining, processing, smelting, G&A and indirect tax costs together. Results are presented in Table 22-5, Table 22-6 and Figure 22-2.

Table 22-5: Base Case after-tax NPV8 sensitivity (US$ M)

Driver	−30%	−20%	−10%	Base	+10%	+20%	+30%
Metal prices	80.6	538.8	993.8	1,448.7	1,903.6	2,358.2	2,811.8
Total operating cost	2,092.9	1,879.0	1,664.1	1,448.7	1,233.3	1,017.8	802.4
Smelter operating cost	1,486.9	1,474.1	1,461.4	1,448.7	1,435.9	1,423.2	1,410.5
Capital cost	1,657.7	1,588.0	1,518.4	1,448.7	1,379.0	1,309.3	1,239.6

Source: Bravo, 2026; PFS financial model, full re-runs.

Table 22-6: Base Case after-tax IRR sensitivity (%)

Driver	−30%	−20%	−10%	Base	+10%	+20%	+30%
Metal prices	10.0	19.9	28.0	35.1	41.5	47.5	53.0
Total operating cost	44.9	41.7	38.5	35.1	31.7	28.1	24.4
Smelter operating cost	35.7	35.5	35.3	35.1	34.9	34.7	34.5
Capital cost	46.5	42.0	38.3	35.1	32.3	29.9	27.7

Source: Bravo, 2026; PFS financial model, full re-runs.

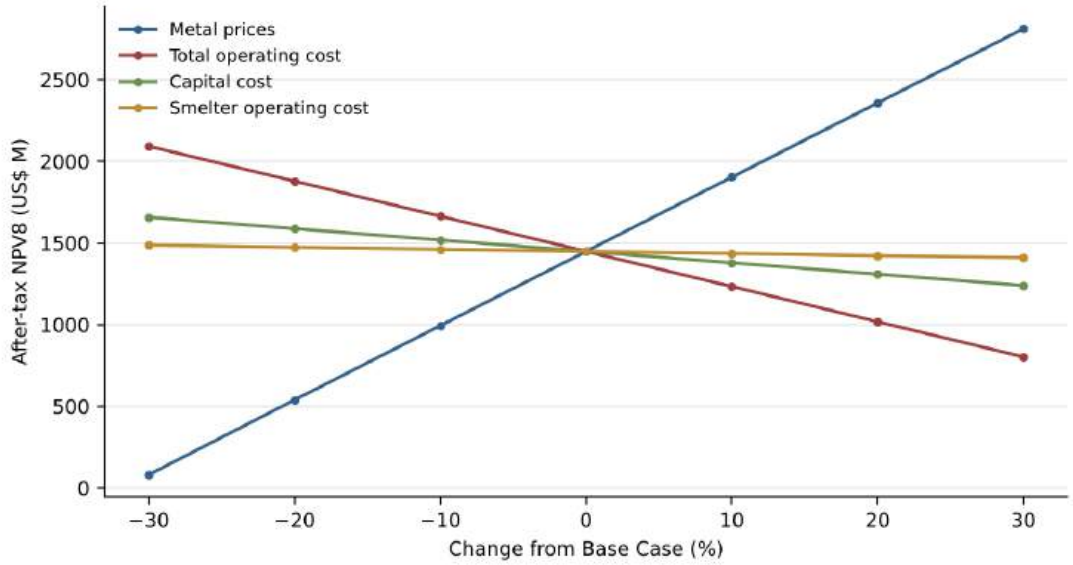


Figure 22-2: Base Case after-tax NPV8 sensitivity

Source: Bravo, 2026; PFS financial model, full re-runs.

The Project is most sensitive to metal prices: a 10% change in prices changes the NPV by approximately US$ 455 M. A 10% change in total operating cost changes the NPV by approximately US$ 215 M, and a 10% change in capital cost by approximately US$ 70 M. The NPV remains positive at metal prices 30% below the Base Case. Because the smelter cost

represents less than 6% of total operating cost, a 30% change in smelter operating cost changes the NPV by less than US$ 40 M.

Table 22-7 presents after-tax NPV sensitivity to discount rate variation.

Table 22-7: Base Case after-tax NPV at alternative discount rates

Discount rate (real)	5%	6%	7%	8%	9%	10%	12%
After-tax NPV (US$ M)	1,858.5	1,710.7	1,574.5	1,448.7	1,332.4	1,224.8	1,032.7

Source: Bravo, 2026; PFS financial model.

22.7 Conclusions – Base Case

At the PFS metal price assumptions and cost estimates, the Base Case generates an after-tax net present value at an 8% discount rate (NPV8) of US$ 1,448.7 M and an IRR of 35.1% on growth capital of US$ 784.9 M, from a 10-year production schedule based solely on Proven and Probable Mineral Reserves. The economics depend on the availability of the SUDAM income tax incentive and the ZPE regime, on the alloy offtake terms, and above all on PGM prices.

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.

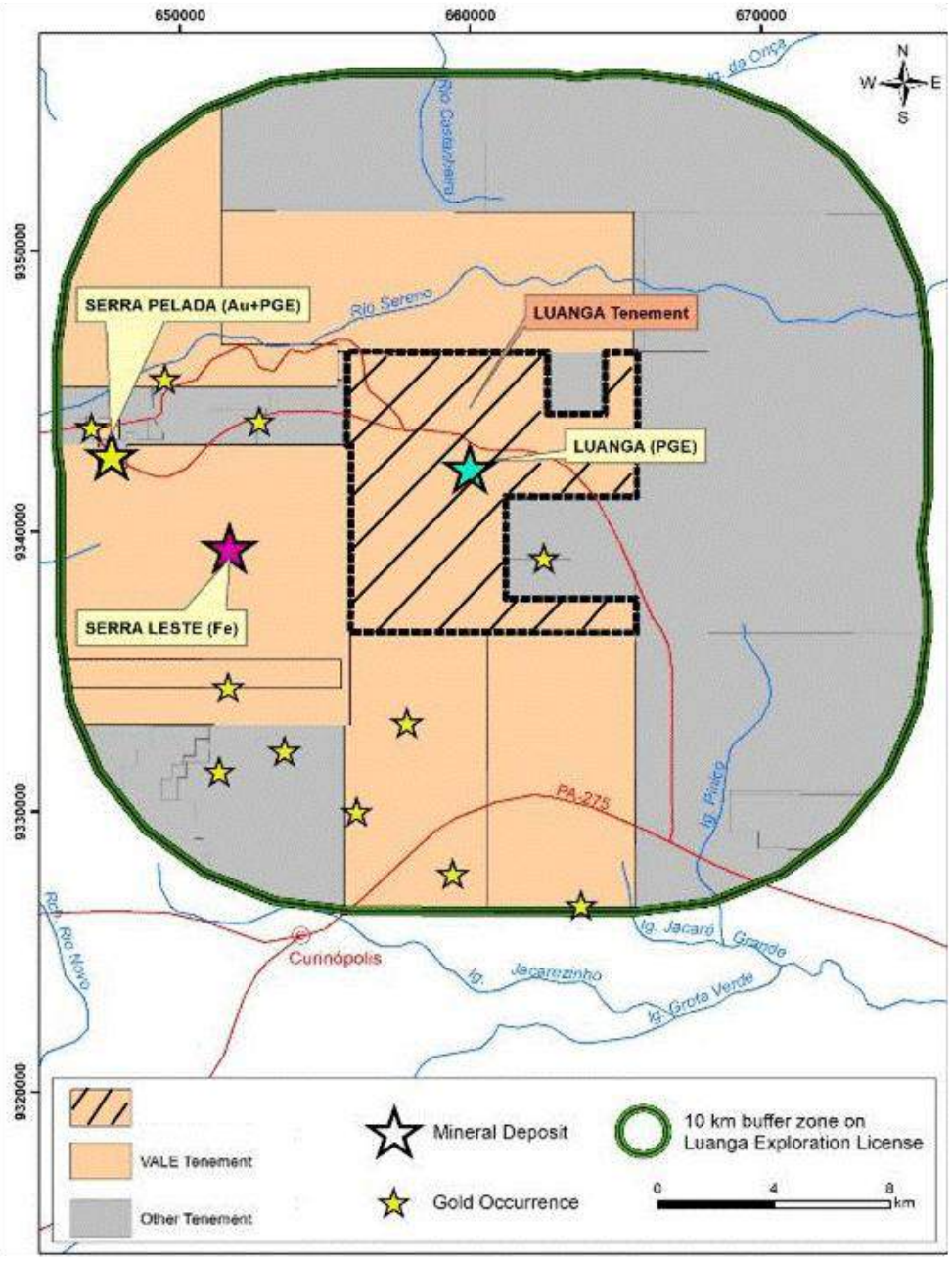


Figure 23-1: Mineral deposits adjacent to Luanga Project

Source: Bravo, 2023.

The Serra Pelada Au + PGM deposit occurs 8 km west of Luanga in a tenement with a Mining Licence 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 (Vale public records).

There is no open ground for new exploration claims surrounding the Luanga licence, and Vale is the major holder of exploration claims in the region.

The QP has not been able to independently verify the information on the adjacent properties, which is drawn from public sources, and observes that the information in Section 23 is not necessarily indicative of the mineralization on the property that is the subject of this Technical Report.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Introduction

In addition to the Base Case presented in Section 21 and Section 22, the PFS financial model was used to evaluate three alternative development cases, and the PFS identified a number of opportunities that could improve the Project and that are not included in the Base Case. This Section describes:

- **Alternate Case A**, which applies the Base Case mine and concentrator to a staged development of the smelter. Alternate Case A was prepared at pre-feasibility level (AACE Class 4) and forms part of the PFS;
- **Alternate Case B**, which applies the Base Case configuration at a reduced throughput of 5 Mtpa. Alternate Case B is a preliminary economic assessment (PEA) with AACE Class 5 cost estimates and does not form part of the PFS;
- **The Opportunity Case**, which applies the Base Case modifying factors, inputs and cost bases to the whole current Mineral Resource block model, including Inferred Mineral Resources. The Opportunity Case is a PEA with AACE Class 5 cost estimates and does not form part of the PFS;
- The Project opportunities identified in the PFS: Mineral Resource and Mineral Reserve growth, a copper credit, vertical integration flexibility, an Owner-operated mining fleet, the oxide Mineral Resources and the copper-gold discoveries outside the Luanga deposit;
- The exploration potential of the Luanga property for PGM and for copper +/- nickel +/- PGMs +/- Au mineralization, summarized from Section 9.

24.1.1 Cautionary Statement on Alternate Case B and the Opportunity Case

For the purposes of NI 43-101, Alternate Case B and the Opportunity Case are preliminary economic assessments and are identified as such wherever they are referred to in this Report, consistent with CSA Staff Notice 43-307, Mining Technical Reports – Preliminary Economic Assessments. They do not form part of the PFS and are included in this Section as Other Relevant Data and Information under Item 24 of Form 43-101F1. Readers are cautioned as follows:

- The Opportunity Case has been developed solely to illustrate the potential economic effect on the Project of a hypothetical future conversion of part of the Inferred Mineral Resources to the Measured and Indicated categories and, subject to the appropriate modifying factors, to Mineral Reserves, and to assist Bravo in assessing whether further Mineral Resource and Mineral Reserve definition drilling is warranted. It is not a basis for investment decisions.

- The Opportunity Case includes Inferred Mineral Resources in its production schedule and economic analysis, as permitted for a PEA by section 2.3(3) of NI 43-101. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. The economic viability of the Mineral Resources in the Opportunity Case has not been demonstrated, and there is no certainty that the Opportunity Case, or any part of it, will be realized.
- There is no certainty that any of the Inferred Mineral Resources in the Opportunity Case will be upgraded to Measured or Indicated Mineral Resources, either on completion of the current infill drilling programme and the resulting Mineral Resource update or at any later time, and no certainty that any resulting Measured and Indicated Mineral Resources would be converted to Mineral Reserves under the modifying factors applicable at that time.
- The Opportunity Case does not declare, and is not to be construed as declaring, any Mineral Reserves. Section 2.3(1)(b) of NI 43-101 does not permit Inferred Mineral Resources to be included in a prefeasibility-level economic analysis, and no such use has been made in the PFS.
- The Class 5 cost estimates used in Alternate Case B and the Opportunity Case have an expected accuracy range of −30% to +50%.
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

24.2 Alternate Development Cases

24.2.1 Summary Comparison

Each case is calculated in the same financial model as the Base Case, with the same metal prices, exchange rate, discount rate, valuation date, royalties, fiscal regime and SUDAM and ZPE assumptions, except where stated below. Table 24-1, Figure 24-1 and Figure 24-2 compare the cases.

Table 24-1: Comparison of the Base Case and alternative development cases

Indicator	Base Case	Alternate Case A	Alternate Case B	Opportunity Case
Description	Mine, concentrator and smelter built concurrently	Base Case mine and concentrator; staged smelter, first alloy in Year 3	Base Case configuration at 5 Mtpa	Base Case modifying factors applied to M, I and Inferred
Study class (AACE)	Class 4	Class 4	Class 5	Class 5
Study status	**PFS**	**PFS**	**PEA**	**PEA**
Mine life (years)	10	10	20	13
Ore processed, LOM (Mt)	86.7	86.7	86.8	123.3
Waste mined, LOM (Mt)	575.9	575.9	544.0	781.9
Strip ratio (t/t)	6.64	6.64	6.27	6.34
Peak mining rate (Mtpa)	81.4	81.4	41.6	104.0
Peak plant throughput (Mtpa)	10.03	10.03	5.00	10.00
Payable Pd, LOM (koz)	2,088.9	2,031.8	2,069.8	2,912.1
Payable Pt, LOM (koz)	1,507.5	1,463.9	1,495.2	1,985.2
Payable Rh, LOM (koz)	160.7	155.7	159.6	207.1
Payable Au, LOM (koz)	180.9	174.6	181.0	229.4
Payable Ni, LOM (kt)	98.2	93.7	99.2	128.0

Indicator	Base Case	Alternate Case A	Alternate Case B	Opportunity Case
Average annual payable 4E (koz)	393.8	382.7	195.3	410.3
Growth capital (US$ M)	784.9	603.6	491.1	784.9
Sustaining capital, LOM (US$ M)	98.2	265.2	98.2	98.2
Closure and rehabilitation (US$ M)	25.6	25.6	25.6	25.6
Total capital, LOM (US$ M)	908.7	894.4	614.9	908.7
After-tax NPV8 (US$ M)	1,448.7	1,213.5	1,151.2	1,777.1
After-tax IRR (%)	35.1	31.5	30.6	32.0
Payback, post-tax (years)	2.0	2.5	2.5	2.5
Peak funding (US$ M)	877	718	547	948
C1 cash cost (US$/oz 4E)	478	526	488	509
AISC (US$/oz 4E)	706	797	718	729

Note: sustaining capital excludes closure, which is shown separately. In Alternate Case A, sustaining capital includes the US$ 181.3 M smelter capital spent after the start of production. Payback is from the start of production. Alternate Case B and the Opportunity Case are PEAs; see the cautionary statement in Section 24.1.1.
Source: GE21 and Bravo, 2026; PFS financial model.

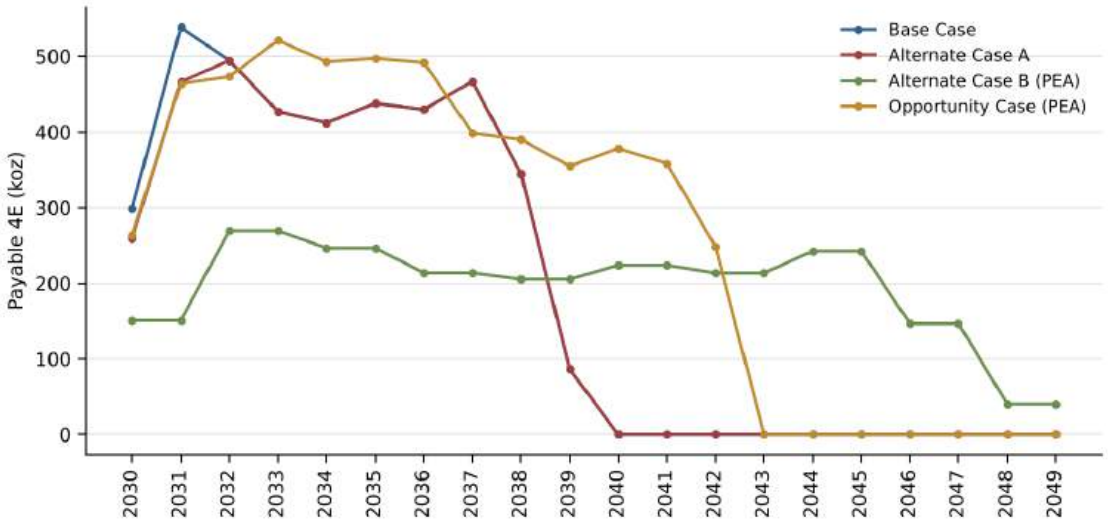


Figure 24-1: Annual payable 4E production by case

Source: GE21 and Bravo, 2026; PFS financial model.

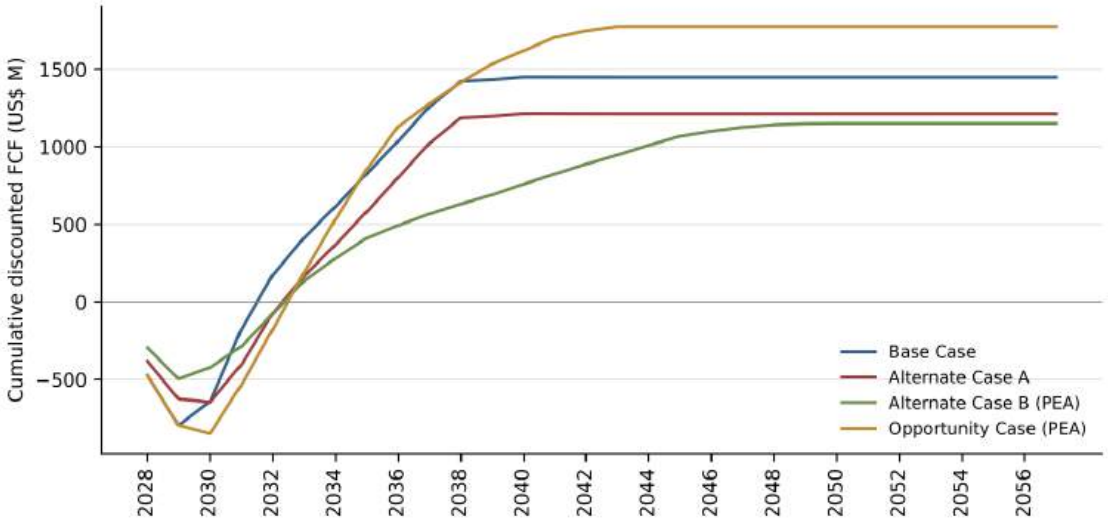


Figure 24-2: Cumulative discounted free cash flow by case

Source: GE21 and Bravo, 2026; PFS financial model.

24.2.2 Alternate Case A – Staged Smelter Development

Alternate Case A keeps the Base Case mine plan, concentrator, development schedule and production scale and stages the development of the Barcarena ZPE smelter. Its purpose is to test the potential to reduce the initial capital and up-front funding requirement, with the smelter funded from operating cash flow. Mine-site construction is assumed to start in January 2028 and first plant feed is assumed to occur in January 2030, as in the Base Case. Smelter construction is assumed to start in January 2030 and first alloy is assumed to be produced in January 2032 (Year 3). Alternate Case A was evaluated at pre-feasibility level, with the same Class 4 capital and operating cost estimates as the Base Case.

During 2030 and 2031, it is assumed that the 269.7 kt of concentrate produced is sold at the mine gate to third-party smelters and refiners. The concentrate sales terms in the model are payabilities of 85% for Pd and Pt, 84% for Rh and Au and 72% for Ni, no treatment charge, a nickel refining charge of US$ 400/t of payable nickel, penalties for MgO above 20% and Cr2O3 above 2% (not triggered by the concentrate specification), road and rail haulage to Barcarena of US$ 47.73/t, ocean freight to an overseas smelter of US$ 359.16/t and insurance at 0.15% of value. These terms are indicative and no concentrate offtake has been negotiated. From 2032, it is assumed that all concentrate is smelted on the Base Case terms.

The smelter capital of US$ 181.3 M (ZPE basis) is assumed to be spent in 2030 (US$ 95.0 M) and 2031 (US$ 86.2 M). Because it is incurred after the start of production and is funded from operating cash flow, it is reported as sustaining rather than growth capital. This reclassification does not change the cash flow, tax or NPV, but it raises the AISC. The smelter refractory campaign carried in the seventh year of smelter operation in the Base Case (US$ 14.3 M) is not included in Alternate Case A.

The key differences between Alternate Case A and the Base Case are presented in Table 24-2.

Table 24-2: Alternate Case A – key differences from the Base Case

Item	Unit	Base Case	Alternate Case A	Difference
Concentrate sold at mine gate	kt	–	269.7	+269.7
Concentrate smelted	kt	1,466.7	1,197.1	(269.7)
Alloy produced	kt	146.7	119.7	(27.0)
Sulphuric acid produced	kt	878.1	716.7	(161.4)
Total payable 4E	koz	3,937.9	3,826.0	(111.9)
Payable Ni	kt	98.2	93.7	(4.5)
Gross revenue	US$ M	9,322.9	8,934.5	(388.3)
Net revenue	US$ M	8,346.4	7,894.9	(451.5)
Total operating cost	US$ M	4,124.2	4,072.5	(51.7)
Growth capital	US$ M	784.9	603.6	(181.3)
Sustaining capital (incl. smelter in A)	US$ M	98.2	265.2	+167.0
Peak funding requirement	US$ M	877	718	(159)
After-tax NPV8%	US$ M	1,448.7	1,213.5	(235.2)
After-tax IRR	%	35.1	31.5	(3.6)

Note: brackets denote a reduction. Totals may not add due to rounding.
Source: GE21 and Bravo, 2026; PFS financial model.

Alternate Case A reduces the growth capital by US$ 181.3 M and the peak funding requirement by US$ 159 M, at a cost of US$ 235.2 M of NPV. The NPV reduction comes from the lower payabilities, higher freight and lower acid revenue during the two years of concentrate sales, and from deferring the higher-margin smelter cash flows. The concentrate assumed to be sold in 2030 and 2031 also carries the shipping, payability and counterparty risk of the concentrate market, which the Base Case avoids. NPV/peak funding is similar for the two cases (1.69 for Alternate Case A against 1.65 for the Base Case).

24.2.3 Alternate Case B – Reduced Throughput of 5 Mtpa (PEA)

Alternate Case B applies the Base Case configuration, with the concentrator and the Barcarena ZPE smelter constructed concurrently, but at a peak plant throughput of 5 Mtpa instead of 10 Mtpa, as a lower up-front capital alternative. It was prepared to a PEA level of confidence and does not form part of the PFS. The assumed production schedule maps the Base Case Mineral Reserve schedule onto half the annual rate, giving 86.8 Mt processed over a 20-year mine life at a peak mining rate of 41.6 Mtpa. The schedule was derived from the Base Case schedule rather than re-optimized for the lower rate; the waste mined is 544.0 Mt against 575.9 Mt in the Base Case. Metallurgical recoveries and per-tonne variable costs are unchanged from the Base Case.

The capital cost was factored from the Base Case estimate by area, using the capacity ratio raised to an area-specific scaling exponent (Table 24-3). For the smelter areas, the ratio of the reduced smelter nameplate (100 ktpa) to the Base Case nameplate (200 ktpa) was used. Mine-site sustaining capital and closure keep their Base Case totals, spread over twice the number of years. Fixed G&A costs were scaled by 0.5 raised to the power 0.75, and fixed mine management costs similarly. Alternate Case B assumes the SUDAM incentive is renewed for a second ten-year term, which the Base Case does not assume, and includes an embedded 15% contingency in all areas.

Table 24-3: Alternate Case B – capital cost factorization

Area	Scaling exponent	Base Case (US$ M)	Alternate Case B (US$ M)
Mining and pre-production	1.00	19.7	9.9
Process plant – mechanical equipment	0.70	280.6	172.8
Civil works, buildings and earthworks	0.75	51.6	30.7
Electrical, substation and distribution	0.75	65.3	38.8
Tailings storage facility (starter)	0.90	14.5	7.8
Water storage dam	0.90	15.2	8.1
Indirect costs	0.70	30.2	18.6
Owner’s costs	0.80	47.7	27.4
EPCM	0.80	71.6	41.1
Insurance and taxes	0.80	7.2	4.1
Mine site growth capital		603.6	359.3
ZPE smelter growth capital (ZPE basis)	0.60	181.3	131.8

Area	Scaling exponent	Base Case (US$ M)	Alternate Case B (US$ M)
Total growth capital		784.9	491.1
Sustaining capital		98.2	98.2
Closure and rehabilitation		25.6	25.6
Total capital		**908.7**	**614.9**

Note: Class 5 estimate. Capacity ratio 0.50 for mine site areas and 0.59 for smelter areas.
Source: GE21 and Bravo, 2026; PFS financial model.

Alternate Case B gives an after-tax NPV8 of US$ 1,151.2 M and an IRR of 30.6%, with growth capital of US$ 491.1 M, peak funding of US$ 547 M and average payable production of 195.3 koz/y of 4E over 20 years. The NPV is US$ 297.4 M below the Base Case, as the same Mineral Reserve is recovered over twice the period, while the NPV/peak funding ratio rises to 2.10. The total operating cost of US$ 4,163.0 M is slightly above the Base Case, as the fixed G&A accrues over a longer life (US$ 541.9 M against US$ 475.1 M).

24.2.4 Opportunity Case (PEA)

The Opportunity Case tests the potential effect on the Project of converting part of the current Inferred Mineral Resources to the Measured and Indicated categories and is therefore prepared on the basis of a PEA and does not form part of the PFS. A separate pit optimization, mine design and production schedule was prepared on the whole current Mineral Resource block model, applying the updated PFS modifying factors, inputs and assumptions, and the resulting schedule was evaluated in the PFS financial model with the same economic and financial parameters as the Base Case. The mine schedule contains 123.3 Mt of mineralized material, of which 33.1 Mt (27%) is Inferred, equal to 44% of the current non-oxide Inferred Mineral Resource. Waste mined is 781.9 Mt, a strip ratio of 6.34, and the mine life is 13 years at a plant throughput of 10 Mtpa, with a peak mining rate of 104 Mtpa.

The Opportunity Case applies the Base Case capital cost estimate without change: growth capital of US$ 784.9 M, sustaining capital of US$ 98.2 M and closure of US$ 25.6 M. No allowance has been made for the additional TSF capacity, WRSF and stockpile preparation and closure that the larger tonnage would require. Operating costs are calculated from the Base Case unit rates. As in Alternate Case B, the SUDAM incentive is assumed to be renewed for a second term.

Table 24-4 presents the physical and economic results for the Opportunity Case.

Table 24-4: Opportunity Case – physical and economic results

Item	Unit	Base Case	Opportunity Case
Material processed	Mt	86.7	123.3
of which Inferred	Mt	–	33.1
Waste mined	Mt	575.9	781.9
Mine life	years	10	13
Head grade (PdEq)	g/t	2.88	2.70
Concentrate produced	kt	1,466.7	2,085.6
Total payable 4E	koz	3,937.9	5,333.8
Average annual payable 4E	koz/y	393.8	410.3
Payable Ni	kt	98.2	128.0

Item	Unit	Base Case	Opportunity Case
Net revenue	US$ M	8,346.4	11,108.1
Total operating cost	US$ M	4,124.2	5,700.9
Total capital	US$ M	908.7	908.7
After-tax NPV8	US$ M	1,448.7	1,777.1
After-tax IRR	%	35.1	32.0
Peak funding requirement	US$ M	877	948
C1 cash cost	US$/oz 4E	478	509
AISC	US$/oz 4E	706	729

Note: the Opportunity Case is a PEA and includes Inferred Mineral Resources; see Section 24.1.1.
Source: GE21 and Bravo, 2026; PFS financial model.

The Opportunity Case extends the mine life by approximately three years and adds 1,396 koz of payable 4E and 29.8 kt of payable nickel. The after-tax NPV8 increases by US$ 328.4 M to US$ 1,777.1 M. The IRR is lower than the Base Case (32.0% against 35.1%) because the larger pit moves more waste in the first production year (104 Mt in 2030 against 63 Mt in the Base Case) and the head grade is lower (2.70 g/t against 2.88 g/t PdEq). The case supports the infill drilling programme described in Section 24.3.1, whose aim is to convert Inferred Mineral Resources within and below the current pit shells to the Measured and Indicated categories.

24.3 Project Opportunities

24.3.1 Potential Mineral Resource and Mineral Reserve Growth

The Mineral Reserve of 86.7 Mt grading 2.77 g/t PdEq, as detailed in Section 15, is derived from the Measured and Indicated Mineral Resource with an effective date of February 18, 2025, with conversion ratios of 55% on tonnes and 74% on contained PdEq. The risks and opportunities analysis of the 2025 PEA concluded that a significant opportunity exists to increase the Mineral Resources and recommended further drilling and expanded exploration. Bravo's 2026 field programme, announced on February 18, 2026, was designed in response: it comprises 28,000 m of drilling, of which approximately 22,000 m is infill and extensional drilling at the Luanga deposit and approximately 6,000 m is directed at regional and deep targets. The infill component targets the conversion of Inferred Mineral Resources to the Measured and Indicated categories.

The current Mineral Resource was estimated from 531 drill holes for 108,343 m, including 8,714 m of trenching. A further 81 drill holes for 22,100 m, assuming they are all completed as scheduled, are planned for inclusion in the next Mineral Resource update, targeted for the first half of 2027, of which 69 holes for 19,227 m had been completed at the date of the PFS news release of September 21, 2026. Drilling completed after the February 18, 2025 effective date of the 2025 Mineral Resource estimate is not reflected in the Mineral Resource in Section 14 or the Mineral Reserve in Section 15 or in the economic analysis of Section 22 of this Report.

Bravo has extended the drill-defined Main Sulphide Zone to approaching 350 m below surface in the Central Sector, to approximately 200 m in the North Sector and to shallower depths in the Southwest Sector. Deeper drilling in the Central Sector has returned mineralized thicknesses equal to or greater than those of the shallower drilling, with PGM+Au grades generally comparable to, or higher than, those above and sulphide nickel grades increasing with depth. Mineralization

remains open below the current base of drilling along essentially the full 8.1 km strike length of the Luanga Complex and, on Bravo's interpretation, at depths potentially economically accessible by open pit mining. The exploration potential is described further in Section 24.4.

24.3.2 Potential Copper Credit

Copper has not been included in the Mineral Resource estimate, as its grade and contribution to the Project economics were assessed as immaterial, and no copper revenue is included in the Base Case or other cases. Copper is nonetheless present in the mineralized system and has been assayed routinely in the metallurgical test work, in both feed and concentrate. Test work to date has returned copper recoveries to concentrate of 89% to 95% of the head grades, with copper grades of 2% to 3% in the final concentrate. The CZPE authorization of Bravo Metals covers the manufacture of nickel-PGM alloys containing copper.

At the long-term consensus mean copper price of US$ 10,516/t and an assumed smelter payable of 80% and were copper levels to be confirmed at the levels indicated from the metallurgical test work, the recovered copper could generate approximately US$ 29 M to US$ 43 M of additional revenue per year at the peak concentrate production rate, or approximately US$ 290 M to US$ 430 M over the LOM, with no associated increase in mining or processing costs. Table 24-5 sets out the basis.

Table 24-5: Indicative copper credit

Item	Unit	Low	High
Copper grade in concentrate	%	2.0	3.0
Concentrate production (peak year)	ktpa	~170	~170
Payable copper (80%)	tpa	~2,720	~4,080
Copper price	US$/t	10,516	10,516
Annual revenue at peak concentrate production	US$ M/y	~29	~43
LOM revenue, PFS basis (10 years at peak rate)	US$ M	~290	~430

Note: indicative only. Copper is not included in the Mineral Resource, Mineral Reserve or economic analysis.
Source: Bravo, 2026.

Of the approximately 137,500 sample intervals in the drill database, copper has been assayed in about 57%: approximately 99% of the historical Vale drilling but approximately 34% of Bravo's drilling. Bravo intends to evaluate options to include copper in a future Mineral Resource estimate, through additional systematic copper assaying of pulps and core and statistical grade estimation methods.

24.3.3 Vertical Integration Flexibility

Vertical integration improves the net smelter return of the metal contained in concentrate through lower treatment and logistics costs and higher final payabilities. The 10-to-1 reduction in mass from concentrate to alloy reduces the Project's exposure to the volatility of maritime shipping costs and reduces its carbon footprint accordingly. The smelter is designed for approximately 200 ktpa of concentrate against a Base Case peak of approximately 170 ktpa. This spare capacity offers two opportunities:

- Reducing the concentrate grade target in the flotation circuit to increase metal recovery. Higher recoveries at a lower concentrate grade may improve the overall Project economics, given the low operating cost environment of the ZPE. This would require further flotation test work and a trade-off of concentrator recovery against smelter throughput and cost;
- Toll treatment of third-party nickel or PGM concentrates within the ZPE, with profits from toll treatment potentially offsetting some of the smelter operating costs.

24.3.4 Owner-Operated Mining Fleet

The Base Case assumes contract mining. A trade-off study compared an Owner-operated fleet with the contractor fleet and concluded that mine unit costs could be reduced by up to 43%, for an additional up-front capital cost of approximately US$ 91.5 M and sustaining capital of approximately US$ 361 M over the LOM. An Owner's fleet was not adopted in the Base Case because of the relatively short mine life, but a more detailed trade-off, including hybrid fleet options, is planned for the next study phase. An Owner's or hybrid fleet may reduce the C1 cash cost, and the case for it strengthens with any extension of the mine life. A further opportunity to be considered in future studies is leasing an owner-operated mining fleet, deferring the additional capital cost associated with an a purchased Owner's fleet.

24.3.5 Oxide Mineral Resources

The oxide Mineral Resource, 10 Mt grading 1.51 g/t PdEq in the Measured and Indicated categories and 3 Mt grading 1.57 g/t PdEq in the Inferred category, is excluded from the Base Case. Oxide material is mined and stockpiled during pre-stripping, and its mining cost is included in the Base Case. Metallurgical testing indicates that economic recoveries are possible (the 2025 Mineral Resource estimate assumed recoveries of 81% Pd, 23% Pt, 54% Rh and 90% Au from oxide to a PGM residue). Further test work is planned. Processing oxide material would require additional plant components that are not included in the PFS.

24.3.6 Copper-Gold Discoveries Outside the Luanga Deposit

Copper-gold discoveries on the Project outside the Luanga deposit, principally the T5 and T16 targets as well as others summarized in Section 9 of this Report, are not included in the PFS and may present additional opportunities. They are summarized in Section 24.4.2.

24.4 Exploration Upside

This Section summarizes the exploration potential of the Luanga property as described in Section 9, to which the reader is referred for the supporting data, figures and sample results. The potential quantity and grade of the exploration targets described below is conceptual in nature. There has been insufficient exploration to define a Mineral Resource in respect of them, and it is uncertain whether further exploration will result in the estimation of a Mineral Resource. None of the exploration potential is included in the Mineral Resource, the Mineral Reserve, the mine plan or the economic analysis of this Report.

24.4.1 PGM Exploration Potential

The Luanga mafic-ultramafic Complex has a strike length of approximately 8.1 km, divided from north to south into the North, Central and Southwest Sectors. Vale's drilling between 1992 and 2003 tested principally the upper 100 m to 150 m of the deposit. The exploration potential for PGM mineralization falls into three groups (Table 24-6).

Table 24-6: PGM exploration targets

Target	Mineralization style	Favourable indicators	Stage
Central Sector at depth	Stratiform PGM+Au+Ni	Main Sulphide Zone open below current drilling, within potential open pit depths	Resource extension
North Sector at depth	Stratiform PGM+Au+Ni	Defined to ~200 m; grades and widths improving with depth	Resource extension
Southwest Sector	Stratiform PGM+Au+Ni	Shallowest drill coverage; pit shell reaches limit of drilling; further mineralized zones higher in the stratigraphy	Resource extension
LU330 target (North Sector)	Magmatic Ni-Cu-PGM massive sulphide	Massive sulphide with chromitite in the Transition Zone; BHEM plates	Drilling / BHEM
Babylon target	Hydrothermal Ni-Cu (± PGM) massive sulphide	Magnetic high, EM conductor and gravity high; deep IP anomaly	Drilling / BHEM
Crescent Zone	Stratiform PGM+Au	Repeated Transition Zone within the Mafic Zone; first drilled 2026	Early drill testing
Luanga South Complex	Stratiform PGM+Au	Same stratigraphy as Luanga over 6.5 km of strike; 12 of 24 historical holes mineralized	Not yet explored by Bravo

Source: Bravo, 2026; section 9 of this Report.

24.4.1.1 Extension of the deposit

In the Central Sector, which contains the largest proportion of the Mineral Resource, the 2026 programme has extended the Main Sulphide Zone from approximately 200 m to approaching 350 m below surface, with equal or greater thickness, equal or higher PGM+Au grade and increasing sulphide nickel grade with depth. Intercepts from the 2026 programme include 51.0 m at 3.67 g/t PGM+Au and 0.33% sulphide Ni from 259.25 m in DDH26LU312, including 23.0 m at 5.59 g/t PGM+Au, and 50.0 m at 2.55 g/t PGM+Au from 296.60 m in DDH26LU316. The North Sector is defined only to about 200 m, in a setting that the Central Sector shows to be mineralized below 350 m, and it carries localized high-grade rhodium (for example 58.3 m at 3.88 g/t PGM+Au including 0.25 g/t Rh from 21.50 m in DDH24LU235). In the Southwest Sector, the constraining pit shell of the 2025 Mineral Resource reaches the limit of the drilling data, and recent drilling has confirmed that the Main Sulphide Zone keeps a substantial and continuous thickness where it had been interpreted to narrow.

24.4.1.2 Massive sulphide styles within the deposit area

Drill holes DDH26LU330 and DDH26LU347, in the southern part of the North Sector, intersected magmatic massive to semi-massive Ni-Cu-PGM sulphide associated with a chromitite interval in the Transition Zone, including 13.44 m at 1.55% Ni, 0.33% Cu and 2.02 g/t PGM+Au from 205.58 m in DDH26LU347. Euhedral chromite enclosing sulphide blebs indicates the coexistence of chromite with an immiscible Ni-PGE sulphide liquid, a magmatic origin directly

related to the Luanga intrusion. DDH26LU347 was sited on a BHEM conductor plate modelled from DDH26LU330 and intersected the sulphide as predicted. At the Babylon target, stratigraphically below the North Sector, drilling intersected sulphide-matrix breccias reaching semi-massive and massive textures in a Fe-Ca-K metasomatic zone, including 11.04 m at 2.04% Ni, 1.23% Cu and 4.24 g/t PGM+Au from 131.11 m in DDH22LU047. A deep IP traverse completed in August 2026 identified strong chargeability with coincident low resistivity at depth, interpreted as the Babylon sulphide extending deeper, and more steeply, than drilled to date. At Crescent, a repetition of the Transition Zone within the norites of the Mafic Zone, drilling in 2026 intersected PGM mineralization comparable to that of the deposit.

24.4.1.3 Luanga South Complex

The mineral property covers 6.5 km of strike of the Luanga South Complex, which carries the same stratigraphy as the Luanga Complex. Vale completed a limited IP survey and scout drilling programme: 12 of 24 holes returned PGM+Au mineralization, at average grades of 0.29 g/t to 14.17 g/t PGM+Au. Bravo has not yet started exploration of this area.

24.4.2 Copper-Related Exploration Potential

A property-wide helicopter-borne time-domain electromagnetic and magnetic (HeliTEM) survey flown in 2023 over 99.7 km² identified 17 Priority 1 and 16 Priority 2 electromagnetic targets which, integrated with the structural and geochemical data, defined a pipeline of copper-gold and magmatic nickel-copper targets outside the Luanga deposit. Drill testing from 2024 discovered high-grade copper-gold mineralization at T5, and massive to brecciated sulphide at T6 and T16. In late 2025, a review of the geology classified the copper-gold mineralization outside the deposit as iron sulphide copper-gold (ISCG), a type closely related to IOCG deposits within the broader copper-gold-iron mineral system, in which iron oxides are replaced by pyrrhotite and pyrite and quartz and tourmaline are more abundant. The principal targets are summarized in Table 24-7.

Table 24-7: Principal copper exploration targets

Target	Style	Status and best results
T5	Cu-Au ISCG	28 holes for 5,926 m; sulphide in 19 holes. 11.48 m at 14.27% Cu and 3.33 g/t Au from 165.62 m (DDH2405T002), including 2.86 m at 22.91% Cu; 17.42 m at 3.49% Cu and 0.95 g/t Au (DDH2405T009). Tabular, near-vertical sulphide body about 250 m long and 150 m deep, controlled by a WNW-ESE fault
T16	Cu-Ni ISCG	7 holes. 6.54 m at 2.25% Cu and 0.55% Ni from 82.96 m (DDH2516T005); 1.29 m at 2.39% Cu and 1.27% Ni (DDH2316T001)
T6	Cu-Au ISCG / Ni-Cu-PGM magmatic	4 holes. 6.27 m at 0.41% Cu and 0.10% Ni (DDH2406T002); coincident EM, magnetic and gravity anomalies
T1	Cu-Au IOCG/ISCG	2 holes. Semi-massive sulphide, 0.19% Cu and 0.13% Ni; very strong HeliTEM conductor
T17	Ni-Cu-PGM magmatic / ISCG	3 holes within the Central Sector. 2.0 m at 0.51% Cu (DDH2517T004); malachite-bearing gossans
Lizard	Cu-Au IOCG/ISCG	Cu-in-soil anomaly over 1,800 m by 600 m; auger to 0.62% Cu over 6.3 m; trench 6.75 m at 0.40% Cu; first two drill holes unmineralized
Taurus	Cu-Au IOCG/ISCG / Ni-Cu-PGM	Four Cu-in-soil zones; trench 14.9 m at 0.46% Cu and 0.10 g/t Au; best drill intercept 5 m at 0.41% Cu

Target	Style	Status and best results
Orion	Cu-Au IOCG/ISCG	800 m east of T5; 8.50 m at 0.58% Cu from 4.1 m (DDH25OR002)
Gemini	Cu-Au IOCG/ISCG	Five Cu-in-soil zones over HeliTEM conductors T39 to T45; drill targets being generated
Jupiter, Scorpion, Lynx, Perseus, Musca	Cu-Au IOCG/ISCG	Magnetic and EM targets on regional structures; Jupiter historical hole 21 m at 0.31% Cu; work to be started

Note: intervals are downhole lengths.
Source: Bravo, 2026; section 9 of this Report.

The 2026 Phase 1 programme comprises target refinement by mapping and soil and rock-chip sampling, approximately 50 line-km of deep IP surveying designed to investigate depths beyond 300 m, fixed-loop and borehole TEM and structural work at T5 and T16 to trace the high-grade shoots along strike and at depth, and a first-phase drill programme of 8,000 m on the highest-ranked targets. The deep IP survey also covers parts of the Luanga deposit, to assess deep extensions of the PGM+Au+Ni mineralization and additional ISCG or IOCG occurrences. No Mineral Resource has been estimated for any copper target.

The copper potential of the Project is therefore twofold: copper contained in the Luanga PGM deposit, which could become a by-product credit through the existing concentrator and smelter (Section 24.3.2), and a pipeline of ISCG copper-gold and magmatic nickel-copper targets that may, in future, provide additional feed or stand-alone opportunities, the closest of which (T5 and Orion) lie within approximately 1 km of the Luanga Complex.

25 INTERPRETATION AND CONCLUSIONS

25.1 Introduction

The QPs note the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for this Report.

25.2 Principal Results and Interpretations

The Luanga deposit hosts stratiform PGM + Au + Ni mineralization, concentrated in the Main Sulphide Zone, along an arc-shaped strike of approximately 8.1 km. Drilling, sampling, QA/QC and data verification meet industry practice, and historical Vale data were validated by resampling. The Mineral Resource Estimate (effective date February 18, 2025) comprises 158 Mt at 2.04 g/t PdEq (10.4 Moz PdEq) in the Measured and Indicated categories and 78 Mt at 2.01 g/t PdEq (5.0 Moz PdEq) in the Inferred category, as detailed in Section 14. Mineralization remains open at depth along essentially the full strike length, and exploration has identified additional Ni-Cu-PGM and Cu-Au targets within the property that warrant further work.

As detailed in Section 15, Proven and Probable Mineral Reserves total 86.7 Mt at 2.77 g/t PdEq (7.7 Moz PdEq), supporting a 10-year conventional open pit operation at a strip ratio of 6.6:1 and a plant feed rate of 10 Mtpa after a one-year ramp-up.

Laboratory and pilot-scale testwork supports a flowsheet of HPGR, ball and vertical milling to a P80 of 26–30 µm, followed by Jameson Cell flotation, with projected recoveries of 81.5% Pd, 80.0% Pt, 55.0% Rh, 88.0% Au and 70.6% Ni. In the Base Case, the concentrate is treated at a Bravo-owned smelter in the Barcarena ZPE, producing a PGM-nickel alloy and sulphuric acid. The smelting route is supported by pilot and demonstration campaigns on analogous feed and by confirmatory bench-scale testwork on Luanga concentrate.

The site would connect to the National Interconnected System at 230 kV, and plant water demand is met from reclaim water and runoff. Preliminary Licence No. 2062/2025 is valid until February 2030, and the Project does not overlap Conservation Units, Indigenous lands or other protected areas.

In the Base Case, growth capital is estimated at US$ 784.9 M (including US$ 181.3 M for the smelter) and total LOM capital is US$ 908.7 M, estimated to AACE Class 4 (−20% / +30%) with a 15% contingency. The Base Case returns an after-tax NPV8 of US$ 1,448.7 M, an IRR of 35.1% and a payback of 2.0 years, at a C1 cash cost of US$ 478/oz 4E and an AISC of US$ 706/oz 4E. The NPV remains positive with metal prices 30% below the Base Case.

25.3 Significant Risks and Uncertainties

Table 25-1 summarizes the most significant risks to the reliability of the estimates and to the economic viability of the Project. Other risks are typical of mining projects at this stage of study.

Table 25-1: Significant risks and opportunities

Risk area	Description	Potential impact
Metal prices	PGM prices were highly volatile through 2026 and forecasts diverge widely; nickel remains in structural surplus.	Highest sensitivity: a 10% change in prices changes the NPV by about US$ 455 M. At −30%, the NPV falls to US$ 81 M (IRR 10.0%).
Fiscal incentives	The SUDAM income tax incentive requires application and approval. The economics also rely on the ZPE regime. These have not been applied for as yet.	The SUDAM benefit totals US$ 628 M over the LOM. Its loss would materially reduce the NPV.
Mineral Resources	Additional drilling conducted since the effective date of the MRE may affect the MRE once results are incorporated.	An updated MRE may result in a different production profile, which may affect the project economics
Smelter and offtake	Smelter layout is conceptual. Site geotechnical and groundwater studies are pending. Continuous converting has not been piloted. Environmental and construction permits are outstanding. No offtake or toll refining agreements are in place.	If smelter start-up is deferred (Alternate Case A), the NPV falls by about US$ 235 M, with exposure to concentrate placement risk, payability and counterparty risk.
Sulphuric Acid	The Base case assumes sale of sulphuric acid, reducing smelter operating costs.	Were the sulphuric acid prices lower than that assumed, smelter costs could increase.
Metallurgy	Talc, ultra-fine grinding and large-scale Jameson Cell performance carry scale-up risk. The 2026 testwork programmes are not yet complete.	Lower recoveries or concentrate grade would reduce revenue and affect smelter feed.
Permitting and tenure	Design changes since the LP must be submitted to SEMAS/PA, and the LI and LO are pending. The Mining Licence Application (2013) awaits an ANM decision. Land access agreements are renewable every two years.	Delays in permitting would defer construction start beyond the assumed January 2028 start and reduce the NPV.
Geotechnical and hydrological data	Pit slopes are at PFS level. No site investigation has been carried out at the TSF and water dam sites. Hydrological data are preliminary.	Flatter slopes would increase stripping or reduce Reserves. Structure redesign could increase capital.
Capital and operating costs	The estimate is Class 4 accuracy. Mining is by contractor, and waste mining represents about half of LOM operating cost.	A 10% change in operating cost changes the NPV by about US$ 215 M; a 10% change in capital cost changes it by about US$ 70 M.

Source: GE21, 2026.

25.4 Conclusions of the Qualified Persons

The QPs conclude that the exploration data, Mineral Resource and Mineral Reserve estimates, and engineering and cost inputs are adequate to support this Pre-feasibility Study. The Base Case shows positive economics and is resilient to cost variations. The Project is most exposed to PGM prices, the availability of fiscal incentives, and the execution and permitting of the Barcarena smelter. None of the risks identified is considered a fatal flaw, are typical of projects at this stage and can be reduced through the work recommended in Section 26.

The QPs conclude that the results support advancing the Project to a Feasibility Study.

26 RECOMMENDATIONS

The results presented in this Technical Report demonstrate that the Luanga project is technically and economically viable, based on the assumptions contained herein. It is recommended to continue developing the Project through additional studies, including a Definitive Feasibility Study (DFS), to de-risk the project and enhance confidence.

26.1 Sampling

- Acquire certified or standard reference materials certified for sulphide Ni assay methods.
- Inclusion of coarse and fine duplicates as control samples of its QA/QC program. Coarse and fine duplicates might be collected as aliquots from the material after the mass-reduction processes employed following crushing and milling, respectively. This measure should help to evaluate the quality of the sample preparation processes.

26.2 Mineral Resource

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 2026 metallurgical test work programs;
- Mineral Resource estimation by the conditional simulation method, defining the SMU to determine recoverable Resource.
- Evaluate the inclusion of copper in a future Mineral Resource estimate, through additional systematic copper assaying of pulps and core and statistical grade estimation methods.
- Additional targets with potential for IOCG/ISCG and magmatic Ni-Cu-(PGM) deposits have been defined at the Luanga property and require additional exploration work for their potential to be assessed.

26.3 Mining Methods

- Additional geotechnical investigations and stability analyses are recommended to refine the pit, dumps, stockpiles, TSF and water dam designs criteria in the next phase of study.
- Hydrogeological and hydrological studies for drainage, dewatering and runoff coefficient assumptions to be optimized.
- Sensitivity to the degree of alteration (saprolite) is high; therefore, ore blending is recommended as a “geometallurgical safeguard” to protect the concentrate grade against dilution in zones of greater alteration.

- Conduct a dedicated drill and blast study for the oxide and transition zones. It should define the free dig/blast boundary, appropriate blast designs and powder factors, and expected fragmentation, supported by additional geotechnical and geomechanical characterization of these materials.
- A detailed cost-benefit analysis is also recommended, considering the following:
  - Adding multiple waste dumps closer to the mining fronts is expected to reduce haulage distances for waste disposal and, consequently, minimize overall equipment requirements along the LOM.
  - Trade-off study on wider haul roads and ramps that would allow larger-class loading and haulage equipment. The study should weigh the potential reduction in unit mining costs against the additional waste stripping and the effect on pit slope geometry.
  - Assess dedicated haul roads for the different truck classes, separating the production fleet from auxiliary and light vehicle traffic. This would improve operational safety and haulage efficiency.

26.4 Recovery Methods

- Additional validation test work regarding:
  - Metallurgical recoveries.
  - Equipment selection and sizing.
  - Detailed comminution test work to the Central and Southwest sector global composites.
  - Further test work for oxide material.
- Evaluation of the use of paste thickeners to reduce the volume of water sent to the Tailings Storage Facility.

26.5 Project Infrastructure

- Geotechnical drilling with in-situ permeability tests to confirm the values of the parameters adopted for infiltration. Long-term hydrological models using Monte Carlo simulation, based on 50-year historical data series, to validate the resilience of the water reservoir in the face of multi-year drought cycles.
- Drilling, characterisation and strength testing of borrow materials, permeability testing, and testing of the tailings, to support stability, seepage and liquefaction analyses in the next engineering phase.
- Carry out a dedicated study to define the optimal separation and interface between the WRSF and the TSF. The study should assess the geometry of both structures, interaction effects on slope stability, and surface and contact water management at the interface. It should also consider the potential to use the waste rock for buttressing or embankment raises. The objective is to reduce construction and maintenance costs while ensuring that both structures operate safely throughout the life of mine.

26.6 Market Studies and Contracts

- Marketing strategy to evaluate and approach:

- Established South African and international toll smelters and PGM/nickel trading houses with the technical capability to treat a Ni-Cu-PGM concentrate of Luanga's expected mineralogy, were that option to be considered.
- Engineering, construction and technology partners capable of delivering smelting/converting capacity at Barcarena.
- Evaluate sulphuric acid offtake options.

26.7 Environmental Studies, Permitting, and Social or Community Impacts

- Updates to the project need to be subject of a specific assessment during subsequent licensing stages with SEMAS/PA. Since these updates alter the design and configuration evaluated during the licensing process that supported LP No. 2062/2025, they must be formally submitted to SEMAS/PA.

26.8 Smelter Studies

- Study to assess the reduction on concentrate grade target in the flotation circuit to increase metal recovery. Higher recoveries at a lower concentrate grade may improve the overall Project economics, given the low operating cost environment of the ZPE. This would require further flotation test work and a trade-off of concentrator recovery against smelter throughput and cost.
- Larger-scale confirmatory roasting and smelting test work on Luanga concentrate, of sufficient scale to close a reliable mass balance and to support vendor performance guarantees for the roaster, the furnace and the converter.
- Confirmation of the deportment of iron between alloy and slag at the selected operating point, since the degree of iron reduction sets both the alloy make and the converter duty.
- Confirmation of gold deportment through roasting, smelting and converting, which is presently taken to follow that of the platinum group metals.
- Selection of the converting technology. Continuous converting offers a single steady product composition and longer refractory life but has not been piloted, so batch converting on Peirce-Smith, top-blown rotary or top submerged lance equipment is the basis adopted.
- Selection of the carbonaceous reductant, between imported coke or coal and locally sourced charcoal, on the basis of further test work and the chemical requirements of the smelting configuration.
- A ground geotechnical investigation of the smelter site and a groundwater study to confirm sustainable yields, both of which affect the capital estimate.
- Inclusion of chlorine, fluorine and arsenic in the process model, and confirmation of their deportment to the off-gas cleaning circuit and to the acid product specification.

Table 26-1 summarizes the recommendations described in this section, and the corresponding estimated budgets proposed to advance the project through the DFS stage.

Table 26-1: Proposed budget to advance the Project to the Definitive Feasibility Study stage

PHASE 5A – Metallurgical testwork at Luanga	
Completion of any outstanding metallurgical testwork and optimization:	
Estimate	US$ 0.50 M
Continuation of carbon sequestration study:	
Estimate	US$ 0.05 M
Continued Metallurgical testwork and optimization	
Estimate	US$ 0.20 M
Sub-total – Phase 5A	**US$ 0.75 M**
PHASE 5B – Pre-feasibility Study (PFS) following favorable results from a PEA	
Completion of a PFS:	
Estimate	US$ 1.0 M
Sub-total – Phase 5B	**US$ 1.0 M**
PHASE 5C – Deep Drilling below the Luanga PGM+Au+Ni deposit	
Deep drilling at the Luanga PGM+Au+Ni deposit.	
8 holes @ ~500 m = 4,000 m @ US$ 450/m	
Estimate	US$ 1.8 M
Sub-total – Phase 5C	**US$ 1.8 M**
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$ 0.1 M
Drilling: 70 holes x 200 m for 14,000 m @ US$ 400/m (all inclusive)	US$ 5.6 M
Sub-total – Phase 5D	**US$ 5.7 M**
PHASE 5E – Deep Drilling outside the Luanga PGM+Au+Ni deposit	
Deep drilling at the Luanga PGM+Au+Ni deposit.	
3 holes @ ~800 m = 2,400 m @ US$ 450/m (all inclusive)	US$ 1.08 M
Sub-total – Phase 5E	**US$ 1.08 M**
PHASE 5F – Extension and Infill Drilling at Luanga	
Drilling to convert existing MRE Inferred resources to MRE Indicated resources, and define new MRE Inferred Resources:	
Drilling: 33 holes x 300 m for 9,900 m @ US$ 400/m (all inclusive)	US$ 3.96 M
Sub-total – Phase 5F	**US$ 3.96 M**
TOTAL – PHASE 5	**US$ 14.29 M**
PHASE 6 – Definitive Feasibility Study (DFS) following favorable results from a PFS	
Completion of a FS, including any required infill drilling, geotechnical drilling:	
Estimate	US$ 5.0 M
TOTAL – PHASE 6	**US$ 5.0 M**

Source: GE21, 2026.

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.

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APPENDIX A – CERTIFICATE OF QUALIFIED PERSON

I, **Porfirio Cabaleiro Rodriguez**, FAusIMM (#3136091), FAIG (#3708), as author of the technical report entitled “NI 43-101 Technical Report Pre-Feasibility Study on Luanga Project – Pará State, Brazil” (the Report), dated November 4, 2026, with an effective date of September 16, 2026, prepared for Bravo Mining Corp. (the Issuer), hereby certify that:

1. I am a mining engineer at GE21 Consultoria Mineral Ltda., located at 3130 Afonso Pena Ave., 9th Floor, Belo Horizonte, MG, 30130-910, Brazil.
2. I have the following academic qualifications:
   - B.Sc. in Mining Engineering from the Federal University of Minas Gerais, Belo Horizonte, Brazil.
3. I am a professional mining engineer with over 45 years of experience in the mining industry, with Mineral Resource and Mineral Reserves estimation, which includes numerous mineral properties in Brazil, including iron ore and manganese properties.
4. I am a registered professional fellow of the Australian Institute of Geoscientists (FAIG #3708) and fellow of the Australasian Institute of Mining and Metallurgy (FAusIMM #3136091).
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, my affiliation with a professional association as defined in NI 43-101, and my past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I visited the Project site between July 4 and 7, 2023, between October 3 and 6, 2023, between January 27 and 31, 2025, and between February 10 and 13, 2026.
7. I am responsible for Sections 14, 15, 16, 19, 21, 22, 23 and 24, and co-responsible for Sections 1, 18, 25, 26 and 27 of the Report.
8. As of the effective date of the Report, to the best of my knowledge, information and belief, the sections of which I am the author and for which I am responsible contain all the scientific and technical information that must be disclosed so the Report is not misleading.
9. As of the date of this certificate, I have no knowledge of any material fact or material change that is not reflected in the Report.
10. I am independent of the Issuer and of the property, applying all the tests set out in Section 1.5 of NI 43-101. I hold no material interest in the property, in the Issuer or in any of their related entities, and my relationship with the Issuer is strictly professional, being that of a client and an independent consultant. My fees are based on rates set by commercial agreement and are in no way dependent on the findings of the Report.
11. I have had prior involvement with the property that is the subject of the Report; the nature of my prior involvement is the preparation of the Mineral Resource Estimates for the Project with effective dates of October 22, 2023, and February 18, 2025, the latter presented in the technical report titled “Independent Technical Report on the Luanga PGM+Au+Ni Project, Pará State, Brazil”.
12. I have read NI 43-101 and Form 43-101F1 – Technical Report, and I believe the Report has been prepared in accordance with those documents.
13. I consent to the public filing of the Report by the Issuer with any stock exchange or other regulatory authority, and to its publication by them for regulatory purposes, including electronic publication in the public company files on their websites.

Belo Horizonte, Brazil, November 4, 2026.

<Signed and sealed in original>

_____

Porfirio Cabaleiro Rodriguez

FAusIMM #3136091, FAIG #3708

I, **Bernardo Horta de Cerqueira Viana**, FAIG (#3709), as author of the technical report entitled “NI 43-101 Technical Report Pre-Feasibility Study on Luanga Project – Pará State, Brazil” (the Report), dated November 4, 2026, with an effective date of September 16, 2026, prepared for Bravo Mining Corp. (the Issuer), hereby certify that:

1. I am a geologist at GE21 Consultoria Mineral Ltda., located at 3130 Afonso Pena Ave., 9th Floor, Belo Horizonte, MG, 30130-910, Brazil.
2. I have the following academic qualifications:
   - B.Sc. in Geology from the Federal University of Minas Gerais, Belo Horizonte, Brazil.
3. I am a professional geologist with over 25 years of experience in the mining industry. My experience relevant to the Report includes:
   - 25 years of experience in Mineral Resource estimation and geological exploration, including numerous mineral properties in Brazil, particularly manganese deposits.
4. I am a registered professional fellow of the Australian Institute of Geoscientists (FAIG #3709).
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, my affiliation with a professional association as defined in NI 43-101, and my past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I visited the Project site between October 3 and 6, 2023, and between January 27 and 31, 2025.
7. I am responsible for Sections 2 to 12, and co-responsible for Sections 1, 25, 26 and 27 of the Report.
8. As of the effective date of the Report, to the best of my knowledge, information and belief, the sections of which I am the author and for which I am responsible contain all the scientific and technical information that must be disclosed so the Report is not misleading.
9. As of the date of this certificate, I have no knowledge of any material fact or material change that is not reflected in the Report.
10. I am independent of the Issuer and of the property, applying all the tests set out in Section 1.5 of NI 43-101. I hold no material interest in the property, in the Issuer or in any of their related entities, and my relationship with the Issuer is strictly professional, being that of a client and an independent consultant. My fees are based on rates set by commercial agreement and are in no way dependent on the findings of the Report.
11. I have had prior involvement with the property that is the subject of the Report; the nature of my prior involvement is participation in the preparation of the Mineral Resource Estimates for the Project with effective dates of October 22, 2023, and February 18, 2025, the latter presented in the technical report titled “Independent Technical Report on the Luanga PGM+Au+Ni Project, Pará State, Brazil”, and of the Preliminary Economic Assessment presented in the technical report titled “NI 43-101 Preliminary Economic Assessment (PEA) Independent Technical Report for the Luanga PGM+Au+Ni Project Pará, Brazil”, dated November 28, 2025, with an effective date of July 7, 2025.
12. I have read NI 43-101 and Form 43-101F1 – Technical Report, and I believe the Report has been prepared in accordance with those documents.
13. I consent to the public filing of the Report by the Issuer with any stock exchange or other regulatory authority, and to its publication by them for regulatory purposes, including electronic publication in the public company files on their websites.

Belo Horizonte, Brazil, November 4, 2026.

<Signed and sealed in original>

_____

Bernardo Horta C. Viana

FAIG #3709

I, **Paulo Roberto Bergmann Moreira**, FAusIMM (#333121), as author of the technical report entitled “NI 43-101 Technical Report Pre-Feasibility Study on Luanga Project – Pará State, Brazil” (the Report), dated November 4, 2026, with an effective date of September 16, 2026, prepared for Bravo Mining Corp. (the Issuer), hereby certify that:

1. I am a mining engineer at GE21 Consultoria Mineral Ltda., located at 3130 Afonso Pena Ave., 9th Floor, Belo Horizonte, MG, 30130-910, Brazil.
2. I have the following academic qualifications:
   - B.Sc. in Mining Engineering from the Federal University of Minas Gerais, Belo Horizonte, Brazil.
3. I am a professional mining engineer with over 42 years of experience in the mining industry. My experience relevant to the Report includes:
   - 30 years in mining and plant operation management, including AngloGold, Yamana, Jaguar Mining and Buritirama Mineração;
   - 10 years as engineering development and consultancy in the mining industry, including gold, iron, manganese, rare earth elements and others.
4. I am a registered professional fellow of the Australasian Institute of Mining and Metallurgy (FAusIMM #333121).
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, my affiliation with a professional association as defined in NI 43-101, and my past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
6. As of the date of the Report, I have not visited the Project site.
7. I am responsible for Section 13, and co-responsible for Sections 1, 17, 18, 25, 26 and 27 of the Report.
8. As of the effective date of the Report, to the best of my knowledge, information and belief, the sections of which I am the author and for which I am responsible contain all the scientific and technical information that must be disclosed so the Report is not misleading.
9. As of the date of this certificate, I have no knowledge of any material fact or material change that is not reflected in the Report.
10. I am independent of the Issuer and of the property, applying all the tests set out in Section 1.5 of NI 43-101. I hold no material interest in the property, in the Issuer or in any of their related entities, and my relationship with the Issuer is strictly professional, being that of a client and an independent consultant. My fees are based on rates set by commercial agreement and are in no way dependent on the findings of the Report.
11. I have had prior involvement with the property that is the subject of the Report; the nature of my prior involvement is participation in the preparation of the Mineral Resource Estimate for the Project with an effective date of February 18, 2025, presented in the technical report titled “Independent Technical Report on the Luanga PGM+Au+Ni Project, Pará State, Brazil”.
12. I have read NI 43-101 and Form 43-101F1 – Technical Report, and I believe the Report has been prepared in accordance with those documents.
13. I consent to the public filing of the Report by the Issuer with any stock exchange or other regulatory authority, and to its publication by them for regulatory purposes, including electronic publication in the public company files on their websites.

Belo Horizonte, Brazil, November 4, 2026.

<Signed and sealed in original>

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Paulo Roberto Bergmann Moreira

FAusIMM #333121

I, **Rodney Trevor Jones**, Pr Eng (#880129), FSAIMM (#31229), as author of the technical report entitled “NI 43-101 Technical Report Pre-Feasibility Study on Luanga Project – Pará State, Brazil” (the Report), dated November 4, 2026, with an effective date of September 16, 2026, prepared for Bravo Mining Corp. (the Issuer), hereby certify that:

1. I am an independent consulting chemical engineer and metallurgist, located at 16 Weybridge, 50 Isipingo Road, Paulshof, Sandton, 2191, South Africa.
2. I have the following academic qualifications:
   - BSc (Eng) in Chemical Engineering from the University of the Witwatersrand, Johannesburg, South Africa.
   - BA in Logic and Philosophy from the University of South Africa, Pretoria, South Africa.
   - MSc (Eng) in Metallurgical Engineering from the University of the Witwatersrand, Johannesburg, South Africa.
   - PhD in Metallurgical Engineering from the University of the Witwatersrand, Johannesburg, South Africa.
3. I am a professional chemical and metallurgical engineer with over 40 years of experience in the minerals industry. My experience relevant to the Report includes:
   - 37 years of experience as a chemical and metallurgical engineer specializing in pyrometallurgy, working for Mintek (a national research organization in South Africa), where my work involved a wide variety of processes being taken from concept to industrial implementation. Included in this time is a period of approximately twenty years focused on the production of platinum group metals (PGMs).
   - 6 years of experience as an independent Consulting Pyrometallurgist.
4. I am a registered Professional Engineer (Pr Eng #880129) with the Engineering Council of South Africa, and a Fellow of the Southern African Institute of Mining and Metallurgy (FSAIMM #31229).
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, my affiliation with a professional association as defined in NI 43-101, and my past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
6. As of the date of the Report, I have not visited the Project site.
7. I am co-responsible for Sections 1, 17, 25, 26 and 27 of the Report.
8. As of the effective date of the Report, to the best of my knowledge, information and belief, the sections of which I am the author and for which I am responsible contain all the scientific and technical information that must be disclosed so the Report is not misleading.
9. As of the date of this certificate, I have no knowledge of any material fact or material change that is not reflected in the Report.
10. I am independent of the Issuer and of the property, applying all the tests set out in Section 1.5 of NI 43-101. I hold no material interest in the property, in the Issuer or in any of their related entities, and my relationship with the Issuer is strictly professional, being that of a client and an independent consultant. My fees are based on rates set by commercial agreement and are in no way dependent on the findings of the Report.
11. I have had no prior involvement with the property that is the subject of the Report.
12. I have read NI 43-101 and Form 43-101F1 – Technical Report, and I believe the Report has been prepared in accordance with those documents.
13. I consent to the public filing of the Report by the Issuer with any stock exchange or other regulatory authority, and to its publication by them for regulatory purposes, including electronic publication in the public company files on their websites.

Sandton, November 4, 2026.

<Signed and sealed in original>

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Rodney Trevor Jones

Pr Eng #880129, FSAIMM #31229

I, **Branca Horta de Almeida Abrantes**, MAIG (#8145), as author of the technical report entitled “NI 43-101 Technical Report Pre-Feasibility Study on Luanga Project – Pará State, Brazil” (the Report), dated November 4, 2026, with an effective date of September 16, 2026, prepared for Bravo Mining Corp. (the Issuer), hereby certify that:

1. I am an environmental consultant, geography and analysis at GE21 Consultoria Mineral Ltda., located at 3130 Afonso Pena Ave., 9th Floor, Belo Horizonte, MG, 30130-910, Brazil.
2. I have the following academic qualifications:
   - B.Sc. in Geography and Environmental Analysis from the UNI-BH, Belo Horizonte, Brazil.
   - MBA in Project Management from the Foundation Getúlio Vargas, Belo Horizonte, Brazil.
3. I am a professional environmental consultant, geography and environmental analysis with over 20 years of experience in the mining industry. My experience relevant to the Report is in the industrial, mining, energy, and sanitation sectors.
4. I am a registered professional member of the Australian Institute of Geoscientists (MAIG #8145).
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, my affiliation with a professional association as defined in NI 43-101, and my past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
6. As of the date of the Report, I have not visited the Project site.
7. I am responsible for Section 20, and co-responsible for Sections 1, 25, 26 and 27 of the Report.
8. As of the effective date of the Report, to the best of my knowledge, information and belief, the sections of which I am the author and for which I am responsible contain all the scientific and technical information that must be disclosed so the Report is not misleading.
9. As of the date of this certificate, I have no knowledge of any material fact or material change that is not reflected in the Report.
10. I am independent of the Issuer and of the property, applying all the tests set out in Section 1.5 of NI 43-101. I hold no material interest in the property, in the Issuer or in any of their related entities, and my relationship with the Issuer is strictly professional, being that of a client and an independent consultant. My fees are based on rates set by commercial agreement and are in no way dependent on the findings of the Report.
11. I have had prior involvement with the property that is the subject of the Report; the nature of my prior involvement is participation in the technical studies prepared by GE21 for the Project with effective dates of October 22, 2023, and February 18, 2025, the latter presented in the technical report titled “Independent Technical Report on the Luanga PGM+Au+Ni Project, Pará State, Brazil”, in which my responsibility was limited to the matters covered by Section 20.
12. I have read NI 43-101 and Form 43-101F1 – Technical Report, and I believe the Report has been prepared in accordance with those documents.
13. I consent to the public filing of the Report by the Issuer with any stock exchange or other regulatory authority, and to its publication by them for regulatory purposes, including electronic publication in the public company files on their websites.

Belo Horizonte, Brazil, November 4, 2026.

<Signed and sealed in original>

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Branca Horta de Almeida Abrantes

MAIG #8145