COMPETENT PERSONS' REPORT ON THE POSSE GOLD PROJECT, STATE OF MINAS GERAIS, FEDERATIVE REPUBLIC OF BRASIL

Prepared For

Hochschild Mining PLC

Report Prepared by

SRK Consultores do Brasil Ltda

001-22

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Project Number:		001-22
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Commercial Client		
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– EXECUTIVE SUMMARY

1 INTRODUCTION

1.1 Background

SRK Consultores do Brasil Ltda ("**SRK**") has been requested by Hochschild Mining PLC ("**Hochschild**" and also the "**Client**" and or the "**Company**") to author a Competent Persons' Report (the "**CPR**") in respect of the "**Posse Gold Project**" (also the "**PGP**") a Development Property (defined below) located in the State of Goiás, Federative Republic of Brazil ("**Brazil**").

SRK has been informed that Hochschild has entered into a definitive agreement (the "**Agreement**") with Amarillo Gold Corporation ("**Amarillo Gold**") to acquire all of the issued and outstanding shares of Amarillo Gold (the "**Transaction**") at a price of C$0.40 per share in cash (the "**Cash Offer**"). Pursuant to the Transaction, Hochschild will acquire a 100% interest in Amarillo Gold's PGP located in Goiás State, Brazil. In addition, shareholders of Amarillo will receive shares in a newly formed company, Lavras Gold Corp. ("**Lavras Gold**"), which will hold a stake in the Lavras do Sul project (the "**LDS Project**"), C$10m of cash, and a 2.0% net smelter revenue royalty on certain exploration properties owned by Amarillo Gold and located outside the current PGP Mineral Resource and Mineral Reserve at Amarillo Gold's Mara Rosa Property comprising a land area of 2,552ha across three mining concessions plus numerous exploration leases in areas surrounding the PGP.

Amarillo Gold is a public Company whose ordinary shares are listed on the TSX Venture Exchange ("**TSXV**") which files all of its regulatory submissions on the System for Electronic Document Analysis and Retrieval ("**SEDAR**"): an electronic filing system established by the Canadian Securities Administrators (the "**CSA**") that allows listed companies to report their securities-related information with the authorities concerned with securities regulation in Canada. The previous regulatory technical submission filed by Amarillo Gold in respect of the PGP is the *"Amended and Restated NI 43-101 Technical Report Definitive Feasibility Study Posse Gold Project, Brazil"* published on 03 August 2020 (the "**PGP 2020 43-101 TR**") in accordance with the provisions adopted by the Canadian Institute of Mining and Metallurgy ("**CIM**") Definition Standards on Mineral Resources and Reserves (the "**CIM Definition Standards**") and incorporated into Canadian National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("**NI 43-101**") adopted by CIM Council in May 2014.

During 2021, SRK was further commissioned by Amarillo Gold to prepare an update of PGP 2020 43-101 TR to incorporate updates to the following items inter alia: capital expenditure and operating expenditures; construction and commissioning timelines; and commodity prices (hereinafter "**PGP 2022 43-101 TR**"). The PGP 2022 43-101 TR was filed on SEDAR on 21 February 2022 and has been re-reported in the format of a CPR as noted herein.

Hochschild is a public company whose ordinary shares are listed on the London Stock (the "**LSE**") a market operated by the London Stock Exchange Group plc. SRK has also been informed that the Acquisition as defined above is classified as a Class 1 transaction in respect of the Requirements as defined below. As such SRK has been requested to author the CPR which is required solely in respect of the PGP and accordingly any other Mineral Assets including other Exploration Properties which the Company may acquire/obtain as part of the Transaction are specifically excluded from the CPR.

Hochschild is a leading underground precious metals producer focusing on high grade silver and gold deposits, with over 50 years' operating experience in the Americas. The Company operates three underground mines, two located in southern Peru and one in southern Argentina. All of the Company's underground operations are epithermal vein mines and the principal mining method used is cut and fill. The ore at its operations is processed into silver-gold concentrate or doré. For the twelve-month period ended 31 December 2021 the Company reported attributable siler production of 12.17Moz, attributable gold production of 221koz and an All In Sustaining Cost ("**AISC**") from operations of US$1,241/oz.

The salient features of the PGP as reported herein reflect:

- Mineral Resources reported assuming an in-situ cut-off grade of 0.35g/tAu and comprising:
  - Measured and Indicated Mineral Resources of 32.0Mt grading 2.80g/tAu and containing 1,200kozAu,
  - Inferred Mineral Resources of 0.10Mt grading 2.40g/tAu and containing 1.7kozAu;
- Mineral Reserves reported assuming a cut-off grade of 0.37g/tAu and incorporating a long-term gold price of US$1,450/oz and reporting a total of 23.8Mt grading 1.18g/tAu and containing 902kozAu and comprising:
  - Proven Mineral Reserves of 11.8Mt grading 1.20g/tAu and containing 456kozAu,
  - Probable Mineral Reserves of 11.8Mt grading 1.20g/tAu and containing 436kozAu;
- The results of the various technical studies completed in respect of the PGP including:
  - the 2020 Definitive Feasibility Study (the "**2020 DFS**") as reported in the PGP 2020 43-101 TR published by SRK in August 2020,
  - the PGP 2022 43-101 TR which includes various updates to commodity prices, macro-economic assumptions, operating and capital expenditure estimates, project construction and commissioning schedules completed during H2 2021; and
- Post-Tax Pre-Finance cashflow analysis which indicates the following:
  - Total gold production of 811koz produced over a 10-year Life-of-Mine (**"LoMp**"),
  - Gross Sales Revenue of US$1,297.6m assuming a constant gold price of US$1,600/oz,
  - Operating expenditure of (US$638.3m,
  - Earnings before Interest Depreciation and Amortisation ("**EBITDA**") of US$639.3m,
  - Initial capital expenditure of US$194.0m,
  - Sustaining capital expenditure of US$43.4m,
  - Free cashflow of US$262.8m,
  - All In Sustaining Costs (**"AISC**") of US$841/oz of gold.

This CPR presents the following key technical information as at the Effective Date defined below):

- Mineral Resources and Mineral Reserve statements (the "**2021 Statements**") for the PGP reported in accordance with the terms and definitions of the CIMM Definition Standards (2014) also defined in Section 1.2.2 below;

- The Life-of-Mine plan ("**LoMp**") for the PGP reflecting depletion of the Mineral Reserves including assumed production, sales, sales revenue, operating and capital expenditure commencing 1 January 2021;
- The "**Environmental and Social Liabilities**" for the Mineral Assets inclusive of all mine closure related expenditures and retrenchment costs for the LoMp Scenarios; and
- Financial Modelling of the Mineral Assets undertaken to support the technical and economic viability of the Ore Reserves and the LoMp Scenarios as reported herein.

For the avoidance of doubt, this CPR is limited to the Mineral Assets and specifically exclude all assets and liabilities relating to the Group's activities external to the Mineral Assets as defined herein. Notwithstanding the aforementioned, this CPR does include the results of the Financial Modelling of the Mineral Assets which relies on certain inputs including TEPs as provided by the Company and as appropriate, modified and adjusted by SRK. Certain units of measurements and technical terms defined in the CIM Definition and Standards (defined in 1.2.2 of the Main Report) are defined in the glossaries, abbreviations and units included at the end of this CPR.

1.2 Requirement, Reporting Standard and Reliance

The CPR will be published in a "**Shareholder Circular**" being issued to all Hochschild shareholders in order to convene a general meeting to vote on a resolution approving the Transaction. Hochschild has engaged RBC Capital Markets ("**RBC**") as its financial advisor, sole sponsor and corporate broker, Stikeman Elliott LLP ("**Stikeman**") as its Canadian legal counsel, Pinheiro Neto Advogados as its Brazilian legal counsel, and Linklaters LLP ("**Linklaters**") as its UK legal counsel in connection with the Transaction.

Requirement

The CPR is to be prepared in compliance with the following requirements which together comprise the "**Requirements**":

- The "**Listing Rules**" published by the FCA from time to time and under Part VI of the Financial Services and Markets Act 2000 of the United Kingdom (the "**FSMA**"); and
- The "***ESMA update of the CESR recommendations: The consistent implementation of Commission Regulation (EC) No 809/2004 implementing the Prospectus Directive***", published on 20 March 2013: specifically paragraphs 131 to 133, section 1b – mineral companies, Appendix I – Acceptable Internationally Recognised Mining Standards, and Appendix II – Mining Competent Persons' Report – recommended content, hereinafter and collectively referred to as the "**CESR Recommendations**" and published on 20 March 2013.

Accordingly, whilst Amarillo Gold is in accordance with its regulatory reporting requirements publishing the PGP 2022 43-101 TR, the CPR as published by the Company in respect of the PGP will contain the same technical information as incorporated into the Technical Report and largely presented in the same format of a NI 43-101, but with appropriate references to the required Rules and Regulations and other presentational amendments.

With respect of paragraphs 132(a)-(e) of the CESR Recommendations SRK notes that all relevant details are included in the discipline technical Sections for the PGP. In respect of compliance with "**Appendix II**" of the CESR Recommendations, specifically the recommended content of the Competent Persons' Reports SRK respectfully highlights the following:

- **Scope of the CPR:** The primary focus of the CPR is with respect to the provision of independently audited and current: Mineral Resources and Mineral Reserves; Life-of-Mine plans (limited to Ore Reserves only); Environmental and Social Liabilities; and Financial

Modelling of the PGP as reported herein; and

- **Compliance Cross Reference** for similar groupings noted for paragraphs 132(a)-(e) above, the following items are referenced in Section 4 Property Description And Location, Section 5 Accessibility, Climate, Local Resources, Infrastructure And Physiography, Section 6 History, Section 7 Geological Setting And Mineralisation, Section 8 Deposit Types, Section 9 Exploration, Section 10 Drilling, Section 11 Sample Preparation, Analyses And Security, Section 12 Data Verification, Section 13 Mineral Processing And Metallurgical Testing, Section 14 Mineral Resource Estimates, Section 15 Mineral Reserve Estimates, Section 16 Mining Methods, Section 17 Recovery Methods, Section 18 Project Infrastructure, Section 19 Market Studies And Contracts, Section 20 Environmental Studies, Permitting And Social Or Community Impact, Section 21 Capital And Operating Costs, Section 22 Economic Analysis, Section 23 Adjacent Properties, Section 24 Other Relevant Data And Information, Section 25 Interpretation And Conclusions, and Section 26 Recommendations:
  - Item (i) Legal and Geological Overview of the Mineral Assets including (1) and (2),
  - Item (ii) Geological Overview,
  - Item (iii) Mineral Resources and Mineral Reserves including (1) (2), (3), (4 and 5), (6), (7), (8a), (8b), 8 (c and d),
  - Item (iv) Valuation of Mineral Reserves/Mineral Assets. This CPR includes a Valuation of the Mineral Reserves,
  - Item (v) Environmental, Social and Facilities: (1), (2), (3),
  - Item (vi) Historic Production/Expenditures,
  - Item (vii) Infrastructure,
  - Item (viii) Maps,
  - Item (ix) Special Factors.

Reporting Standard – Mineral Resources and Mineral Reserves

The reporting standard adopted for the reporting of the Mineral Resource and Ore Reserve statements included in the CPR is that adopted by the Canadian Institute of Mining and Metallurgy ("**CIM**") Definition Standards on Mineral Resources and Reserves (the "**CIM Definition Standards**") and incorporated into Canadian National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("**NI 43-101**") adopted by CIM Council in May 2014.

Reporting Standard – Technical Study Standards

A Feasibility Study is a comprehensive technical and economic study of the selected development option for a mineral project that includes appropriately detailed assessments of applicable Modifying Factors together with any other relevant operational factors and detailed financial analysis that are necessary to demonstrate, at the time of reporting, that extraction is reasonably justified (economically mineable). The results of the study may reasonably serve as the basis for a final decision by a proponent or financial institution to proceed with, or finance, the development of the project.

Reporting Standard – Environmental and Social Standards

Environmental and Social Standards as considered in this CPR has been, where practically possible, assessed with due consideration for national legislation and regulation as currently applicable in Brazil. SRK notes, however that the PGP has not been assessed in respect of international standards and guidance. In respect of the latter standards and guidance SRK has not considered adherence or alignment with the International Financial Corporation's Performance Standards ("**IFC PS**") and relevant World Bank Group's Environmental Health and Safety Guidelines.

Accordingly, the principal focus of the Environmental and Social review in respect of the Mineral Assets comprised a review of the Environmental Management Practices and Environmental and Social Liabilities (Bio-Physical and Social) at the Mineral Assets with specific focus on the primary regulatory documentation and compliance with the conditions of approval, including emissions and discharges in respect of local standards. It is however important to note that this review did not constitute a detailed Environmental Audit, does not extend to provide a detailed opinion and development of any Equator Principles Action Plan capable of bringing the technical studies into compliance with the Equator Principles, nor indicate when compliance is not possible as typically required for a Project Finance facility: for all Category A and, as appropriate, Category B Projects.

Responsible sourcing regulations are an increasing focal point for stakeholders in the international mining and metals sector and in addition to national legislation, there are also a number of regulations and guidance that specifically cover the responsible souring of gold. For example, the "**Dodd-Frank**" legislation in the United States (Section 1502) and the "**EU Conflict Free Minerals**" regulations require due diligence within the supply chain in order to ensure that mining and production of gold does not fund conflict. One of the most widely recognised is the "**OECD Due Diligence Guidance for Responsible Supply Chains of Minerals from Conflict-Affected and High-Risk Areas**". The guidance was operationalised by the World Gold Council for the mining sector, the London Bullion Market Association for the refining sector and the Responsible Jewellery Council for this sector.

With respect to "**Mine Closure"** related liabilities key international standards include those which are focused on a combination of technological and engineering solutions which reflect Good International Industry Practice ("**GIIP**") and "**Best Available Technology**" to where practicable achieve "**Ground Zero**" or "**Walk Away**" remediation status. Guiding standards which reinforce these objectives include: the International Council on Mining and Minerals ("**ICMM**") Planning for Integrated Mine Closure: Toolkit (2008); World Bank in Mining and Development, It's Not Over When It's Over: Mine Closure Around the World (2002); European Commission's Reference Document on *"Best Available Techniques for Management of Tailings and Waste-Rock in Mining Activities"* published in 2009; *"IFC EHS Guidelines on Construction and Decommissioning"* published in 2007; and *"Mining for Closure: Policies and Guidelines for Sustainable Mining Practice and Closure of Mines"* published by United Nations Environment Programme ("**UNEP**"), United Nations Development Programme ("**UNDP**"), Organization for Security and Co-operation in Europe ("**OSCE**") and the North Atlantic Treaty Organization ("**NATO**") in 2005.

Reporting Standard – Mineral Asset Valuation

This CPR includes a Valuation of the Mineral Reserves and is reported in accordance with the general disclosure principles and process as defined by the ***"CIMVAL Code for the Valuation of Mineral Properties"***, prepared by the Special Committee of the Canadian Institute of Mining, Metallurgy and Petroleum on the Valuation of Mineral Properties ("**CIMVAL**") and adopted by the CIM Council on November 29, 2019, ("**CIMVAL 2019**").

Reporting Standard – Cash Cost Reporting

The determination of cash costs in the metals and mining sector varies both within and between commodity focus companies. Furthermore, it would appear that with respect to reporting standards, that defined by the World Gold Council ("**WGC**") and published (2018) ("**WGC 2018**") in its guidance noted on "*all-in sustaining costs*" and *"all-in costs*" metrics would appear to be the most comprehensive. This was an advance from the cash cost reporting methodology introduced in 1996 which focused solely on the mining and processing costs incurred. In contrast WGC 2018 focuses on costs incurred in the complete mining life cycle from exploration to closure. In this instance SRK notes the following industry standard definitions:

- Cash Costs reported per ounce gold sold and reported on a by-product basis, where expenditures are determined net of silver sales where relevant. Cash costs are defined as:
  - Adjusted Operating Costs ("**AOC**") comprising on-site mining costs, on-site general and administrative costs, royalties and production taxes, realised gains/losses on hedges due to operating costs, community costs related to current operations, refining and transport costs, non-cash remuneration (site-based), stockpile/leach pad and product inventory write down, operational waste stripping costs and by-product credits;
  - All In Sustaining Costs ("**AISC**") comprising corporate general & administration costs (including share-based remuneration), reclamation and remediation accretion and amortisation (operating sites), exploration and study costs (sustaining), capital exploration (sustaining), capitalised stripping & underground mine development (sustaining), sustaining capital expenditure and sustaining leases;
  - All-in Costs ("**AIC**") comprising growth and development costs not related to current operations, community costs not related to current operations, permitting costs not related to current operations, reclamation and remediation costs not related to current operations, exploration and study costs (non-sustaining), capital exploration (non-sustaining), capitalised stripping & underground mine development (non-sustaining), non-sustaining capital expenditure and non-sustaining leases.

In respect of the above items it is important to note that the following expenditures are typically not included in the WGC guidance: corporate income tax; working capital (except for adjustments to inventory on a sales basis); all financing charges (including capitalised interest); costs related to business combinations, asset acquisitions and asset disposals; items needed to normalise earnings, for example impairments on non-current assets, one-time material severance charges or legal costs or settlements or legal costs or settlements related to significant lawsuits.

Reliance

This CPR is addressed to and may be relied on by the Directors of the Company and the "**Advisors**", specifically in compliance with the Requirements and the Reporting Standard. Accordingly, SRK has confirmed in writing (the "**Consent letter**"), dated on the Publication Date which confirms:

- Reliance as regards the CPR for any benefit of the Company and its Advisors;
- Consent to the inclusion of the CPR, and to the inclusion of any extracts from the CPR in the Prospectus;
- Confirmation that all information contained in the Prospectus which is extracted from the CPR or based upon information contained in the CPR has been reviewed by SRK and that such information as presented is accurate, balanced, complete and not inconsistent with the CPR; and
- Responsibility for the CPR and declares that it has taken all reasonable care to ensure that the information contained in the CPR is, to the best of its knowledge, in accordance with the facts and makes no omission likely to affect its import.

SRK has no obligation or undertaking to advise any person of any development in relation to Mineral Assets which comes to its attention after the date of this CPR or to review, revise or update the CPR or opinion in respect of any such development occurring after the date of this CPR.

1.3 Effective date, Base Technical Information Date and Publication Date

Modelling of the PGP reflect SRK's assessments of the:

- Mineral Resource and Ore Reserves statements as noted in the 2021 Statements and reported by SRK in accordance with the CIM Definition Standards;
- LoMp Scenarios with projected production from 1 January 2022;
- Detailed schedules of activities and expenditures relating to the derivation and support of the forecast TEPs as included in the LoMp Scenario for the PGP including, production, sales, sales revenue, operating expenditure and capital expenditure;
- Cashflow Model for the PGP incorporating annual forecasts of the TEPs and resulting post-tax pre-finance cashflows;
- Mine closure costs relating to the PGP s comprising the Environmental and Social Liabilities reported herein; and
- Cashflow Modelling of the Mineral Assets to assess the technical and economic viability of the Ore Reserves.

The Base Technical Information Date is defined as 1 January 2022 which is co-incident with the reporting date for the 2021 Statements, this being 31 December 2021. The Publication Date of the CPR is assumed to be 4 March 2022. As advised by the Company, as at the Publication Date of the Circular no material change has occurred as of the Effective Date of the CPR inclusive of: the 2021 Statements; the LoMp and accompanying TEPs; the Environmental and Social Liabilities; and the Cashflow Modelling of the PGP.

1.4 Verification and Validation

This CPR is dependent upon technical, financial and legal input from the Company, Amarillo and its third-party consultants. Following publication of the PGP 2020 43-101 TR, SRK has undertaken a detailed review of various updates to the 2020 DFS to reflect changes with respect to commodity prices and macro-economics, operating and capital expenditure assumptions and construction and commissioning schedules completed during H2 2021. The results of this review are reported in the PGP 2022 43-101 TR, recently filed on SEDAR, and the results of which are reproduced in this CPR.

The Qualified Person who takes overall responsibility for the CPR and the Mineral Reserve as reported herein is Mr Paulo Laymen who undertook a site visit in September 2018. SRK confirms that whilst it has not undertaken any site visits since September 2018 and given the current greenfield status of the PGP and limited site activity since this date, the technical data and technical opinion as expressed in this CPR remain valid as at the Effective Date of the CPR, that being 31 December 2021. Furthermore, SRK notes that as part of the original 2020 DFS, SRK authored the Mineral Reserve statement and all underlying mining engineering work streams required to support the 2021 Statements.

SRK confirms that it has performed all necessary validation and verification procedures deemed necessary and/or appropriate to place a suitable level of reliance on such technical information. SRK considers that with respect to all material technical-economic matters, it has undertaken all necessary investigations to ensure compliance with the Requirements including the Reporting Standards (specifically the CIM Definition Standards and the CIMVAL Code).

In consideration of all legal aspects relating to the PGP, SRK has placed reliance on the representations by the Company and Amarillo that the following are correct as at the Effective Date of the CPR and remain correct until the date of the Public Document:

- That save as disclosed in the CPR, the Directors of the Company are not aware of any legal proceedings that may have an influence on the rights to explore for minerals in respect of the Mineral Assets;

- That Amarillo is the legal owner of all relevant mineral and surface rights as reported in the CPR; and
- That save as expressly mentioned in the CPR, no significant legal issue exists which would affect the likely viability of the PGP and/or the estimation and classification of the Mineral Resources and Mineral Reserves, the LoMp, the Environmental and Social Liabilities, and the Cashflow Modelling.

The Mineral Resource and Mineral Reserve statements as included in the 2021 Statements are reported with a date of depletion of 31 December 2021. For the avoidance of doubt, the 2021 Statements are the "current statements" and any historical statements as reported herein are done so solely for comparative purposes to provide context with respect to any significant changes and to support the reconciliation process between reporting periods.

1.5 Limitations, Reliance on Information, Declaration, Consent and Cautionary Statements

Limitations

Save as set out in Section 1.2.3 of the Main Report and for the responsibility arising under the Requirements to any person and to the extent there provided, to the fullest extent permitted by law, SRK does not assume any responsibility and will not accept any liability to any other person for any loss suffered by any such other person as a result of, arising out of, or in connection with this CPR or statements contained therein, required by and given solely for the purpose of complying with the Requirements, consenting to its inclusion in the Circular.

SRK notes that this CPR has been prepared in accordance with the Requirements as defined herein. For the avoidance of doubt SRK notes that the contents of this CPR including the technical opinion as expressed herein must be read in association with the **Error! Reference source not found.**, Reliance on Information, Declarations and Consent as reported herein.

The achievability of the projections as reported in this CPR, are neither warranted nor guaranteed by SRK, specifically the: TEPs including assumed production, sales volumes, sales revenue, operating and capital expenditure relating to depletion of the Ore Reserves from 1 January 2022; the Environmental and Social Liabilities; and the Cashflow Modelling relating to the PGP. The projections as presented and discussed herein have been proposed by the Company's management and adjusted where appropriate by SRK to reflect its opinion but cannot be assured. Notably, for example, they are necessarily based on economic and market assumptions, many of which are beyond the control of the Company.

Future cashflows and profits derived from any projections reflected by the TEPs in the LoMp, the Environmental and Social Liabilities are inherently uncertain and actual results may be significantly more or less favourable.

Unless otherwise expressly stated all the opinions and conclusions expressed in this report are those of SRK. It should also be noted that this report reflects SRK's review of information generated, and/or technical work completed, by others. As a result of this, the projections presented here may not directly reflect that previously presented by the Company or in public announcements made by the Company as they also incorporate judgements made by SRK not necessarily incorporated into the Company's assessments.

This CPR specifically excludes all aspects of legal issues, marketing, commercial and financing matters, insurance, land titles and usage agreements, and any other agreements and/or contracts that the Company may have entered into.

Responsibility Statement

For the purpose of, and in compliance with, the Requirements, SRK accepts responsibility for the information provided in the CPR and for all information in the Prospectus which is extracted or sourced from the CPR. SRK declares that the information contained in the CPR and the Prospectus is, to the best of the knowledge of SRK, in accordance with the facts and makes no omission likely to affect its import. SRK has given and has not withdrawn its written consent to the publication of the CPR.

SRK accepts responsibility for the 2021 Statements, the LoMp Scenario and associated TEPs, the 2021 Environmental and Social Liabilities, the Cashflow Modelling of the PGP as reported herein. Where applicable, SRK confirms that:

- the 2021 Statements are reported in accordance with the terms and definitions of the CIM Definition Standards;
- the various technical studies supporting the Production Scenarios have been completed in accordance with the Technical Study standards as defined in Section 1.2.2. of the Main Section of this CPR;
- that the Environmental and Social Liabilities are derived and reported in accordance with local standards; and
- the Cashflow Modelling for the PGP as reported herein are reported in accordance with the CIMVAL (2019).

The scope of the CPR is limited to the PGP as reported herein and expressly excludes all other mineral assets relating to the Transaction or currently owned by the Company.

Reliance on Information

SRK believes that its opinion must be considered as a whole and that selecting portions of the analysis or factors considered by it, without considering all factors and analyses together, could create a misleading view of the process underlying the opinions presented in this CPR. The preparation of a CPR is a complex process and does not lend itself to partial analysis or summary.

SRK's opinions given in this document with respect to the 2021 Statements, the LoMp and accompanying TEPs, the Environmental and Social Liabilities, and the Cashflow Modelling are effective at 31 December 2021 and are based on information provided by the Company and Amarillo throughout the course of SRK's investigations, which in turn reflects various technical-economic conditions prevailing at the date of this report and the Company's expectations regarding the gold market, gold prices and exchange rates as at the date of this report. These and the underlying TEPs, comprising projections of production, sales, sales revenue, operating and capital expenditures can change significantly over relatively short periods of time. Should these change materially, the 2021 Statements, the LoMp Scenarios and accompanying TEPs, the Environmental and Social Liabilities, and the Cashflow Modelling of the CPR could be materially different in these changed circumstances.

Whilst SRK has exercised all due care in reviewing the supplied information, SRK does not accept responsibility for finding any errors or omissions contained therein and disclaims liability for any consequences of such errors or omissions.

This CPR includes technical information, which requires subsequent calculations to derive subtotals, totals and weighted averages. Such calculations may involve a degree of rounding and consequently introduce an error. Where such errors occur, SRK does not consider them to be material.

Declarations

SRK will receive a fee for the preparation of this CPR in accordance with normal professional consulting practice. This fee is not contingent on the outcome of any transaction and SRK will receive no other benefit for the preparation of this report. SRK does not have any pecuniary or other interests that could reasonably be regarded as capable of affecting its ability to provide an unbiased opinion in relation to 2021 Statements, the principal findings regarding the LoMp Scenario, the Environmental and Social Liabilities and the Cashflow Modelling of the PGP as reported herein.

Neither SRK, the Qualified Persons (as identified under Section 1.7, below) who are responsible for authoring this CPR, nor any Directors of SRK have at the date of this report, nor have had within the previous two years, any shareholding in the Company, the PGP or the Advisors of the Company, or any other economic or beneficial interest (present or contingent) in any of the assets being reported on. SRK is not a group, holding or associated company of the Company. None of SRK's partners or officers are officers or proposed officers of any group, holding or associated company of the Company. Further, no Qualified Person involved in the preparation of this CPR is an officer, employee or proposed officer of the Company or any group, holding or associated company of the Company. Consequently, SRK, the Qualified Persons and the Directors of SRK consider themselves to be independent of the Company, its directors, senior management and Advisors.

Consent

SRK has given and has not withdrawn its written consent to the publication of this CPR and has authorised the contents of its report and context in which they are respectively included and has authorised the contents of its report for the purposes of compliance with the Requirements.

Copyright

Except where SRK has agreed otherwise (including pursuant to an agreement between SRK and the Company dated 14 February 2022 or any subsequent agreement (each, the "**Hochschild Agreement**")):

- neither the whole nor any part of this report nor any reference thereto may be included by any party other than the Company, any of its direct and indirect subsidiaries or a competent state authority in the United Kingdom of Great Britain and Northern Ireland or any other relevant jurisdiction, as may be applicable (together, the "**Recipients**"), in any other document without the prior written consent of SRK save that in the case that the report is not included in full in any other document, the Recipient shall present a draft of any document produced by it that may incorporate a part of this report to SRK for review so that SRK may ensure that this is presented in a manner which accurately and reasonably reflects any results or conclusions contained in this report; and
- copyright of all text and other matters in this document, including the manner of presentation, is the exclusive property of SRK. It is an offence to publish this document or any part of the document under a different cover, or to reproduce and/or use, without written consent (whether granted by virtue of an Hochschild Agreement or otherwise), any technical procedure and/or technique contained in this document. The intellectual property reflected in the contents resides with SRK and shall not be used for any activity that does not involve SRK, without the written consent of SRK.

Neither the whole nor any part of this report nor any reference thereto may be included in any other document without the prior written consent of SRK regarding the form and context in which it appears.

1.6 Indemnities Provided by the Company

The Company has provided the following indemnities to SRK:

- The Company has agreed that, to the extent permitted by law, it will indemnify SRK and its employees and officers in respect of any liability suffered or incurred as a result of or in connection with the preparation of this report albeit that this indemnity will not apply in respect of (i) fraud, bad faith, gross negligence wilful misconduct or breach of law on the part of SRK or its employees or officers; or (ii) breach of this Agreement on the part of SRK. The Company has also agreed to indemnify SRK and its employees and officers for time incurred and any costs in relation to any inquiry or proceeding initiated by any person albeit that this indemnity will not apply in respect of (i) fraud, bad faith, gross negligence wilful misconduct or breach of law on the part of SRK or its employees or officers; or (ii) breach of this Agreement on the part of SRK; and
- In order to assist SRK in the preparation of this CPR the Company may be required to receive and process information or documents containing personal information in relation to SRK's project personnel. The Company has agreed to comply strictly with the provisions of the Data Protection Act 1998 of the United Kingdom ("**DPA 1998**") and all regulations and statutory instruments arising from the DPA 1998, and the Company will indemnify and keep indemnified SRK in respect of all and any claims and costs caused by breaches of the DPA 1998.

1.7 Qualifications of Consultants and Competent Persons

SRK is an associate company of the international group holding company SRK Consulting (Global) Limited (the "**SRK Group**"). The SRK Group comprises some 1,400 professional staff offering expertise in a wide range of resource and engineering disciplines with 45 offices located in 20 countries.

The SRK Group's independence is ensured by the fact that it holds no equity in any project. This permits the SRK Group to provide its clients with conflict-free and objective recommendations on crucial judgment issues. The SRK Group has a demonstrated track record in undertaking independent assessments of resources and reserves, project evaluations and audits, Mineral Resource and Ore Reserve audits and independent feasibility studies on behalf of exploration and mining companies and financial institutions worldwide. The SRK Group has also worked with a large number of major international mining companies and their projects, providing mining industry consultancy service inputs.

This CPR has been prepared by SRK Brasil and relies on various technical inputs to the recently published PGP 2022 43-101 TR which in turn relies on a number of historical documents, namely the prior PGP 2020 43-101 TR and the 2020 DFS. The PGP 2022 43-101 TR refers to a total of 10 consultants who are specialists in the fields of exploration, geology, Mineral Resource and Mineral Reserve estimation and reporting, open-pit mining, mining geotechnics, water management (hydrogeology/hydrology), mineral processing, tailings engineering, infrastructure, environmental and social, financial modelling and mineral asset valuation. The individuals listed in Table ES 1 have provided the material input to the PGP 2022 43-101 TR and the historical documents upon which this CPR is based, have extensive experience in the mining industry and are members in good standing of appropriate professional institutions.

The Qualified Person who has overall responsibility for the CPR and the Mineral Reserves as reported in the CPR will be Mr Paulo Laymen, MSc, Registered Member in good standing of Chilean Mining Commission (Comisión Calificadora de Competencias en Recursos y Reservas de Chile: Membership number 0320) and member of the Australasian Institute of Mining and

Metallurgy (membership number 320077). In being a registered member of the Chilean Mining Commission, Paulo Laymen is a qualified member of Accepted Foreign Associations and Membership Designations within the meaning of Appendix A of NI 43-101. Mr Paulo Laymen is a full-time employee of SRK and is independent of the Company as defined herein and as sufficient relevant experience in the commodity, type of deposit and situation as reflected by the Mineral Reserve statement.

The Qualified Person who has responsibility for reporting of Mineral Resources in the CPR will be Mr Gregory Keith Whitehouse, B.Sci, MAusIMM (CP). In being a registered Chartered Professional Member of the Australian Institute of Mining and Metallurgy, Gregory Whitehouse is a qualified member of Accepted Foreign Associations and Membership Designations within the meaning of Appendix A of NI 43-101. Mt Gregory Whitehouse is a full-time employee of Australian Exploration Field Services Pty Ltd ("**AEFS**") and is independent of the Company as defined herein and as sufficient relevant experience in the commodity, type of deposit and situation as reflected by the Mineral Resource statement.

Table ES 1 Team members[1]

Responsible Discipline	Consultant	Designation	Registration, Membership, Qualification	Years' Experience
Geology/Mineral Resources	**Gregory Keith Whitehouse[2]**	**Principal**	**MAusIMM, CP, BSci**	**46**
	John Watts	Principal	BSc	54
	John Collier	Principal	BSc	22
Mining & Mineral Reserves, Geotechnical Engineering, Human Resources	**Paulo Laymen[2]**	**Principal**	**MCMMC (RM), BEng**	**20**
Metallurgy, Mineral Processing and Infrastructure	**Stuart Smith[2]**	**Principal**	**FAusIMM, Ba.App.Sci**	**35**
	Tommaso Roberto Raponi[2]	**Principal**	**APEG, Pr.Eng., BA.Sc.**	**38**
Waste and Water Management	Paulo Paiva	Principal	BEng., LLB	49
Environmental and Social	Nelson Siqueira	Principal	BSc.	42
Mine Closure	Cristina Simonetti	Principal	PhD Geol Sci	35
Financial Modelling	Luiz Confúcio	Consultant	MBA Fin	23

(1) Keith Whitehouse and John Watts are employees of Australian Exploration Field Services Pty Ltd ("**AEFS**"); John Collier is an employee of Conarco Consulting (Pty) Ltd ("**Conarco**"); Paulo Laymen is an associate of SRK; Stuart Smith is an employee of Aurifex Pty Ltd ("**Aurifex**"), Tommaso Roberto Raponi is an employee of Ausenco Engineering Canada Inc ("**Ausenco**"); Paulo Paiva is a full time employee of GeoHydroTech Engenharia; Nelson Siqueria is a full time employee of DBO Engenharia Ltda ("**DBO**"); Cristina Simonetti is a full time employee of the Ramboll Group ("**Ramboll**"); and Luiz Confúcio is a full time employee of SRK

(2) Qualified Persons within the meaning of NI 43-101.

2 POSSE GOLD PROJECT

2.1 Property Description and Ownership

The Mara Rosa Property (also generally known and referred to as the Posse Deposit, the Posse Gold Project and the Project) is located in the State of Goiás, central Brazil, approximately 6km north of the town of Mara Rosa. The Project encompasses a land area of 2,552ha across three mining concessions plus numerous exploration leases in areas surrounding the Project mine area.

Amarillo visited the Project in August 2003 and in October 2003 signed a letter of intent with Metallica Brasil Ltda (MBL) to purchase MBL and 100% of the Posse Gold Project. Amarillo currently owns 100% of the Posse Gold Project.

2.2 Geology and Mineralization

Amarillo's land position within the Mara Rosa District primarily covers the Eastern Belt greenstone assemblage with some coverage of the Western and Central belts as well. The Eastern Belt, has a maximum thickness of 6km, generally strikes to the northeast and dips moderately to steeply to the northwest.

The Posse Deposit occurs in a regional thrust that probably acted as one of the primary dewatering conduits during the Neo-Proterozoic Brasiliano orogeny. The geophysical, geological and geochemical data available demonstrate that the Posse Deposit occurs within a 50km long shear zone with potassium alteration and lower order gold-copper-molybdenum mineralization. The Posse deposit has grey gneiss in the hanging wall of the fault and amphibolite, "greenstone" in the footwall. Shearing of the Gray Gneiss has resulted in the formation of a distinct lithologic unit, a quartz-feldspar-mica schist (Posse Schist) that is characteristic of the Posse ore zone. This unit has been identified in several other areas including the Posse footwall and on strike extensions of the Posse Ore Zone to the northeast. Shearing is most intense in the footwall.

The mineralization envelope at Posse is about 30m thick and over 1km long. It has a mylonitic appearance that is most noticeable in the footwall where shearing is the most intense. Higher intensity of shearing is associated with increased sulphide mineralization (up to about 4%), and a slight increase in metamorphic grade from greenschist to high greenschist facies in the hanging wall through to high greenschist/low amphibolite facies in the footwall (biotite flakes and garnet alteration). Higher gold values are associated with increasing intensity of shearing and higher levels of silicification and sulphide mineralization.

2.3 Status of Exploration

Numerous drilling campaigns have been completed on the property: BHP Billiton (1982 – 1987), WMC (1988 – 1995), Amarillo (2005 – 2006), Amarillo (2008), Amarillo (2010 – 2011), Amarillo (2011 – 2012), Amarillo (2018 – 2019) and Amarillo (2021). In all, the drillhole data base contains 423 drill holes totalling 64,749m of drilling.

During the period from late 2012 until June 2018 no drilling was carried out or samples submitted for assay. Amarillo completed a 63-hole drilling program at the Posse Gold Project in February of 2019. The program consisted of 49 diamond drillholes, 18P047 – 18P087 and 19P088 – 19P095, with a total length of 15,195m and 14 reverse circulation (**"RC"**) drillholes, 18PRC001 – 18PRC014, for a total length of 1,295m, a further program of 10 diamond drillholes (21P112 – 21P121) with a total length of 2,519m was completed in 2021.

2.4 Mineral Processing and Metallurgical Testing

The test work completed provided support for the proposed flowsheet to be applied at the Posse Gold Project and is considered adequate to take into process design. The flowsheet being to crush, grind, leach at 53µm for 36 hours at a pH of 12.0 at anticipated temperatures of +35°C generated as a consequence of grinding effort. The work has shown the carbon characteristics remain in the range typical of the industry, even though elevated pH is present. The work has also shown that SO_2/air cyanide detoxification is applicable using reagent doses and residence times again typical of the gold industry.

To reduce capital cost, the decision to take the tailings thickener out of the flowsheet has been made. Filtration testing at a nominal pulp density of 40% and 50% solids has shown filtered solids can be generated at moisture contents that will allow handling and placement. Press type filter technologies appearing the most appropriate.

The samples used in the test work have been sourced from a large number of drill holes and from varying depths along strike. The basic work (both earlier work by Coffey and later work managed by Amarillo directly) to define the flowsheet has been conducted on a number of composites suggesting average or "typical" performance will provide high leach extractions in the 90% range. As the test work programs have progressed, and as test work control has improved, the Locality Composites tested have provided very consistent results in both extraction outcomes and reagent demands. This lack of variability suggests the Mara Rosa material can be expected to provide consistent leach extractions in the 90% range and also supports adequate coverage of the deposit by the samples selected. That is sensitivity to sample location is minor and is not a key driver with regard to the metallurgical responses.

The derived gold recovery expression is:

- Recovery % = [(Au – 0.0854 x Au0.8718 – 0.023) / Au] x 100% here Au is the head grade of the ore.

There do not appear to be any deleterious elements or compounds present. An exception may be considered to be the presence of auriferous tellurides themselves. However, as the flowsheet has provided high leach extractions, these tellurides are no longer considered deleterious. The extractions achieved are high even by typical free milling ores in this head grade range.

2.5 Mineral Resource Estimate

A Mineral Resource can only be declared for material which is considered to have potential for economic extraction at some point in the future. The cut-off at which a resource is reported should also meet this criterion, it should not include material which does not have reasonable potential to be mined and processed. The definition of a Mineral Reserve on the other hand applies a specific set of economic parameters to a mineral resource to determine which portions of the Resource can be mined under those economic conditions.

In the case of the Posse Deposit economic modelling of the blocks in the model has indicated that the lowest grade block to be mined as ore has a grade of 0.37g/tAu. On this basis the cut-off grade for the mineral resource has been set at 0.35g/tAu. The Mineral Resource above a cut-off of 0.35g/tAu declared for the Posse Deposit is summarized in Table ES 2.

Table ES 2 Posse Gold Project Mineral Resource Statement 31 December 2021[(1)]

Category	Tonnes (Mt)	Au grade (g/t)	Troy Ounces (koz)
Measured Mineral Resource	14	1.2	510
Indicated Mineral Resource	19	1.1	640
Total of Measured and Indicated Mineral Resource	**32**	**1.1**	**1,200**
Inferred Mineral Resource	0.10	0.52	1.7

(1) Note that Tonnes, Grade and Ounces in the 2020 Resource Estimate summarised in Table ES 2 have been reported to 2 significant figures only to reflect the uncertainty inherent in any Mineral Resource Estimate. A cut-off grade of 0.35g/tAu has been used for the Mineral Resource Estimate. The Mineral Resource is quoted inclusive of Mineral Reserves.

Drilling completed in 2019 and reported as part of this report has significantly increased the confidence in the current mineral resource estimate compared to that reported in 2018. Extensive work in 2020 and 2021 to test the validity of historic assays was undertaken by Amarillo, following a risk assessment by an independent Technical Engineering ("**ITE**") Consultant. The re-assay program was followed by targeted drilling in 2021. This work has not materially altered the resource estimate completed in 2020 and the 2020 resource estimate forms the basis of this report. The work undertaken post the declaration of the 2020 resource is outlined in the body of this report.

The opinion of AEFS is that the character of the Mara Rosa Property, the Posse Deposit and the Mineral Resource Estimate reported herein is appropriate to support the continued development of the Posse project and valuations which may be derived from the current knowledge of the project.

2.6 Mineral Reserve Estimate

The Mineral Reserve is derived from Measured and Indicated Resources based on CIM guidelines. As mentioned in Section 14, the Mineral Resources have not been updated since the DFS 2020.

To convert Mineral Resources to Mineral Reserves, consideration was given to forecasts and estimates of gold price, metallurgical recovery, mining dilution and ore loss factors, royalties and costs associated to mining, processing, overhead, refining and logistics. After the completion of the DFS 2020, some of these parameters were updated to reflect more accurately the current economic conditions of the Project, including:

- Long term gold price;
- Processing operating costs;
- Mining operating costs;
- G&A costs; and
- Project implementation plan and mine schedule.

SRK verified the effect of these changes on the economic cut-off grades and pit design. No material impact was noted. Therefore, the Mineral Reserve estimated in the DFS 2020 remained unchanged. Specifically, the Mineral Reserve estimated in 2020 reached 23.8Mt (dry) at an average grade of 1.18g/t. The detailed breakdown of the Mineral Reserve is presented in Table ES 3. It is SRK's opinion that the Mineral Reserve estimation is compliant with the CIM Definition Standards.

This Mineral Reserve is estimated on the basis of currently available information. The Reserve classification reflects the level of accuracy of the updated DFS.

Table ES 3 Posse Gold Project Mineral Reserve Statement 31 December 2021[(1)]

Mineral Reserve	Diluted tonnes (Mt dry)	Diluted grade (g/t Au)	Contained metal (koz Au)	Estimated recovery (%Au)	Recoverable metal (koz Au)
Proven	11.8	1.20	456	89.9%	410
Probable	12.0	1.16	446	89.8%	401
Total Mineral Reserve	23.8	1.18	902	89.9%	811

(1) A gold price of US$1,450/oz is assumed. An exchange rate of R$5.05 to US$1.00 is assumed. Mineral Reserves are based on Measured and Indicated Mineral Resources only. Mineral Reserves above an economic cut-off grade of 0.37g/tAu. The Mineral Reserve is included in the Mineral Resource quoted in Table 14-17.

2.7 Mining Methods

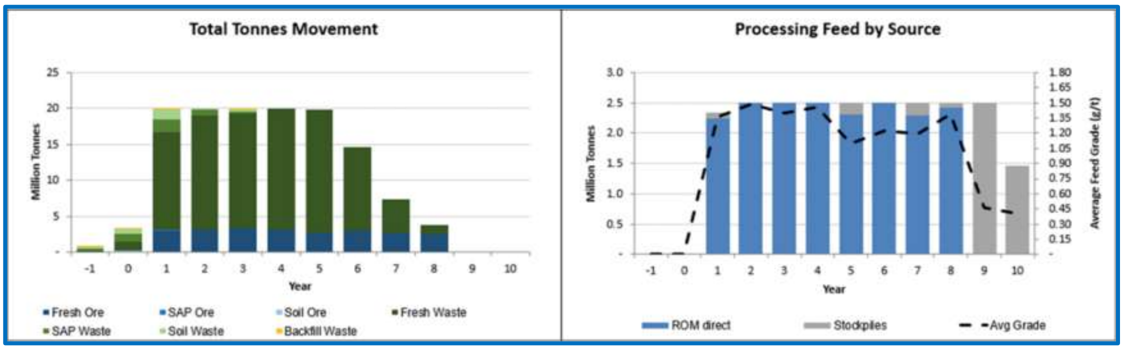
The Posse Gold Project is based on a mining concept that uses conventional drill, blast, load and haul techniques for all mining areas and rock types. One hundred per cent of the fresh rock and 30% of the saprolite will be blasted and loaded with small excavators (74-t op. weight) into on-road mining trucks (45-t capacity), and hauled to final destinations, i.e., primary crusher, low grade stockpiles or waste dumps. Direct mining will be applied to soft material such as soil and fill materials.

The ore and ore/waste contact materials will be mined in 5-m high benches for selectivity purposes, while double benches of 10-m high will be adopted for waste where there is no risk of dilution or ore loss. The mining method will generate variable quantities of low grade that will require the use of stockpiles. Front-end loaders ("**FELs**") will provide RoM feed and stockpile re-handling. The mined waste will be distributed into six waste dumps.

It is SRK's opinion that the method is appropriate to the orebody geometry, mineralization style, production rate, and is benchmarked with similar mining operations.

The mine schedule achieved a production target of 2.5Mtpa with a maximum annual rock movement (ore and waste) of 20.0Mtpa (Figure ES 1). A variable cut-off grade strategy was implemented by which the high grades were mined in the early periods while leaving the low grades for the end of the mining sequence. The LoM sequence encompasses a 15-month pre-stripping phase between October 2022 and December 2023 followed by 8 years of primary ore mining and, finally, 2 years of re-handling low grade ore.

Figure ES 1: Mine Schedule

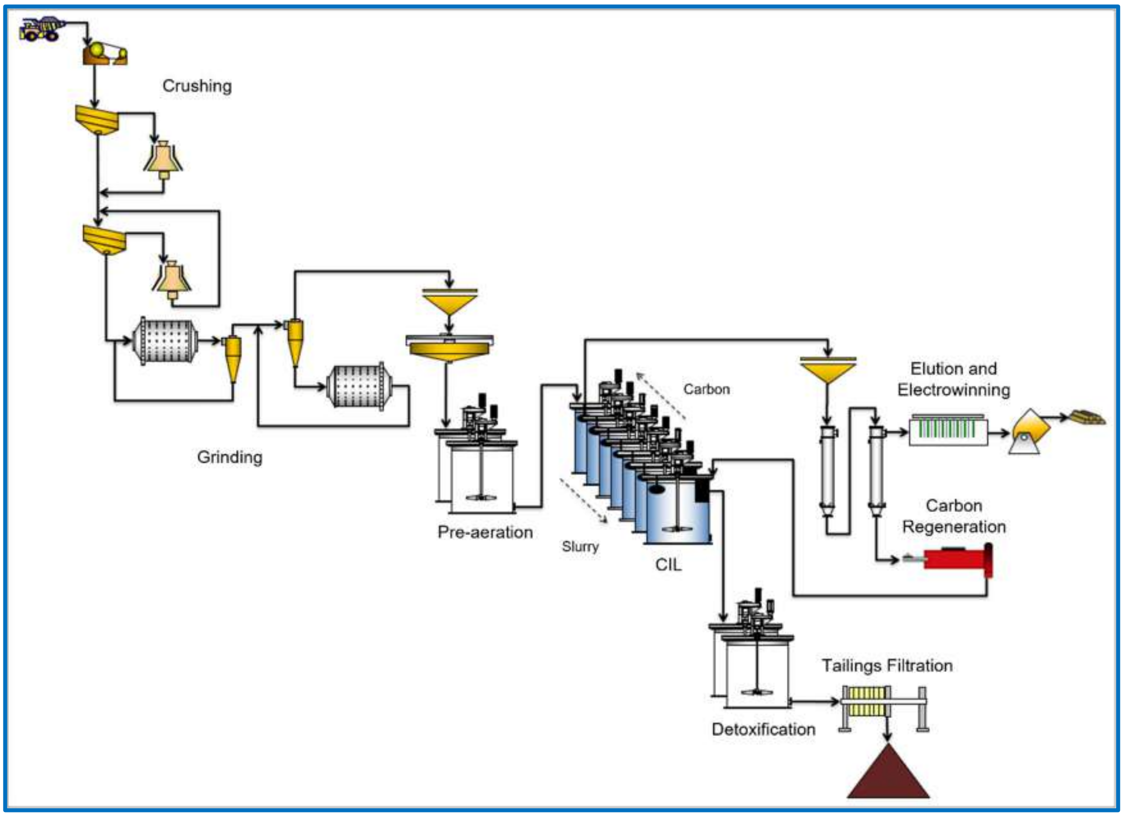


2.8 Recovery Methods

The Project process plant will have a capacity of 2.5Mtpa. The process plant includes crushing, milling, pre-leach thickening, pre-oxidation and CIL adsorption, desorption, regeneration and gold room (Figure ES 2). The process plant also includes tailings detoxification and filtration. The filtered tailings are transported and stored in a tailings pile.

The process flow sheet proposed for the Posse Gold Project applies well proven unit processes in the gold/silver processing industry. Novel recirculation and high shear technology is included for oxygen addition in the pre-oxidation and CIL circuit.

Figure ES 2: Process Flowsheet

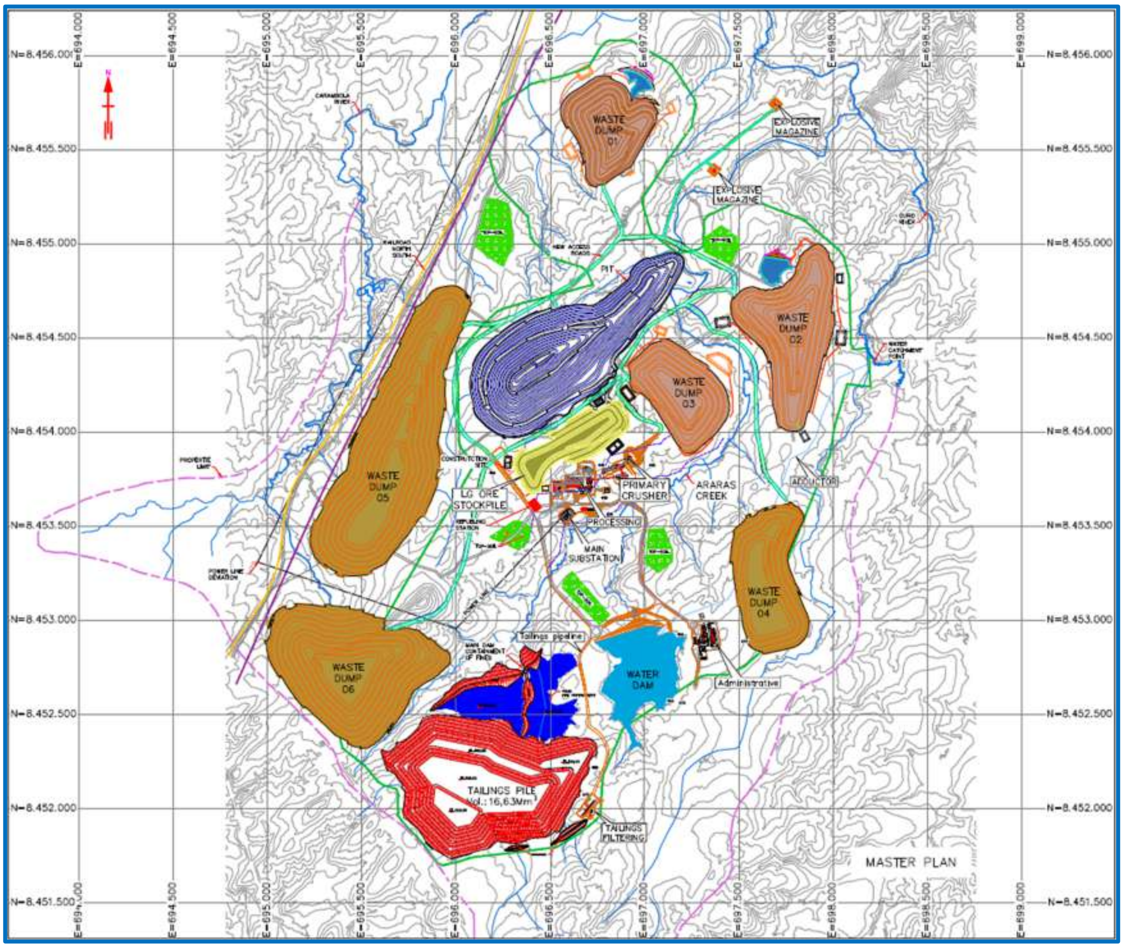


2.9 Project Infrastructure

The Project infrastructure consists mainly of the process plant, buildings, power line, water dam, filtered tailings pile, waste dumps and low grade stockpile (Figure ES 3).

The Project access and most of service roads are existing roads, minimizing earthworks and clearing vegetation. The construction of 67km of a 138kV transmission line to link Porangatu and the mine site will be required.

Figure ES 3: Mine Site Layout and Infrastructure



2.10 Environmental Studies and Permitting

Amarillo has completed an Environmental Impact Assessment to fulfil initial requirements for the licensing process. This study was concluded in 2015 and submitted to the regulatory agencies to apply for the Preliminary License (LP), which was granted to Amarillo in 2016 by the State of Goiás environmental agency (LP #792/2016). The Installation License (LI) application was filed on December 13, 2019. During 2021 and the first quarter of 2022, Amarillo received the LI of several components. All the licenses granted to date are summarized in Table ES 4.

Table ES 4 Licenses and Water Grants for the Posse Mine

Type	Number	Issuance	Expiration	Comment
LP	792/2016	05/05/16	-	Posse Mine
LI	06/2021	02/02/21	02/02/22	Substation Expansion in Porangatu City
	45/2021	02/03/21	01/29/27	Construction Site, Access Roads, and Topsoil Deposition
	226/2021	05/18/21	05/18/22	Waste Rock Pile 1
	245/2021	05/28/21	01/29/27	Waste Rock Pile 2
	309/2021	06/30/21	01/29/27	Waste Rock Pile 4
	418/2021	10/15/21	10/15/27	Waste Rock Pile 3
	421/2021	10/19/21	10/19/31	138 kV Power Line
	474/2021	12/14/21	12/14/27	Low-Grade Ore Pile
	34/2022	02/02/22	02/02/28	Mine Pit
Environmental Registry	-	10/18/21	-	69 kV Power Line
Water license	1412/2020	07/15/20	04/08/30	Rio do Ouro - River
Authorization	17/2021	05/03/21	05/03/23	IPHAN – Archeology ordinance
Archeology Registry	-	10/05/21	-	Term of commitment – 69kv
Authorization	2649/2020	08/12/20	-	Rescue and Conservation of Terrestrial Fauna

The implementation of a Dry Stack Tailings Storage Facility (TSF) will significantly reduce the Project water demand out of the Rio do Ouro from 720m^3/h to 136m^3/h, which means an 80%

reduction in water consumption. Amarillo requested a permit for this uptake from Rio do Ouro from the National Water Agency in 2019 and was granted in July 2020.

The Project has acquired 926.2 hectares of the 1070.2 hectares required for the Project. The remaining 144 hectares are under judicial negotiation and are expected to be finalized before the beginning of construction. The land where the pit is located has been acquired.

A preliminary mine closure plan was developed in 2020 by Ramboll which includes closure activities for each phase of the Project. The plan cost has been updated in 2021.

2.11 Capital and Operating Costs

The capital cost estimate for the Posse Gold Project is broken down by area including mining, crushing, processing plant and associated infrastructure. Processing and on-site infrastructure were developed by Ausenco. The filtered tailings pile, water dam, waste dumps, power transmission line, mining and other owner costs were estimated by Amarillo. SRK reviewed the capital cost build-up and quotes for all areas except the process plant, tailings filtration plant, and on-site infrastructure. Contingency has been included in the estimate.

All pre-production costs are considered as capital cost.

The capital cost estimates are based primarily on quotes by vendors (materials, supplies, equipment, and installation) and mining contractors (drilling, blasting and mining during the pre-stripping). Table ES 5 summarizes the overall Project capital.

The mine closure cost is estimated at US$20.0m, including all activities related to pre-closure, closure, and post-closure phases.

Table ES 5 Capital Cost Estimate Summary

Item	Initial Capex (US$k)	Sustaining (US$k)	Total (US$k)
Processing plant and infrastructure(1)	112,882	0	112,882
Power line	13,805	0	13,805
Mining (pre-stripping)	9,299	0	9,299
Waste dumps and low-grade stockpile	19,503	24,703	44,206
Araras creek diversion	-	212	212
Water dam	2,000	0	2,000
Filtered tailings pile	0	9,951	9,951
Owner costs	13,369	5,000	18,369
Subtotal	**170,857**	**39,866**	**210,723**
Contingency	14,284	3,487	17,770
Subtotal	**185,141**	**43,352**	**228,493**
Working Capital	8,876	0	8,876
Total capital cost	**194,017**	**43,352**	**237,369**
Mine closure w/ 10% contingency	-	-	20,000

(1) With exception of owner cost, electric equipment and working capital

The operating costs are broken down by area including mining, processing, general and administrative ("**G&A**"), owner costs, and tailings management. The processing, G&A, mining and tailings logistics operating costs were estimated by Amarillo based on updated quotes. The operating costs are reported in US$.

Table ES 6 shows the operating cost summary, which amounts to US$23.06/t processed over the LoM.

Table ES 6 Operating Cost Estimate Summary

Item	Unit	Operating Cost
Mining	US$/t processed	9.97
Processing w/ 5% allowance	US$/t processed	10.89
G&A w/ 5% allowance	US$/t processed	1.20
Tailings Haulage and Disposal	US$/t processed	1.00
Total	**US$/t processed**	**23.06**

Table ES 7 shows the estimated cash cost over the LoM for a total gold production of 811koz.

Table ES 7 LoM Cash Cost Estimate

LOM Cash Cost Estimate	Total Cost (US$k)	Unit Cost (US$/oz)
Operating Cost Estimate		
Mining	237,431	292.8
Processing w/ 5% allowance	259,234	319.6

LOM Cash Cost Estimate	Total Cost (US$k)	Unit Cost (US$/oz)
G&A w/ 5% allowance	28,566	35.2
Tailings Haulage and Disposal	23,805	29.4
Operating Cost	**549,036**	**677.0**
Adjusted Operating Cost Estimate		
Refining, Transportation, Insurance	9,732	12.0
Royalties	79,553	98.09
Adjusted Operating Cost	**638,321**	**787.1**
All-in Sustaining Cost (“AISC”) Estimate		
Sustaining Capital	43,352	53.5
AISC	**681,673**	**840.6**

2.12 Economic Analysis

The following economic analysis contains forward-looking information with regard to the Mineral Reserve estimates, commodity prices, exchange rates, proposed mine production plan, projected recovery rates and processing costs, infrastructure construction costs and schedule. The results of the economic analysis are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here.

The discounted cash flow (“**DCF**”) economic analysis is based on the following:

- A Base Case gold price of US$1,600/oz;
- An exchange rate of R$5.05/US$;
- A 100% equity financing with no debt component;
- All revenues and costs are reported in ‘real’ constant US$ terms without escalation; and
- SRK’s economic analysis is for the purpose for Mineral Reserve estimates only.

The Net Present Value (“**NPV**”) @ 5% annual real discount rate is US$154.6m and the resulting internal rate of return (“**IRR**”) is 19%. The payback period based on the undiscounted cash flow is 3 years from the start-up date.

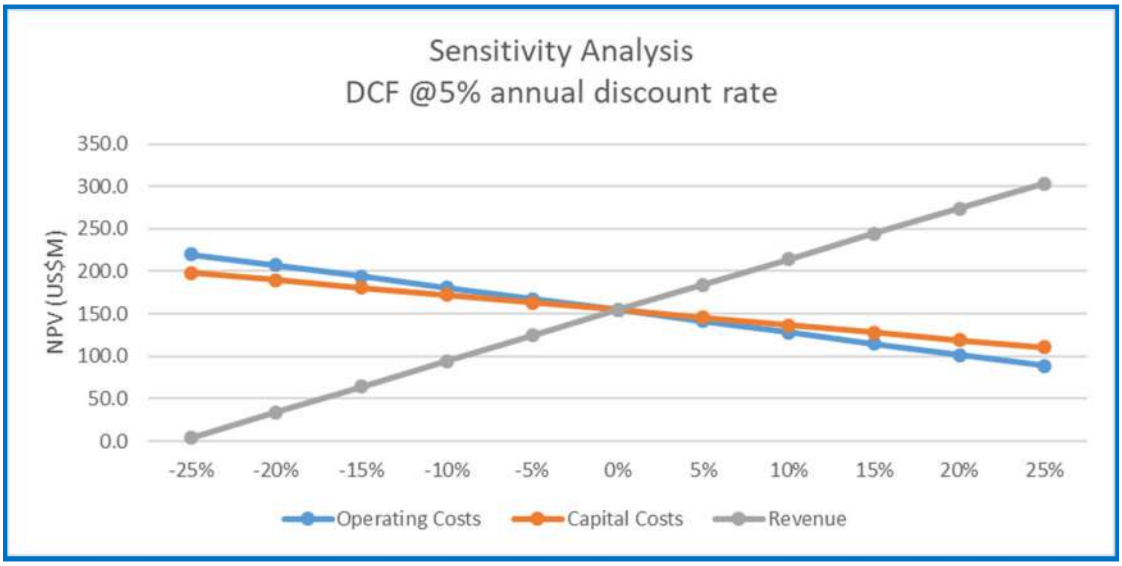
The results of the DCF analysis are shown in Table ES 8.

Table ES 8 DCF Results for the Base Case

Results	Annual discount rate			
	5%	**8%**	**10%**	**15%**
Pre-tax NPV (US$m)	**269.6**	204.9	169.1	99.6
Pre-tax IRR (%)	**28%**	28%	28%	28%
After-tax NPV (US$m)	**154.6**	107.5	81.6	31.4
After-tax IRR (%)	**19%**	19%	19%	19%
Tax rate (%)	**34%**	34%	34%	34%

Figure ES 4 shows the results of the sensitivity analysis. The Project is most sensitive to revenue, and least sensitive to capital expenditure.

Figure ES 4: Sensitivity Spider Chart (Opex, Capex and Revenue)



2.13 Comparison to Previous Studies

Table ES 9 compares the updated DFS study with previous reports.

Table ES 9 Comparison to Previous Studies

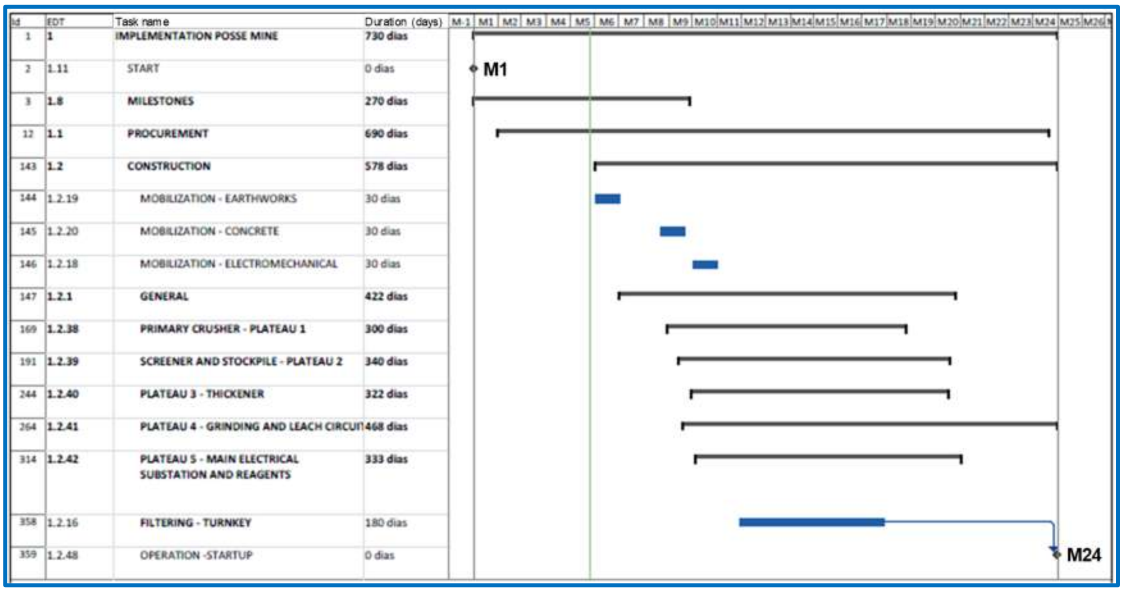
Category	Units	2017 SRKBR PFS	2018 SRKAU PFS	2020 SRKBR DFS	2021 SRKBR DFS Update
Exchange rate	US$ / R$	3.2	3.6	4.2	5.05
Initial capital (including initial working capital)	US$M	132.3	122.9	145.2	194.0
Sustaining capital	US$M	16.5	17.4	20.5	43.4
Total LOM capital	US$M	148.8	140.3	165.7	237.4
After-tax NPV @ 5%	US$M	178.3	244.3	183.1	154.6
After-tax IRR	%	35.2	50.8	25.1	19.4
Cash operating cost (excluding royalty & refining)	US$/oz	545	545	615	677
Cash operating cost (including royalty & refining)	US$/oz	603	633	706	787
AISC (including sustaining capital & closure)	US$/oz	627	655	738	841
Tonnes of ore processed	Mt dry	19.0	23.8	23.8	23.8
Grade of ore processed	g/t	1.63	1.42	1.18	1.18
LOM strip ratio (waste: ore)	t:t	4.5: 1	4.84: 1	4.44: 1	4.44: 1
Resources Measured & Indicated	contained koz	1,260	1,300	1,200	1,200
Resources cut-off grade	g/t	0.35	0.20	0.35	0.35
Resources average grade (M&I)	g/t	1.50	1.30	1.10	1.10
Reserves Proven & Probable	contained koz	998	1,087	902	902
Reserves cut-off grade	g/t	0.38	Variable	0.37	0.37
Reserves average grade	g/t	1.63	1.42	1.18	1.18
Gold price	US$/oz	US$1,200/oz 0.80 Revenue Factor Pit Shell US$1,200/oz Financials	US$1,300/oz 0.85 Revenue Factor Pit Shell US$1,300/oz Financials	US$1,400/oz 0.92 Revenue Factor Pit Shell US$1,400/oz Financials	US$1,450/oz 0.81 Revenue Factor Pit Shell US$1,600/oz Financials
Mining dilution & loss	%	3% dilution & 3% loss factors	3% dilution & 3% loss factors	Regularized mining model (4% dilution & 4% loss)	Regularized mining model (4% dilution & 4% loss)
Metallurgical recovery	%	92%	Variable 90.6% (LOM Average)	Variable 89.9% (LOM Average)	Variable 89.9% (LOM Average)

2.14 Project Implementation

An implementation plan was developed that addresses the Posse Gold Project schedule, engineering and construction management, procurement, logistics, construction, construction contracting, temporary facilities, project planning/execution/reporting, pre-commissioning and commissioning, and start-up/turnover.

The project plan has been developed for the duration of two years as shown in Figure ES 5. The activities related to grinding mills define the implementation critical path, therefore they will have special focus from the management and execution teams to ensure project targets are achieved.

Figure ES 5: Implementation Summary Schedule



2.15 Conclusions

The economic model for the Project demonstrates that under the current set of economic assumptions the Posse Gold Project provides a robust positive post-tax Net Present Value (“**NPV**”) of US$154.6m @ 5% annual discount rate over the LoM and an Internal Rate of Return (“**IRR**”) of 19.4%. Thus, it can be concluded that the Posse Gold Project is economically viable under the Base Case technical, legal and economic parameters.

2.16 Recommendations

2.16.1 Mineral Processing and Metallurgical Testing

The metallurgical performance is a function of gold head grade, the deportment of which is understood per the reserve model. Gold head grade providing a means to estimate recovery per the algorithm presented herein. The test work also suggests a telluride association as would be expected given the mineralogy of the ore and this may improve the accuracy of the recovery estimates. There is no tellurium model available for the reserve at this time.

To understand the metallurgy in the operating stage of the Project, it is recommended:

- Grade controls samples be subjected to a standardised leach test and include tellurium head assay so as to establish a data set of gold and tellurium head grades and extraction behaviour;
- That some grade control samples be subjected to the same leach test but at two alternative pH levels. This will allow the operations to associated gold and tellurium grade with benefit of higher and lower pH considering reagent demands and extraction; and
- Grade control sample viscosity also be determined. This being the only physical characteristic of the samples tested noted to be potentially problematic, albeit sporadic. A simple viscosity funnel test could be employed to simplify the data collection, combined with periodic cross-checks with a proprietary viscometer capable for presenting variable shear rates.

In the pre-operational stage and during operations, it is recommended:

- Future drilling of the resource/reserve includes sulphide sulphur and tellurium assays with a view to build a tellurium and possibly sulphide model in the future; and
- Some metallurgical test work be conducted to establish gold-tellurium-sulphide influences

on extraction and improve the prediction thereof.

2.16.2 Geology and Mineral Resources

The work undertaken to calculate the current Mineral Resource has indicated the need for further work including the following:

- Ensure that future diamond drilling is conducted in such a way that geological information is maximised and recorded in an appropriately structured database so that it can be used for future mineral resource development;
- Ensure that accurate rock density data is collected as a regular part of diamond drilling;
- Carry out further drilling to test the areas under the old Posse north pit to upgrade Indicated resource to Measured;
- Test drill the historic waste dumps to test the degree of mineralization in waste dumps;
- Ensure the check drilling of the backfill in the historic Posse pits is conducted early in the mine development to determine if the material is mineralised and represents unrecognised mineralisation and to confirm volumes;
- Updating of lithological and mineralisation wireframes;
- All re-assay results should be incorporated into the drillhole database as preferred assays and used for future modelling work together with the results of the 2021 drilling;
- The volume of underground workings, while small, should be recognised and removed from future models;
- The newly acquired SG data should be modelled as part of any future resource; and
- There is now assay data for a range of elements other than Au, those that have potential to interfere with metallurgy or which may indicate potential for AMD should be modelled as part of any future model.

2.16.3 Mining and Mineral Reserves

- The geotechnical study is based on a limited number of geotechnical boreholes. It is recommended that additional geotechnical boreholes be drilled to collect additional data to update the geotechnical characterization;
- Major structures need to be mapped in the old pit once access is re-established and used to develop a working structural geological model to assist pit design;
- Standard ground control/slope management procedures need to be adopted so that the design assumptions are validated during mining and the design is further optimized. Mapping of the footwall structures will be very important to maintain the optimal pit production as well as checking for the potential for adverse footwall structures that could be unstable;
- The intermediate pushbacks were designed using slope angles recommended for the walls of the ultimate pit. Good quality blasting of final walls and major intermediate cutbacks will be critical to good performance, so pre-splitting (or similar blasting techniques) should be adopted;
- A mine-to-mill approach should be considered to optimize the overall costs of mining and processing operations; and
- Develop grade control procedures to improve the mining model accuracy and grade estimates.

2.16.4 Recovery Methods

- Lime slurry capacity should be studied to assess the benefit of an increased capacity to add lime to the grinding pump boxes;
- For oxygen addition some testwork should be considered to verify the target dissolved oxygen concentrations can be achieved with the recirculation pump and high shear mixer. The total cost of ownership of the mixing system should also be investigated further to verify the technology; and
- The water treatment system should be evaluated to determine its suitability for cyanide detoxification.

2.16.5 Project Infrastructure

- The tailings generated from the ore processing plant will be accommodated in a Dry Stacking Facility ("**DSF**") after filtering. The DSF design is based on tests of tailings samples to determine resistance characteristics. The following additional studies are recommended for tailings characterization:
  - Improve the knowledge of the physical indexes and geotechnical parameters of the tailings to better estimate the safety and economic factors. The solid size distribution and the mineralogy of the fine fraction of tailings are essential to optimize the performance of filtering.
  - Specify the tailings compaction conditions, such as moisture content, as they are intrinsically associated with the pile configuration.
  - Undertake detailed studies of densification / compressibility, due to the impact of these parameters on the undrained behaviour of the material and on the predicted pore-pressure conditions.
  - Further investigate the liquefaction effect. Static liquefaction is activated in saturated tailings when these are subject to shear stress, mainly, within dykes at certain levels of disposal rates.
  - Design experimental fills to create the conditions for testing the Normal and Modified Proctor;
- Executive projects for the dry stacking pile and waste dumps were developed for the first two years of production. Additional studies and designs at a PFS level were then completed to accommodate the remaining materials until the end of the life of the mine. Future engineering iterations should increase the level of accuracy of these studies as required by the mine;
- Twenty-one drill holes were completed within the limits of the planned tailings pile to assess the geotechnical conditions. However, no samples were collected for laboratory analysis. The geotechnical parameters of the foundation need to be better understood through shear stress tests under dry and wet conditions covering all concerned lithologies;
- The stability analysis showed safety factors above the minimum limits established by the legal regulations under the specified premises. Hence, it is suggested that additional evaluations under pseudo-static conditions be performed for both the pile and the dyke;
- Thirty-seven drill holes were undertaken to assess the foundation of the planned waste dumps WD1, WD2 and WD3 and define the required excavation. It is recommended that laboratory tests be performed to estimate the geotechnical parameters; and
- The waste dumps WD1, WD2, WD3 and WD4 account for 36% of the total waste dumping capacity of the Project, which is sufficient to meet 3.5 years of operation, including the pre-

stripping phase. These waste dumps have already been granted the Installation License ("**LI**"). Additional areas will be required for future waste dumping (waste dumps WD5 and WD6). SRK recommends planning early to obtain the required environmental licenses so these waste dumps can be built in a timely manner.

2.16.6 Environment

- Develop a plan to obtain the operation license ("**LO**") according to schedule; and
- SRK recommends periodic updates of the mine closure plan to consider any changes in the socio-environmental conditions of the region, seeking to ensure post-closure sustainability in the generation of income and conservation of the environment and to comply with ANM 68/2021 which requires an updated every five years.

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COMPETENT PERSONS' REPORT ON THE POSSE GOLD PROJECT, STATE OF MINAS GERAIS, FEDERATIVE REPUBLIC OF BRASIL

1 INTRODUCTION

1.1 Background

SRK Consultores do Brasil Ltda ("**SRK**") has been requested by Hochschild Mining PLC ("**Hochschild**" and also the "**Client**" and or the "**Company**") to author a Competent Persons' Report (the "**CPR**") in respect of the "**Posse Gold Project**" (also the "**PGP**") a Development Property (defined below) located in the State of Goiás, Federative Republic of Brazil ("**Brazil**").

SRK has been informed that Hochschild has entered into a definitive agreement (the "**Agreement**") with Amarillo Gold Corporation ("**Amarillo Gold**") to acquire all of the issued and outstanding shares of Amarillo Gold (the "**Transaction**") at a price of C$0.40 per share in cash (the "**Cash Offer**"). Pursuant to the Transaction, Hochschild will acquire a 100% interest in Amarillo Gold's PGP located in Goiás State, Brazil. In addition, shareholders of Amarillo will receive shares in a newly formed company, Lavras Gold Corp. ("**Lavras Gold**"), which will hold a stake in the Lavras do Sul project (the "**LDS Project**"), C$10m of cash, and a 2.0% net smelter revenue royalty on certain exploration properties owned by Amarillo Gold and located outside the current PGP Mineral Resource and Mineral Reserve at Amarillo Gold's Mara Rosa Property comprising a land area of 2,552ha across three mining concessions plus numerous exploration leases in areas surrounding the PGP.

Amarillo Gold is a public Company whose ordinary shares are listed on the TSX Venture Exchange ("**TSXV**") which files all of its regulatory submissions on the System for Electronic Document Analysis and Retrieval ("**SEDAR**"): an electronic filing system established by the Canadian Securities Administrators (the "**CSA**") that allows listed companies to report their securities-related information with the authorities concerned with securities regulation in Canada. The previous regulatory technical submission filed by Amarillo Gold in respect of the PGP is the *"Amended and Restated NI 43-101 Technical Report Definitive Feasibility Study Posse Gold Project, Brazil"* published on 03 August 2020 (the "**PGP 2020 43-101 TR**") in accordance with the provisions adopted by the Canadian Institute of Mining and Metallurgy ("**CIM**") Definition Standards on Mineral Resources and Reserves (the "**CIM Definition Standards**") and incorporated into Canadian National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("**NI 43-101**") adopted by CIM Council in May 2014.

During 2021, SRK was further commissioned by Amarillo Gold to prepare an update of PGP 2020 43-101 TR to incorporate updates to the following items inter alia: capital expenditure and operating expenditures; construction and commissioning timelines; and commodity prices (hereinafter "**PGP 2022 43-101 TR**"). The PGP 2022 43-101 TR was filed on SEDAR on 21 February 2022 and has been re-reported in the format of a CPR as noted herein.

Hochschild is a public company whose ordinary shares are listed on the London Stock Exchange (the "**LSE**") a market operated by the London Stock Exchange Group plc. SRK has also been informed that the Acquisition as defined above is classified as a Class 1 transaction in respect of the Requirements as defined below. As such SRK has been requested to author the CPR which is required solely in respect of the PGP and accordingly any other Mineral Assets including other Exploration Properties which the Company may acquire/obtain as part of the Transaction are specifically excluded from the CPR.

Hochschild is a leading underground precious metals producer focusing on high grade silver and gold deposits, with over 50 years' operating experience in the Americas. The Company operates three underground mines, two located in southern Peru and one in southern Argentina. All of the Company's underground operations are epithermal vein mines and the principal mining method used is cut and fill. For the twelve-month period ended 31 December 2021 the Company reported attributable siler production of 12.17Moz, attributable gold production of 221koz and an All In Sustaining Cost ("**AISC**") from operations of US$1,241/oz.

The salient features of the PGP as reported herein reflect:

- Mineral Resources reported assuming an in-situ cut-off grade of 0.35g/tAu and comprising:
  - Measured and Indicated Mineral Resources of 32.0Mt grading 2.80g/tAu and containing 1,200kozAu,
  - Inferred Mineral Resources of 0.10Mt grading 2.40g/tAu and containing 1.7kozAu;
- Mineral Reserves reported assuming a cut-off grade of 0.37g/tAu and incorporating a long-term gold price of US$1,450/oz and reporting a total of 23.8Mt grading 1.18g/tAu and containing 902kozAu and comprising:
  - Proven Mineral Reserves of 11.8Mt grading 1.20g/tAu and containing 456kozAu,
  - Probable Mineral Reserves of 11.8Mt grading 1.20g/tAu and containing 436kozAu;
- The results of the various technical studies completed in respect of the PGP including:
  - the 2020 Definitive Feasibility Study (the "**2020 DFS**") as reported in the PGP 2020 43-101 TR published by SRK in August 2020, and
  - the PGP 2022 43-101 TR which includes various updates to commodity prices, macro-economic assumptions, operating and capital expenditure estimates, project construction and commissioning schedules completed during H2 2021;
- Post-Tax Pre-Finance cashflow analysis which indicates the following:
  - Total gold production of 811koz produced over a 10-year Life-of-Mine (**"LoMp**"),
  - Gross Sales Revenue of US$1,297.6m assuming a constant gold price of US$1,600/oz,
  - Operating expenditure of (US$638.3m,
  - Earnings before Interest Depreciation and Amortisation ("**EBITDA**") of US$639.3m,
  - Initial capital expenditure of US$194.0m,
  - Sustaining capital expenditure of US$43.4m,
  - Free cashflow of US$262.8m,
  - All In Sustaining Costs (**"AISC**") of US$841/oz of gold.

This CPR presents the following key technical information as at the Effective Date defined below):

- Mineral Resources and Mineral Reserve statements (the "**2021 Statements**") for the PGP reported in accordance with the terms and definitions of the CIMM Definition Standards (2014) also defined in Section 1.2.2 below;
- The Life-of-Mine plan ("**LoMp**") for the PGP reflecting depletion of the Mineral Reserves including assumed production, sales, sales revenue, operating and capital expenditure commencing 1 January 2021;

- The “**Environmental and Social Liabilities**” for the Mineral Assets inclusive of all mine closure related expenditures and retrenchment costs for the LoMp Scenarios; and
- Financial Modelling of the Mineral Assets undertaken to support the technical and economic viability of the Ore Reserves and the LoMp Scenarios as reported herein.

For the avoidance of doubt, this CPR is limited to the Mineral Assets and specifically exclude all assets and liabilities relating to the Group’s activities external to the Mineral Assets as defined herein. Notwithstanding the aforementioned, this CPR does include the results of the Financial Modelling of the Mineral Assets which relies on certain inputs including TEPs as provided by the Company and as appropriate, modified and adjusted by SRK. Certain units of measurements and technical terms defined in the CIM Definition and Standards (defined in 1.2.2 below) are defined in the glossaries, abbreviations and units included at the end of this CPR.

1.2 Requirement, Reporting Standard and Reliance

The CPR will be published in a “**Shareholder Circular**” being issued to all Hochschild shareholders in order to convene a general meeting to vote on a resolution approving the Transaction. Hochschild has engaged RBC Capital Markets (“**RBC**”) as its financial advisor, sole sponsor and corporate broker, Stikeman Elliott LLP (“**Stikeman**”) as its Canadian legal counsel, Pinheiro Neto Advogados as its Brazilian legal counsel, and Linklaters LLP (“**Linklaters**”) as its UK legal counsel in connection with the Transaction.

1.2.1 Requirement

The CPR is to be prepared in compliance with the following requirements which together comprise the “**Requirements**”:

- The “**Listing Rules**” published by the FCA from time to time and under Part VI of the Financial Services and Markets Act 2000 of the United Kingdom (the “**FSMA**”); and
- The “***ESMA update of the CESR recommendations: The consistent implementation of Commission Regulation (EC) No 809/2004 implementing the Prospectus Directive***”, published on 20 March 2013: specifically paragraphs 131 to 133, section 1b – mineral companies, Appendix I – Acceptable Internationally Recognised Mining Standards, and Appendix II – Mining Competent Persons’ Report – recommended content, hereinafter and collectively referred to as the “**CESR Recommendations**” and published on 20 March 2013.

Accordingly, whilst Amarillo Gold is in accordance with its regulatory reporting requirements publishing the PGP 2022 43-101 TR, the CPR as published by the Company in respect of the PGP will contain the same technical information as incorporated into the Technical Report and largely presented in the same format of a NI 43-101, but with appropriate references to the required Rules and Regulations and other presentational amendments.

With respect of paragraphs 132(a)-(e) of the CESR Recommendations SRK notes that all relevant details are included in the discipline technical Sections for the PGP. In respect of compliance with “**Appendix II**” of the CESR Recommendations, specifically the recommended content of the Competent Persons’ Reports SRK respectfully highlights the following:

- **Scope of the CPR:** The primary focus of the CPR is with respect to the provision of independently audited and current: Mineral Resources and Mineral Reserves; Life-of-Mine plans (limited to Ore Reserves only); Environmental and Social Liabilities; and Financial Modelling of the PGP as reported herein; and
- **Compliance Cross Reference** for similar groupings noted for paragraphs 132(a)-(e) above, the following items are referenced in Section 4 Property Description And Location, Section

5 Accessibility, Climate, Local Resources, Infrastructure And Physiography, Section 6 History, Section 7 Geological Setting And Mineralisation, Section 8 Deposit Types, Section 9 Exploration, Section 10 Drilling, Section 11 Sample Preparation, Analyses And Security, Section 12 Data Verification, Section 13 Mineral Processing And Metallurgical Testing, Section 14 Mineral Resource Estimates, Section 15 Mineral Reserve Estimates, Section 16 Mining Methods, Section 17 Recovery Methods, Section 18 Project Infrastructure, Section 19 Market Studies And Contracts, Section 20 Environmental Studies, Permitting And Social Or Community Impact, Section 21 Capital And Operating Costs, Section 22 Economic Analysis, Section 23 Adjacent Properties, Section 24 Other Relevant Data And Information, Section 25 Interpretation And Conclusions, and Section 26 Recommendations:

- Item (i) Legal and Geological Overview of the Mineral Assets including (1) and (2),
- Item (ii) Geological Overview,
- Item (iii) Mineral Resources and Mineral Reserves including (1) (2), (3), (4 and 5), (6), (7), (8a), (8b), 8 (c and d),
- Item (iv) Valuation of Mineral Reserves/Mineral Assets. This CPR includes a Valuation of the Mineral Reserves,
- Item (v) Environmental, Social and Facilities: (1), (2), (3),
- Item (vi) Historic Production/Expenditures,
- Item (vii) Infrastructure,
- Item (viii) Maps,
- Item (ix) Special Factors.

1.2.2 Reporting Standard

Mineral Resources and Mineral Reserves

The reporting standard adopted for the reporting of the Mineral Resource and Ore Reserve statements included in the CPR is that adopted by the Canadian Institute of Mining and Metallurgy ("**CIM**") Definition Standards on Mineral Resources and Reserves (the "**CIM Definition Standards**") and incorporated into Canadian National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("**NI 43-101**") adopted by CIM Council in May 2014.

Technical Study Standards

A Feasibility Study is a comprehensive technical and economic study of the selected development option for a mineral project that includes appropriately detailed assessments of applicable Modifying Factors together with any other relevant operational factors and detailed financial analysis that are necessary to demonstrate, at the time of reporting, that extraction is reasonably justified (economically mineable). The results of the study may reasonably serve as the basis for a final decision by a proponent or financial institution to proceed with, or finance, the development of the project.

Environmental and Social Standards

Environmental and Social Standards as considered in this CPR has been, where practically possible, assessed with due consideration for national legislation and regulation as currently applicable in Brazil. SRK notes, however that the PGP has not been assessed in respect of international standards and guidance. In respect of the latter standards and guidance SRK has not considered adherence or alignment with the International Financial Corporation's Performance Standards ("**IFC PS**") and relevant World Bank Group's Environmental Health and Safety Guidelines.

Accordingly, the principal focus of the Environmental and Social review in respect of the Mineral

Assets comprised a review of the Environmental Management Practices and Environmental and Social Liabilities (Bio-Physical and Social) at the Mineral Assets with specific focus on the primary regulatory documentation and compliance with the conditions of approval, including emissions and discharges in respect of local standards. It is however important to note that this review did not constitute a detailed Environmental Audit, does not extend to provide a detailed opinion and development of any Equator Principles Action Plan capable of bringing the technical studies into compliance with the Equator Principles, nor indicate when compliance is not possible as typically required for a Project Finance facility: for all Category A and, as appropriate, Category B Projects.

Responsible sourcing regulations are an increasing focal point for stakeholders in the international mining and metals sector and in addition to national legislation, there are also a number of regulations and guidance that specifically cover the responsible souring of gold. For example, the "**Dodd-Frank**" legislation in the United States (Section 1502) and the "**EU Conflict Free Minerals**" regulations require due diligence within the supply chain in order to ensure that mining and production of gold does not fund conflict. One of the most widely recognised is the "**OECD Due Diligence Guidance for Responsible Supply Chains of Minerals from Conflict-Affected and High-Risk Areas**". The guidance was operationalised by the World Gold Council for the mining sector, the London Bullion Market Association for the refining sector and the Responsible Jewellery Council for this sector.

With respect to "**Mine Closure"** related liabilities key international standards include those which are focused on a combination of technological and engineering solutions which reflect Good International Industry Practice ("**GIIP**") and "**Best Available Technology**" to where practicable achieve "**Ground Zero**" or "**Walk Away**" remediation status. Guiding standards which reinforce these objectives include: the International Council on Mining and Minerals ("**ICMM**") Planning for Integrated Mine Closure: Toolkit (2008); World Bank in Mining and Development, It's Not Over When It's Over: Mine Closure Around the World (2002); European Commission's Reference Document on *"Best Available Techniques for Management of Tailings and Waste-Rock in Mining Activities"* published in 2009; *"IFC EHS Guidelines on Construction and Decommissioning"* published in 2007; and *"Mining for Closure: Policies and Guidelines for Sustainable Mining Practice and Closure of Mines"* published by United Nations Environment Programme ("**UNEP**"), United Nations Development Programme ("**UNDP**"), Organization for Security and Co-operation in Europe ("**OSCE**") and the North Atlantic Treaty Organization ("**NATO**") in 2005.

Mineral Asset Valuation

This CPR includes a Valuation of the Mineral Reserves and is reported in accordance with the general disclosure principles and process as defined by the ***"CIMVAL Code for the Valuation of Mineral Properties"***, prepared by the Special Committee of the Canadian Institute of Mining, Metallurgy and Petroleum on the Valuation of Mineral Properties ("**CIMVAL**") and adopted by the CIM Council on November 29, 2019, ("**CIMVAL 2019**").

Cash Cost Reporting

The determination of cash costs in the metals and mining sector varies both within and between commodity focus companies. Furthermore, it would appear that with respect to reporting standards, that defined by the World Gold Council ("**WGC**") and published (2018) ("**WGC 2018**") in its guidance noted on "*all-in sustaining costs*" and *"all-in costs*" metrics would appear to be the most comprehensive. This was an advance from the cash cost reporting methodology introduced in 1996 which focused solely on the mining and processing costs incurred. In contrast WGC 2018 focuses on costs incurred in the complete mining life cycle from exploration to closure. In this instance SRK notes the following industry standard definitions:

- Cash Costs reported per ounce gold sold and reported on a by-product basis, where expenditures are determined net of silver sales where relevant. Cash costs are defined as:
  - Adjusted Operating Costs (“**AOC**”) comprising on-site mining costs, on-site general and administrative costs, royalties and production taxes, realised gains/losses on hedges due to operating costs, community costs related to current operations, refining and transport costs, non-cash remuneration (site-based), stockpile/leach pad and product inventory write down, operational waste stripping costs and by-product credits;
  - All In Sustaining Costs (“**AISC**”) comprising corporate general & administration costs (including share-based remuneration), reclamation and remediation accretion and amortisation (operating sites), exploration and study costs (sustaining), capital exploration (sustaining), capitalised stripping & underground mine development (sustaining), sustaining capital expenditure and sustaining leases;
  - All-in Costs (“**AIC**”) comprising growth and development costs not related to current operations, community costs not related to current operations, permitting costs not related to current operations, reclamation and remediation costs not related to current operations, exploration and study costs (non-sustaining), capital exploration (non-sustaining), capitalised stripping & underground mine development (non-sustaining), non-sustaining capital expenditure and non-sustaining leases.

  In respect of the above items it is important to note that the following expenditures are typically not included in the WGC guidance: corporate income tax; working capital (except for adjustments to inventory on a sales basis); all financing charges (including capitalised interest); costs related to business combinations, asset acquisitions and asset disposals; items needed to normalise earnings, for example impairments on non-current assets, one-time material severance charges or legal costs or settlements or legal costs or settlements related to significant lawsuits.

1.2.3 Reliance

This CPR is addressed to and may be relied on by the Directors of the Company and the “**Advisors**”, specifically in compliance with the Requirements and the Reporting Standard. Accordingly, SRK has confirmed in writing (the “**Consent letter**”), dated on the Publication Date which confirms:

- Reliance as regards the CPR for any benefit of the Company and its Advisors;
- Consent to the inclusion of the CPR, and to the inclusion of any extracts from the CPR in the Prospectus;
- Confirmation that all information contained in the Prospectus which is extracted from the CPR or based upon information contained in the CPR has been reviewed by SRK and that such information as presented is accurate, balanced, complete and not inconsistent with the CPR; and
- Responsibility for the CPR and declares that it has taken all reasonable care to ensure that the information contained in the CPR is, to the best of its knowledge, in accordance with the facts and makes no omission likely to affect its import.

SRK has no obligation or undertaking to advise any person of any development in relation to Mineral Assets which comes to its attention after the date of this CPR or to review, revise or update the CPR or opinion in respect of any such development occurring after the date of this CPR.

1.3 Effective Date, base Technical Information Date, Declarations and Copy right

The effective date of the CPR is 31 December 2021 (the "**Effective Date**"). The 2021 Statements, the LoMps, the TEPs, the Environmental and Social Liabilities and Financial Modelling of the PGP reflect SRK's assessments of the:

- Mineral Resource and Ore Reserves statements as noted in the 2021 Statements and reported by SRK in accordance with the CIM Definition Standards;
- LoMp Scenarios with projected production from 1 January 2022;
- Detailed schedules of activities and expenditures relating to the derivation and support of the forecast TEPs as included in the LoMp Scenario for the PGP including, production, sales, sales revenue, operating expenditure and capital expenditure;
- Cashflow Model for the PGP incorporating annual forecasts of the TEPs and resulting post-tax pre-finance cashflows;
- Mine closure costs relating to the PGP s comprising the Environmental and Social Liabilities reported herein;
- Cashflow Modelling of the Mineral Assets to assess the technical and economic viability of the Ore Reserves.

The Base Technical Information Date is defined as 1 January 2022 which is co-incident with the reporting date for the 2021 Statements, this being 31 December 2021. The Publication Date of the CPR is assumed to be 4 March 2022. As advised by the Company, as at the Publication Date of the Circular no material change has occurred as of the Effective Date of the CPR inclusive of: the 2021 Statements; the LoMp and accompanying TEPs; the Environmental and Social Liabilities; and the Cashflow Modelling of the PGP.

1.4 Verification, Validation and Reliance

This CPR is dependent upon technical, financial and legal input from the Company, Amarillo and its third-party consultants. Following publication of the PGP 2020 43-101 TR, SRK has undertaken a detailed review of various updates to the 2020 DFS to reflect changes with respect to commodity prices and macro-economics, operating and capital expenditure assumptions and construction and commissioning schedules completed during H2 2021. The results of this review are reported in the PGP 2022 43-101 TR, recently filed on SEDAR, and the results of which are reproduced in this CPR.

The Qualified Person who takes overall responsibility for the CPR and the Mineral Reserve as reported herein is Mr Paulo Laymen who undertook a site visit in September 2018. SRK confirms that whilst it has not undertaken any site visits since September 2018 and given the current greenfield status of the PGP and limited site activity since this date, the technical data and technical opinion as expressed in this CPR remain valid as at the Effective Date of the CPR, that being 31 December 2021. Furthermore, SRK notes that as part of the original 2020 DFS, SRK authored the Mineral Reserve statement and all underlying mining engineering work streams required to support the 2021 Statements.

SRK confirms that it has performed all necessary validation and verification procedures deemed necessary and/or appropriate to place a suitable level of reliance on such technical information. SRK considers that with respect to all material technical-economic matters, it has undertaken all necessary investigations to ensure compliance with the Requirements including the Reporting Standards (specifically the CIM Definition Standards and the CIMVAL Code).

In consideration of all legal aspects relating to the PGP, SRK has placed reliance on the representations by the Company and Amarillo that the following are correct as at the Effective Date of the CPR and remain correct until the date of the Public Document:

- That save as disclosed in the CPR, the Directors of the Company are not aware of any legal proceedings that may have an influence on the rights to explore for minerals in respect of the Mineral Assets;
- That Amarillo is the legal owner of all relevant mineral and surface rights as reported in the CPR; and
- That save as expressly mentioned in the CPR, no significant legal issue exists which would affect the likely viability of the PGP and/or the estimation and classification of the Mineral Resources and Mineral Reserves, the LoMp, the Environmental and Social Liabilities, and the Cashflow Modelling.

The Mineral Resource and Mineral Reserve statements as included in the 2021 Statements are reported with a date of depletion of 31 December 2021. For the avoidance of doubt, the 2021 Statements are the "current statements" and any historical statements as reported herein are done so solely for comparative purposes to provide context with respect to any significant changes and to support the reconciliation process between reporting periods.

1.5 Limitations, Responsibility Statement, Reliance on Information, Declarations and Copyright

1.5.1 Limitations

Save as set out in Section 1.2.3 above and for the responsibility arising under the Requirements to any person and to the extent there provided, to the fullest extent permitted by law, SRK does not assume any responsibility and will not accept any liability to any other person for any loss suffered by any such other person as a result of, arising out of, or in connection with this CPR or statements contained therein, required by and given solely for the purpose of complying with the Requirements, consenting to its inclusion in the Circular.

SRK notes that this CPR has been prepared in accordance with the Requirements as defined herein. For the avoidance of doubt SRK notes that the contents of this CPR including the technical opinion as expressed herein must be read in association with the Responsibility Statement, Reliance on Information, Declarations and Consent as reported herein.

The achievability of the projections as reported in this CPR, are neither warranted nor guaranteed by SRK, specifically the: TEPs including assumed production, sales volumes, sales revenue, operating and capital expenditure relating to depletion of the Ore Reserves from 1 January 2022; the Environmental and Social Liabilities; and the Cashflow Modelling relating to the PGP. The projections as presented and discussed herein have been proposed by the Company's management and adjusted where appropriate by SRK to reflect its opinion but cannot be assured. Notably, for example, they are necessarily based on economic and market assumptions, many of which are beyond the control of the Company.

Future cashflows and profits derived from any projections reflected by the TEPs in the LoMp, the Environmental and Social Liabilities are inherently uncertain and actual results may be significantly more or less favourable.

Unless otherwise expressly stated all the opinions and conclusions expressed in this report are those of SRK. It should also be noted that this report reflects SRK's review of information generated, and/or technical work completed, by others. As a result of this, the projections presented here may not directly reflect that previously presented by the Company or in public announcements made by the Company as they also incorporate judgements made by SRK not necessarily incorporated into the Company's assessments.

This CPR specifically excludes all aspects of legal issues, marketing, commercial and financing matters, insurance, land titles and usage agreements, and any other agreements and/or contracts that the Company may have entered into.

1.5.2 Responsibility Statement

For the purpose of, and in compliance with, the Requirements, SRK accepts responsibility for the information provided in the CPR and for all information in the Prospectus which is extracted or sourced from the CPR. SRK declares that the information contained in the CPR and the Prospectus is, to the best of the knowledge of SRK, in accordance with the facts and makes no omission likely to affect its import. SRK has given and has not withdrawn its written consent to the publication of the CPR.

SRK accepts responsibility for the 2021 Statements, the LoMp Scenario and associated TEPs, the 2021 Environmental and Social Liabilities, the Cashflow Modelling of the PGP as reported herein. Where applicable, SRK confirms that:

- the 2021 Statements are reported in accordance with the terms and definitions of the CIM Definition Standards;
- the various technical studies supporting the Production Scenarios have been completed in accordance with the Technical Study standards as defined in Section 1.2.2. of this CPR;
- that the Environmental and Social Liabilities are derived and reported in accordance with local standards; and
- the Cashflow Modelling for the PGP as reported herein are reported in accordance with the CIMVAL (2019).

The scope of the CPR is limited to the PGP as reported herein and expressly excludes all other mineral assets relating to the Transaction or currently owned by the Company.

1.5.3 Reliance on Information

SRK believes that its opinion must be considered as a whole and that selecting portions of the analysis or factors considered by it, without considering all factors and analyses together, could create a misleading view of the process underlying the opinions presented in this CPR. The preparation of a CPR is a complex process and does not lend itself to partial analysis or summary.

SRK's opinions given in this document with respect to the 2021 Statements, the LoMp and accompanying TEPs, the Environmental and Social Liabilities, and the Cashflow Modelling are effective at 31 December 2021 and are based on information provided by the Company and Amarillo throughout the course of SRK's investigations, which in turn reflects various technical-economic conditions prevailing at the date of this report and the Company's expectations regarding the gold market, gold prices and exchange rates as at the date of this report. These and the underlying TEPs, comprising projections of production, sales, sales revenue, operating and capital expenditures can change significantly over relatively short periods of time. Should these change materially, the 2021 Statements, the LoMp Scenarios and accompanying TEPs, the Environmental and Social Liabilities, and the Cashflow Modelling of the CPR could be materially different in these changed circumstances.

Whilst SRK has exercised all due care in reviewing the supplied information, SRK does not accept responsibility for finding any errors or omissions contained therein and disclaims liability for any consequences of such errors or omissions.

This CPR includes technical information, which requires subsequent calculations to derive subtotals, totals and weighted averages. Such calculations may involve a degree of rounding and consequently introduce an error. Where such errors occur, SRK does not consider them to be material.

1.5.4 Declarations

SRK will receive a fee for the preparation of this CPR in accordance with normal professional consulting practice. This fee is not contingent on the outcome of any transaction and SRK will receive no other benefit for the preparation of this report. SRK does not have any pecuniary or other interests that could reasonably be regarded as capable of affecting its ability to provide an unbiased opinion in relation to 2021 Statements, the principal findings regarding the LoMp Scenario, the Environmental and Social Liabilities and the Cashflow Modelling of the PGP as reported herein.

Neither SRK, the Qualified Persons (as identified under Section 1.7, below) who are responsible for authoring this CPR, nor any Directors of SRK have at the date of this report, nor have had within the previous two years, any shareholding in the Company, the PGP or the Advisors of the Company, or any other economic or beneficial interest (present or contingent) in any of the assets being reported on. SRK is not a group, holding or associated company of the Company. None of SRK's partners or officers are officers or proposed officers of any group, holding or associated company of the Company. Further, no Qualified Person involved in the preparation of this CPR is an officer, employee or proposed officer of the Company or any group, holding or associated company of the Company. Consequently, SRK, the Qualified Persons and the Directors of SRK consider themselves to be independent of the Company, its directors, senior management and Advisors.

1.5.5 Consent

SRK has given and has not withdrawn its written consent to the publication of this CPR and has authorised the contents of its report and context in which they are respectively included and has authorised the contents of its report for the purposes of compliance with the Requirements.

1.5.6 Copyright

Except where SRK has agreed otherwise (including pursuant to an agreement between SRK and the Company dated 14 February 2022 or any subsequent agreement (each, the "**Hochschild Agreement**")):

- neither the whole nor any part of this report nor any reference thereto may be included by any party other than the Company, any of its direct and indirect subsidiaries or a competent state authority in the United Kingdom of Great Britain and Northern Ireland or any other relevant jurisdiction, as may be applicable (together, the "**Recipients**"), in any other document without the prior written consent of SRK save that in the case that the report is not included in full in any other document, the Recipient shall present a draft of any document produced by it that may incorporate a part of this report to SRK for review so that SRK may ensure that this is presented in a manner which accurately and reasonably reflects any results or conclusions contained in this report; and
- copyright of all text and other matters in this document, including the manner of presentation, is the exclusive property of SRK. It is an offence to publish this document or any part of the document under a different cover, or to reproduce and/or use, without written consent (whether granted by virtue of an Hochschild Agreement or otherwise), any technical procedure and/or technique contained in this document. The intellectual property reflected in the contents resides with SRK and shall not be used for any activity that does not involve SRK, without the written consent of SRK.

Neither the whole nor any part of this report nor any reference thereto may be included in any other document without the prior written consent of SRK regarding the form and context in which it appears.

1.6 Indemnities Provided by the Company

The Company has provided the following indemnities to SRK:

- The Company has agreed that, to the extent permitted by law, it will indemnify SRK and its employees and officers in respect of any liability suffered or incurred as a result of or in connection with the preparation of this report albeit that this indemnity will not apply in respect of (i) fraud, bad faith, gross negligence wilful misconduct or breach of law on the part of SRK or its employees or officers; or (ii) breach of this Agreement on the part of SRK. The Company has also agreed to indemnify SRK and its employees and officers for time incurred and any costs in relation to any inquiry or proceeding initiated by any person albeit that this indemnity will not apply in respect of (i) fraud, bad faith, gross negligence wilful misconduct or breach of law on the part of SRK or its employees or officers; or (ii) breach of this Agreement on the part of SRK; and
- In order to assist SRK in the preparation of this CPR the Company may be required to receive and process information or documents containing personal information in relation to SRK's project personnel. The Company has agreed to comply strictly with the provisions of the Data Protection Act 1998 of the United Kingdom ("**DPA 1998**") and all regulations and statutory instruments arising from the DPA 1998, and the Company will indemnify and keep indemnified SRK in respect of all and any claims and costs caused by breaches of the DPA 1998.

1.7 Qualifications of Consultants and Qualified Persons'

SRK is an associate company of the international group holding company SRK Consulting (Global) Limited (the "**SRK Group**"). The SRK Group comprises some 1,400 professional staff offering expertise in a wide range of resource and engineering disciplines with 45 offices located in 20 countries.

The SRK Group's independence is ensured by the fact that it holds no equity in any project. This permits the SRK Group to provide its clients with conflict-free and objective recommendations on crucial judgment issues. The SRK Group has a demonstrated track record in undertaking independent assessments of resources and reserves, project evaluations and audits, Mineral Resource and Ore Reserve audits and independent feasibility studies on behalf of exploration and mining companies and financial institutions worldwide. The SRK Group has also worked with a large number of major international mining companies and their projects, providing mining industry consultancy service inputs.

This CPR has been prepared by SRK Brasil and relies on various technical inputs to the recently published PGP 2022 43-101 TR which in turn relies on a number of historical documents, namely the prior PGP 2020 43-101 TR and the 2020 DFS. The PGP 2022 43-101 TR refers to a total of 10 consultants who are specialists in the fields of exploration, geology, Mineral Resource and Mineral Reserve estimation and reporting, open-pit mining, mining geotechnics, water management (hydrogeology/hydrology), mineral processing, tailings engineering, infrastructure, environmental and social, financial modelling and mineral asset valuation The individuals listed in Table 1-1 have provided the material input to the PGP 2022 43-101 TR and the historical documents upon which this CPR is based, have extensive experience in the mining industry and are members in good standing of appropriate professional institutions.

The Qualified Person who has overall responsibility for the CPR and the Mineral Reserves as reported in the CPR will be Mr Paulo Laymen, MSc, Registered Member in good standing of

Chilean Mining Commission (Comisión Calificadora de Competencias en Recursos y Reservas de Chile: Membership number 0320) and member of the Australasian Institute of Mining and Metallurgy (membership number 320077). In being a registered member of the Chilean Mining Commission, Paulo Laymen is a qualified member of Accepted Foreign Associations and Membership Designations within the meaning of Appendix A of NI 43-101. Mr Paulo Laymen is a full-time employee of SRK and is independent of the Company as defined herein and as sufficient relevant experience in the commodity, type of deposit and situation as reflected by the Mineral Reserve statement.

The Qualified Person who has responsibility for reporting of Mineral Resources in the CPR will be Mr Gregory Keith Whitehouse, B.Sci, MAusIMM (CP). In being a registered Chartered Professional Member of the Australian Institute of Mining and Metallurgy, Gregory Whitehouse is a qualified member of Accepted Foreign Associations and Membership Designations within the meaning of Appendix A of NI 43-101. Mt Gregory Whitehouse is a full-time employee of Australian Exploration Field Services Pty Ltd ("**AEFS**") and is independent of the Company as defined herein and as sufficient relevant experience in the commodity, type of deposit and situation as reflected by the Mineral Resource statement.

Table 1-1: Team members[1]

Responsible Discipline	Consultant	Designation	Registration, Membership, Qualification	Years' Experience
Geology/Mineral Resources	**Gregory Keith Whitehouse[2]**	**Principal**	**MAusIMM, CP, BSci**	**46**
	John Watts	Principal	BSc	54
	John Collier	Principal	BSc	22
Mining & Mineral Reserves, Geotechnical Engineering, Human Resources	**Paulo Laymen[2]**	**Principal**	**MCMMC (RM), BEng**	**20**
Metallurgy, Mineral Processing and Infrastructure	**Stuart Smith[2]**	**Principal**	**FAusIMM, Ba.App.Sci**	**35**
	Tommaso Roberto Raponi[2]	**Principal**	**APEG, Pr.Eng., BA.Sc.**	**38**
Waste and Water Management	Paulo Paiva	Principal	BEng., LLB	49
Environmental and Social	Nelson Siqueira	Principal	BSc.	42
Mine Closure	Cristina Simonetti	Principal	PhD Geol Sci	35
Financial Modelling	Luiz Confúcio	Consultant	MBA Fin	23

(1) Keith Whitehouse and John Watts are employees of Australian Exploration Field Services Pty Ltd ("**AEFS**"); John Collier is an employee of Conarco Consulting (Pty) Ltd ("**Conarco**"); Paulo Laymen is an associate of SRK; Stuart Smith is an employee of Aurifex Pty Ltd ("**Aurifex**"), Tommaso Roberto Raponi is an employee of Ausenco Engineering Canada Inc ("**Ausenco**"); Paulo Paiva is a full time employee of GeoHydroTech Engenharia; Nelson Siqueria is a full time employee of DBO Engenharia Ltda ("**DBO**"); Cristina Simonetti is a full time employee of the Ramboll Group ("**Ramboll**"); and Luiz Confúcio is a full time employee of SRK

(2) Qualified Persons within the meaning of NI 43-101.

1.8 Report Format

This CPR is structured on a technical discipline basis as follows:

- Section 1 Introduction;
- Section 2 Terms Of Reference, Qualifications And Site Visits;
- Section 3 Reliance On Other Experts;
- Section 4 Property Description And Location;
- Section 5 Accessibility, Climate, Local Resources, Infrastructure And Physiography;
- Section 6 History;
- Section 7 Geological Setting And Mineralisation;
- Section 8 Deposit Types;
- Section 9 Exploration;
- Section 10 Drilling;
- Section 11 Sample Preparation, Analyses And Security;
- Section 12 Data Verification;
- Section 13 Mineral Processing And Metallurgical Testing;

- Section 14 Mineral Resource Estimates;
- Section 15 Mineral Reserve Estimates;
- Section 16 Mining Methods;
- Section 17 Recovery Methods;
- Section 18 Project Infrastructure;
- Section 19 Market Studies And Contracts;
- Section 20 Environmental Studies, Permitting And Social Or Community Impact;
- Section 21 Capital And Operating Costs;
- Section 22 Economic Analysis;
- Section 23 Adjacent Properties;
- Section 24 Other Relevant Data And Information;
- Section 25 Interpretation And Conclusions; and
- Section 26 Recommendations

2 TERMS OF REFERENCE, QUALIFICATIONS AND SITE VISITS

2.1 Terms of Reference

SRK was retained by the Amarillo to update, with input from other engineering companies and consultants, the 2020 DFS for the Posse Gold Project, located in the municipality of Mara Rosa in the state of Goiás, Brazil, 360km to the north of the state capital, Goiânia.

Once constructed, the Posse Gold Project (the "**Project**") will consist of an open pit gold mine and related processing facilities for approximately 23.8Mt of mill feed (dry basis) at a rate of 2.5Mtpa.

In 2011, a PFS was prepared by Coffey Consultoria e Serviços Ltda (Coffey). Successive updates of the PFS were undertaken in 2017 and 2018 by SRK. A definitive feasibility study of the project was completed in 2020. Following publication of this report considerable work was completed to assess and ameliorate risks associated with the Project. This report discusses that work and the effect on the 2020 Mineral Resource and Mineral Reserve statements as reflected in the 2021 Statements.

Amarillo has previously filed the following NI 43-101 technical reports which include Mineral Resource Estimates for the Project as follows:

- **Caracle Creek International Consulting, 2008:** Independent Technical Report and Preliminary Economic Assessment, Mara Rosa Gold Property, Goiás State, Brazil. Report prepared for Amarillo Gold Corporation dated 29 February 2008;
- **Hoogvliet Contract Services and Australian Exploration Field Services PL. 2010:** Independent Mineral Resource Estimate and Preliminary Economic Assessment, Posse Deposit, Mara Rosa, Goiás State, Brazil. Report prepared for Amarillo Gold Corporation dated 30 June 2010;
- **Hoogvliet Contract Services and Australian Exploration Field Services PL. 2011:** Report on Independent Site Visit and Resource Estimate. Posse Deposit, Mara Rosa, Goiás State, Brazil. Report prepared for Amarillo Gold Corporation dated 30 July 2011;
- **Coffey Mining Pre-Feasibility Study, Mara Rosa Project, Goiás State, Brazil:** Report prepared for Amarillo Gold Corporation, dated 28 October 2011;
- **Australian Exploration Field Services PL. 2016:** Posse Deposit, Mara Rosa, Goiás State,

Brazil, Mineral Resource Update prepared for Amarillo Gold Corporation dated 21 July 2016;

- **SRK Consultores do Brasil Ltda, 2017 Updated PFS, Posse Mine Project:** Mara Rosa GO prepared for Amarillo Gold Corporation dated 11 April 2017;
- **SRK Consulting (Australasia) Pty Ltd, 2018:** Technical Update on the Posse Gold Project, Brazil, prepared for Amarillo Gold Corporation dated 12 September 2018;
- **SRK Consultores do Brasil Ltda:** 2020 Amended and Restated Definitive Feasibility Study Posse Gold Project, Brazil; and
- **SRK Consultores do Brasil Ltda:** 2022 Updated Definitive Feasibility Study Posse Gold Project, Brazil;

The Mineral Resource and Mineral Reserves disclosed in this report supersede all previous estimates for the Project. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to SRK at the time of preparing this CPR;
- Assumptions, conditions, and qualifications as set forth in this CPR; and
- Data, reports, and other information supplied by Amarillo and other third-party sources.

2.2 Qualifications and Site Visits

The following individuals, by virtue of their education, experience, and professional association, are considered Qualified Persons as defined in the NI 43-101 standard, for this report, and are members in good standing of appropriate professional institutions. The QPs are responsible for the specific sections as follows:

- Paulo Laymen, BEng Mining, Member of the Chilean Mining Commission (RM), SRK Principal Consultant (Mining), is the QP responsible for the following Sections: Executive Summary, 1, 2, 3, 15, 16, 18.1, 18.11, 18.12, 18.13, 19, 20, 22, 24, and portions of 21, 25, 26 and 27;
- Keith Whitehouse, MAusIMM CP (Geo), PCert JORC, AEFS Principal Consultant (Geology), is the QP responsible for the following Sections: 4, 5, 6, 7, 8, 9, 10, 11, 12, 14, 23, and portions of 25, 26 and 27;
- Stuart Smith, FAusIMM, Aurifex Principal Consultant (Metallurgy), is the QP responsible for Section 13, and portions of 25 and 26; and
- Tommaso Robert Raponi, Professional Engineer (Ontario), Ausenco Principal Consultant (Metallurgy), is the QP responsible for the following Sections: 17, 18.2 to 18.10, and portions of 21.1.3, 21.2.2, 25.4 and 26.4.

With respect to site visits completed to the Posse Gold Project, SRK notes that:

- Paulo Laymen, QP (Mineral Reserves), visited the Project site in September 2018;
- Keith Whitehouse, QP (Mineral Resources), conducted a site visit of the Project in July 2012; and
- Stuart Smith, QP (Metallurgy), and Tommaso Roberto Raponi, P.Eng. (Processing), have not visited the Project site.
- Stuart Smith, QP (Metallurgy), and Tommaso Roberto Raponi, P.Eng. (Processing), have not visited the Project site.

3 RELIANCE ON OTHER EXPERTS

In taking overall responsibility for this CPR, Mr Paulo Laymen, a Qualified Person within the meaning of NI 43-101 confirms that the recently authored PGP 2022 43-101 TR, from which this CPR is derived, was published in compliance with the methodology and format outlined in National Instrument 43-101, companion policy NI 43-101CP and Form 43-101F1. In so doing Mr Paulo Laymen has relied on the following experts in the writing of this report:

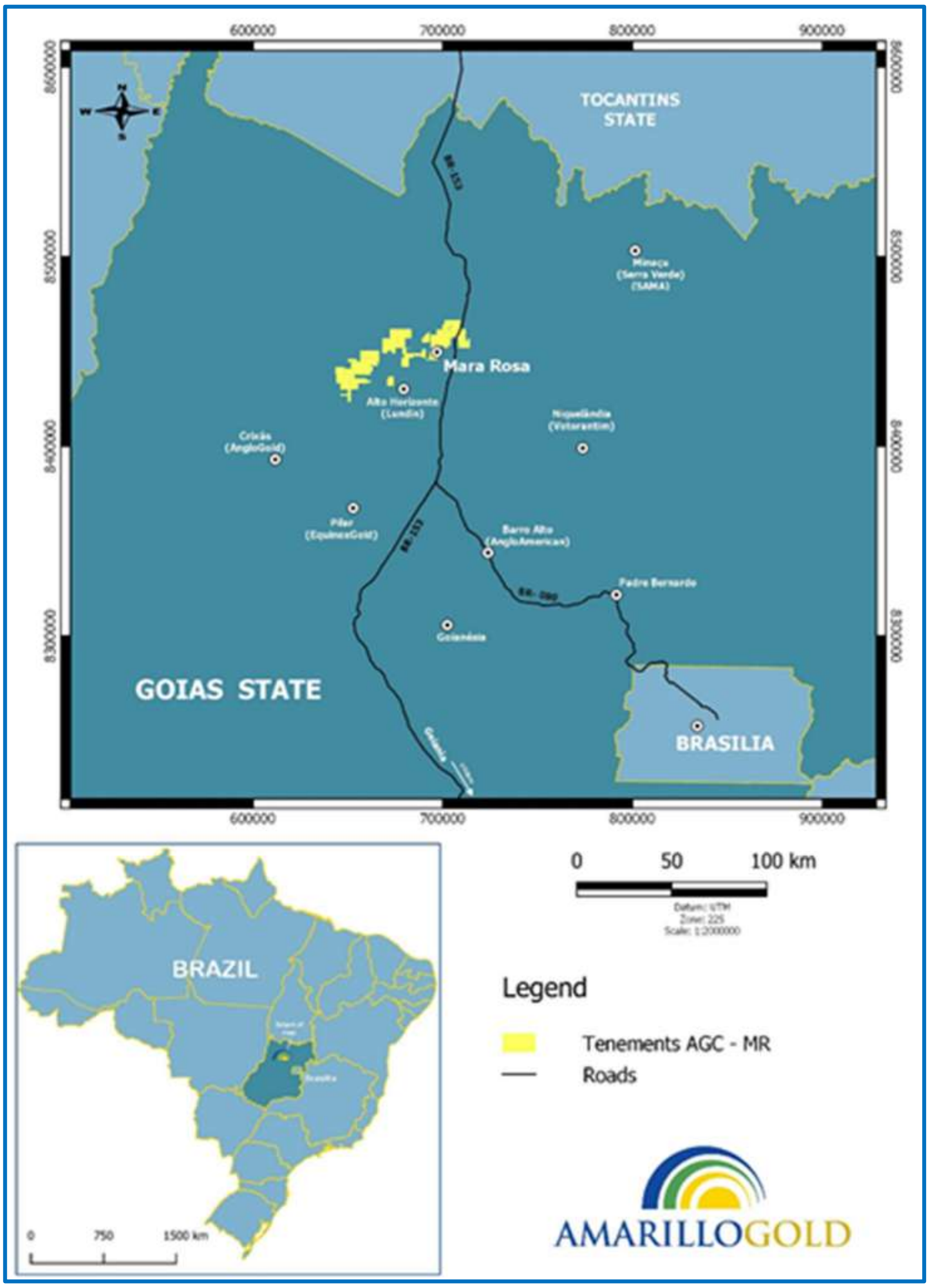
- Keith Whitehouse of Australian Exploration Field Services of Bendigo, Australia, for the Mineral Resource statement;
- John Watts of Australian Exploration Field Services of Perth, Australia, for a site visit in 2018 that informed the mineral resource statement;
- John Collier of Conarco Consulting of Bendigo, Australia, for geostatistical analysis used as part of the mineral resource statement;
- Stuart Smith of Aurifex, Australia, for the ore characterization;
- Tommaso Roberto Raponi, P.Eng. of Ausenco Engineering Canada Inc., for the mineral process and infrastructure;
- Nelson Siqueira Junio of DBO, Brazil, for the environmental and permitting studies;
- Paulo Paiva of GHT, Brazil, for the tailings pile, waste dumps, water dam and creek diversion;
- Cristina Simonetti of Ramboll, Brazil, for the mine closure; and
- Luiz Confúcio of SRK, Brazil, for the economic analysis.

4 PROPERTY DESCRIPTION AND LOCATION

The Mara Rosa Property (also generally known and referred to as the Posse Deposit, Posse Gold Project and the Project) is located in Goiás state, central Brazil, approximately 6km north of the town of Mara Rosa.

The Posse Deposit is centred at approximate Latitude 13° 58.395′ S, Longitude 49° 10.690′ W (Datum WGS84) or 696,900mE, 8454,500mN (Datum WGS84, Projection UTM, Zone 22 South), as shown in Figure 4-1. The Project encompasses a land area of 2,552 ha across three mining concessions plus numerous exploration leases in areas surrounding the Project mine area.

Figure 4-1: Location of Amarillo's Mara Rosa Properties



Western Mining Corporation ("**WMC**") operated a small open pit mine at the Project site during the 1990s. Two pits, Posse South and Posse North, were developed over a five-year period. The ore, along with feed from the nearby Zacarias mine, was processed on site. The processing, beginning with heap leach and later Carbon-in-Leach ("**CIL**"), was conducted on approximately 10ha of freehold property adjacent to the mining leases. Local infrastructure included adequate power and water to run a 600 tonnes per day CIL plant and heap leach operation.

As of November 2006, the mine and mill site had been reclaimed and no site infrastructure remained. According to Amarillo, the required remediation for mine closure had been met and accepted by the relevant government agencies. No significant environmental liabilities are known to exist at the former mine site.

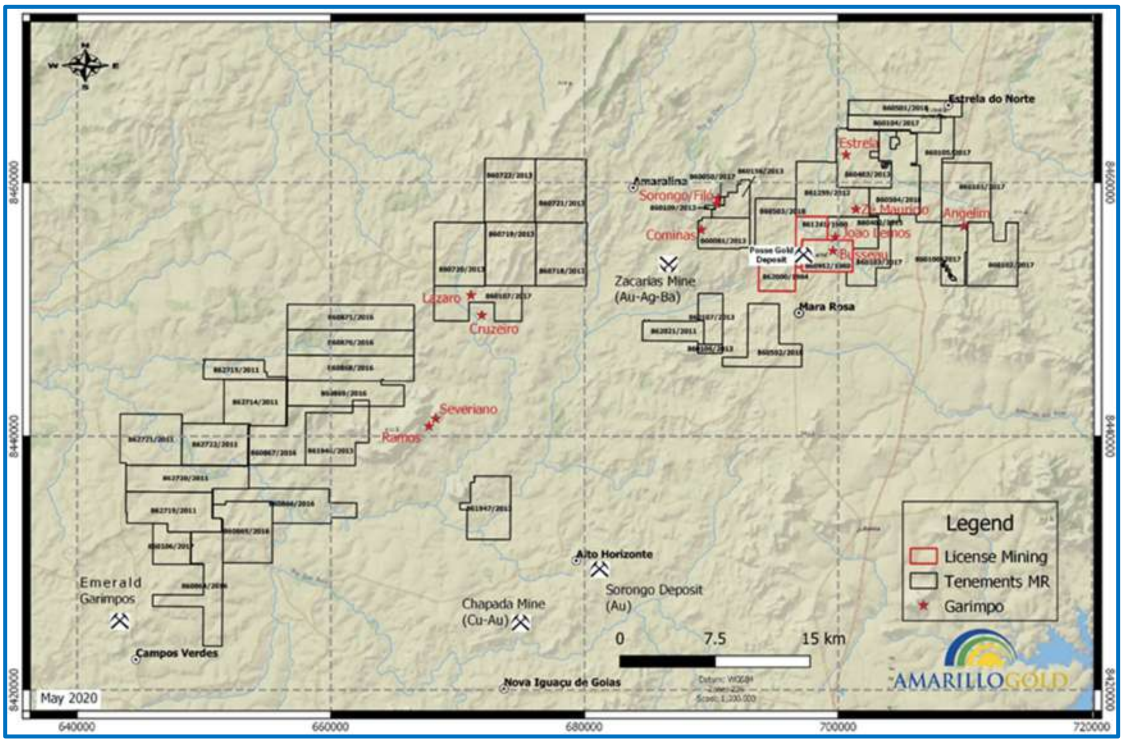
WMC maintained a core logging and storage facility, sample preparation laboratory, assay laboratory, and office complex immediately north of the town of Mara Rosa. The facilities, which occupy 8ha of freehold land, have been used by Amarillo during their exploration programs. As of October 2018, when Mr Watts visited the Project, the structures remain in excellent condition. The offices were used by Mr Watts during his site visit and Amarillo staff at site use the offices as their base. Amarillo also owns two houses on contiguous pieces of land on São Paolo Street in the town of Mara Rosa.

Table 4-1 shows a list of current concessions and tenements, owned by Amarillo, that make up the Posse Gold Project. Amarillo has stated to AEFS that all the concessions and tenements shown in the table are currently valid. A plan showing the location of individual tenements listed in Table 4-1 is shown in Figure 4-2.

Table 4-1: Concession and tenement schedule, December 2021

Item	ANM Number	Status	Tenement granted	Remarks	Location	Area (in Ha)
1	861.241/1980	M. Concession		Mining Suspension will expire September 23, 2023. The Mining Suspension extension requests were submitted by Amarillo to the ANM on October 28, December 10, and December 22, 2020.	Mara Rosa	566.62
2	860.952/1980	M. Concession			Mara Rosa	1,000.00
3	862.000/1984	M. Concession			Mara Rosa	986.00
4	862.021/2011	Tenement	Sep14 2015	Final Report must be filed by June 17 2023	South Zacarias	768.10
5	862.714/2011	Tenement	Sep 24 2015	Final Report must be filed by June 17 2023	C. Verdes	1,762.66
6	862.715/2011	Tenement	Sep 24 2015	Final Report must be filed by June 17 2023	C. Verdes	780.51
7	862.719/2011	Tenement	Sep 24 2015	Final Report must be filed by June 17 2023	C. Verdes	1,987.81
8	862.720/2011	Tenement	Sep 24 2015	Final Report must be filed by June 17 2023	C. Verdes	1,970.12
9	862.721/2011	Tenement	Sep 24 2015	Final Report must be filed by June 17 2023	C. Verdes	1,955.67
10	862.722/2011	Tenement	Sep 24 2015	Final Report must be filed by June 17 2023	C. Verdes	1,719.04
11	861.947/2013	Tenement	Sep 03 2015	Final Report must be filed by March 25 2023	Alto Horizonte	1,669.55
12	861.948/2013	Tenement	Sep 03 2015	Final Report must be filed by March 25 2023	C. Verdes	1,954.36
13	860.718/2013	Tenement	May 02 2016	Final Report must be filed by October 21 2023	Amaralina	1,999.75
14	860.719/2013	Tenement	May 02 2016	Final Report must be filed by October 21 2023	Amaralina	1,982.33
15	860.720/2013	Tenement	May 02 2016	Final Report must be filed by October 21 2023	Amaralina	1,999.88
16	860.721/2013	Tenement	May 02 2016	Final Report must be filed by October 21 2023	Amaralina	1,999.85
17	860.722/2013	Tenement	May 02 2016	Final Report must be filed by October 21 2023	Amaralina	1,999.86
18	860.864/2016	Tenement	Feb 17 2017	Final Report must be filed by September 12 2024	C. Verdes	1,971.78
19	860.865/2016	Tenement	Feb 17 2017	Final Report must be filed by September 12 2024	C. Verdes	1,968.46
20	860.866/2016	Tenement	Mar 20 2017	Final Report must be filed by September 12 2024	C. Verdes	1,987.49
21	860.867/2016	Tenement	Feb 17 2017	Final Report must be filed by September 12 2024	C. Verdes	1,801.58
22	860.868/2016	Tenement	Feb 17 2017	Final Report must be filed by September 12 2024	C. Verdes	1,996.97
23	860.869/2016	Tenement	Mar 20 2017	Final Report must be filed by September 12 2024	C. Verdes	1,723.62
24	860.870/2016	Tenement	Feb 17 2017	Final Report must be filed by September 12 2024	C. Verdes	2,000.00
25	860.871/2016	Tenement	Feb 17 2017	Final Report must be filed by September 12 2024	C. Verdes	2,000.00
26	860.100/2017	Tenement	Mar 02 2017	Final Report must be filed by September 12 2024	Mara Rosa	878.07
27	860.101/2017	Tenement	Apr 06 2017	Final Report must be filed by September 12 2024	Mara Rosa	1,880.02
28	860.102/2017	Tenement	Apr 06 2017	Final Report must be filed by September 12 2024	Mara Rosa	1,853.41
29	860.103/2017	Tenement	Mar 02 2017	Final Report must be filed by September 12 2024	Mara Rosa	1,074.53
30	860.104/2017	Tenement	May 22 2017	Final Report must be filed by September 12 2024	Mara Rosa	874.61
31	860.105/2017	Tenement	May 22 2017	Final Report must be filed by September 12 2024	Mara Rosa	1,632.50
32	860.106/2017	Tenement	Mar 02 2017	Final Report must be filed by September 12 2024	C. Verdes	839.01
33	860.107/2017	Tenement	Mar 02 2017	Final Report must be filed by September 12 2024	Amaralina	1,632.34
34	860.501/2018	Tenement	Aug 30 2018	Partial Report must be filed by January 11 2023	Mara Rosa	1,062.26
35	860.502/2018	Tenement	Aug 30 2018	Partial Report must be filed by January 11 2023	Mara Rosa	1,686.54
36	860.503/2018	Tenement	Aug 30 2019	Partial Report must be filed by January 11 2023	Mara Rosa	1,247.74
37	860.504/2018	Tenement	Aug 30 2019	Partial Report must be filed by March 19 2023	Mara Rosa	1,922.60
38	861.259/2012	Tenement	Dec 08 2015	Final Report must be filed by May 29 2023	Mara Rosa	1,964.11
39	860.081/2013	Tenement	Dec 08 2015	Final Report must be filed by June 01 2023	Mara Rosa	982.82
40	860.106/2013	Tenement	Dec 08 2015	Final Report must be filed by June 01 2023	Mara Rosa	135.07
41	860.107/2013	Tenement	Dec 08 2015	Final Report must be filed by June 01 2023	Mara Rosa	690.43
42	860.109/2013	Tenement	Dec 08 2015	Final Report must be filed by June 01 2023	Mara Rosa	21.71
43	860.156/2013	Tenement	Dec 08 2015	Final Report must be filed by June 01 2023	Mara Rosa	565.68
44	860.402/2013	Tenement	Dec 08 2015	Final Report must be filed by June 01 2023	Mara Rosa	865.94
45	860.403/2013	Tenement	Dec 08 2015	Final Report must be filed by June 01 2023	Mara Rosa	1,673.71
46	860.050/2017	Tenement	Mar 20 2017	Final Report must be filed by September 29 2024	Mara Rosa	40.28
47	860.706/2021	Tenement	July 27 2021	Partial Report must be filed by August 01 2024		
	Total in Ha.					**6,6074.9**
All properties are held by Amarillo Mineração do Brasil Ltda.						

Figure 4-2: Amarillo Tenements, December 2021



5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

Details of the of surface rights, availability of power, water, labour and both waste disposal and process plant locations are discussed in more detail in separate relevant sections of this report. Suffice it to note that the Project is not seriously constrained by space or other factors necessary for mining activities.

5.1 Accessibility

The Municipality of Mara Rosa, see Figure 5-1, is located 356km north of Goiânia in the Porangatu Microregion, 11km west of the Belém-Brasília highway, between the basins of the Araguaia River and the Tocantins River. According to a 2019 estimate, Mara Rosa has a population of approximately 16,300 people of whom 9,200 live in the town (Instituto Brasileiro de Geografia e Estatistica (web)., 2019).

5.2 Climate

Average annual rainfall is approximately 1,500mm, resulting in a relatively wet climate. The year is defined by two principal seasons, a dry season from April to September and a wet season from October to March. The mean temperature is 24°C during the dry season and 28°C during the wet season. Annual temperatures typically range from approximately 4°C to 45°C. The climate does not impose any limitations on exploration or potential mining operations, which can continue throughout the year.

5.3 Local Resources

Local facilities include several public and private elementary and high schools, two hospitals, a public health centre, three banks, three gas stations, several small motels and numerous shops. Agriculture (saffron, corn, rice, manioc, sugarcane, soybeans, and bananas) and cattle ranching are the primary commercial activities in the region. Mara Rosa is a regional support community for these activities.

5.4 Infrastructure

The municipality has an excellent network of local farm roads, the majority of which are unpaved but generally in good condition. The municipality is also serviced by an 800 metre-long, unpaved airstrip. Access to Mara Rosa is via Federal Highway BR-153, the main north-south highway in central Brazil leading to the city of Belém at the mouth of the Tocantins River. Mara Rosa is 356km, or 4 hours driving time, north from the state capital of Goiânia, and 320km, or 4 hours driving time from the national capital, Brasilia. Highway access, see Figure 5 1, to Goiânia is via GO-080 / Nerópolis / São Francisco de Goiás / BR-153 / Jaraguá / GO-080 / Goianésia / Barro Alto / GO-342 / BR-153 / Uruaçu / Campinorte / GO-239.

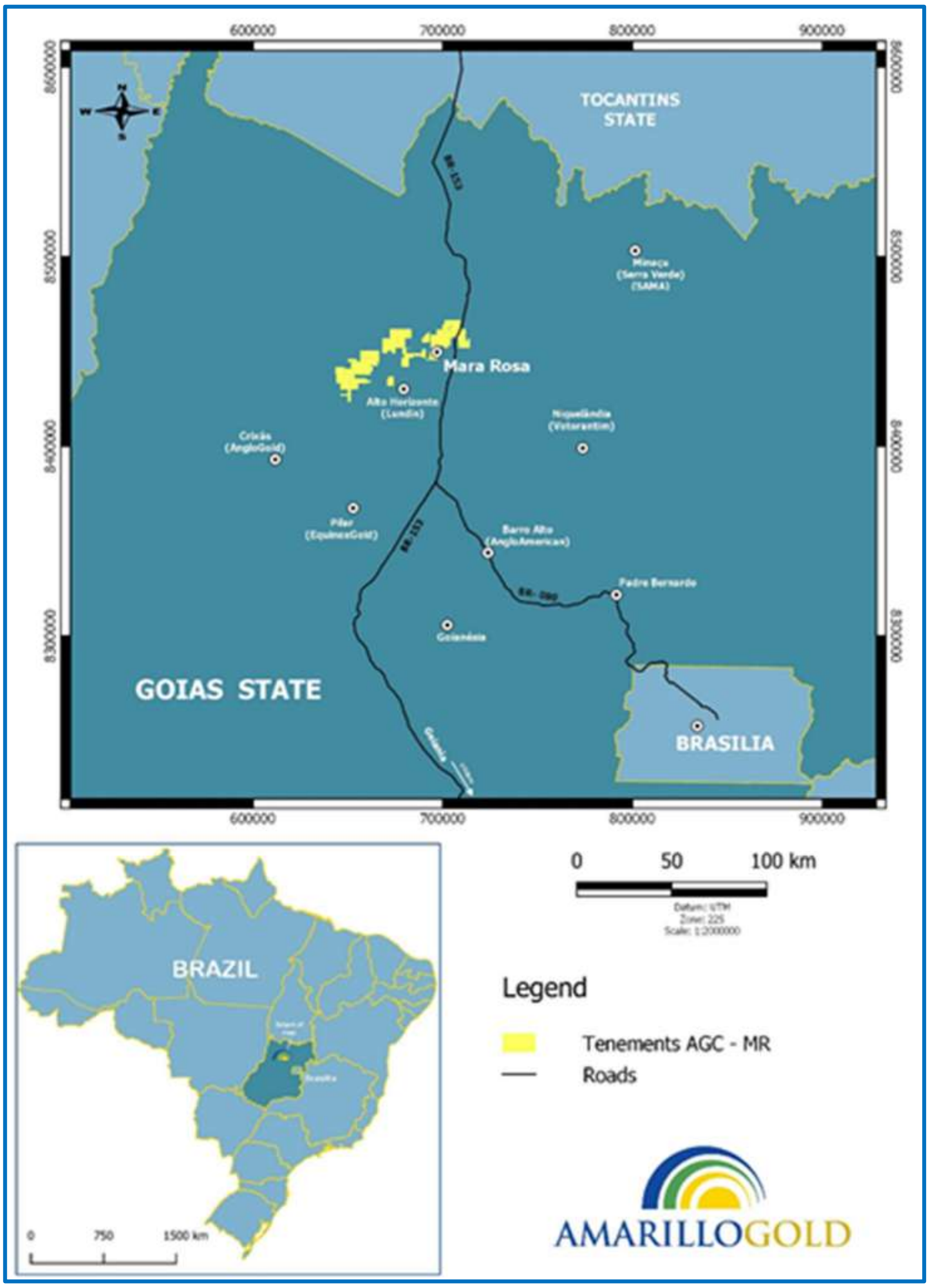
Electric power is supplied by CELG, the State of Goiás Energy Authority. The local electricity grid has an installed capacity of 14MW supplied to the area via a 69kV line. Should the proposed mine be developed, a new transmission line of 67km at 138kV will be installed to supply the mine. The water supply is metered and is provided by SANEAGO, the state water company. Water for the Posse Gold Project as well as ranches in the surrounding region is derived from a combination of local streams and artesian wells. Telephone service, both local and international, is provided by TELEGOIAS. Cellular telephone service is available in the area.

5.5 Physiography

The region is characterized by tropical savannah of low to moderate topographic relief ranging from approximately 400m to 500m above sea level ("**ASL**"). The town itself has a mean elevation of 520m ASL. Much of the area has been cleared for farming and as a result is open savannah grassland.

Trees occur along the abundant water courses.

Figure 5-1: Mara Rosa and surrounding towns



6 HISTORY

This section summarises the work carried out prior to the release of this CPR.

Evidence of small scale surficial-alluvial mining along the Rio do Ouro in the historic Amaro Leite area indicates mining activity in the Mara Rosa District dates to the mid-1700s. More recent activity dating from the early 1970s to early 1980s began with the successful discoveries by INCO (now Vale S.A. or "Vale") of the Chapada gold-copper and Crixás gold deposits. These deposits are located approximately 30km and 100km to the south-west of the town of Mara Rosa, respectively.

During the early 1980s, BHP-Utah Mines (now BHP Limited), through its subsidiary Mineração Colorado Ltda., initiated a grass roots reconnaissance program that covered the Chapada district and the Mara Rosa area, and eventually led to the discovery of the Posse gold and

Zacarias gold-silver-barite deposits. From 1981 to 1987, BHP completed 12,300m of diamond and reverse circulation ("**RC**") drilling at Posse and Zacarias. At Posse, a 107m exploration shaft was sunk and 400m of lateral drifting was completed to test mineralization.

As a result of Brazilian restrictions on foreign ownership in 1988 BHP chose to joint venture the Mara Rosa properties with Western Mining Corp ("**WMC**"). In 1990, WMC set up a subsidiary, Mineração Jenipapo S.A. ("**MJSA**"), to acquire a 100% interest in Posse, and to explore, develop, and operate the asset. The Posse mine was opened in 1992 and operated until July 1995, during which time two pits, Posse North and Posse South, were developed. The on-site mill processed approximately 750,000 tonnes of ore grading a combined 3.5g/tAu. Zacarias, which was significantly higher grade, operated at roughly the same time as Posse and was processed through the same mill.

In order to provide cash flow for its activities in Brazil, WMC focussed much of its attention on development of the Posse and Zacarias mines between 1990 and 1995. This work is understood to have been completed as a result of a corporate decision to make each business unit self-funding and to encourage efforts to develop known deposits. In addition, efforts to replace mined reserves were directed toward both the Eastern and Central Belt exploration targets generated previously by BHP as well as new targets identified to the east and north of Mara Rosa.

By June 1995, a combination of factors, including low gold prices, the exhaustion of reserves at the higher grade Zacarias deposit, and the failure to discover any additional, near-surface reserves, caused WMC to discontinue mining and exploration activities at Mara Rosa. As the primary exploration objective had been the discovery of near-surface mineralization that could be fast tracked into production, most of the exploration targets identified by BHP and WMC had only been evaluated to depths within, approximately, 50m from surface.

Upon suspension of its mining and exploration activities, WMC was approached by several companies interested in exploring the property under lease-option agreements. The Zacarias deposit and the rights to its tailings were eventually sold to Minere Mineração Ltda. ("**Minere**"), a small Brazilian company interested in exploiting the deposit's very high barite content. The Project has since been on-sold to a company called Baribras Mineração Ltda.

In 1996, Barrick do Brasil ("**Barrick**") completed a full due diligence study of the remaining Posse Gold Project concessions (the Eastern Belt claims). The due diligence involved a team of at least 14 people and a significant program of test sampling, re-logging of core, soil sampling, reinterpretation of geophysics, and an estimate of the mineral resource for the Posse Deposit. Although this program subsequently led to a preliminary offer by Barrick to purchase the property in full, negotiations stalled prior to execution of the agreement. Barrick provided WMC with a copy of its due diligence report and related correspondence after the failure to execute a deal.

Following Barrick's withdrawal, Metallica Brasil Ltda. ("**MBL**") entered into negotiations with WMC for the purchase of the Eastern Belt properties, and in November 1997, successfully completed an agreement that called for a total purchase price of US$1.5m. As part of the previous buy-out agreement between BHP and WMC, BHP held a 1% NSR royalty interest on the property. This now sits with Royal Gold Inc. ("**RG**") after a royalty portfolio sale by BHP. Euro-Nevada Gold Corporation (later absorbed into Newmont) held an additional 1% NSR royalty. This now sits with Franco Nevada Corporation, after this royalty focused corporation folded out of Newmont.

Following a compilation of data and a review of the Project, MBL completed a systematic soil geochemistry and geological mapping program north-east of the Posse Deposit. Induced polarisation ("**IP**") and ground magnetic geophysical surveys were completed over some of the more promising areas. MBL suspended exploration operations in September 1998 and placed the Project on care and maintenance. In 2001, MBL revisited the Project and completed a review of the regional potential. At this time, 5 holes, totalling 940m, were drilled into three separate targets on the northern extensions to the Posse mine trend. Following this work, a corporate decision was made to focus on properties in Mexico and Chile and MBL decided to sell the Project.

Amarillo visited the Project in August 2003 and in October 2003 signed a letter of intent with MBL to purchase MBL and 100% of the Posse Gold Project. In June 2018, Amarillo entered into an agreement for the sale of an additional 1.75% NSR on the Posse Gold Project to RG Royalties, LLC, a wholly-owned subsidiary of Royal Gold. The Project thus remains subject to the 1.0% NSR royalty to Franco Nevada Corporation and a further 2.75% royalty to Royal Gold. Since gaining control of the Property, Amarillo has done considerable work to define the extent and nature of the Posse Deposit with the aim of developing the primary or fresh (non-oxidised) mineralisation.

7 GEOLOGICAL SETTING AND MINERALISATION

Information is this section is largely derived from an unpublished report by Micon International Limited for Amarillo dated 2003, (Micon International Limited, 2003). The Mara Rosa District is situated within the Goiás Magmatic Arc ("**GMA**") which forms part of the Tocantins physiographic province, an intercratonic mobile belt that separates the Amazonas and São Francisco cratons, located to the northwest and southeast respectively. The GMA is a 100km wide, northeast-trending granite-greenstone terrane that extends for approximately 700km. The geology in the Mara Rosa District is principally delineated by three northeast striking, moderately to steeply northwest dipping belts of metamorphosed volcano-sedimentary and associated intrusive rocks. These belts, referred to as the Western, Central, and Eastern Belts, are separated by broad zones of tonalitic orthogneiss.

The Eastern Belt is bounded to the southeast by the Rio dos Bois fault, which also defines the southeastern limit of the GMA.

Amarillo's land position within the Mara Rosa District primarily covers the Eastern Belt greenstone assemblage together with some coverage of the Western and Central belts as well. The Eastern Belt, has a maximum thickness of 6km, generally strikes to the northeast and dips moderately to steeply to the northwest. Surface topography is characterised by moderate relief and locally dissected drainages that follow lithologic or structural weaknesses. Depth to fresh bedrock is generally shallow, ranging from 0m to15m. The upper portion of the weathered profile consists of clay-rich latosol and saprolite derived from the underlying bedrock.

Rocks of the Eastern Belt are locally intruded by quartz-feldspar-muscovite and biotite granitic rocks and associated aplite and pegmatite dykes, small stocks and dykes of hornblende, biotite and magnetite diorite, and, in its north-central portion, a large body of hornblende-plagioclase gabbro. All units exhibit varying degrees of foliation that typically range from weak to moderate, and generally intensify along sheared contacts. The tonalitic orthogneiss that separates the Eastern and Central Belts is composed of coarse-grained plagioclase, hornblende, and biotite with localised patches of biotite schist near its contact with the Eastern Belt.

Structurally the Eastern Belt is dominated by well-developed, penetrative foliation that strikes 30° to 50° and dips 40° to 70° north-west – an orientation subparallel to stratigraphy. Major structural systems include 50° to 65° striking shears and thrusts and associated drag folds. Shears are most commonly developed along zones of elastic disparity such as lithologic contacts. Shear sense is typically reverse-dextral oblique although a sinistral sense is locally observed. A second set of structures consist of late cross cutting north-west to east-northeast striking brittle faults and fractures that locally offset stratigraphy in apparent dextral strike-slip sense.

Uranium-lead isotopic age determination of zircons from some of the principal lithologic units within the district indicates timing of initial rock formation for both the belt rocks and the tonalite gneiss to be between approximately 870Ma to 850Ma (Viana, 1995). Subsequent amphibolite facies metamorphism is estimated to have occurred between 700Ma to 600Ma based on U-Pb and Rb-Sr dating of recrystallised titanate. The latter date corresponds to peak metamorphism related to the Brasiliano orogenic event.

Several significant mineral deposits occur within 50km of Mara Rosa town including the Posse gold deposit, the Zacarias gold-silver deposit and the Chapada copper-gold deposit, together with numerous historic prospects and small-scale historic mines known locally as garimpos.

7.1 Local Geology

The Posse Deposit occurs in a regional thrust that probably acted as one of the primary dewatering conduits during the Neo-Proterozoic Brasiliano orogeny. The geophysical, geological and geochemical data available demonstrate that the Posse Deposit occurs within a 50km long shear zone with potassium alteration and lower order gold-copper-molybdenum mineralization. The Posse Deposit has a metamorphosed granodiorite traditionally called a grey gneiss or “Biotite gneiss” in the hanging wall of the fault and amphibolite, “greenstone” in the footwall. Shearing and hydrothermal alteration, of the meta granodiorite has resulted in the formation of mylonitic zones that form a distinct lithologic unit, a quartz-feldspar-mica schist, known as the Posse Schist, that is characteristic of the Posse ore zone. This unit has been identified in several other areas including the Posse footwall and on strike extensions of the Posse Ore Zone to the northeast. Shearing is most intense in the footwall. It is speculated that the rheological contrast between the hanging wall and footwall rock types captured the regional thrust (movement west to east) for a 2km segment of the shear. It is also possible that the chemical contrast between the hanging wall and footwall rocks may have aided in focusing mineralizing fluids. Observations from drill core suggest that an earlier potassic event with quartz veining, chalcopyrite, molybdenum, biotite and K-feldspar was followed by a later phyllic (sericite) event with pyrite, iron-telluride, and gold. Gold occurs as native gold and also with telluride and pyrite.

In general, mineralization at Posse is developed along a 050° to 065° striking fault zone. Mineralisation tends to be strongest within mylonitic zones that follow more northerly striking (approximately 030° to 050°) shear strands and dilatant jogs that obliquely transect the contact between the hanging wall and footwall rocks.

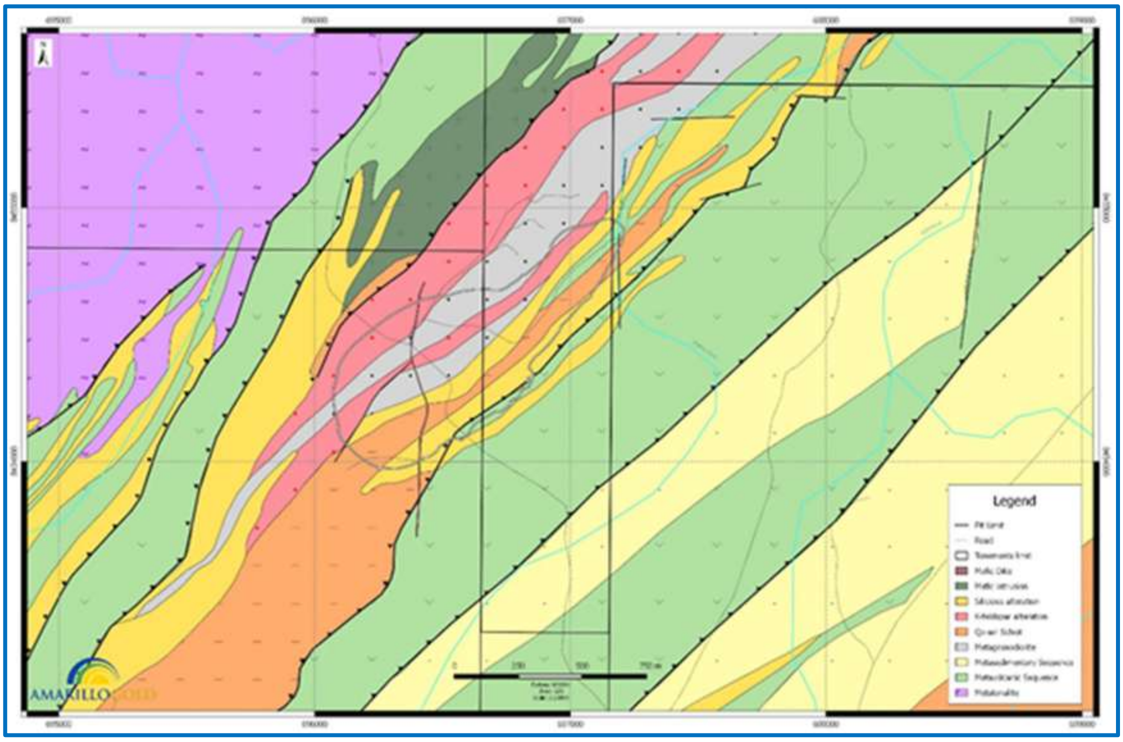
The mineralisation envelope at Posse is about 30m thick and over 1km long, Figure 7-1. It has mylonitic appearance that is most noticeable adjacent to the footwall where shearing is the most intense. Higher intensity of shearing is associated with increased sulphide mineralization (up to about 4%), and a slight increase in metamorphic grade from greenschist to high greenschist facies in the hanging wall through to high greenschist/low amphibolite facies in the footwall (biotite flakes and garnet alteration). Higher gold values are associated with increasing intensity of shearing and higher levels of silicification and sulphide mineralization.

Aside from the slight increase in metamorphic grade, there appears, based on Inductively Coupled Plasma (“**ICP**”) analyses obtained from the 2005/2006 drilling program, to be a chemical difference in lithology between the hanging wall and footwall. However, this is not visually obvious.

The shear zone may be more complicated than a simple main shear near the footwall with gradually decreasing intensity towards the hanging wall. Based on geochemical evidence there is some reason to believe that different portions of the shear zone were active at different times.

A thin basaltic dyke that does not offset the mineralization has been intersected in some drill holes.

Figure 7-1: Geology of the Posse Deposit



8 DEPOSIT TYPES

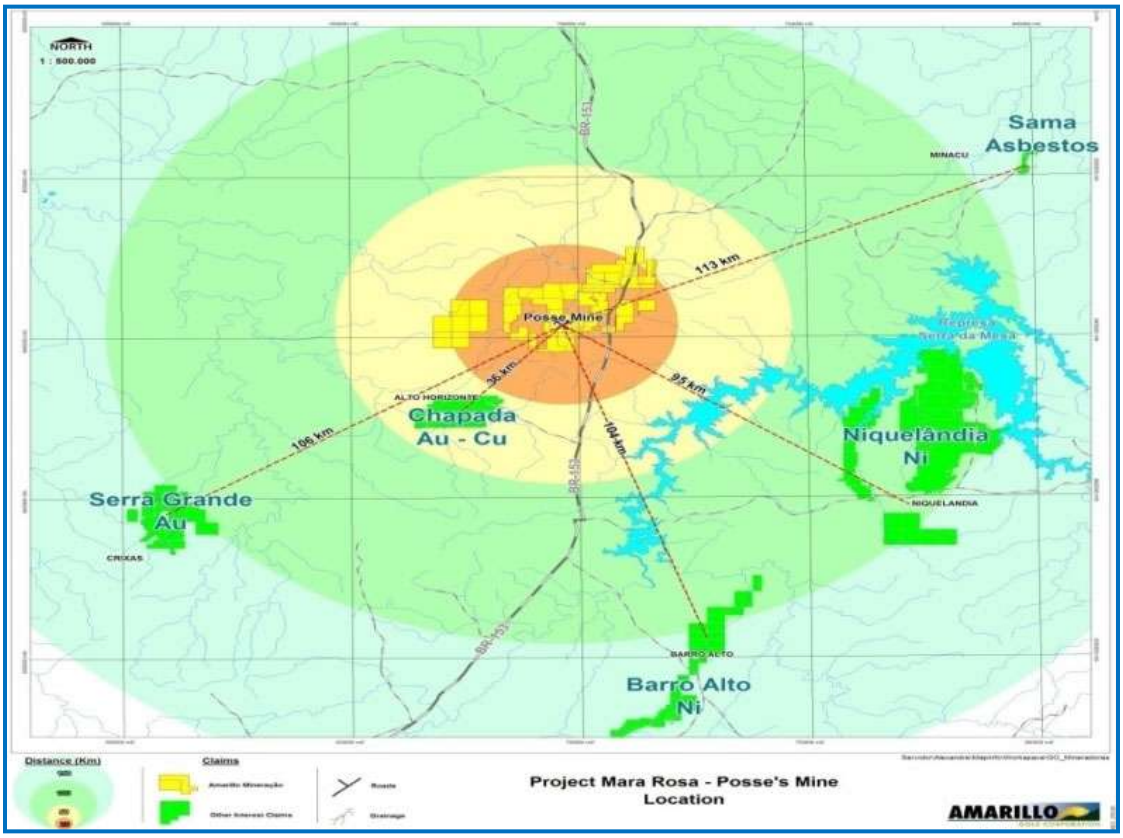
As discussed above, several significant mineral deposits occur in the Mara Rosa region. These include the Posse gold deposit, the former Zacarias gold-silver-barite deposit, and the Chapada copper-gold deposit, in addition to numerous historic prospects and garimpos, Table 8-1.

Table 8-1: Significant Deposits of the Mara Rosa Region

Deposit	Deposit Class	References
Posse Au (Eastern Belt)	Shear-hosted mesothermal lode-gold.	MBL data (Mara Rosa files) and Amarillo website
Zacarias Au-Ag-Ba (Central Belt)	Stratiform syngenetic exhalative or shear related epigenetic high sulphidation?	WMC data (MR files); Poll, 1994. R. Shaw/M. Petersen
Chapada Cu-Au (Eastern Belt)	Volcanogenic exhalative? Wall rock porphyry copper system?	Kuyumjian, 1991. Richardson, et. al., 1986; 1988.

Other current mining projects within 120km of Mara Rosa are shown in Figure 8-1.

Figure 8-1: Current Mining Projects in Mara Rosa Region



The Posse Deposit is hosted in a regional thrust that probably acted as one of the primary dewatering conduits during the Neo-Proterozoic Brasiliano orogeny. The available geophysical, geological and geochemical data demonstrate that the Posse Deposit occurs within a 50km long structural zone with potassium alteration and lower order gold-copper-molybdenum mineralization. The Posse Deposit has a hanging wall of grey gneiss and the foot wall of amphibolites, "greenstone", and it is speculated that the rheological contrast between the two rock types captured the regional thrust (movement West to East) for a 2km segment. It is also possible that the chemical contrast between the acid hanging wall and basic foot wall may have aided in focusing the mineralizing fluids. Observations in the core suggest that an earlier potassic event with chalcopyrite, molybdenum, quartz veining, biotite and K-feldspar was followed by a later auriferous phyllic event with gold occurring as both free grains and associated with the telluride and pyrite.

9 EXPLORATION

During the 1990's WMC operated a small open pit mine at the Project site. Two pits, Posse South and Posse North, were developed over a 5-year period and oxide ore was processed on-site. The mine and mill site were reclaimed, and no site infrastructure remained by November 2006. No significant environmental liabilities are known to exist at the former mine site, and it is understood that the required remediation for mine closure had been met and accepted by the appropriate government agencies.

Numerous drilling campaigns have been completed on the property:

- BHP Billiton: 1982 – 1987;
- WMC: 1988 – 1995;
- Amarillo: 2005 – 2006;

- Amarillo: 2008;
- Amarillo: 2010 – 2011;
- Amarillo: 2011 – 2012;
- Amarillo: 2018 – 2019; and
- Amarillo: 2021.

The complete drillhole data base contains 423 drill holes totalling 64,749m of drilling. Fourteen of these holes were drilled on targets outside the Posse mineralization.

During the period from late 2012 until June 2018 no drilling was carried out or samples submitted for assay. Amarillo completed a 63-hole drilling program at Posse in February of 2019. The program consisted of 49 diamond drillholes, 18P047 – 18P087 and 19P088 – 19P095, with a total length of 15,195m and 14 reverse circulation ("**RC**") drillholes, 18PRC001 – 18PRC014, for a total length of 1,295m.

A further program of 10 diamond drillholes, 21P112 – 21P121 was completed in 2021 as part of a program of work to stress test and de risk the current (2020 Resource). Results of these drill programs are discussed under Section 10, Drilling.

All exploration prior to June 2018 is covered in the 2016 Resource report and earlier reports referred to in Section 2.

9.1 Topography

9.1.1 Lidar

In 2019 as part of renewed work at the Posse site Amarillo conducted a detailed LIDAR survey of the Posse mine and surrounding area. This survey provided a detailed model of the surface topography together with a detailed orthophoto. LIDAR survey is particularly useful where vegetation obscures the actual ground surface as the lasers used penetrate even very dense vegetation to provide a ground return. There is also some penetration of water allowing the collection of some information from the flooded portions of the historic pits. The survey was conducted by BASE Aerofotogrametria e Projetos SA, based in Sao Paulo, Brazil (baseaerofoto.com.br) using an aircraft mounted LIDAR Riegl sensor Model LMS Q560. Observations were processed by BASE using RiPROCCESS software to rectify and stitch together the LIDAR swaths and associated orthophoto imagery and to match the dataset to a network of ground truth points.

As part of the data validation work carried out by AEFS the digital terrain model and associated orthophoto imagery produced from the survey was tested by comparing coordinates of points in the survey which could be matched to points on Google Earth. There was a very close match and AEFS are satisfied that this data can be used as an accurate survey base for future work.

An accurate topography and associated imagery allowed historical plans of the pits and stockpiles to be cross referenced to the Lidar survey in order to determine areas of backfill in the old pits together with location, extent and volume of stockpiles and dumps. This work resolved a suspected mismatch in the location of the pits as shown on the end of mining survey from WMC and the actual location of the pits. A mismatch in data had been suspected from 2018 when the 2015 topographic surface used over the period 2015 – 2017 was amended to discount backfill in the pits as part of the 2018 resource update. At that time, it was not possible to confirm or resolve the data mismatch. However, the orthophoto imagery provided as part of the 2019 LIDAR survey allowed accurate location of mine infrastructure. This in turn allowed the historic Posse mining survey in the form of a scanned plan of the mine and infrastructure to be compared with the orthophoto. Using values on the plan reference grid (in UTM coordinates) to geolocate the plan revealed a mismatch with the position of both the pit and infrastructure in the orthophoto. The locations shown on the orthophoto were known to be a good match with the coordinates of the same points in Google Earth. Cross matching the same points in the post mining plan indicated that the coordinates of UTM grid on the plan were incorrect, see Figure 9-1. A correction of -25m in East and +20m in North accurately matched the infrastructure on post mining plan to the orthophoto. With an accurately located mine plan it was a relatively straightforward process to determine the volume of backfill in the pits.

Figure 9-1: Current Mining Projects in Mara Rosa Region(1)

(1) Accurate registration of the Posse Mine Plan over the Orthophoto required the plan to be shifted by -25m in East and +20m in North.

In addition to determining the amount of backfill in the pits the location of stockpiles and dumps shown on the mine plan allowed the reconstruction of the pre-mining surface. Previously the pre-mining surface has been constructed from 5m contours digitised from government topographic mapping compiled in the mid 1980's. The surface reconstructed from the LIDAR survey after removal of the pits, dumps and stockpiles showed a reasonable fit to the historic digitised surface in areas where there was no ground disturbance and gave confidence that the reconstructed surface was reasonably accurate. A volumetric assessment was made of the stockpiles shown on the mining infrastructure plan as shown in Figure 9 2 with the volumes tabulated in Table 9 1

Figure 9-2: Stockpile and dump locations

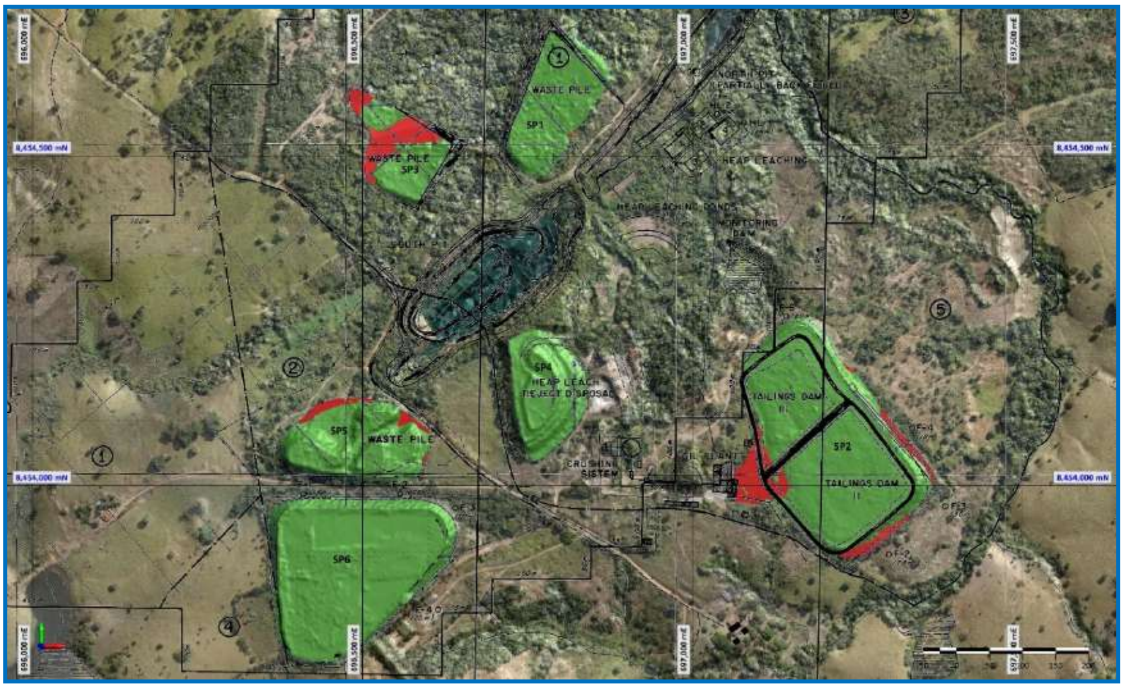


Table 9-1: Historic stockpile volumes(1)

Stockpile	Volume		Thickness	
	Fill	Cut	Fill	Cut
SP1	86,000	-2	4.2	0.0
SP2	207,000	-5,000	3.5	0.6
SP3	24,000	-8,000	2.2	1.4
SP4	181,000	0	10.6	0.0
SP5	128,000	-1,000	6.0	0.7
SP6	382,000	0	7.0	0.0
Total	**1,006,000**	**-14,000**	**5.5**	**0.9**

(1) Volumes have been rounded to the nearest 1,000 and thicknesses to 1 decimal place.

9.1.2 Bathymetry

The LIDAR surface was only able to provide limited information on ground surface in the areas of the old pits which were flooded. To resolve this Amarillo staff conducted a bathymetric survey over the old pits. The survey used a lead line to take soundings of the pits with the location of sounding points being recorded by a handheld GPS. A total of 595 soundings were made, 119 in the North Pit and 476 in the South Pit. The soundings were converted to elevations by subtracting the observed depth at each sounding location from the elevation of the water surface in the pits. The elevation of the surface of the water in the pits was estimated from the LIDAR survey and was able to be established within a few centimetres. The soundings were them merged into the LIDAR topography to produce a combined topography and bathymetry. The resulting surface was compared with the historical final pit shape and found to be in close agreement. Some areas of minor slumping of the historic profile and consequent backfilling of the base of the pit were observed but the volumes of material concerned were small.

9.1.3 Drillhole elevations

With a detailed topographic surface, it was then possible to update the elevations of drillhole collars. Prior to 2018 drillhole elevations had been by reference to various surfaces representing the best elevation available at the time various geological interpretations were made. The drillhole database contained holes ranging in age from very recent drilling through to holes drilled in the mid 1980's. As a result, holes drilled prior to development of the open pits in 1992 were referenced to the reconstructed original topography. In areas which had not been disturbed by mining activities this matched the current LIDAR surface. Holes drilled as mining progressed were referenced to the surface that existed when they were drilled, for those holes in undisturbed areas, this was the current LIDAR Surface. For holes drilled in areas that had been borrow areas for subsequent pit backfill, this was the reconstructed original surface. For holes drilled inside the pit boundaries the surface was the base of the pit when they were drilled. In many cases this matched the pit shape before backfilling. There were also holes that were drilled from intermediate levels. For these the elevation was established as the height that gave the best fit to the mineralised zones as observed in adjacent holes whose elevation was known. Holes drilled post mining could be referenced to the current LIDAR surface.

When adjustments were made to holes, reference was also made to an old copy of the drillhole database which dated from 2006. In general, this database had a very good fit to the adjusted elevations of holes drilled prior to 2006. Unless there was good reason, if the difference between adjusted elevation and the 2006 elevation was less than 1m, then the 2006 elevation was used for holes in the pit area.

The nature of adjustments made is summarised in Table 9-2.

Table 9-2: Adjustments to drillhole elevation by period

Period	Number of holes	Reference surface
1985 – 1992	153	Reconstructed original topography *
1992 – 1996	20	Match to original topography
1992 – 1996	23	In pit matched to intermediate level
1992 – 1996	5	Match to mined topography
1996 – 2019	211	Current LIDAR surface
2021	10	Current LIDAR surface

Figure 9-3: Location of drillholes in the 2018-2019 drill program

Figure 9-4: Location of drillholes in the 2021 drill program

(1) Accurate registration of the Posse Mine Plan over the Orthophoto required the plan to be shifted by -25m in East and +20m in North.

10 DRILLING

A large number of drilling programs have taken place at Posse since the early 1980's when BHP originally commenced work on the property. Following completion of the initial stage of mining in 1996 drilling of the Posse mineralization restarted in 2003, when the Project was acquired by Amarillo. During the time that Amarillo has held the Project there was an extended hiatus in drilling from early 2012 until June 2018. Starting in June 2018 a major drilling campaign that ran until February 2019 (Drilling 2018 - 2019) was conducted. Following that work, further exploration drilling started at Posse in November 2019 all holes were outside the current Resource envelope and this work did not impact on the mineral resource estimate discussed in this report. In 2021 as the result of stress testing and de risking work undertaken on the project and the 2020 mineral resource, a further 10 diamond holes were designed to test areas of the resource which were not well tested by existing diamond drilling. Results from the 2018 - 2019 drill campaign, together with relevant results from earlier drilling programs provided a basis for a new estimation of the mineral resources at Posse, the 2020 resource, as discussed elsewhere in this report. The results of the 2021 drilling have been reviewed against the 2020 model and it is the view of the Qualified Person that a revised model incorporating this data would not materially affect the 2020 resource. The stress testing and de risking as it applied to the 2020 resource are discussed under Section 12.7.

10.1 Drilling prior to 2018

Drill campaigns to the end of 2009 were discussed in the 2010 Resource Report (HCS & AEFS, 2010) while those to the end of 2012 were discussed in the 2011 (HCS & AEFS, 2011) and 2016 (AEFS, 2016) Resource Reports. Drilling prior to 2018 is summarised in Table 10-1 below.

Table 10-1: Drill Programs to end 2012 (1)

Company	Program	Type	Start Date	End Date	Start Hole	Finish Hole	Number	Metres
BHP	1983_F	Diamond	24/10/1983	10/12/1983	F001	F005	5	5,551
BHP	1984_F	Diamond	25/05/1984	26/09/1984	F006	F018	13	1,441
BHP	1984_W	Percussion	15/07/1984	21/07/1984	W001	W004	4	320
BHP	1985_W	Percussion	18/01/1985	9/11/1985	W005	W032	27	2,085
BHP	1985_FW	Diamond	11/02/1985	15/12/1985	FW019	FW059	41	8,358
BHP	1987_FS	UG	4/04/1987	4/09/1987	FS001	FS010	10	314
BHP	1987_W	Percussion	23/11/1987	24/11/1987	W034	W036	3	160
WMC	1988_MRD	Diamond	17/10/1988	15/12/1989	MRD001	MRD073	40	2,570
WMC	1988_MRC	RC	2/01/1989	25/01/1989	MRC035	MRC038	5	317
WMC	1990_MRC	RC	8/03/1990	9/03/1990	MRC094	MRC095	2	179
WMC	1991_MRC	RC	13/05/1991	18/05/1991	MRC125	MRC127	3	285
WMC	1992_MRC	RC	6/02/1992	29/06/1992	MRC175	MRC191	8	302
WMC	1993_MRC	RC	12/05/1993	9/11/1993	MRC200	MRC235	35	967
WMC	1993_MRD	Diamond	27/09/1993	30/09/1993	MRD196	MRD197	2	130
WMC	1994_MRD	Diamond	16/08/1994	1/10/1994	MRD199	MRD346	4	358
Amarillo	2006_SPETI	Diamond	25/11/2005	27/09/2006	SPETI01	SPETI28	28	3,510
Amarillo	2008_FMR	Diamond	26/03/2008	9/07/2008	FMR0001	FMR0009	9	1,235
Amarillo	2008_W	Diamond	15/05/2008	21/05/2008	W002A	W002A	1	136
Amarillo	2008_MRP	Diamond	21/05/2008	16/10/2008	MRP0001	MRP0014	14	3,293
Amarillo	2010_MRP	Diamond	13/10/2010	25/03/2011	MRP0015	MRP0045	33	8,524
Amarillo	2011_MRPA	Diamond	17/06/2011	14/12/2011	2011MRP0001	2011MRP0013	13	2,591
Amarillo	2012_MRPA	Diamond	7/01/2012	4/07/2012	2012MRP0001	2012MRP0034	34	5,029
Amarillo	2012_P	Diamond	6/07/2012	17/12/2012	12P035	12P046	12	2,080

(1) Total holes 346, total metres 49,735. Hole numbers in some of the WMC programs were not consecutive due to hole numbers being shared with the nearby Zacharas project. The FMR series holes and Holes SPETI 24 – 26 together with 2012MRP024, 2012MRP026, 2012MRP027, 2012MRP029 and 2012MRP031, did not target the Posse mineralisation.

10.2 Drilling 2018 – 2019

As noted elsewhere there was no drilling conducted from late 2012 until June 2018. A new drill campaign summarised in Table 10 2 was started in June 2018 and ran until February 2019. Data from this drilling program was combined with data from earlier campaigns and used for the estimation of the current Posse resource detailed in this report.

Table 10-2: 2018 - 2019 Drilling Program(1)

Company	Program	Type	Start Date	End Date	Start Hole	Finish Hole	Number	Metres
Amarillo	2018_P	Diamond	06/06/2018	01/11/2018	18P047	18P087	41	12150
Amarillo	2018_RC	RC	26/06/2018	30/07/2018	18PRC001	18PRC014	14	1295
Amarillo	2019_P	Diamond	08/01/2019	26/02/2019	19P088	19P095	8	3045

(1) Total holes 63, total metres 16,490.

Mr. John Watts of Australian Exploration Field Services visited Mara Rosa, on behalf of Mr Keith Whitehouse, and the Posse Mine site in October 2018 and has confirmed that the drilling operations were being carried out in a competent and professional manner.

Drilling setup and sample handling procedures were generally the same as for the last drilling campaign carried out by Amarillo and were outlined in Section 14 of the 2016 Mineral Resource Update (AEFS, 2016). The drilling companies were Rothes Prospecção Mineral and Geosol – Geologia e Sondagens S.A. for the diamond drilling work and Geosedna Perfurações Especiais S.A for the RC drilling.

For downhole survey work Rothes Prospecção Mineral used a Devco, PeeWee Downhole Survey tool while other downhole survey used a Reflex Instruments, Maxibor tool.

10.3 Drilling 2021

Following the completion to the 2020 DFS an Independent Technical Engineering ("**ITE**") Consultant carried out a risk review the DFS. This work is discussed further under Section 12.7. One of the outcomes of the risk review was the decision to drill further diamond holes into the resource envelope to target areas of the model which were not well tested by existing diamond drilling. A program of 10 holes was designed by AEFS to accomplish this, several additional holes were dropped from the program due to the lack of suitable collar locations. The campaign started in February 2021 and concluded in March 2021; the campaign is summarised in Table 10-3.

Table 10-3: 2021 Drilling Program

Company	Program	Type	Start Date	End Date	Start Hole	Finish Hole	Number	Metres
Amarillo	2021	Diamond	05/02/2021	31/03/2021	21P112	21P121	10	2519

The drilling company was Geosol – Geologia e Sondagens S.A, collar locations were surveyed by DGPS, and downhole surveys were taken using a Reflex Deviflex tool. Drilling setup and sample handling procedures were generally the same as for the last drilling campaign carried out by Amarillo and were outlined in Section 14 of the 2016 Mineral Resource Update (AEFS, 2016). Hole locations and details are shown in Table 10-4.

Table 10-4: 2021 Drilling Program Details

Hole_ID	Azimuth	Dip	Length	East	North	Elevation
21P112	134.91	-55.76	359.81	696,447.601	8,454,555.589	442.849
21P113	130.48	-55.37	329.49	696,301.098	8,45,4461.669	447.168
21P114	130.18	-57.13	349.48	696,267.053	8,454,423.405	446.987
21P115	130.18	-57.49	237.24	696,292.345	8,454,272.965	441.873
21P116	114.79	-54.33	165.25	696,815.333	8,454,663.382	435.677
21P117	109.84	-59.26	127.14	696,719.639	8,454,416.210	436.919
21P118	136.59	-55.18	276.36	696,572.285	8,454,453.636	440.801
21P119	122.85	-56.01	271.87	696,530.488	8,454,518.504	438.644
21P120	130.74	-50.34	168.79	696,535.617	8,454,095.561	440.457
21P121	114.74	-50.32	233.89	696,364.051	8,454,203.628	436.546
Total metres			**2,519.32**			

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

The sample preparation, analyses, and security have had only minor changes since the 2016 Report (AEFS, 2016) where the procedure is described in detail. Importantly the laboratory service preparing assays has changed from ACME Laboratories to ALS. ALS has retained the former ACME Laboratories sample preparation facility in Goiânia. After initial preparation the sample pulps are then sent to an ALS laboratory in Peru for Assay.

As samples are cut (or in the case of RC samples split) at site, they are stored in plastic bags with two copies of a sample ticket from a pre-printed sample number book. The details for each sample are recorded in the sample book, including hole ID and sample interval. Samples are then despatched to the laboratory in lots of 150. The plastic bags containing each sample are packed into canvas bags with 4-5 samples in each bag for transport to the laboratory in Goiânia. The canvas bags are sealed and marked with the requisition number, address, sender's details, and marked 1/28, 2/28 etc. One copy of the sample manifest is retained and stored in the Company archives, and another goes to the driver taking the samples to the laboratory. A copy of the manifest is also sent to ALS, the laboratory carrying out the assay work, and to Amarillo staff by email. An assay request form is also completed and submitted to the laboratory by email and with the samples.

On receipt by the laboratory in Goiânia the samples are prepared by the laboratory. Preparation consists of:

- Sorting and checking against the request sheet;
- Drying at 60°C;
- Washing with a granite wash to scour the equipment before the client's first sample is crushed;
- Crushing of the samples to 70% passing 10 mesh (2mm);
- Samples homogenised and riffle split to 500g subsample;
- Subsamples are pulverised to 85% passing 200 mesh (75micron);
- Equipment cleaning by brush and pressurised air; and
- A granite wash is used to scour the equipment after high grade samples, between changes in rock colour, and at the end of each file.

The subsamples are then split to approximately 100g and sent to ALS Laboratories in Lima Peru for analysis using Fire Assay with an AAS finish.

12 DATA VERIFICATION

12.1 General Drillhole Verification

Following completion of drilling on the Posse Deposit in 2012 extensive data validation and correction of pre-2013 drilling data was carried out by AEFS in 2015 and 2016 with data corrections being incorporated in the Amarillo geological database used for the 2016 model. The 2016 model used all data to the end of 2012 when drilling stopped. Resource updates in 2017 and 2018 did not include new modelling or further drilling, rather they were based on altering cut-off grades and constraints such as historic pit shapes within the 2016 modelling

The extensive data validation carried out in the lead up to the estimation of the 2016 Mineral Resource was detailed in the 2016 Resource report (AEFS, 2016). Between 2016 and the preparation of the current resource estimate changes were made to the drillhole database and consequently the historical data (holes to the end of 2012) contained in the 2019 drilling database (dated 22 March 2019) used for the 2020 Resource were compared to archival copies of the verified data used for the 2016 mineral resource estimate.

The historic data showed only a few discrepancies, and these were resolved with site. Several of the discrepancies related to survey coordinates of 2012 drillholes, with those provided by site (and recorded in the drillhole database) being used in preference to previous coordinates as the coordinates in the drillhole database recorded actual location pickups by differential DGPS rather than handheld GPS. There were also some differences in lithology that were mainly due to additional detail being provided by site.

The one area where there were differences between the data provided by Amarillo in 2019 and the validated datasets used for 2016 modelling was in downhole surveys of some of the drillholes from 2010. These had suspect downhole surveys which after extensive discussion with Amarillo were updated in 2011 to use either values derived from downhole surveys associated with acoustic televiewer runs collected by Weatherford in 2011 or calculated surveys derived from the Weatherford survey. The 2019 site drill database had reverted to the suspect Maxibor surveys. These suspect Maxibor surveys were replaced with the Weatherford and calculated surveys used in earlier (post 2011) modelling. The holes with suspect Maxibor survey are shown in red in Figure 12-1. The corrected trajectory plotted using the surveys derived from the Weatherford surveys are shown in black.

In 2021 a program of limited drilling, Section 10.3, was carried out to help understand grade variability in selected areas of the 2020 Resource. Data from the 2021 drilling program was provided by Amarillo to AEFS and was incorporated into a copy of the drilling data used for the 2020 resource to allow evaluation of the potential impact of the 2021 drill results on the 2020 resource, Section 12.8. The trajectory of the drilled holes was checked against the planned trajectories and the holes hit the target zones with only small margins of error as shown in Figure 12-2.

Figure 12-1: Holes from 2011 drilling with incorrect downhole survey

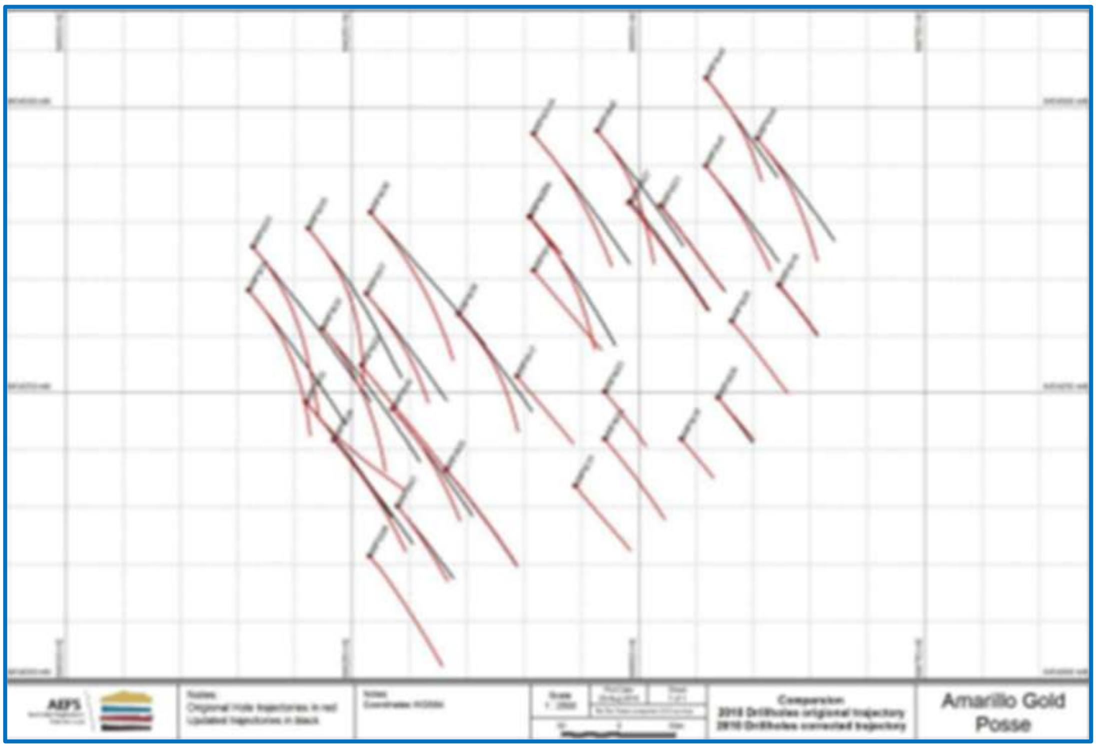


Figure 12-2: 2021 drilling planned and actual(1)

(1) Planned holes in blue, actual in red. Planned holes RD10, RD11 and RD14 were not drilled.

12.2 Topographic Verification

Extensive work was also undertaken, as described in Section 9.1 of this CPR, when the LIDAR survey was received in 2019 to update the elevation of drillhole collars. Drillhole collar elevations are now considered to be accurate to +/- 0.25m and this has allowed better definition of the wireframes controlling the mineralization estimates.

The 2019 LIDAR survey together with the bathymetric survey and accurately referenced historic plans allowed clear definition of backfill in the pits, areas adjoining the pits used for fill (borrow areas) and the extent and volume of waste dumps and historic stockpiles. Within the flooded pit boundaries, the bathymetric survey together with the post mining pit survey has determined the amount of backfill. The precise volume of backfill will not be known until the pits are dewatered and the backfill excavated, however volumetric errors are expected to be small.

The drill hole database provided by Amarillo included updated lithological coding covering the upper portions of the drillholes in-order to better resolve the depth of Soil and / or Saprolite over the mineralization. This information together with information from the 2019 LIDAR survey provided a basis for constructing wireframes of both Soil and Saprolite over the area of mineralization.

12.3 Geological Modelling

The general geological setting for the Posse mineralization is described as a meta granadiorite, “biotite gneiss” hanging wall with amphibolite, “greenstone” in the footwall. Shearing and associated hydrothermal alteration of the ”biotite gneiss” has resulted in the formation of a distinct lithologic unit, a quartz-feldspar-mica schist, “Posse Schist” that is characteristic of the Posse ore zone. The mineralization envelope at Posse is about 30m thick and over 1km long developed along a 050° to 065° striking fault zone. Mineralization tends to be strongest within mylonitic zones that follow more northerly striking (approximately 030° to 050°) shear strands and dilatant jogs that obliquely transect the contact between the hanging wall and footwall rocks. Higher intensity of shearing is associated with increased sulphide mineralization (up to about 4%), and a slight increase in metamorphic grade from greenschist to high greenschist facies in the hanging wall through to high greenschist/low amphibolite facies in the footwall (biotite flakes and garnet alteration). Higher gold values are associated with increasing intensity of shearing and higher levels of silicification and sulphide mineralization.

The Project has been running for approximately 35 years and there have been multiple logging styles used through the exploration program as various companies have held the Project. This has resulted in a complex list of lithological codes, however using the general geological description of the deposit as a guide, the list in Table 12-1 has been reduced into codes used for resource modelling.

Table 12-1: Lithology Coding and Mapping

Logging Code	Remarks	Grouped Code	Comments
Sol	Soil	Sol	
Sap	Saprolite	Sap	
Dmaf	Mafic Dike	Dmaf	
Qv	Quartz Vein	Qv	Minor occurrence
PEG	Pegmatitic	PEG	
Dpeg	Pegmatite	Dpeg	
GNb	Biotite Gneiss	Gnb	Meta granadoirite, “Biotite gneiss” with little mylonitization and hydrothermalism, Hanging Wall, HW
Metagrauvaca	Metagrauvaca	Gnb	“Biotite gneiss” (old logs)
Bisex	Bio- musc xisto	Sc	Posse Schist
SCHb	Biotite Schist	Sc	
SCHm	Muscovite Schist	Sc	
SCHmb	Muscovite Biotite Schist	Sc	
SCHqbs	Quartz Biotite Sericite Schist	Sc	
SCHqs	Quartz Sericite Schist	Sc	
FMtf	Felsic Metatuff	Sc	
Qsex	Quartz-Sericite Schist	Sc	
SCHbc	Biotite Chlorite Schist	Sc	

Logging Code	Remarks	Grouped Code	Comments
SCHc	Chlorite Schist	Sc	
Amph	Amphibolite	Amph	Amphibolite, FW, Footwall
HYkqs	Kspar-Sil-Ser Hydrothermal Rock	HYkqs	Minor occurrence
HYq	Silica Hydrothermal Rock	HYq	
Dapl	Aplite	Dapl	

The reduced list of lithologies was then modelled. The Soil and Saprolite units were digitised on sections through the deposit to lay sub parallel to topography. Either or both of these units may be absent locally. The sectional interpretations of Base of Soil and Base of Saprolite were then modelled as wireframe surfaces.

The “Biotite Gneiss” and Amphibolite units were modelled in 2019 initially using Micromine’s Implicit Modelling functions. The resulting wireframes were them examined on sections with drillholes and geology and modified to produce an acceptable Lithological model. The zone between the modelled “Biotite Gneiss” hanging wall and the Amphibolite, footwall was assumed to be Posse Schist, it was not explicitly modelled.

A Mafic dyke which crosses the southern portion of the deposit was modelled on sections and wireframed.

The following surfaces and wireframes in Table 12-2 were used to record the Geology.

Table 12-2: Modelled Lithological Units

Unit	Wireframe Type
Soil	Open Surface
Saprolite	Open Surface
“Biotite Gneiss” (HW)	Closed Surface
Amphibolite (FW)	Closed Surface
Mafic Dyke	Closed Surface
Posse Schist	Not modelled, implied as the gap between the HW and FW

12.4 Specific Gravity

The Mineral Resource reporting from 2016 - 2018 had used SG values based on gamma logs obtained from geophysical survey work carried out by Weatherford in 2011 that confirmed an average SG of 2,730kg/m^3. This is the same estimate figure, derived by other means, that was used in the 2010 Resource Report (HCS & AEFS, 2010) and subsequent reports to 2015.

For the 2020 resource rather than using the average SG of 2,730kg/m^3 for portions of the mineralization that were not in Soil or Saprolite, the Weatherford data was again interrogated to derive SG values for all the major lithological units recognized by AEFS in the deposit, Figure 12 3 and Table 12 3.

Figure 12-3: SG values derived from Weatherford Data

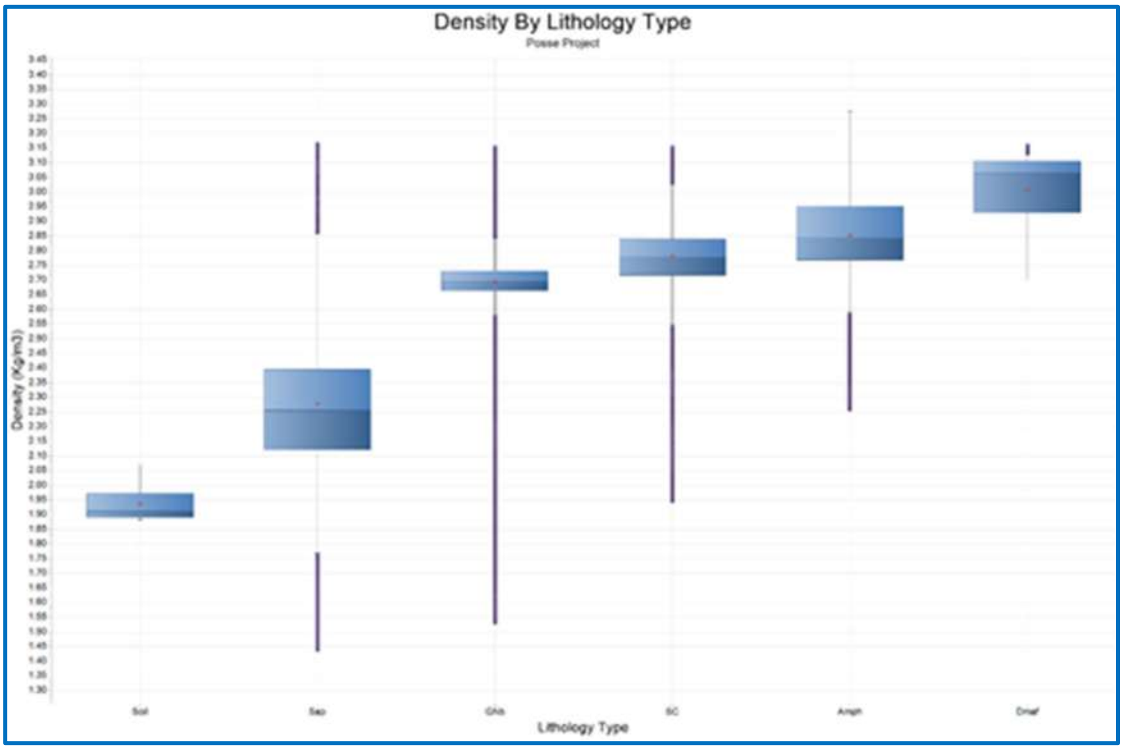


Table 12-3: SG values by rock type from Weatherford Data

Rock type	Count	Mean	Median	Min	Max
Soil	35	1.94	1.91	1.88	2.07
Saprolite	6,545	2.27	2.26	1.46	3.165
"Biotite Gneiss"	79,1671	2.70	2.70	1.531	3.154
Amphibolite	234,008	2.85	2.84	2.26	3.28
Mafic Dyke	3,977	3.01	3.07	2.70	3.16
Posse Schist	218,290	2.78	2.78	1.95	3.15

During 2019, work was also carried out by SRK Consulting on the SG of the deposit. As part of their work, they obtained SG values derived from measuring 129 samples spread across the deposit using the weight in water / weight in air method (SRK, 2019). The analysis work was carried out at the ALS laboratory in Vespasiano, Minas Gerais, Brazil and produced results as shown in Table 12-4.

Table 12-4: SG values by rock type from Wet Weight / Dry Weight Analysis

Rock	Count	Mean	Min	Max
"Biotite Gneiss"	43	2.70	2.63	2.74
Mylonite Biotite Gneiss(1)	38	2.72	2.68	2.86
Mylonite Amphibolite(1)	10	2.86	2.75	3.02
Amphibolite	32	2.92	2.73	3.04
Silica Hydrothermal Rock	6	2.71	2.70	2.75

(1) The Mylonite units tested by SRK are the same as the Posse Schist unit recognized by AEFS and when combined give a weighted mean and median of 2.75 and 2.73, respectively

SRK concluded that the differences in density measurement using the Wet Weight / Dry Weight method were within 2.5% and have recommended that the Wet Weight / Dry Weight density measurement method be used going forward. AEFS concurs with this.

The SG values adopted for estimation of the 2019 Mineral Resource are listed in Table 12-5.

Table 12-5: SG values used for 2020 Resource estimation

Rock type	Code	SG g/cc	SG kg/m^3
Soil	Soil	1.91	1,910
Saprolite	SAP	2.27	2,270
Biotite Gneiss	Gnb	2.70	2,700
Schist	Sc	2.78	2,780
Amphibolite	Amp	2.85	2,850
Mafic Dyke	DMaf	3.01	3,010
Air	Air	0.001	1.000
Fill	Fill	2.0	2,000

Subsequent to the release of the 2020 DFS further work as outlined in Section 12.7.2 has been undertaken. The results of this work have confirmed the density data used in the 2020 Resource and provide a basis for future modelling to incorporate a greater degree of spatial density variation within individual lithologies in future modelling work.

12.5 Assay QA/QC

Given the stage of the Project with the current Resource Estimate contributing to the Feasibility Study it is appropriate to review the QA/QC programs that have been run as part of exploration to date. The QA/QC sampling to date is summarised in Table 12-6. Early drilling programs had no or very limited QA/QC sampling in the current drilling database and searches of archival data did not produce any additional information. It is therefore possible that these holes were drilled without QA/QC samples, indeed considering the age of the holes, early 1980's it is quite likely that this is the case. QA/QC data which has not been reviewed in previous published technical reporting is discussed below together with recent resampling programs which provide confirmation of older drilling and associated sampling. The QA/QC sampling associated with drill programs from 2008 – 2012 has been discussed in the technical reports referenced in Table 12-6.

Table 12-6: Summary of QA/QC sampling by Drill Program(1)

Company	Program	Type	Start Hole	Finish Hole	References	Blanks	Duplicates	Comment
BHP	1983_F	Diamond	F001	F005	NR	NR	NR	
BHP	1984_F	Diamond	F006	F018	NR	NR	NR	
BHP	1984_W	Percussion	W001	W004	NR	NR	NR	
BHP	1985_W	Percussion	W005	W032	NR	NR	NR	
BHP	1985_FW	Diamond	FW019	FW059	Some	Some		QA/QC sampling limited to FW057 & FW058
BHP	1987_FS	UG	FS001	FS010	NR	NR	NR	Used for geologic modelling but not for resource estimation
BHP	1987_W	Percussion	W034	W036	NR	NR	NR	
WMC	1988_MRD	Diamond	MRD001	MRD073	Yes	?	NR	While Reference samples were used there is no record of the expected values for the reference samples
WMC	1988_MRC	RC	MRC035	MRC038	NR	NR	NR	
WMC	1990_MRC	RC	MRC094	MRC095	NR	NR	NR	
WMC	1991_MRC	RC	MRC125	MRC127	NR	NR	NR	
WMC	1992_MRC	RC	MRC175	MRC191	NR	NR	NR	
WMC	1993_MRC	RC	MRC200	MRC235	NR	NR	NR	
WMC	1993_MRD	Diamond	MRD196	MRD197	NR	NR	NR	
WMC	1994_MRD	Diamond	MRD199	MRD346	NR	NR	NR	
Amarillo	2006_SPETI	Diamond	SPETI01	SPETI28	Yes	Yes	Yes	While Reference samples were used there is no record of the expected values for the reference samples, see below.
Amarillo	2008_FMR	Diamond	FMR0001	FMR0009	Yes	Yes	NR	Reviewed in the 2010 Resource Report (HCS & AEFS, 2010)
Amarillo	2008_W	Diamond	W002A	W002A	Yes	Yes	NR	Reviewed in the 2010 Resource Report (HCS & AEFS, 2010)
Amarillo	2008_MRP	Diamond	MRP0001	MRP0014	Yes	Yes	NR	Reviewed in the 2010 Resource Report (HCS & AEFS, 2010)
Amarillo	2010_MRP	Diamond	MRP0015	MRP0045	Yes	Yes	Yes	Reviewed in the 2010 Resource Report. (HCS & AEFS, 2011)
Amarillo	2011_MRPA	Diamond	2011MRP0001	2011MRP0013	Yes	Yes	Yes	Reviewed in the 2010 Resource Report. (HCS & AEFS, 2011)
Amarillo	2012_MRPA	Diamond	2012MRP0001	2012MRP0034	Yes	Yes	Yes	Reviewed in the 2010 Resource Report. (AEFS, 2016)
Amarillo	2012_P	Diamond	12P035	12P046	Yes	Yes	Yes	Reviewed in the 2010 Resource Report. (AEFS, 2016)
Amarillo	2018_P	Diamond	18P047	18P087	Yes	Yes	Yes	Reviewed in this report
Amarillo	2018_RC	RC	18PRC001	18PRC014	Yes	Yes	Yes	Reviewed in this report
Amarillo	2019_P	Diamond	19P088	19P095	Yes	Yes	Yes	Reviewed in this report

(1) None Recorded ("**NR**")

Assay QA/QC is of particular importance as the collection of samples for analysis is a complex process that can if not carried out correctly result in biased results. Bias of results can occur as the result of field and core shed procedures or in the laboratory. A modern QA/QC program will use blanks, reference samples and duplicate sampling to test various aspects of the sampling program. In general:

- Blanks can reveal issues with sample recovery systems at site and / or poor laboratory practices resulting in cross contamination of samples, particularly after high grade samples;
- Reference samples should produce a very consistent set of results for each reference material clustered within narrow limits. Deviations from the expected value may indicate

poor procedures either at the laboratory or on site; and

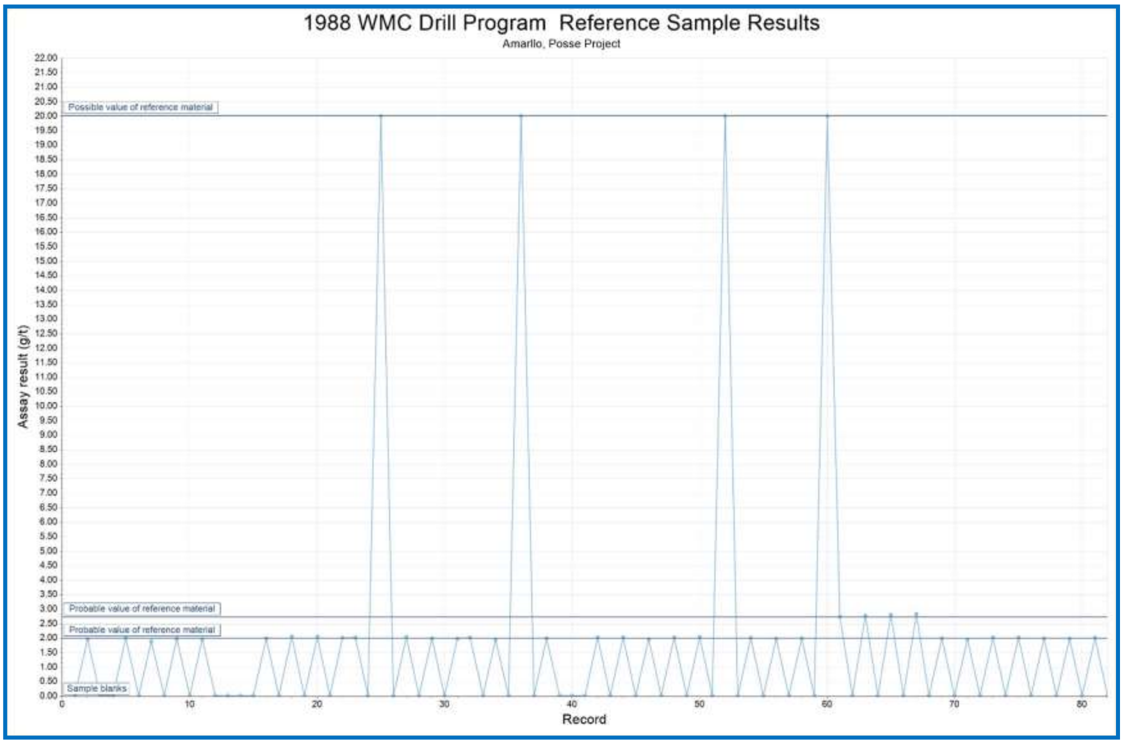
- Duplicate sampling should produce results for each of a pair of duplicates which are within very close agreement. Variations in the results of duplicate sampling can indicate issues with nuggety ore, the sampling process or laboratory processes.

While all laboratories run extensive internal QA/QC programs, the explorer should be running their own QA/QC program to monitor for deficiencies in site procedures and to cross check the laboratory process.

12.5.1 BHP and WMC QA/QC programs

Limited QA/QC sampling was associated with a 40-hole diamond drilling program conducted by BHP in 1985, however the available QA/QC data is limited to 2 holes and the actual identity of the standards is not recorded in the database. Similarly, the 1988 drilling program by WMC recorded reference samples but there is no record of what the reference samples were. A chart, Figure 12-4, of the samples suggests that there was a standard with a grade of around 2.0g/tAu. There were possibly other standards with values of around 2.7g/tAu and 20g/tAu used. It is also apparent that Blank reference material has been used but not labelled as such.

Figure 12-4: WMC QA/QC sampling 1988

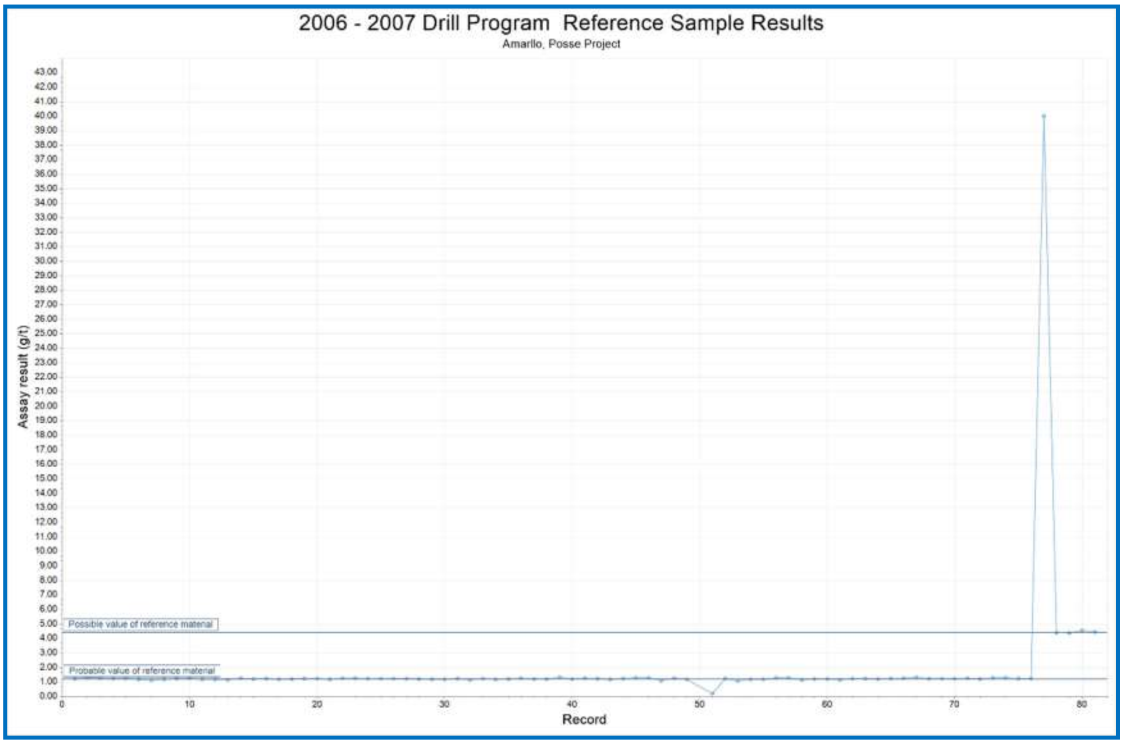


The risk review, discussed in Section 12.7 below considered the use of data associated with WMC and BHP drilling to be an elevated risk to the project, however subsequent re-assay of remaining drill core is considered to have adequately addressed this issue.

12.5.2 Amarillo 2006 drilling

Amarillo's 2006 drill program used standards, blanks and included duplicate sampling. Unfortunately, the standards for that program have not been properly identified and as a result it is not possible to use them, for any meaningful work. A quick chart of the assay results, Figure 12-5, identified as being from reference samples in the 2006 drill program suggest that there were two reference materials used, one with a value of around 1.2g/tAu and the other with a grade of around 4.2g/tAu. One sample produced a very low result and is possible a miscoded blank while the very high result for sample 77 looks to be a typing error when the data was entered with the real value being around 4.01g/tAu.

Figure 12-5: Reference samples 2006



A subsequent search of archival data (January 2020) has returned two assay certificates from ACME laboratories which are for material defined as STD 1. One (Acme Analytical Laboratories S. A, 2006) of these defines STD 1 (referred to as Standard 1a) as having a mean of 1.261 with a standard deviation of 0.067. The other (Acme Analytical Laboratories S.A., 2007) defines STD1 (referred to as Standard 1b) as having a mean of 4.208 with a standard deviation of 0.032. It would appear that these are in-house references, and they have expected (mean) values which closely match the results recorded in the drilling database, Figure 12-6 and Figure 12-7.

Figure 12-6: 2006 Drilling, Standard 1a, Shewhart Plot

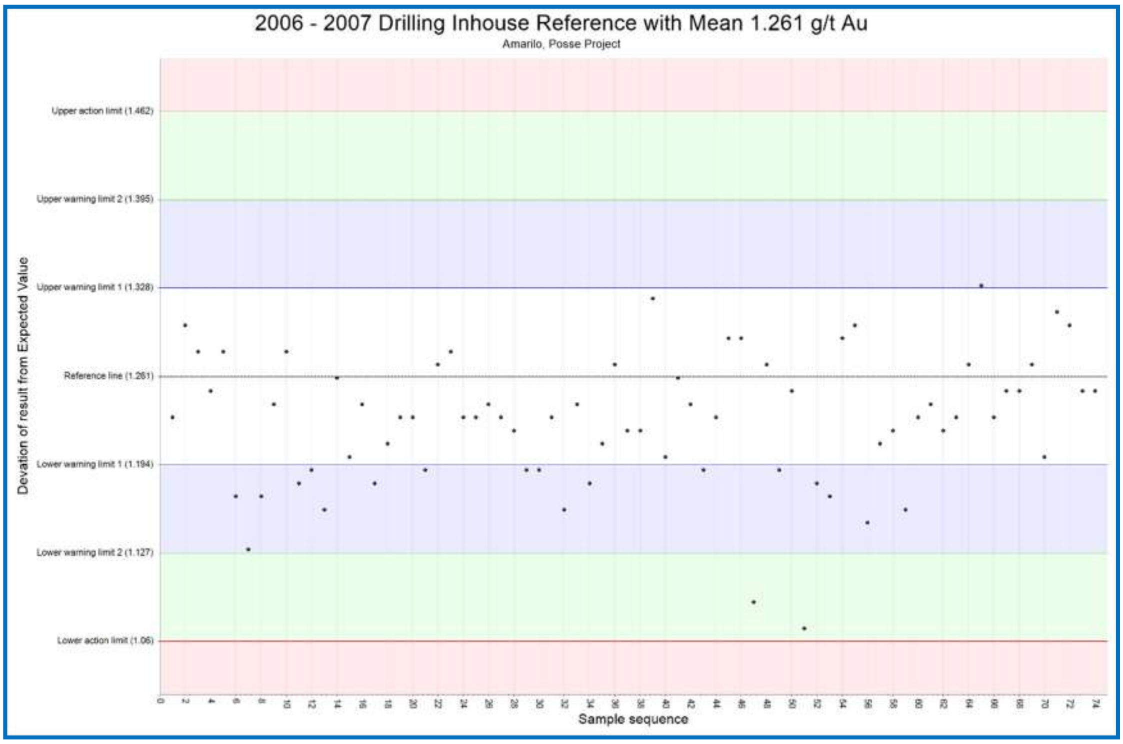
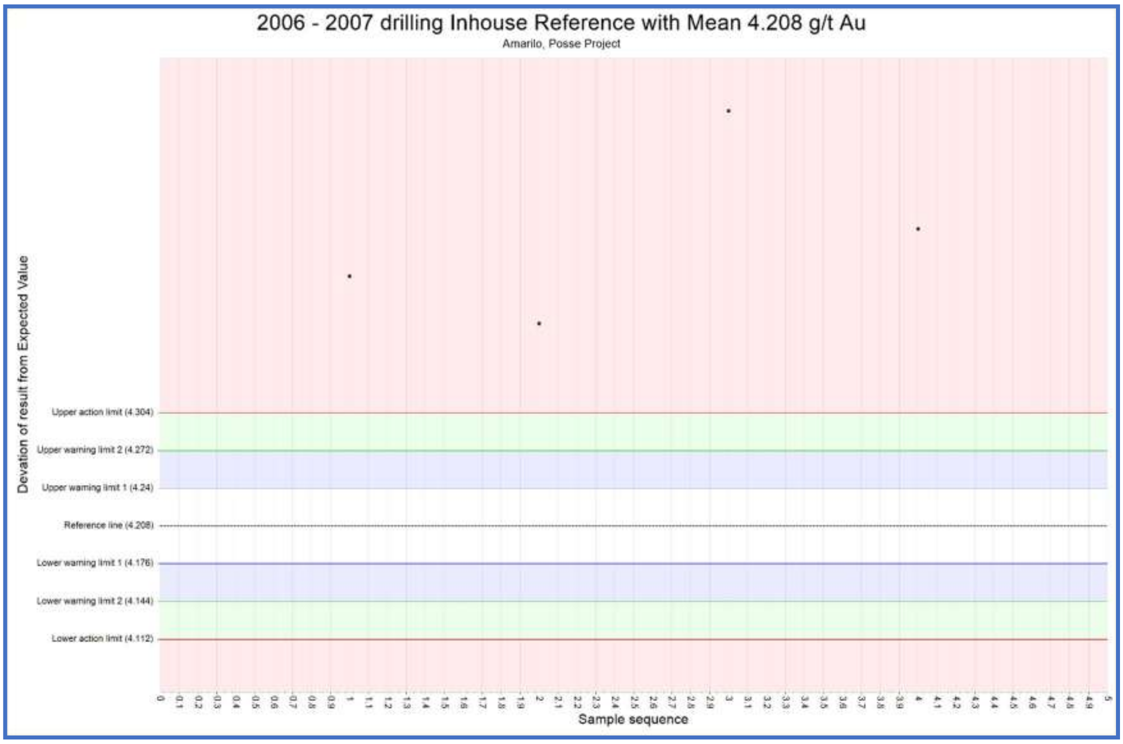


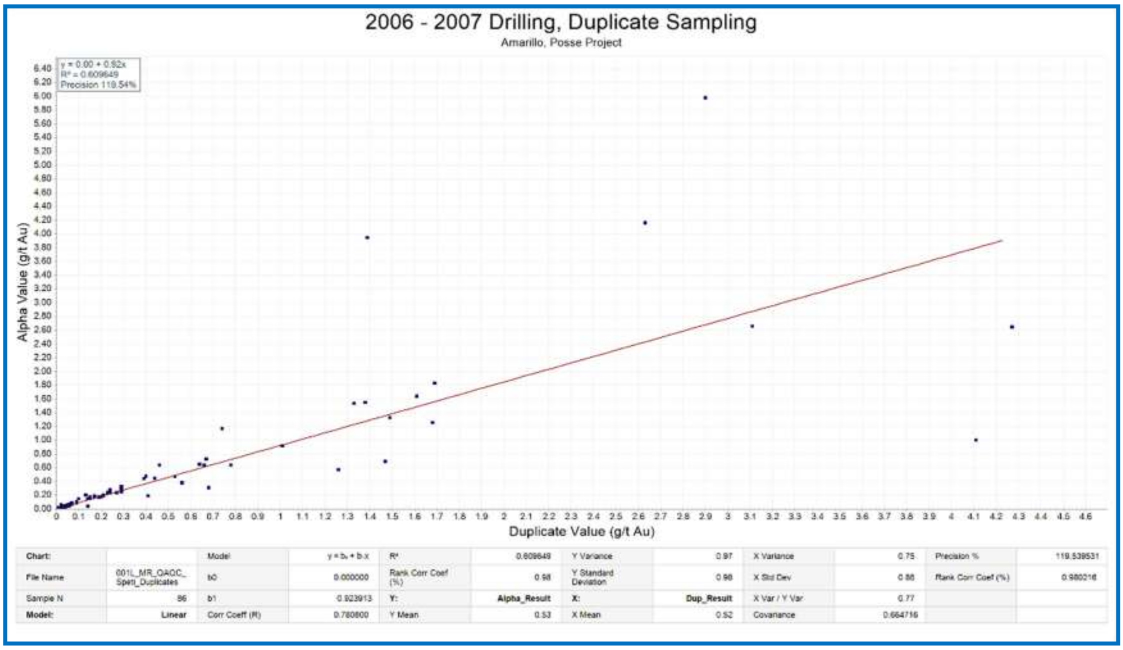
Figure 12-7: 2006 Drilling, Standard 1b, Shewhart Plot



The results for Standard 1a are quite scattered but all fall within acceptable bounds, those for Standard 1b are all out of bounds. A review of the data used shows two sets of results for these samples, one in ppb and one in ppm, with a large disparity between the two sets of values. This suggests that there were errors associated with these QA/QC samples which all relate to hole SEPTI28. It is recommended that, in the absence of any other information on these QA/QC samples, that they be ignored for future work. A check of the assays for the intersection of this hole with the orebody does not reveal any inconsistencies with regard to the width of the intersection or the grade of intervals in relation to adjoining drillholes.

The sample duplicates from the 2006 drill program are charted in Figure 12-8, and shows there is a reasonable correlation between the Alpha Samples and the Duplicates, although it is obvious that at higher grade the correlation is much less well defined.

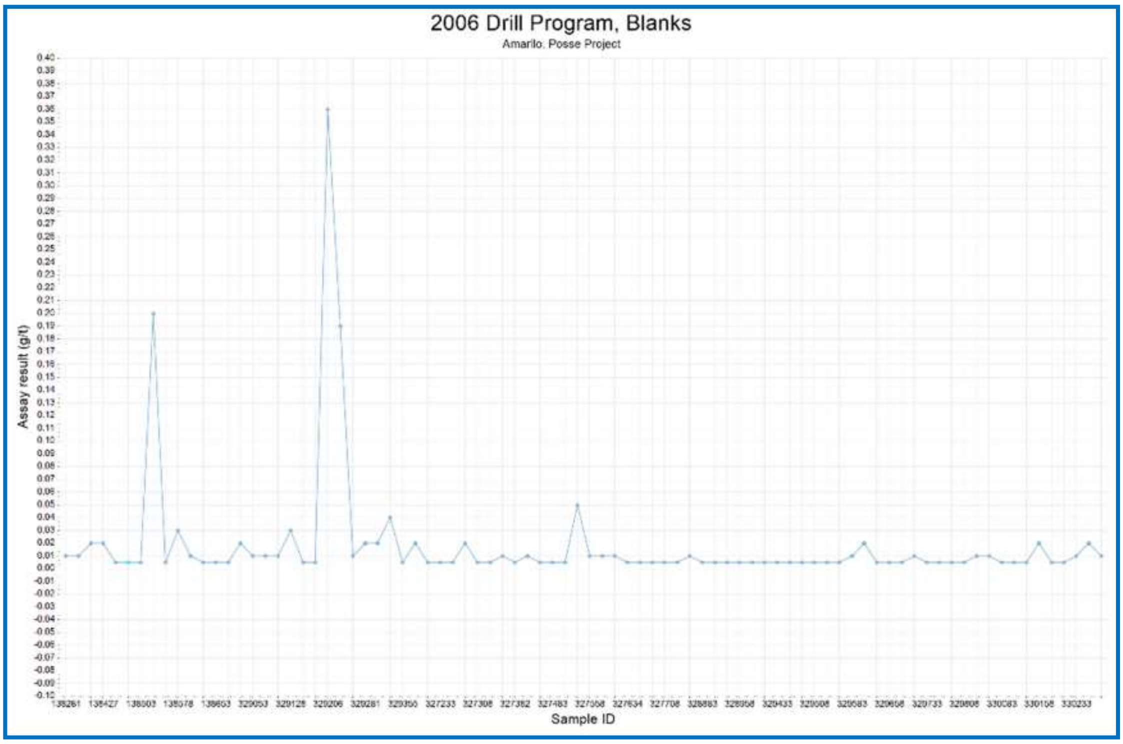
Figure 12-8: 2006 Drilling, Duplicates



There is no information available on the procedure for the duplicate sampling although it is probable that 2 pieces of ¼ core rather than 1 piece of ½ core were submitted to the laboratory for analysis as this was the method used in later duplicate sampling. This being the case it is expected that there would be a breakdown of the regression at higher grades due to nugget effects and small sample size.

The sample blanks from the 2006 Drilling program are charted in Figure 12-9. The detection limit of the Fire Assay analysis technique used was 0.01g/tAu and 95% of the blank samples are in within 3 x of the detection limit. Of the 84 samples in the Blanks program 4 plot above 0.3g/tAu, several of these appear to be close to the value of the 1a Standard that was used, and it may be that these represent a mix up of QA/QC samples at sample despatch. The largest value of 0.36 (sample 329206) on re-analysis returned a value of 0.03g/tAu.

Figure 12-9: 2006 Drilling, Blanks



Overall, there are some issues with the QA/QC sampling program for the 2006 drill campaign which appear to be mainly related to poor record keeping, indeed a number of the same issues were seen in later drilling programs. The issues are minor, and it is the authors' view that data management and associated record keeping has improved with the most recent drill program. Despite the minor issues seen, the Amarillo QA/QC data associated with the 2006 drilling program is considered acceptable and the associated Alpha sampling is considered to be representative of the mineralization.

12.5.3 Amarillo, 2008 – 2012 Drilling

Drilling programs between the years 2008 and 2012 have been reviewed in previous technical reports and this work is not reproduced here. See (HCS & AEFS, 2010) (HCS & AEFS, 2011) and (AEFS, 2016).

12.5.4 Amarillo, 2018 – 2019 Drilling

The 2018 – 2019 drilling programs had an extensive QA/QC program of reference samples (6 standards), blanks and duplicate sampling. The was a change in the method of collecting the sample duplicates, whereas earlier programs had split ½ core into two ¼ cores and submitted both for analysis, the 2018 – 2019 program submitted ½ core samples to the lab and the Lab then took a duplicate sample from the coarse crush.

Results from the reference sampling are shown in Figure 12-10. In general, the assayed result for the reference samples is within acceptable limits. Most results are within 1 SD of the expected value. It is noted that Standard HiSiK2 has 4 out of 22 occurrences where the results plot in the lower action band. The reason for this has not been determined.

Analysis of duplicates, Figure 12-11, and of blanks, Figure 12-12, shows results generally as expected. The duplicate sampling used half core, split into 2 samples after the coarse crush at the laboratory and shows a much higher correlation between alpha samples and duplicates compared to older duplicate sampling where the half core is pre-split into ¼ cores, Figure 12-8, and submitted for analysis. This is to be expected as splitting after the coarse grind will result in a more homogenised sample. The blanks are generally very close to their expected values, with two noticeable exceptions. Sample_ID 700901 has been given a value of 0.1 g/t which matches the assay certificate. Sample_ID 705063 has been given a value of 0.726 which again matches the assay certificate. It is interesting, though, that the proceeding sample 705062 has a value of <0.05 on the assay certificate and it is possible that the blank has been assigned to the incorrect sample number. Evaluation of all assay results and QA/QC results as results are received would ensure that if in fact the samples numbers have been incorrectly assigned it is picked up and corrected.

Figure 12-10: 2018 Drilling, Shewhart Plots of Standards

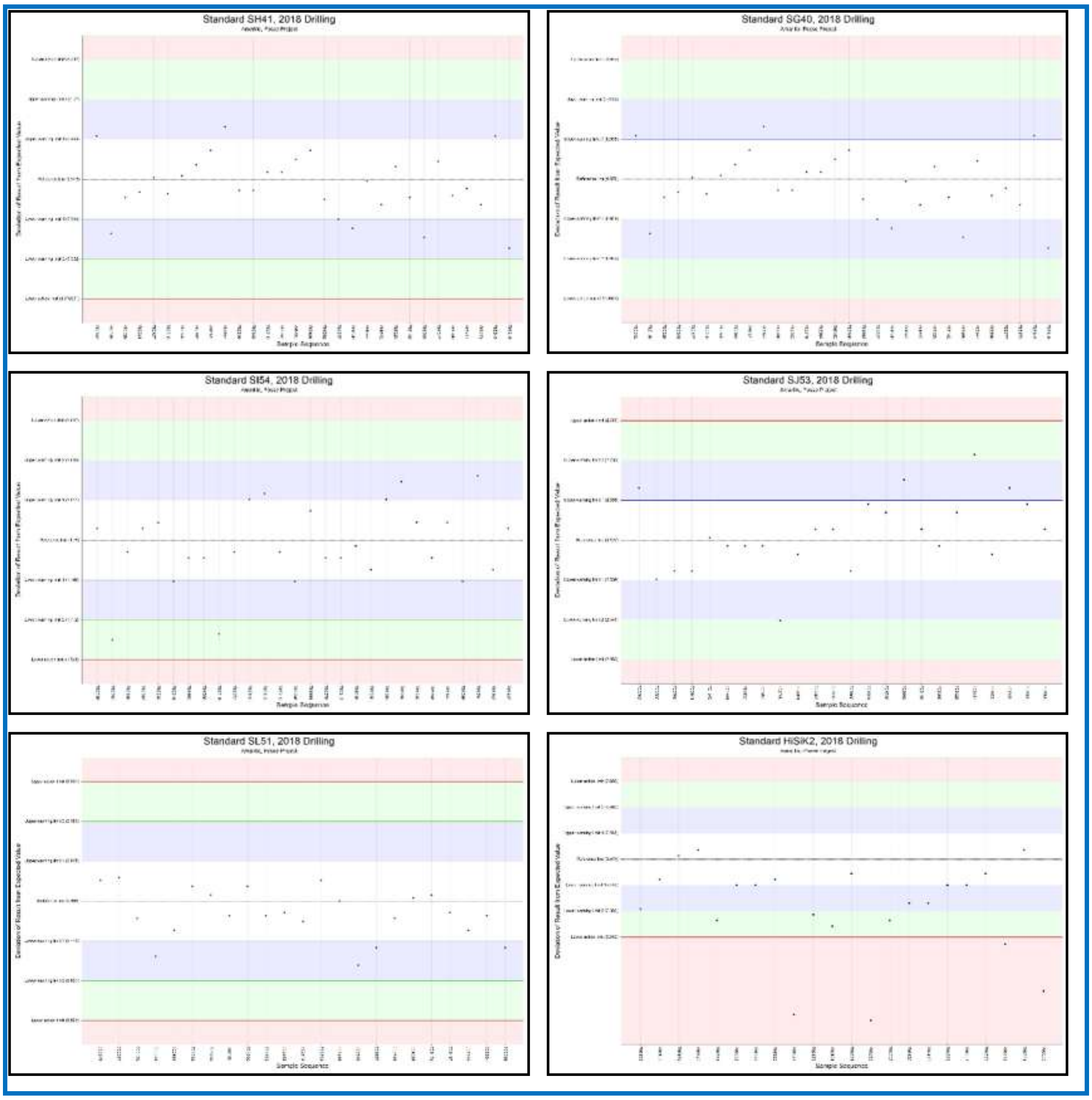


Figure 12-11: 2018 Drilling, Duplicate Sampling

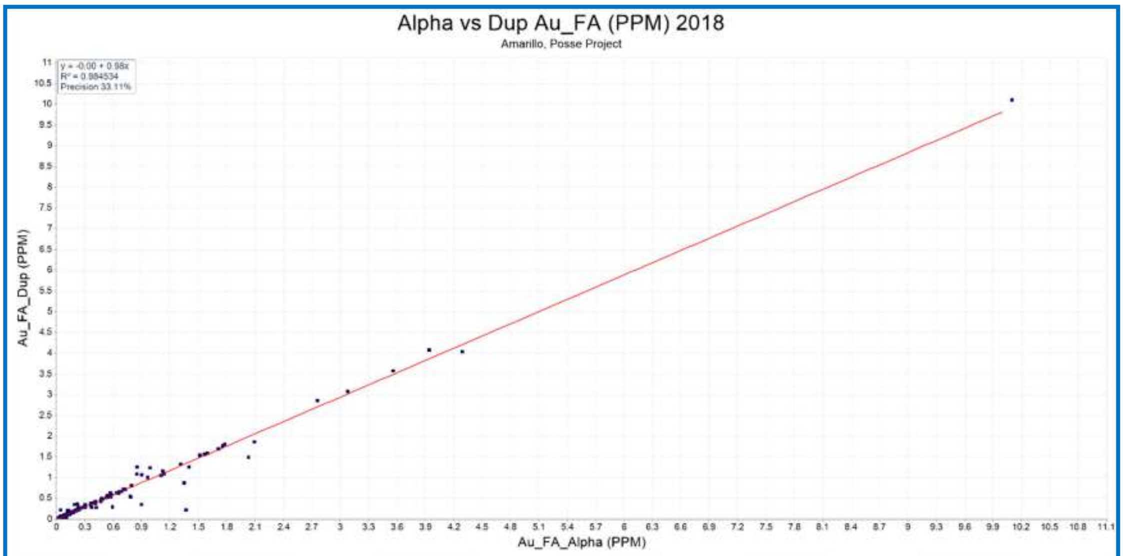
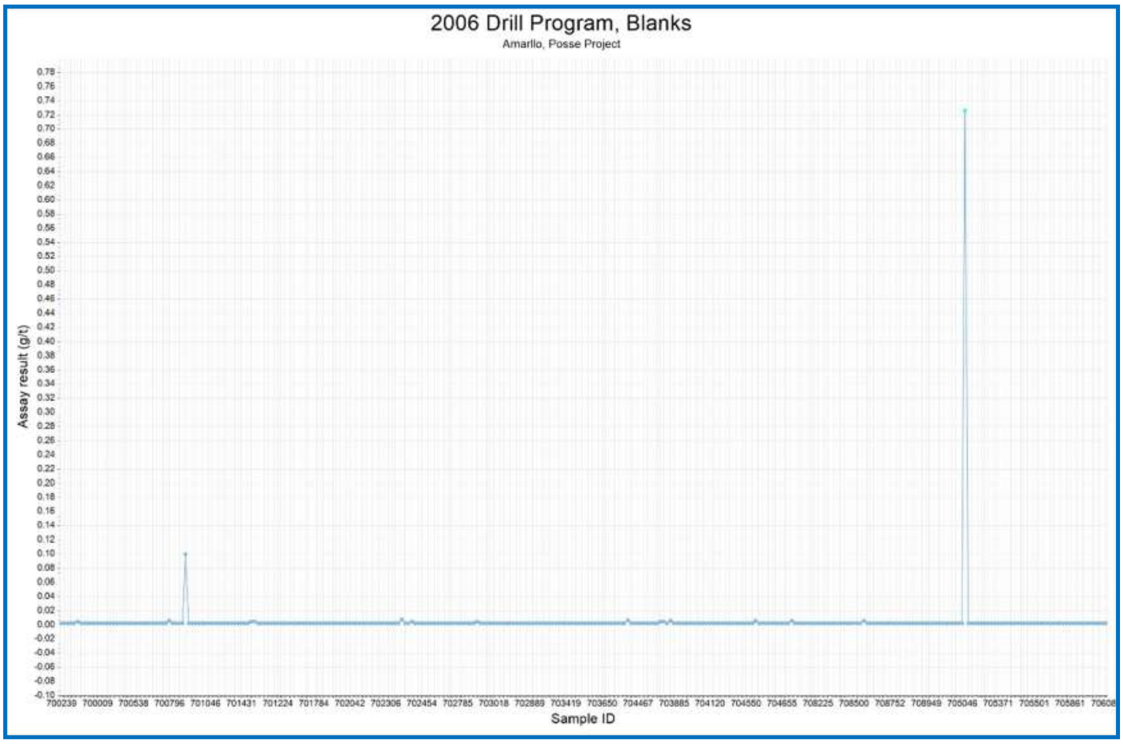


Chart:		Model	y = b₀ + b₁x	R²	0.984534	Y Variance	0.61	X Variance	0.62	Precision %	33.108926
File Name	001F_MR_DH_Assay_Working	b0	-0.002201	Rank Corr Coef (%)	0.99	Y Standard Deviation	0.78	X Std Dev	0.79	Rank Corr Coef (%)	0.989183
Sample N	309	b1	0.981745	**Y:**	**Au_DUP (ppm)**	**X:**	**Au_FA (ppm)**	X Var / Y Var	1.02		
Model:	**Linear**	Corr Coeff (R)	0.992237	Y Mean	0.29	X Mean	0.30	Covariance	0.613480		

Figure 12-12: 2018 Drilling, Blanks



12.5.5 SRK Consulting, QA/QC Review

In September and October 2019, SRK was tasked with examining the quality of assay work at Posse. As a part of this work, results of re-assay work conducted by CCIC (Caracle Creek International Consulting Inc. (Canada)) in 2006 were obtained from CCIC, and a Re-assay Program of 566 samples covering drilling by BHP and Western Mining was conducted by Amarillo staff at SRK's request. Information from SRK's report is summarised below in Table 12 7.

External analytical control samples produced by Amarillo, between 2006 and 2019, for the Posse Gold Project, were aggregated, by SRK, for analysis. Standards and blank data were summarized on time series plots to highlight the performance of the control samples. Paired data (field duplicates) were analysed using bias charts, quantile-quantile and relative precision plots.

Between 2006 and 2012 sample preparation for assay work was carried out at ACME Laboratories sample preparation facility in Goiânia, Brazil with assay determination carried out at ACME Laboratories facilities in Santiago, Chile and Vancouver, Canada. After ACME Laboratories were absorbed into ALS the 2018 – 2019 drilling program used sample preparation at ALS's Goiânia sample preparation facility with assay determination at the ALS laboratory in Lima, Peru.

The performance of the Standards used is shown in Table 12-7, the Blanks are shown in Table 12-8 and the Duplicates are shown in Table 12-9 and Table 12-10.

Table 12-7: External Analytical Standards, Performance, 2006 - 2019

Program	Type	Method	Samples	Failures	Passed
2008	DDH	ICP[1]	125	71	43%
		FA[2]	19	1	95%
		FA[3]	81	17	79%
2010 – 2011	DDH	ICP[1]	283	53	
		FA[4]	289	16	94%
2010 – 2011	RC/DDH	ICP[1]	15	1	87%
		FA[4]	15	1	93%
2012	DDH	ICP[1]	222	73	67%
		FA[5]	126	7	94%
		FA[4]	96	20	79%
2018 – 2019	DDH	FA[3]	290	9	97%
2018	RC	FA[3]	49	1	98%

(1) ICP1: Aqua Regia Digest with determination by Inductively coupled plasma mass spectrometry ("**ICP**"), results in ppb; FA2: Fire Assay with determination by Atomic Absorption Spectroscopy ("**AAS**"), results in ppb; FA3 Fire Assay with determination by AAS, results in ppm; FA4: Fire Assay with determination by either of Inductively coupled plasma atomic emission spectroscopy ("**ICP-ES**") or AS, results in ppm; FA5: Fire Assay with determination by Inductively coupled plasma atomic emission spectroscopy, results in ppm; and Samples assayed by ICP were generally multi element assays which included gold, these results were frequently duplicated by Fire Assay methods which were preferred.

Table 12-8: External Analytical Blanks, Performance, 2006 - 2019

Program	Type	Method	Blanks	Failures	Passed
2006	DDH	ICP[1]	78	39	50%
		FA[3]	82	4	95%
2008	DDH	ICP[1]	149	90	40%
		FA[2]	100	4	96%
		FA[3]	24	2	92%
2010 – 2011	DDH	ICP[1]	294	158	46%
		FA[4]	300	0	100%
2010 – 2011	RC/DDH	ICP[1]	15	9	40%
		FA[4]	15	0	100%
2012	DDH	ICP[1]	222	128	42%
		FA[5]	24	0	100%
		FA[4]	54	0	100%
2018 - 2019	DDH	FA[3]	272	2	99%
2018	RC	FA[3]	48	0	100%

Table 12-9: External Coarse Duplicates, Performance, 2006 – 2019[(1)]

Program	Type	Method	Duplicates	Failures	Passed
2006	DDH	FA[3]	83	17	76%
2010 – 2011	DDH	FA[4]	265	55	79%
2012	DDH	FA[5]	221	56	77%
2018 – 2019	DDH	FA[3]	101	19	81%

(1) Analysis by Half absolute relative difference (HARD) plot.

Table 12-10: External Pulp Duplicates, Performance, 2006 – 2019[(1)]

Program	Type	Method	Duplicates	Failures	Passed
2018 – 2019	DDH	FA[3]	162	8	95%
	RC	FA[3]	47	6	87%

(1) Analysis by Half absolute relative difference (HARD) plot.

12.5.6 SRK Verification of Historical Data

During a site visit in 2006, CCIC collected and re-analysed 84 samples from the BHP and WMC drilling campaigns to assess the quality of the historical database. CCIC was of the opinion that these samples confirmed the veracity of the historic database (CCIC, 2008). Holes sampled are shown in Table 12-11. Amarillo provided to SRK the results of the CCIC sampling data, and SRK found that 82% of the analysed samples presented a Half Average Relative Difference (HARD) lower than 30%.

Table 12-11: CCIC Verification Sampling, 2006

Company	Hole-ID	No. Samples	Year	Type
BHP	FW029	15	1985	DDH
	FW-036	8		
	FW-049	9		
	FW-55	11		
WMC	MRD-015	10	1988	DDH
	MRD-016	10		
	MRD-025	5		
	MRD-028	5		
	MRD-196	5	1993	
	MRD-344	5	1994	
Total		**84**		

12.5.7 SRK Verification of Amarillo 2018 Re-Assay Program

To validate the accuracy of the holes drilled by WMC and BHP Amarillo re-analysed part of the available core data. Of the 5,794 BHP samples in the drilling database 331 samples representing 5.7% of the BHP samples were re-assayed. Of the 1,582 WMC samples in the drilling database 235 samples representing 6.8% of the WMC samples were re-assayed.

As a part of the re-assay program Amarillo inserted QC samples to assess the accuracy of the assay results. These samples represent approximately 13% of the total number of samples analysed. The samples were prepared at the ALS laboratory located in Goiânia, Brazil and the chemical analyses were performed in the ALS laboratory located in Lima, Peru. The re-assay results analysed by SRK, who paired the re-assay results with the original assays, results and analysed with 76% of the BHP samples and 92% of the WMC samples passing the HARD analysis.

12.5.8 QA/QC Summary

When the QA/QC samples analysed are counted approximately 10% of the samples submitted for analysis have been for quality control purposes. The worst results for quality control sampling are related to samples analysed using an aqua regia digest with ICP finish. These samples are generally part of multi element analyses and where the same sample has been analysed by one of the variations on fire assay the fire assay results has been used in preference to the ICP result.

In general, though, the performance of the various reference samples, blanks and duplicates suggest that there are no major problems in the assay program and that the assay results are representative of the mineralization sampled.

12.6 Comparison of Drillhole Sampling

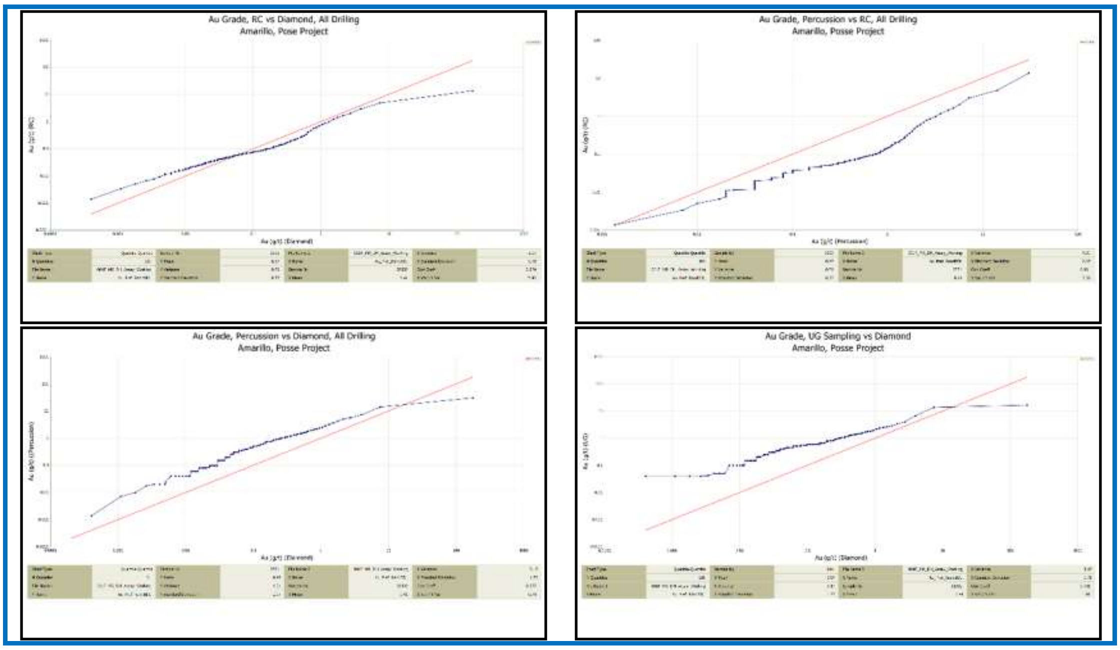
As is normal in a large extended exploration program a variety of drilling methods have been used at Posse. Due to the way the sample is returned from the drill bit to surface different drilling methods can be more or less efficient at returning samples leading to results from one drilling method being biased with respect to another method. To investigate this a series of QQ plots were constructed to compare the key drilling methods and their relationships. There were four major types of sampling investigated, Diamond Drilling, RC Drilling (face sampling hammer), Percussion Drilling and Underground Sampling. The comparisons are shown in Figure 12-13. All methods show an acceptable comparison;

- Sample results from RC drilling closely match those from Diamond drilling (top left image);
- Those from Percussion drilling over-estimate, slightly, in comparison with Diamond Drilling (lower left image);
- The Percussion vs RC samples chart show that Percussion sampling tends to over-estimate grade when compared to RC samples, (top right image) although the degree of over

estimation is slight; and

- Final chart (lower right image) shows the Underground Sampling over-estimates grade compared to Diamond Drilling. Again, the bias is low, however, Underground samples were quarantined and not used in grade modelling.

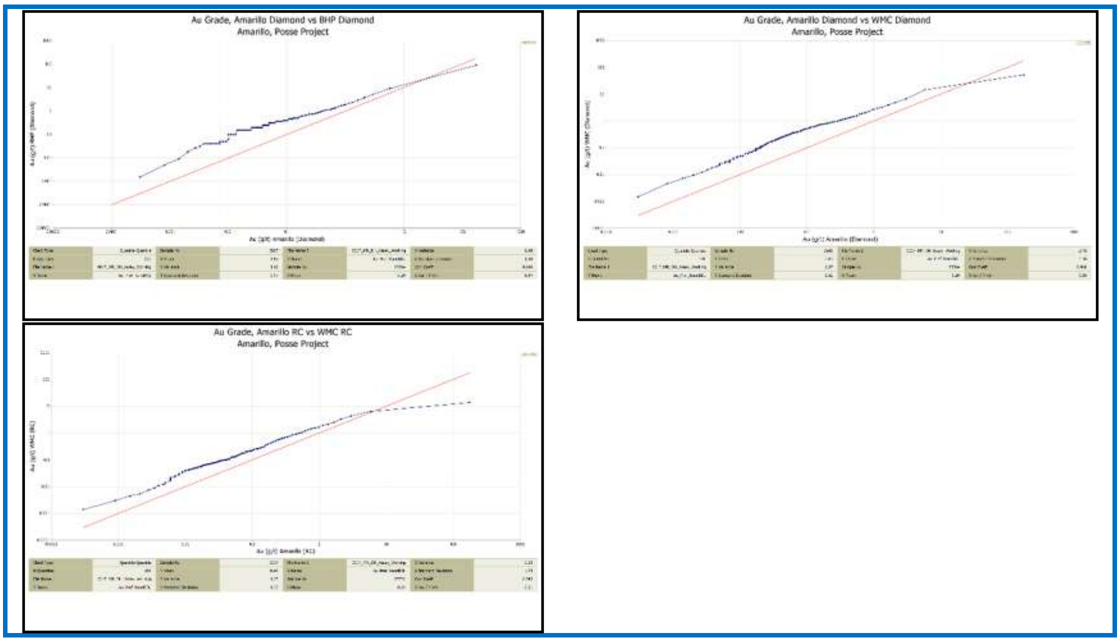
Figure 12-13: Sample Type Comparisons



The second set of images, Figure 12-14, shows comparisons between the same type of sampling and the company that carried out the drilling. All comparisons show a good relationship between datasets:

- The BHP diamond drill program shows a slight positive bias (over -estimate) in comparison with the Amarillo diamond drilling (top left image);
- The WMC diamond drilling program also shows a slight positive bias in comparison to the Amarillo diamond drilling program (top right image); and
- The WMC RC program shows a slight positive bias in comparison with the Amarillo RC drilling (bottom left image).

Figure 12-14: Sample Type Comparisons between Companies



There are no indications of significant bias between any of the sampling methods or between any of the work carried out by different companies. It should also be noted that the different drill sampling types and work by different companies is spread across the whole of the area being investigated. This mixing of sample types and phases of drilling will serve to reduce the effect of any bias on grade modelling.

12.7 Geological Risk Assessment

Following completion of the DFS in 2020 an Independent Technical Engineering ("**ITE**") consultancy reviewed the DFS to identify risks associated with the project together with work that would be needed to reduce the identified risks in order to strengthen the viability of the project. The geological risks identified, and the results obtained from the additional work recommended as part of the risk review have been critical in decisions by the relevant QP's to continue with the use of the 2020 Mineral Resource and the Mineral Reserve which flows from it.

12.7.1 Re-assay

The risk review suggested re-assaying historical holes (if available) or a confirmation drill program of at least 10% of 186 historic drill holes totalling 16,933m which affect the resource and selecting a zone in payback area (around the first 4 years of mine life) and completing a tightly spaced grade control drill program to benchmark the historical drill data. Amarillo elected to complete a combined program of re assay and confirmation drilling to provide the necessary coverage.

An initial re-assay program was carried out using ¼ core samples from retained ½ core samples. The samples were prepared at the ALS sample preparation facility in Goiânia with results determined at ALS in Lima, Peru using Fire assay for determination of Au grades (Au_AA23 (30g charge)) and a 4 – acid digest and ICP analysis of other elements (ME-MS61). A total of 1284 samples were analysed in this program with the samples being preferentially located in the portions of the resource that would be mined in the first 4 years of the proposed mine life. Subsequently a further 555 intervals representing material from the whole of the resource and also including intervals where only ¼ core remained were re-assayed. Additionally in 2019 Amarillo had re-assayed a further 556 points. In total 2,405 historic sample intervals were re-assayed. This represented 39% of the 6,119 sample intervals within the resource wireframes. Results of the re-assay program are summarised below in Table 12-12 and Figure 12-15 and Figure 12-18.

Table 12-12: Comparison between historic and re-assay values

Field Name	Minimum	Maximum	No Points	Mean	Variance	Std Dev	Wgtd Mean	Wgtd Variance	Wgtd Std Dev
Au_Hist	0.020	90.000	2,405	0.850	6.273	2.505	0.774	3.875	1.969
Au_Re-assay	0.003	39.500	2,405	0.800	2.818	1.679	0.755	2.646	1.627
Au_Diff	-0.005	32.800	2,405	-0.049	4.599	2.144	-0.019	2.812	1.677
Field Name	**CoV**	**Median**	**No Outliers**	**Geo Mean**	**Geo Std Dev**	**Ln Variance**	**Ln Std Dev**	**Sichel's T**	
Au_Hist		2.949	246	0.350	3.660	1.683	1.297	0.811	
Au_Re-assay		2.099	221	0.320	4.216	2.070	1.439	0.899	
Au_Diff		-43.397	347	0.118	4.359	2.167	1.472	0.349	

Figure 12-15: Box plots comparing Historic and re-assayed data

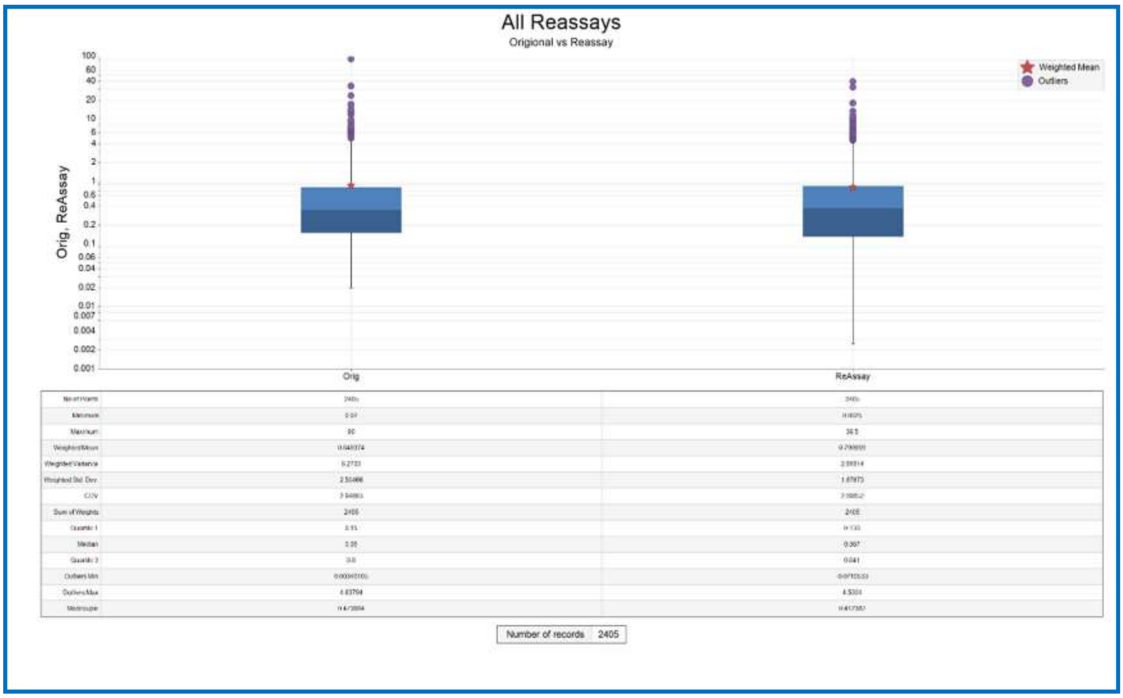


Figure 12-16: Histograms comparing Historic and re Assayed data

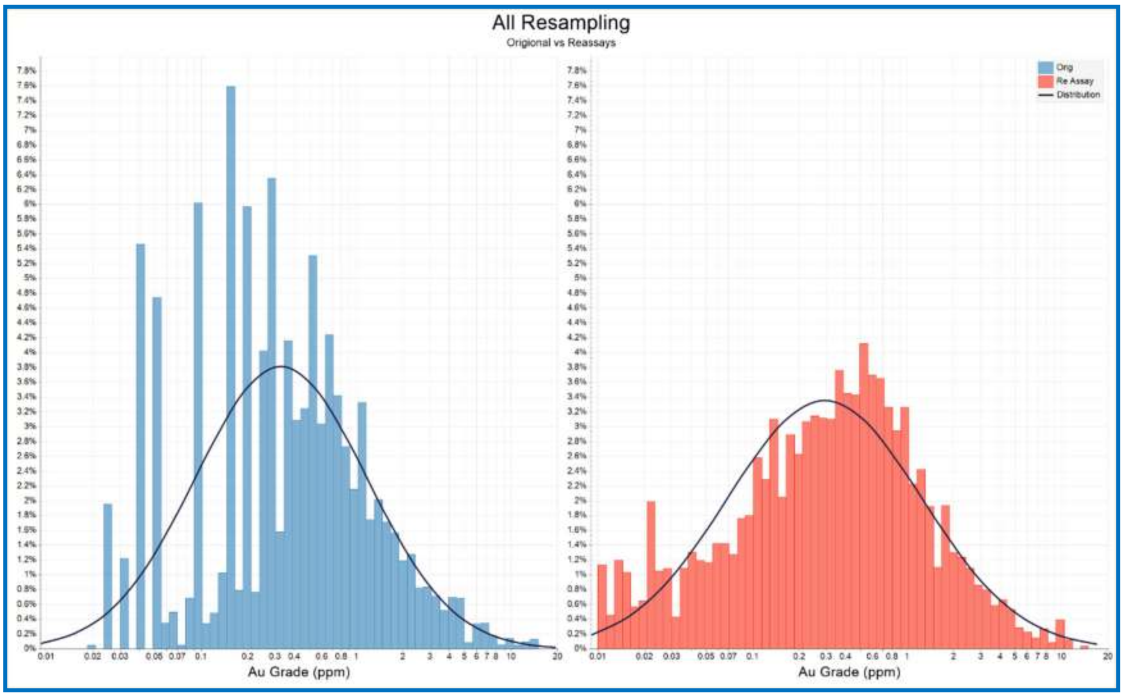


Figure 12-17: QQ plot comparing Historic and re-assayed data

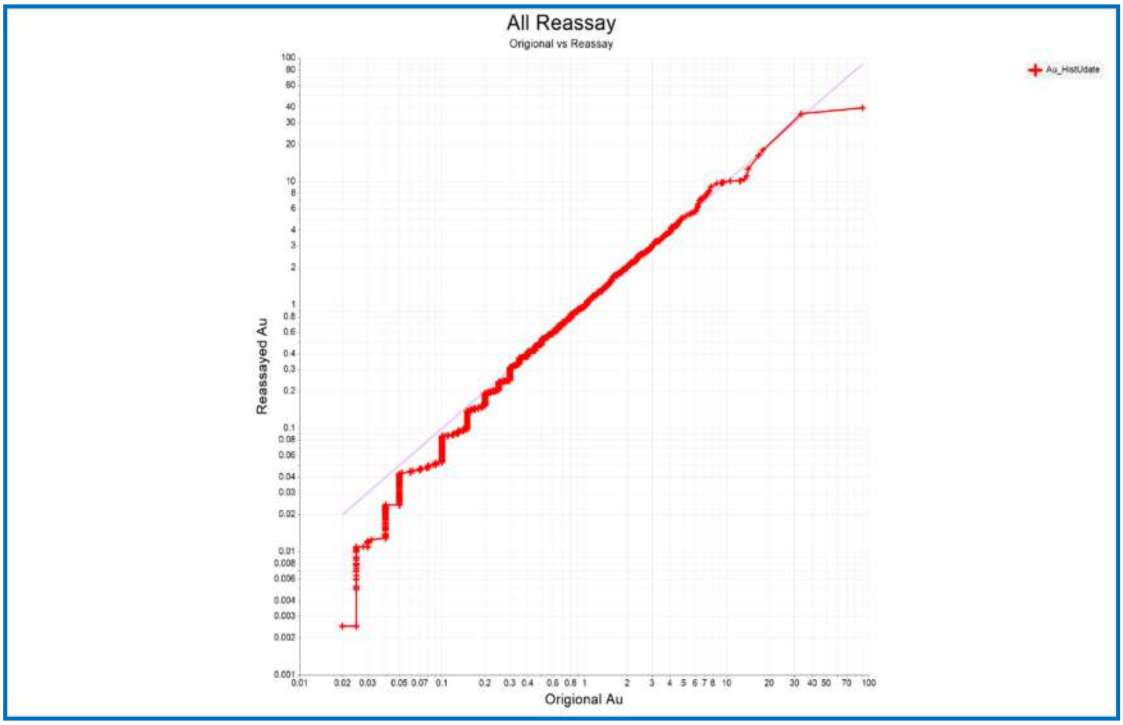
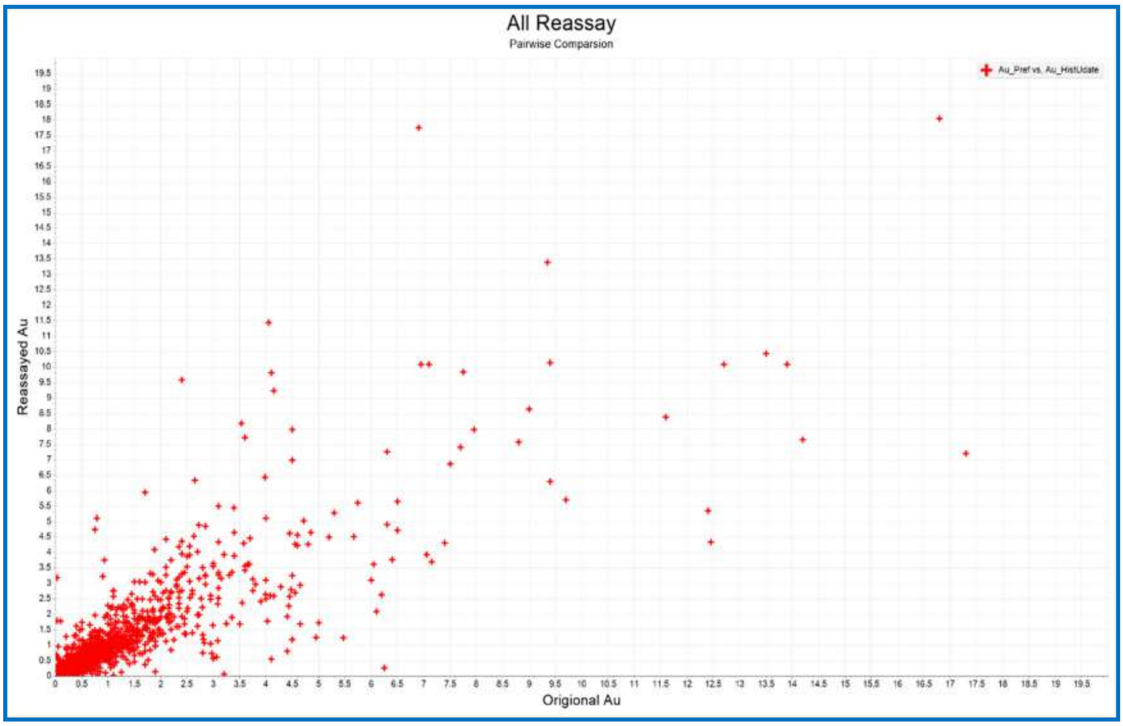


Figure 12-18: Scatterplot comparing Historic and re-assayed data



The re-assayed data set shows a slight decrease in mean and Median grade compared to the historic results but is sufficiently close that they serve to validate the historic assay grades from diamond holes. The histograms in Figure 12-16 show a variation in the shape of the histograms associated with the two data sets. Most of this variation is related to the assay detection limits of the historic data. The historic data had variable detection limits, depending on drill program, with some of the assays having a detection limit as high as 0.05g/tAu, whereas all of the re-assay data had a lower detection limit of 0.005g/tAu. The effect of the difference in detection limits can be seen in the stepping in the lower grades shown Figure 12-17.

On the basis of these results, it was recommended by the ITE group that future resource modelling should use the re-assay values but that where intervals had not been re-assayed it was appropriate to continue to use the historic assays. It would seem then that in the absence of a compelling reason to re model the deposit the historic assays as used in the 2020 resource can be considered to provide an accurate measure of the grades of the sampled intervals.

The re-assay program did not test the grades of RC holes as there were no samples available to re-assay. A review of Historic RC and Percussion grades within the model envelop with both historic diamond drilling and with modern diamond drilling,

Figure 12-19, shows that the RC and Percussion samples comprise 15% of the samples used as input to the model. The QQ plot, Figure 12-20, shows that the RC and Percussion grades are generally higher than the grades from diamond hole samples. The Box and Whisker plot, Figure 12-21, shows the historic RC and Percussion holes have higher grades than either the historic diamond holes or the diamond holes drilled by Amarillo. It can also be seen that the RC and Percussion holes are located in the very top of the resource, Figure 12-19.

Figure 12-19: Resource Envelope, with samples by hole type (Historic RC and Percussion in Red, Historic Diamond in Green, Amarillo Diamond in blue)

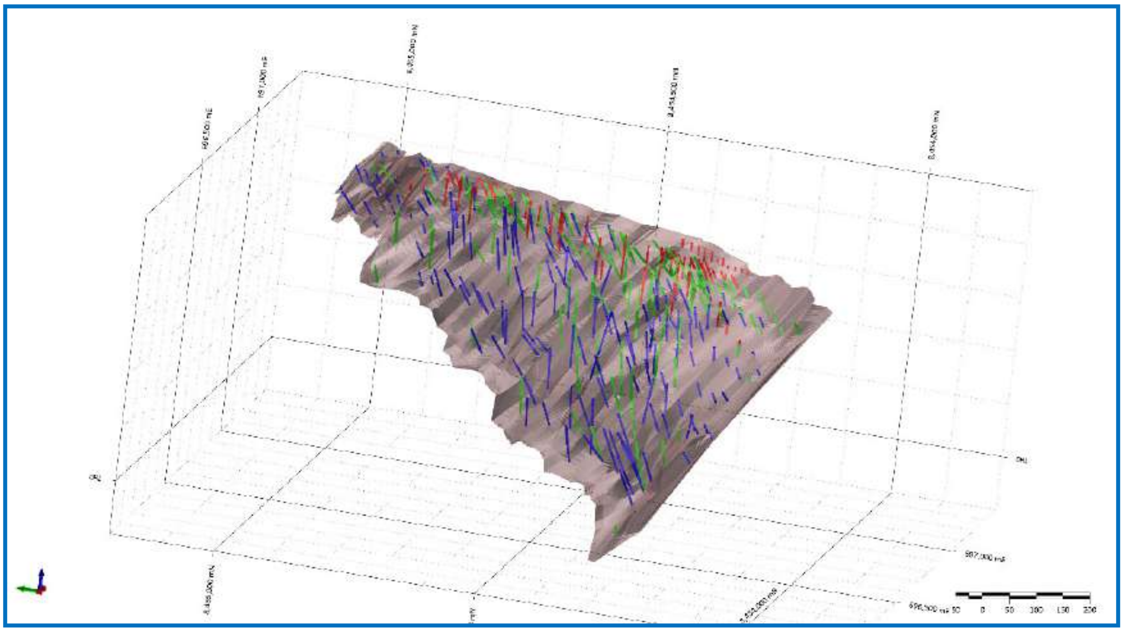


Figure 12-20: QQ Plot, Historic RC & Percussion samples vs all diamond samples

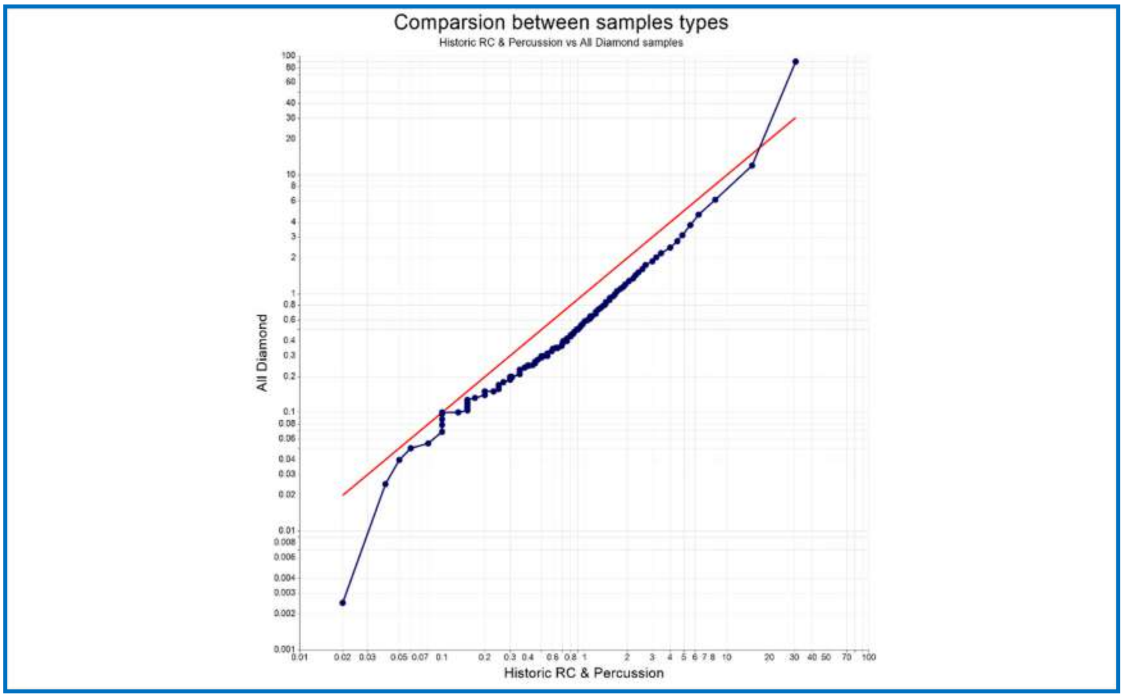


Figure 12-21: Historic RC & percussion historic diamond and Amarillo diamond samples

There are various aspects to this problem, and it may be that the historic RC and Percussion samples due to their larger size and volume variance effects, are giving a more correct measure of grade than has been obtained from the diamond holes. A review of the actual modelled data however suggests that the modelling process itself has, as expected, reduced the differences between blocks with their nearest neighbour (that is the primary input to the grade estimate) from different drillhole types. This can be seen clearly in Figure 12-22 and Figure 12-23.

Figure 12-22: Blocks estimated by historic RC & Percussion samples vs blocks estimated by diamond samples

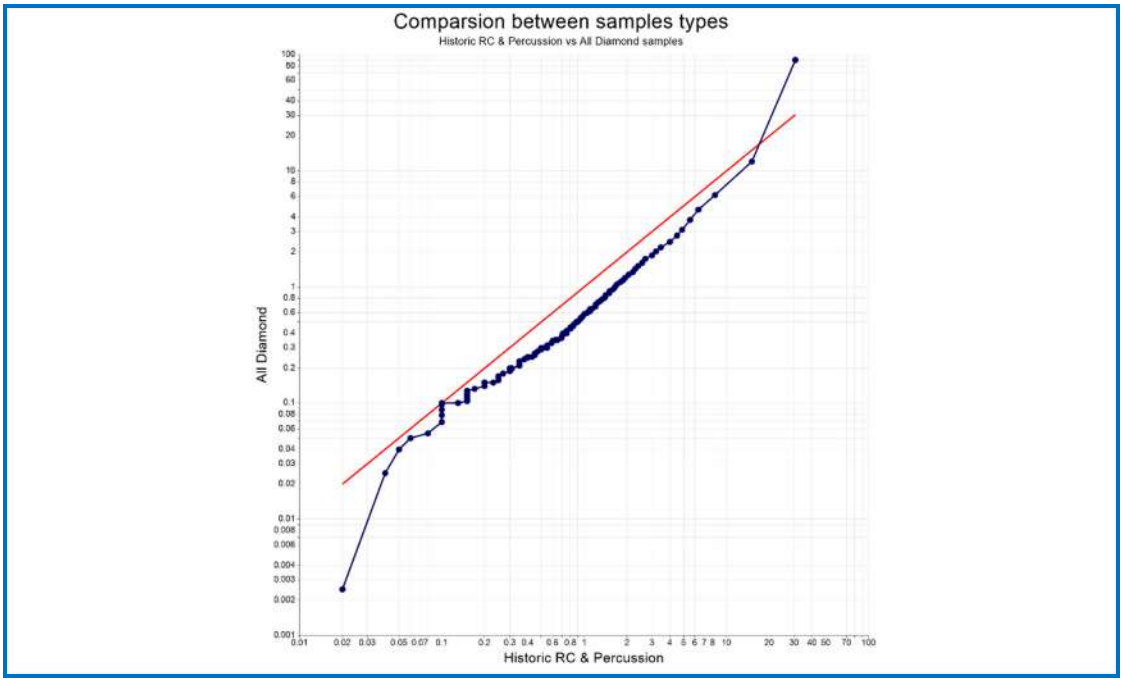
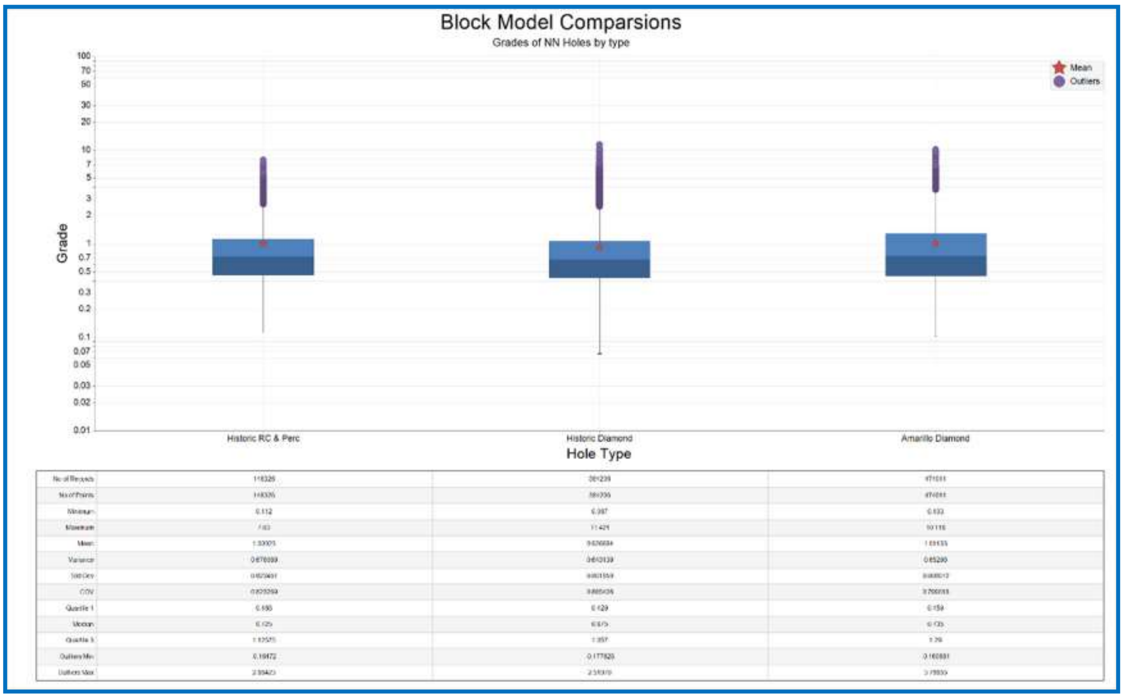


Figure 12-23: Comparison of block estimated by different sample types



On the basis of the comparisons between the different sample types contributing to block estimates it appears that any bias in the block model due to historic RC and Percussion samples being slightly positively biased (higher grade) than samples from diamond drilling is minimal. It will however be appropriate to test the areas of the model by grade control drilling as early as possible in the mine development program to confirm these results. It is noted that it is not currently possible to test these portions of the model with further drilling due to water and backfill in the pits and lack of suitable collar positions adjacent to the historic pits.

12.7.2 Density

There was concern expressed in the risk review that the density measurements used in modelling were from gamma ray logs, and that this is not normally used in the mining industry. This technique is however extensively used in petroleum exploration and development and is also one of the primary methods of density measurement used in mineral processing. In addition, there was no recognition of density variability within lithologies. The use of correct density measures is important as it can have a meaningful impact on the tonnages and hence resources and reserves.

Historically attempts had been made by Amarillo to measure density using a fluid displacement system, the results were however poor and so once the gamma ray density data became available this method was abandoned.

In the 2020 DFS this issue was examined by SRK, see Section 12.4, this work indicated that based on the 129 samples tested the SG values derived from the gamma ray logging were within 2.5% of the values derived from (laboratory based) density measurement by fluid displacement.

To address the issue raised by the risk review samples sent by Amarillo for re-assay were also subjected to a laboratory-based density measurement by fluid displacement. A total of 588 measurements were made by ALS on material supplied by Amarillo. The spread of material was designed to cover the extent of the Posse resource when combined with both the SRK 2019 and Weatherford 2011 (gamma) measurements, Figure 12-24. The 4,035 1m composites of density data derived from the Weatherford gamma density readings were then compared on a rock type basis with the SRK/Amarillo density measurements, Figure 12-25 and Table 12-13.

Figure 12-24: Density measurements used for modelling

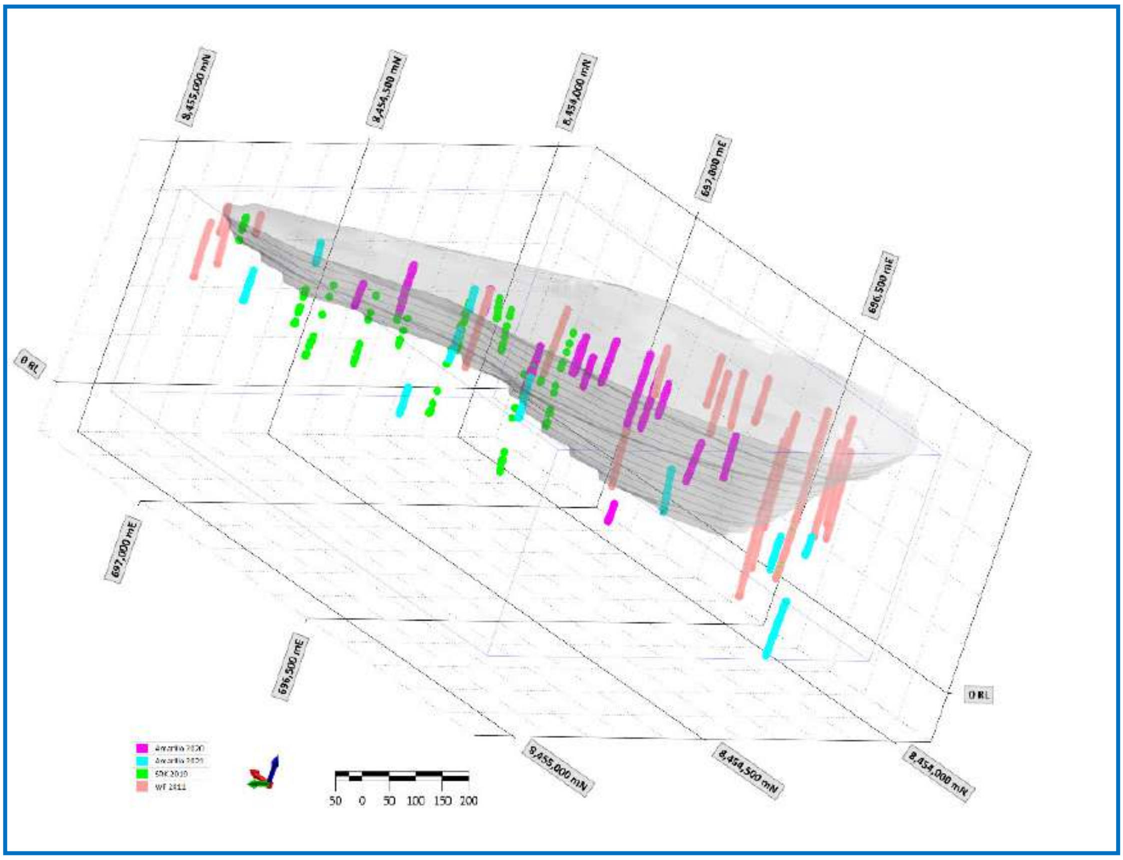


Figure 12-25: Comparison of density readings by rock type

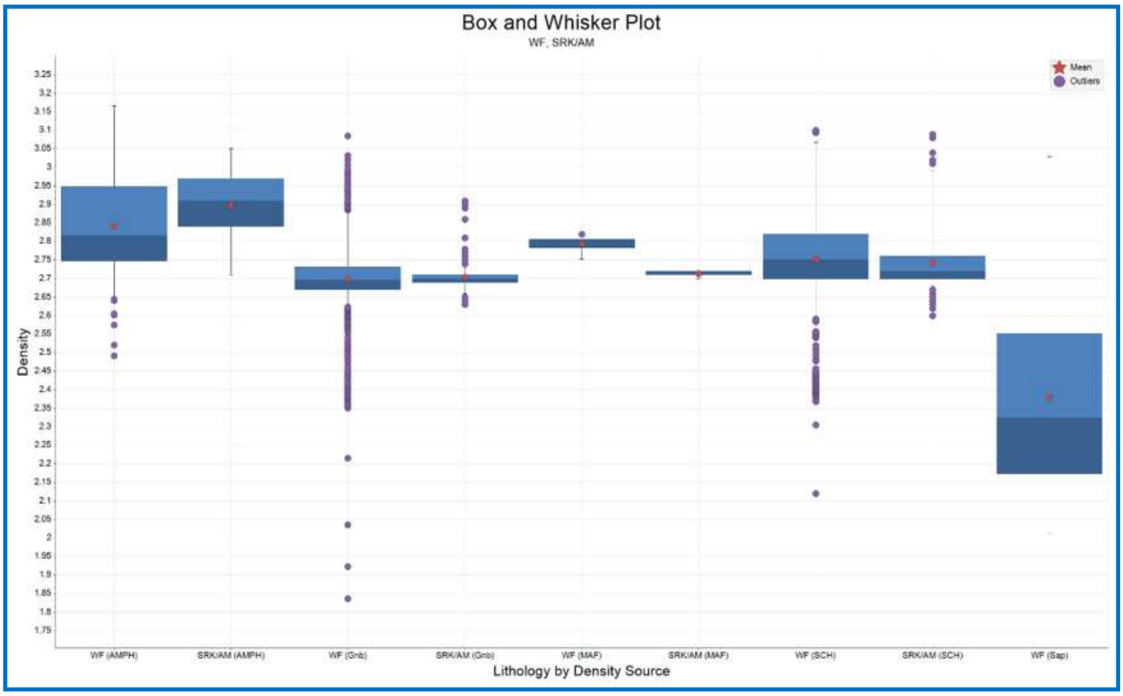


Table 12-13: Comparison of density readings by rock type[1]

Statistic	AMPH		Gnb		MAF		SCH		SAP
	WF	SRK/AM	WF	SRK/AM	WF	SRK/AM	WF	SRK/AM	WF
No of Records	4,387	717	4,387	717	4,387	717	4,387	717	4,387
No of Points	595	67	2,163	126	5	3	1,272	521	20

Statistic	AMPH		Gnb		MAF		SCH		SAP
	WF	SRK/AM	WF	SRK/AM	WF	SRK/AM	WF	SRK/AM	WF
Minimum	2.492	2.710	1.836	2.630	2.753	2.700	2.121	2.600	2.014
Maximum	3.167	3.050	3.085	2.910	2.819	2.720	3.101	3.090	3.029
Mean	2.842	2.899	2.700	2.705	2.794	2.713	2.752	2.743	2.379
Variance	0.014	0.008	0.008	0.002	-0.001	0.000	0.014	0.005	0.086
Std Dev	0.118	0.088	0.088	0.043	0.026	0.012	0.117	0.074	0.293
COV	0.041	0.030	0.033	0.016	0.009	0.004	0.043	0.027	0.123
Quartile 1	2.748	2.840	2.671	2.690	2.784	2.710	2.700	2.700	2.174
Median	2.817	2.910	2.696	2.700	2.806	2.720	2.752	2.720	2.325
Quartile 2	2.948	2.970	2.732	2.710	2.806	2.720	2.820	2.760	2.552
Outliers Min	2.645	2.561	2.624	2.660	2.664	2.643	2.592	2.676	1.917
Outliers Max	3.614	3.091	2.885	2.740	2.812	2.722	3.086	3.005	3.518
Medcouple	0.267	-0.120	0.169	0.000	-0.425	-0.500	0.129	0.333	0.199

(1) Columns in grey have too few data points to be statistically meaningful.

There is a close match between the Weatherford and the SRK/Amarillo data across each of the rock types. In general, the mean and median of the Weatherford results are slightly below the mean and median of the SRK/Amarillo results, it is also obvious from the box plots that there were a number of outlier results that have affected the Weatherford statistics. Significantly the standard deviation of both sets of density results for each rock type is low indicating that there is unlikely to be any significant internal variation in the density of each of the rock types. Check modelling of the Posse deposit, which modelled the SG as one of the inputs showed only minor variation in the overall tonnages of the resource. It is considered that density variation is unlikely to be an issue when the Posse deposit is mined, and it does not provide a compelling reason to update the 2020 Mineral Resource.

12.7.3 Size of Historic Underground Development

The risk review noted that the mined-out underground drift was not accounted for in the resource/reserve model. The underground drift development was recognised from historic plans however, it was not accounted for in modelling as no plans detailing this development had been located. When this issue was raised a search of archival material at the project site turned up several historic plans. These consisted of an historic pit plan showing the location of the mine shaft and three plans showing the underground development. There was sufficient information in the way of grid lines for the four sets of data to be related together and tied to the WGS84 coordinate system used for resource modelling, Figure 12-26 through Figure 12-28. The plans were digitised, and a wireframe produced which allowed the volume and tonnage of mined material to be determined. The volume of the underground shaft and development was 3,667m^3 and the tonnage 10,048t. This amount is not material to the resource which is in excess of 30Mt.

Figure 12-26: Historic Pit plan used to locate UG mine shaft (The plan is georeferenced to the current coordinate system with the shaft location shown as a circle in the top right)

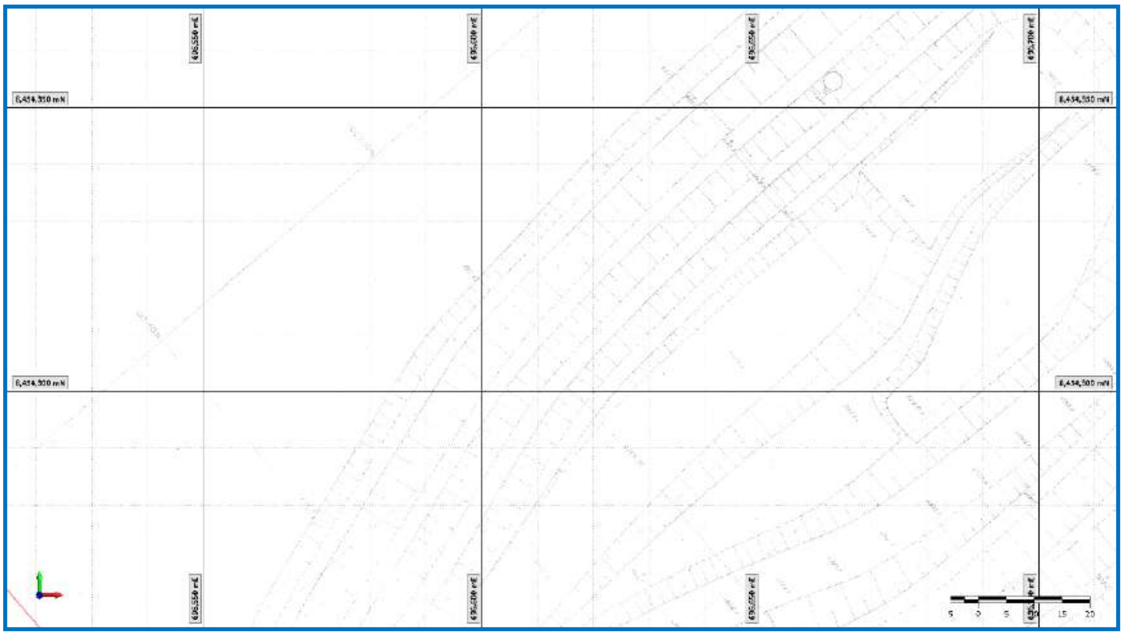


Figure 12-27: UG sampling locations

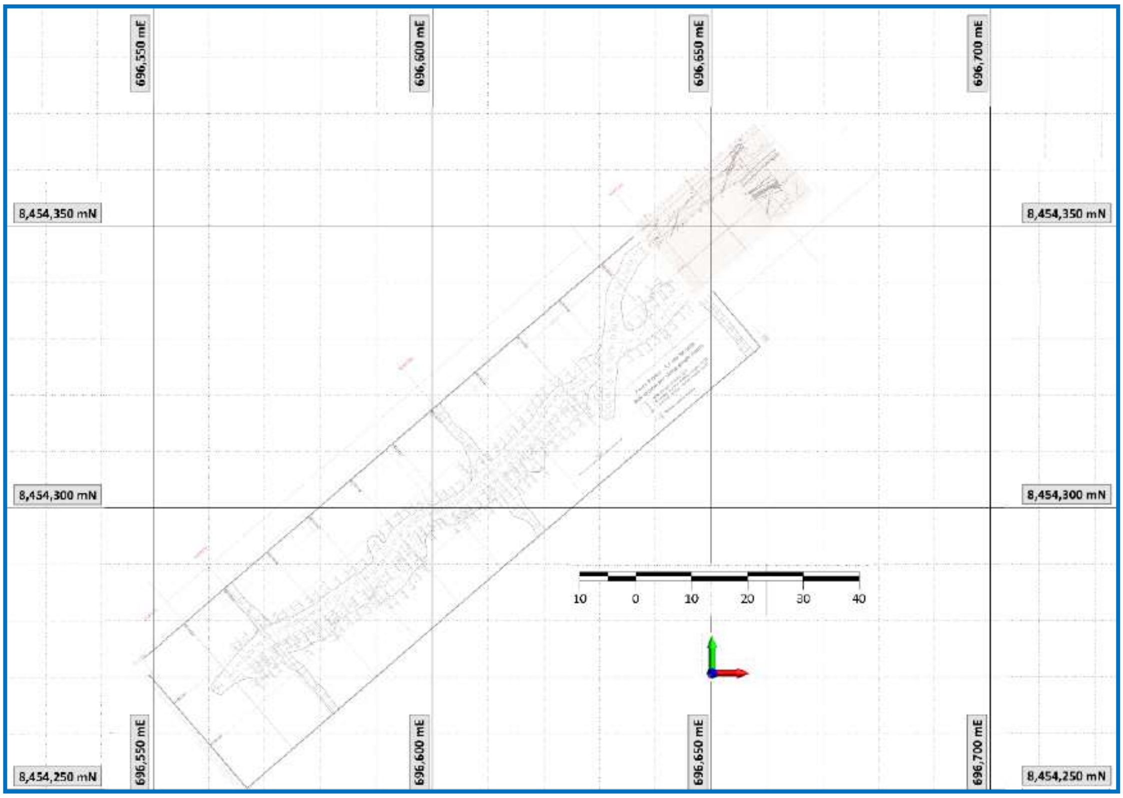
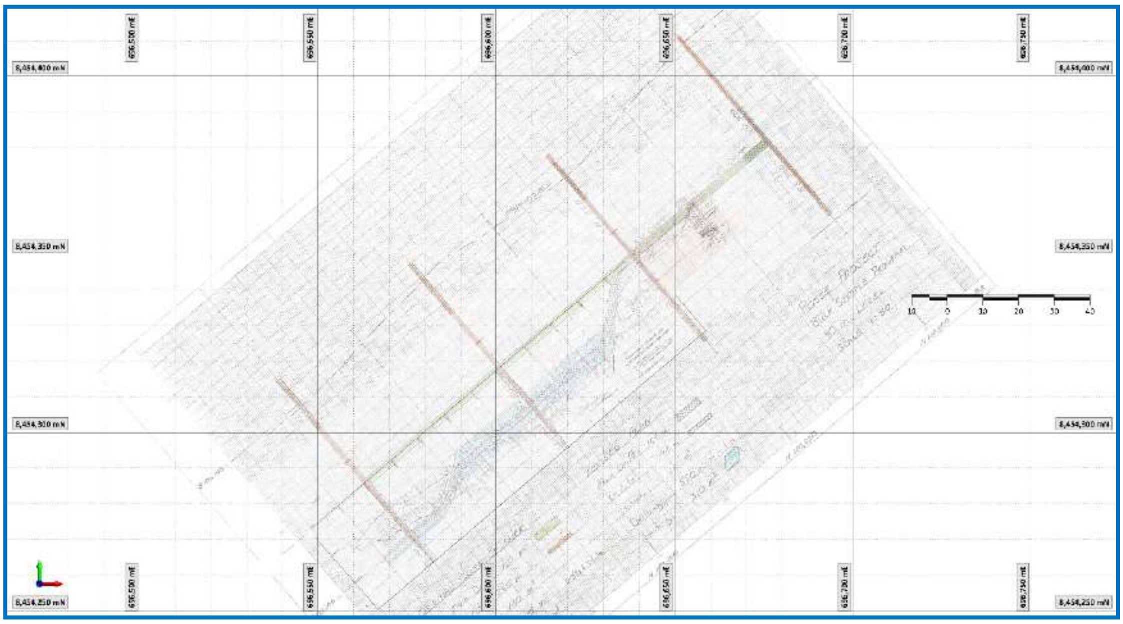


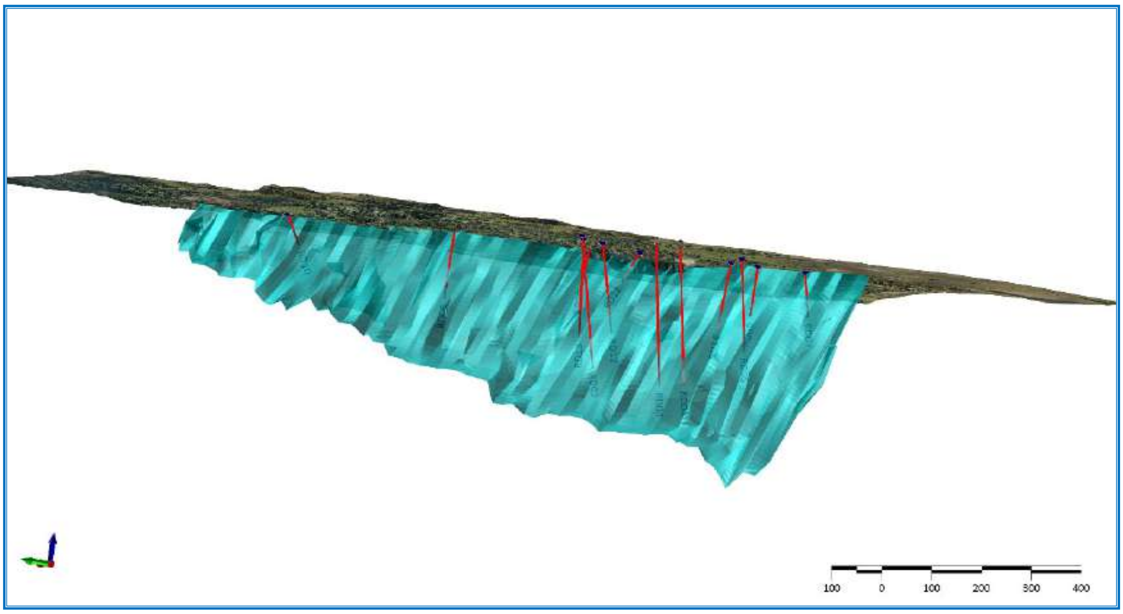
Figure 12-28: UG sampling built plans



12.8 Additional Drilling

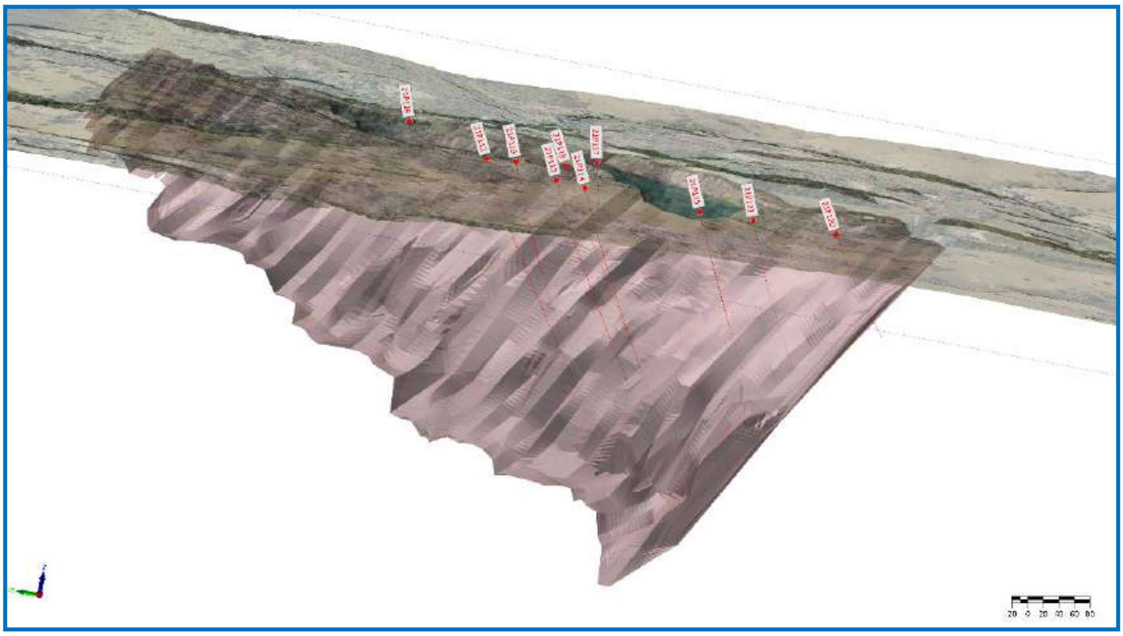
The 2021 drilling was designed to target specific areas of the mineralisation which were considered poorly sampled. The program of 13 holes was designed to target portions of the resource predominantly defined by historic holes which were not available for re-assay, Figure 12-29.

Figure 12-29: Proposed 2021 additional drilling



Once drilling started the program was reduced to a total of 10 holes mainly due to issues with targeting the desired pierce points from available drilling locations. This particularly affected the northern planned holes, Figure 12-30. The planned and actual holes are shown together in Figure 9-4. The actual pierce points of the 2021 holes with the model closely matched the planned locations.

Figure 12-30: Actual 2021 drilling



Once drilled the ore zone as defined by the assays from the 2021 holes was compared with the ore zone, as defined in the 2020 Mineral Resource. Unsurprisingly, there were rather obvious differences between the modelled data (the 2020 Mineral Resource) and the detail of the ore zone as revealed by the assays from the 2021 drilling as shown in Table 12-14 and Figure 12-31 - Figure 12-35.

Table 12-14: 2021 Drilling expected and actual results

Domain	01				05				10			
Hole	Modelled		Calculated		Modelled		Calculated		Modelled		Calculated	
	Thickness	Grade	Thickness	Grade	Thickness	Grade	Thickness	Grade	Thickness	Grade	Thickness	Grade
21P112	56.00	0.984	58.00	0.599	34.00	1.801	10.00	0.884	5.00	3.764	5.00	1.143
							6.00	0.945			4.00	1.017
21P113	69.00	1.732	56.00	1.161	45.00	2.254	8.00	0.703	30.00	2.900	6.00	1.347
							10.00	1.686			8.00	2.450
							30.00	1.352				
21P114	36.00	2.098	52.00	1.570	23.00	2.963	10.00	1.701	16.00	3.932	8.00	1.985
							27.00	2.140			11.00	3.185
											11.00	1.742
21P115	31.00	0.966	26.00	0.896	16.00	1.440	22.00	1.035	12.00	1.496	8.00	1.344
21P116	43.00	0.789	28.00	0.260	9.00	2.603	12.00	1.507	7.00	2.660	10.00	1.663
21P116			18.00	1.058								
21P117	63.00	0.769	44.00	0.786	42.00	0.997	20.00	1.103	2.00	2.280	7.00	1.320
							5.00	1.587			5.00	1.587
21P118	70.00	1.018	9.00	0.333	48.00	1.273	25.00	1.294	34.00	1.940	7.00	2.286
21P118			52.00	0.966			7.00	1.373			5.00	1.446
							5.00	0.502			5.00	1.583
21P119	52.00	0.956	54.00	0.734	36.00	1.295	7.00	0.733	5.00	2.238		
							8.00	1.486				
21P120	11.00	0.359	12.00	0.129	3.00	0.664						
21P121	27.00	0.791	26.00	1.000	10.00	1.504	15.00	1.569	7.00	1.765	5.00	3.058

Figure 12-31: Results holes 21P112 and 21P113

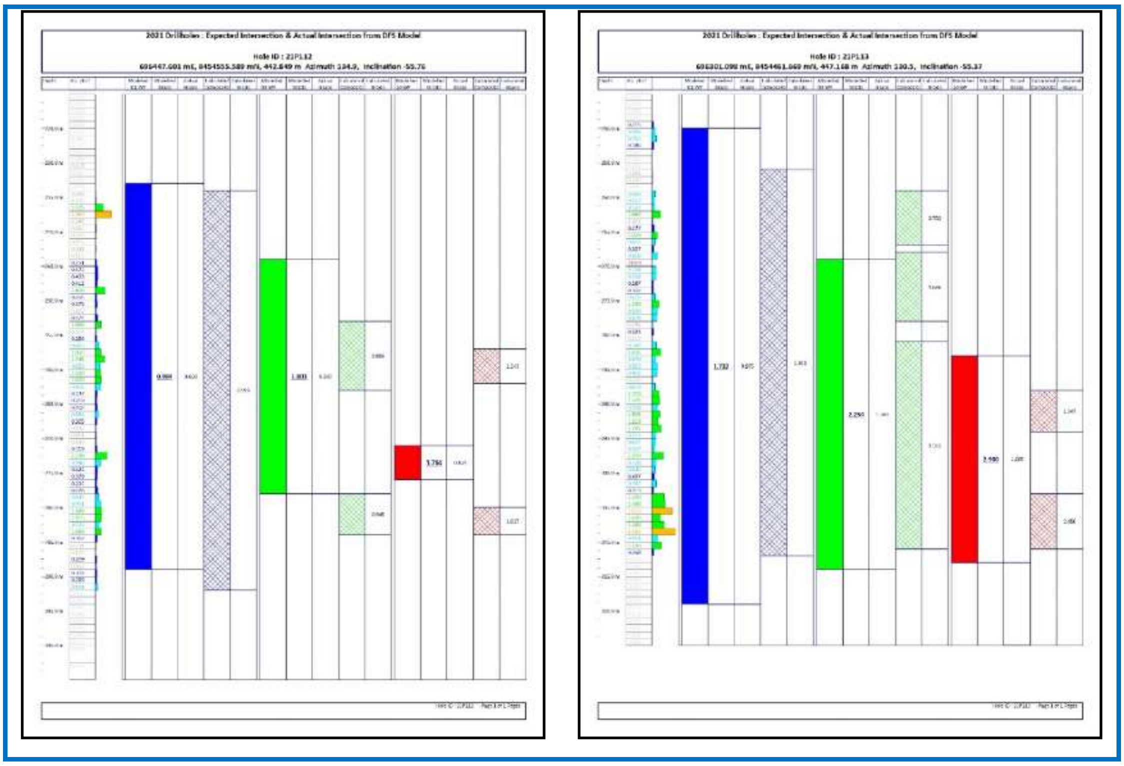


Figure 12-32: Results holes 21P114 and 21P115

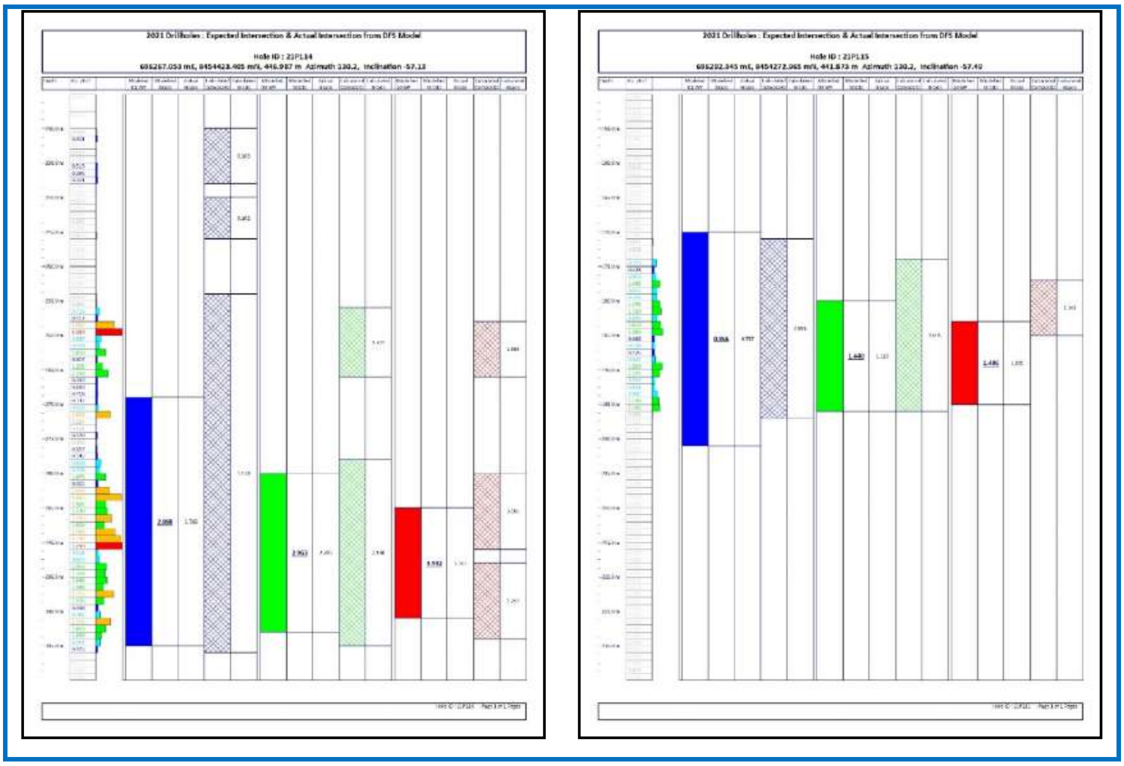


Figure 12-33: Results holes 21P116 and 21P117

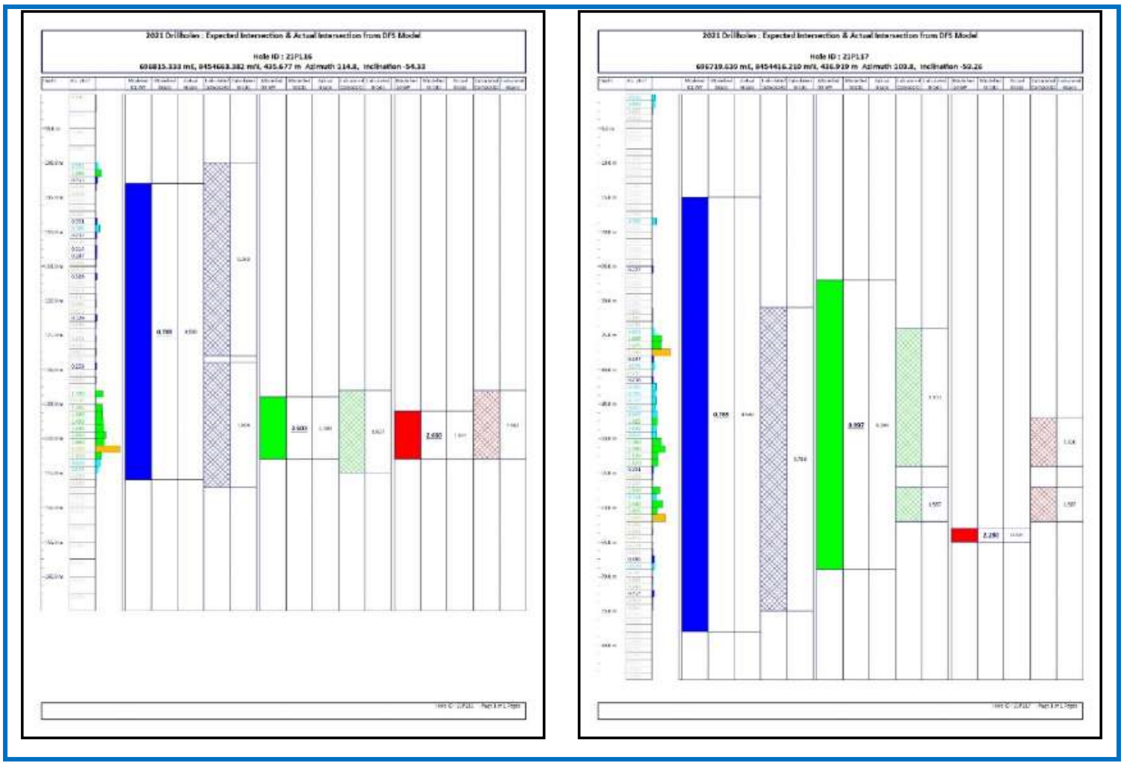


Figure 12-34: Results holes 21P118 and 21P119

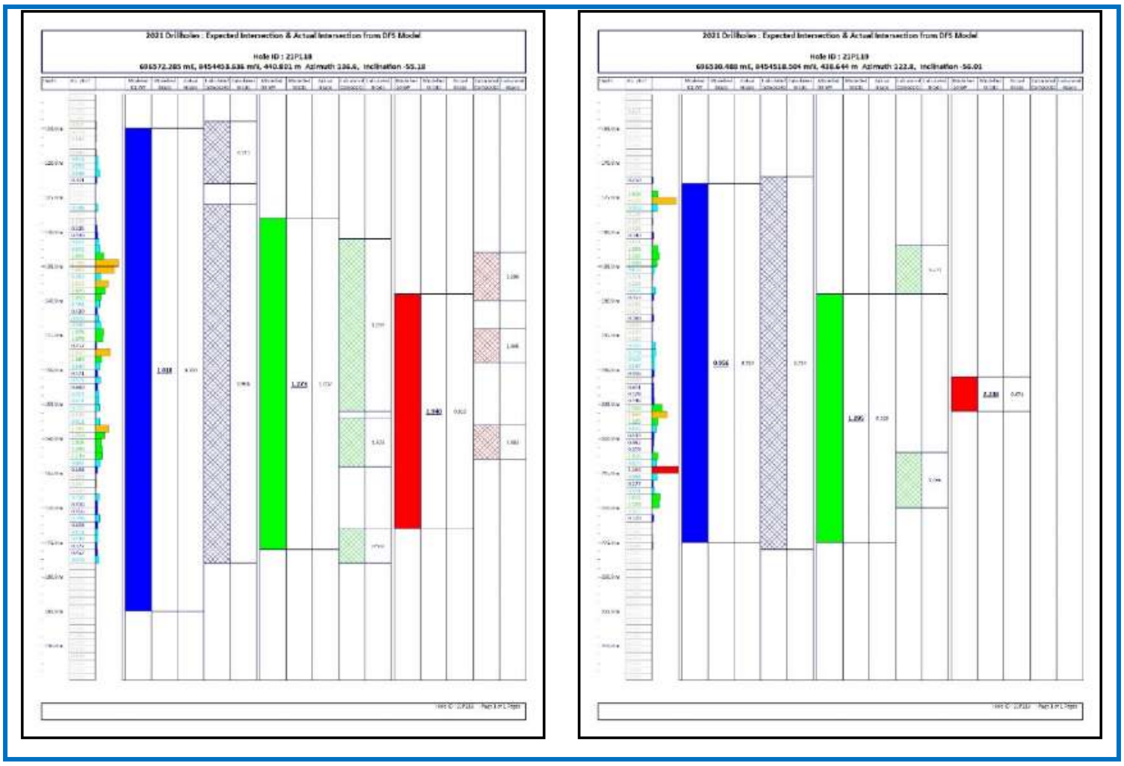
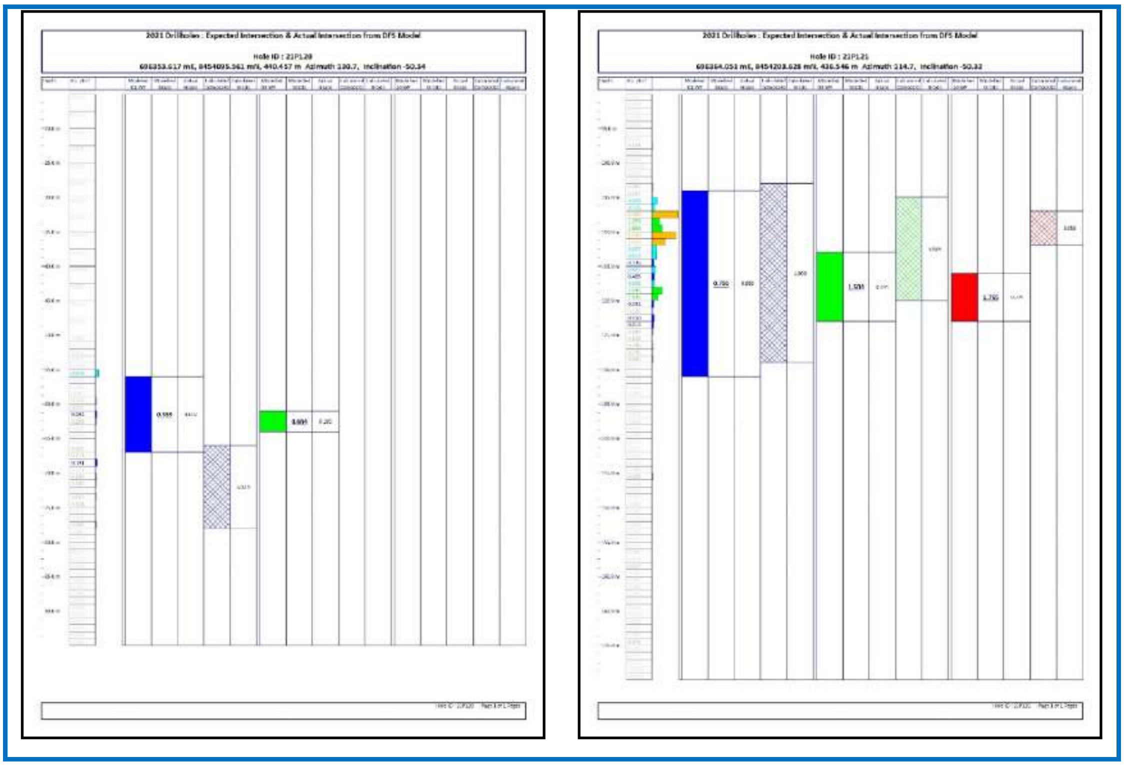


Figure 12-35: Results holes 21P120 and 21P121



A comparison was made on the effect of including the 2021 drillholes in the DFS model by evaluating the statistics for the blocks in the 2020 Mineral Resource which would have been penetrated by the 2021 holes and the actual results obtained from the 2021 holes. These statistics are shown in Table 12-15.

Table 12-15: 2021 Drilling expected and actual results

Statistics	Domain	Mean	Median	SD	RSD	Sichels Mean
DFS holes	01	0.893	0.379	2.070	2.318	0.818
With 2021 holes	01	0.889	0.380	2.042	2.297	0.821
% Difference	01	-0.448	0.264	-1.353	-0.906	0.367
DFS holes	05	1.648	0.900	2.888	1.752	1.525
With 2021 holes	05	1.626	0.890	2.843	1.749	1.513
% Difference	05	-1.335	-1.111	-1.558	-0.171	-0.787
DFS holes	10	2.208	1.348	3.377	1.529	2.126
With 2021 holes	10	2.169	1.326	3.325	1.533	2.137
% Difference	10	-1.766	-1.632	-1.540	0.262	0.517

The statistics suggest the effect of including the 2021 drilling in the resource model will be minimal and do not indicate a need to update the 2020 Resource.

12.9 Other Modelling

In addition to check modelling conducted as part of the follow up to the risk analysis there have been two other models produced for the Posse deposit one by Whittle Consulting and the other by VMG Consultoria e Soluções Ltda.

12.9.1 Whittle Consulting Model

Whittle Consulting has produced a grade tonnage estimate of the Posse project as part of a research project to examine the feasibility of utilising stochastic modelling (conditional simulation) to model a resource which could them be used as input to their optimisation routines to provide improvements in optimisation methodology together with information about the likelihood of the modelled (optimised) outcome. While this work is still in progress and has not been provided to Amarillo, Mr K Whitehouse the QP for the Mineral Resource, was asked to review a draft of the report. The draft report contains results of a stochastic grade evaluation of the mineralisation at Posse, the Etype model. The input data is the set of assays and wireframes used for the 2020 Mineral Resource. The modelling however used different software ("**GSLIB**") and stochastic modelling. The results of this exercise have generated an Etype model with grade tonnage which is very similar to that which forms the basis of the 2020 resource as shown in Table 12-16.

Table 12-16: 2021 Drilling expected and actual results

2020 Mineral Resource									
Category	Ore Tonnes	Vol(m^3)	Au (g/t)	Waste tonnes	Vol(m^3)	Au (g/t)	Total Tonnes	Vol(m^3)	Au (g/t)
1_Hi	8,470,547	3,052,652	2.28			0.00	8,470,547	3,052,652	2.28
2_Med	9,795,414	3,532,612	0.96	1,756	622	0.32	9,797,170	3,533,234	0.96
3_Lo	24,589,299	8,899,568	0.57	14,070,647	5,097,494	0.26	38,659,947	13,997,062	0.46
(blank)			0.00	7,326	2,640	0.00	7,326	2,640	0.00
Grand Total	**42,855,260**	**15,484,832**	**1.00**	**14,079,729**	**5,100,756**	**0.26**	**56,934,989**	**20,585,588**	**0.82**
Whittle E Type Model									
Category	Ore Tonnes	Vol(m^3)	Au (g/t)	Waste tonnes	Vol(m^3)	Au (g/t)	Total Tonnes	Vol(m^3)	Au (g/t)
Hi_10	8,633,981	3,105,862	2.17			0.00	8,633,982	3,105,862	2.17
Med-5	10,018,894	3,607,057	1.03	7,704	2,750	0.29	10,026,598	3,609,807	1.03
Lo_1	25,538,212	9,218,501	0.57	12,971,196	4,692,145	0.27	38,509,407	13,910,646	0.47
Grand Total	**44,191,088**	**15,931,420**	**0.99**	**12,978,899**	**4,694,895**	**0.27**	**57,169,987**	**20,626,315**	**0.82**
Differences									
Category	Ore Tonnes	Vol(m^3)	Au (g/t)	Waste tonnes	Vol(m^3)	Au (g/t)	Total Tonnes	Vol(m^3)	Au (g/t)
(Hi_10/1_Hi)-1	1.9%	1.7%	-0.7%	0.0%	0.0%	0.0%	1.9%	1.7%	-4.7%
(Med_5/2_Med)-1	2.3%	2.1%	7.4%	338.8%	342.1%	-7.6%	2.3%	2.2%	7.3%
(Lo_1/3_Lo)-1	3.9%	3.6%	-0.8%	-7.8%	-8.0%	0.9%	-0.4%	-0.6%	1.3%
Grand Total	**3.1%**	**2.9%**	**-1.2%**	**-7.8%**	**-8.0%**	**1.0%**	**0.4%**	**0.2%**	**0.7%**

The results of the Whittle Consulting exercise would suggest it is appropriate to continue to use the 2020 Mineral Resource as part of this CPR.

12.9.2 VMG 2022 Model

VMG Consultoria e Soluções Ltda ("**VMG**") of Belo Horizonte, Brazil was asked by Amarillo Gold, Brazil to develop a new block model of the Posse deposit which considers all re-assays, the 2021 drilling and the additional density data available. This model and associated report is still considered to be preliminary in nature and does not supersede the 2020 Mineral Resource, it does however provide a useful comparison with the 2020 Mineral Resource model especially as it has been generated using only data in the drillhole database and what is presumably the current Lidar surface. It is therefore independent of the 2020 Mineral Resource. As VMG have not visited the Posse site and completed all due diligence that they would normally conduct they have re-classified the resource as Indicated and Inferred Mineral Resources only. No extensive analysis or comparison of the wireframes used, or the method of estimating density has been made by the QP and no opinion is offered on the accuracy of the model. Nevertheless, the figures reported for the grade tonnage estimate at a 0.35g/tAu cut off are shown in Table 12-17 below.

Table 12-17: VMG 2022 grade tonnage estimate

Zone	Volume	Tonnes	Density	Au	Oz	Au_Cut	Cut_Oz
Soil	175,535	175,535	1.24	1.24	7,011	1.16	6,521
Saprolite	437,495	437,495	1.23	1.23	17,281	1.14	16,039
Fresh Rock	31,647,245	31,647,245	1.17	1.17	1,188,863	1.07	1,091,769
Total Indicated	**32,260,275**	**32,260,275**	**1.17**	**1.17**	**1,213,155**	**1.07**	**1,114,328**
Soil	23,836	41,951	1.76	0.68	916	0.68	916
Saprolite	13,051	25,058	1.92	0.63	507	0.63	507
Fresh Rock	4,758,370	12,895,183	2.71	0.90	371,082	0.88	366,011
Total Inferred	**4,795,257**	**12,962,192**	**2.70**	**0.89**	**372,506**	**0.88**	**367,435**
Total	**16,800,805**	**45,222,467**	**2.69**	**1.09**	**1,585,661**	**1.02**	**1,481,763**

From the volume and tonnage of the VMG model is larger than the 2020 Mineral Resource while the density and grade are slightly lower. Indeed, the density of fresh rock seems as though it may be a little low. The Cut Ounces of 1.4Moz is larger than the 2020 Mineral Resource, this is a function of the increase in volume. See Table 12-18, below. Both sets of data are reported at a 0.35g/tAu cut off. Comparing the increase in volume and hence ounces shown in the VMG model with the 2020 Mineral Resource suggests that if the volumes were the same the ounces would be very similar.

Table 12-18: VMG 2022 grade tonnage estimate

Model	Volume	Tonnes	Density	Au	Oz	Au_Cut	Cut_Oz
VGM grade tonnage	16,800,805	45,222,467	2.69	1.09	1,585,661	1.02	1,481,763
2020 resource	11,717,964	32,475,844	2.77	1.11	1,157,917	1.10	1,150,706
% difference	43.38%	39.25%	-2.92%	-1.71%	36.94%	-7.44%	28.77%

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

Prior to 2011, metallurgical testing of material from the un-oxidised portions of the Posse mineralization had given variable and conflicting results. The poor results reviewed in the 2010 Mineralization Report (Hoogvliet & Whitehouse 2010) were attributed to a lack of understanding of the gold-telluride chemistry of the Posse mineralization.

Coffey Mining's work of 2011 provided an understanding of the processes necessary to maximise extraction of gold-based processes required to deal with telluride ores as often undertaken in the Western Australian gold fields. A pre-oxygenation step of finely ground ore followed by leaching at high pH levels of nominally 12 resulting in high extractions at acceptable reagent consumption.

Subsequent test work programs have proven that the underlying flowsheet is viable and has investigated a number of parameters which influence extraction including pre-aeration/oxygenation, pH, grind size, cyanide concentration, leach residence and temperature. This work has investigated the variability of the ores as well as the significant domains originally investigated in the Coffey 2011 work.

Subsequent programs have similarly investigated flowsheet elements so as to allow them to be defined for future design. That is, the derivation of process design criterion.

Test work conducted prior to 2015 has been summarised and assessed in earlier NI 43-101 reports. Reference is made to these filings and detail discussion of these earlier test work programs is not provided herein.

The final test work programs were completed in 2020. No additional metallurgical test work has been undertaken since that time and consequently, the interpretation of test work has not changed subsequent to the 2020 NI 43-101 released by Amarillo.

13.2 History of Test Work

Early work on the Posse ores was managed by Western Mining Corporation ("**WMC**"), Barrick and more recent programs managed by Amarillo Gold. The earlier Amarillo work conducted by Funmineral (Goias state laboratory), Testwork Desenvolvimento and a series of later test work programs conducted by ALS Metallurgy, a part of the ALS Global group and formerly known as Ammtec.

The most recent and significant programs responsible for defining the proposed flowsheet and conducted on behalf of Amarillo are:

- **2011:** Conducted by Ammtec (ALS) under the direction of Coffey Mining on behalf of Amarillo. Ammtec report A13025 summarising the test work outcomes. This program investigated various leaching conditions to deal with the telluride ores and included some comminution work;
- **2012:** Conducted by ALS under the direction of Coffey Mining on behalf of Amarillo. This work was undertaken to improve confidence in the process route, but also investigated some alternatives. This work was instrumental in negating flotation and gravity unit processes from the flowsheet. Summarised by ALS report A13954;
- **2013:** Comminution testing of waste samples adjacent to ore zones was conducted by ALS

under the direction of Coffey Mining on behalf of Amarillo. One Footwall composite and one Hanging Wall composite being evaluated. Summarised by ALS report A14536;

- **2013:** A pilot test using an enhanced oxygenation system was conducted by ALS under the direction of Coffey Mining on behalf of Amarillo. This work summarised by ALS report A14560;
- **2015:** Kinetic AMD test work conducted by Coffey Mining on request of Amarillo. Summarised by Coffey report MINEWPER00988AD;
- **2017:** Conducted by ALS initially under the management of Amarillo but later under the combined management of Amarillo with input from Aurifex Pty Ltd ("**Aurifex**"). This program further investigated flowsheet options but focussed on the oxygenation-leach route. Other test work as required to define process criteria being included in these programs. Summarised by ALS report A18001;
- **2019:** Conducted by ALS under the combined management of Aurifex and Amarillo. This program investigated variability behaviour for samples selected to represent various area of the resource and provide a basis for extraction forecasting. Work was also conducted to confirm previous process criteria as required for process design development. This program summarised by ALS report A19476;
- **2019:** SGS Geosol undertook comminution and preparation of sample required for other programs. Summarised by SGS Geosol data sheets and JKTech report per Job No: 19004/P16 (SGS Geosol Brasil); and
- **2018:** Outotec investigating thickening and filtration characteristics. Summarised by three Outotec reports S213TA, 318437 and 326264. Samples for this work being prepared by ALS as part of the A18001 and A19476 programs.

Amarillo have also requested a number of vendor suppliers and other laboratories undertake specific work for aspects such as tailings and waste characterization and as requested by Ausenco Limited as part of their flowsheet development. Amarillo managed work described herein includes:

- **2019:** Filtration test work conducted by ANDRITZ Brasil evaluating tailings filtration characteristics and providing equipment sizing and selection. Summarised by ANDRITZ report 31 October 2019, "Amarillo Gold – Rejeito de Ouro";
- **2019:** Filtration test work conducted by Brasfelt (data sheets only provided); and
- **2019:** Filtration test work conducted by TEFSA. Summarised by TEFSA report "***HLT Amarillo Gold HLT-07***", 17 October 2019.

13.3 Sample Collection and Representivity

13.3.1 Coffey Mining Samples

Reference is made to the samples used for the initial Coffey test work program of 2011 and follow-up work managed by Coffey. Whilst these programs are only briefly referenced herein, having been described per previous NI 43-101 filings, samples utilized by the Coffey programs have been carried forward to later programs that are described in more detail.

At the time of sample collection in 2011, Coffey and Amarillo agreed, based on known geology and mineralogical test work, that the most suitable domaining would be based on three domains:

- Foot wall ("**FW**"), which was viewed to make up around 10% of the deposit;
- Main ("**Main**"), which was considered to make up around 60% of the deposit; and

- Hanging Wall ("**HW**"), making up the rest of the deposit at nominally 30%.

The deposit was considered by the Amarillo geological team of the day to display high homogeneity. Consequently, it was considered that these three domains would provide similar outcomes.

Twelve drill holes were used to provide nominally 300kg of core for the Coffey work. Only nine of the holes were finally applied to the program. The holes used are listed per Table 13 1 below.

A follow-up program in 2012 was based around a new Master Composite (and referenced in the test work as such) which was compiled using sample from seventeen drill holes that had not been previously used. As had been anticipated, the Main, HW and FW composites of the first Coffey program had given similar results and so it was considered appropriate to continue exploratory work on a master composite having the same nominal ratio of Main, HW and FW present.

It needs to be noted that at this stage, the ratio of Main:HW:FW had been updated to 80:17:3. Somewhat different to the original 2011 compositing.

This same Master Composite was used for the later 2013 oxygenation pilot testing.

Two composites representing HW, and FW "waste" were compiled in 2013 and were subjected to comminution testing. These composites having theoretical grades of 0.35g/tAu and 0.28g/tAu respectively.

A sample of historical tailings from site were also supplied for basic test work but have been omitted from follow-up work and are not detailed herein.

The details of these composites comprising the drill holes used and intervals selected are noted per Table 13 1.

13.3.2 Geology and Sample Nomenclature

As the Project geology and metallurgy developed, samples were identified by the host lithology dominating the mineralization. These designations being:

- Hanging Wall ("**HW**") - previously described as grey gneiss or biotite gneiss;
- Foot Wall ("**FW**") – amphibolites; and
- Main - a hydrothermal alteration zone that occurs in the contact between the rocks of the HW and the FW, through the thrust shear zone, generating mylonites of these two types of rocks.

With time and better understanding of the geology and metallurgy, the designations of FW, HW and Main were later considered to be inconsistent with the mineralised zone description as is detailed further below. However, these names have been retained per the metallurgical test work for reasons of consistency with historical work.

Recent petrographic studies made by Amarillo during 2019 and 2020 showed that the mineralization is restricted to mylonitic zones in the contact of the meta granodiorite and amphibolites. HW and FW are specific geological definitions of the rocks that are located above and below the main structure.

It is now understood that:

- **Hanging Wall**: The rock of the HW is recognized as meta granodiorite and has traditionally been referred to as a biotite gneiss;
- **Foot Wall:** The rock of the FW is amphibolite; and
- **Main is effectively the ore zone** in the contact between these two rock types of granodiorite and amphibole, there is a shear zone (thrust) where fluids with sulphides, tellurides and gold

percolated. It is in this mineralized zone that gold is found. The rocks within this zone are mylonites. The more intense the mylonitization, the greater the concentration of metals and, consequently, gold, sulphides and tellurides. The dominant gold-bearing rock is the mylonitized granodiorite, which was transformed into a feldspar biotite mica quartz schist. This is the locally recognized "Posse Schist".

In current geological descriptions of the deposit the terms Hanging Wall and Footwall refer to barren units adjoining the mineralization which is contained within the mylonitic Posse Schist. This unit, the Posse Schist, has traditionally been described as having its own Hanging Wall, Main and Footwall zones. These distinctions have been superseded by a geological model with reflects the degree of shearing within the mineralised zone resulting in interfingered zones of higher and lower mineralisation. From a metallurgical point of view the terms HW and FW samples refer to material drawn from lower grade portions of the mineralised zone. Samples drawn from the Main zone refer to samples drawn from the higher-grade portions of the mineralisation.

What the most recent test work has shown is the that metallurgical behaviour of the FW, HW and Main samples is consistent and whilst these descriptors remain, particularly in the earlier sample descriptions, the ore zone is identified and represented by the samples selected.

13.3.3 2018 Samples

Test work initiated by Amarillo in 2017 commenced using the remaining Master Composite and Master Composite components originally compiled for the Coffey work.

As the work progressed, the Master Composite was exhausted. A second master composite was compiled made up from remnant materials used for the original Master Composite. This composite being referred to as Master Composite A.

With time, Master Composite A was exhausted, and a new composite referenced as the MG (medium grade) Composite was compiled from twenty-four drill holes to represent the resource. This composite comprised a different ratio of Main, HW and FW in alignment with updated ratios described by the 2017 PFS. The ratios being Main:HW:FW at 67%:27%:5%. This composite effectively being an updated version of a master composite.

Amarillo also generated a Low-Grade Composite ("**LG Composite**") recipe comprised of Main and HW material which was used for work index and leach testing.

In late 2018 a new suite of samples was selected by Amarillo and sent to ALS in Western Australia to allow for variability testing as well as follow up test work to define process design criteria in the areas of settling, cyanide detoxification, oxygen demand and carbon characterization. The actual test work being conducted through 2019 and into early 2020.

Samples for variability testing were selected by Amarillo to represent predominantly Main and HW and one FW sample from various locations in the proposed open pit along strike and with depth.

Table 13-1 provides a list of the various drill holes used for the various test work programs. It will be noted that several holes are common to the programs.

Table 13-1: Drill Holes used for Various Test Work Programs

Coffey Mining Test Work 2011 and later. ALS program A13025	Coffey Mining Test Work 2012 and later ALS programs A13594, A14560, and 2017 program A18001 (MAIN, HW, FW, Master Composite)	Waste Composites for Comminution Test Work 2012 ALS Program A14536	LG Composite 2018 and later. ALS program A18001	MG Composite 2018 and later ALS program A18001	Variability and Design Test Work 2018 – 2020 ALS program A19476	SGS Geosol used for SMC testing 2019	SGS Geosol prepared at 53 um and then used for filtration testing ANDRITZ, Brasfelt and TEFSA
MAIN	MRP0007	MRP0023	MRP0005	MRP0001	2011MRP0001	MRP0014	MRP0006
MRP0001	MRP0016	MRP0023	MRP0016	MRP0002	2011MRP0003	MRP0015	MRP0014

Coffey Mining Test Work 2011 and later. ALS program A13025	Coffey Mining Test Work 2012 and later ALS programs A13594, A14560, and 2017 program A18001 (MAIN, HW, FW, Master Composite)	Waste Composites for Comminution Test Work 2012 ALS Program A14536	LG Composite 2018 and later. ALS program A18001	MG Composite 2018 and later ALS program A18001	Variability and Design Test Work 2018 – 2020 ALS program A19476	SGS Geosol used for SMC testing 2019	SGS Geosol prepared at 53 um and then used for filtration testing ANDRITZ, Brasfelt and TEFSA
MRP0002	MRP0023	MRP0029A	MRP0040	MRP0003	2011MRP0004	MRP0025	MRP0015
MRP0003	MRP0024	MRP0029A	MRP0042	MRP0004	2012MRP0001	MRP0037	MRP0025
MRP0009	MRP0029A	SPETI – 13	SPETI – 17	MRP0005	2012MRP0009	SPETI - 18	MRP0032
MRP0010	MRP0031A		SPETI - 19	MRP0006	2012MRP0014		MRP0041
HW	MRP0032			MRP0007	2012MRP0015		MRP0044
MRP0006	MRP0033			MRP0008	MRP0017		2012MRP0006
MRP0011	MRP0034			MRP0009	MRP0019		2012MRP0014
MRP0012	MRP0035			MRP0010	MRP0022		SPETI - 03
MRP0014	MRP0040			MRP0012	MRP0043		SPETI - 06
	MRP0042			MRP0016	MRP0045		SPETI - 14
	SPETI – 08			MRP0023	SPETI 011		SPETI - 16
	SPETI – 10			MRP0024	SPETI 017		SPETI - 18
	SPETI – 13			MRP0029A	SPETI027		SPETI - 19
	SPETI – 17			MRP0031A	SPETI028		18 P057
	SPETI - 19			MRP0040	18P052		18 P084
				MRP0042			
				SPETI - 08			
				SPETI - 10			
				SPETI - 13			
				SPETI – 17			
				SPETI – 19			

13.3.4 Sample locations

The various drill holes have been presented per Figure 13-1, Figure 13-2, Figure 13-3, Figure 13-4 and Figure 13-5 which provide detail as to the 3D spatial representation of the various intervals used.

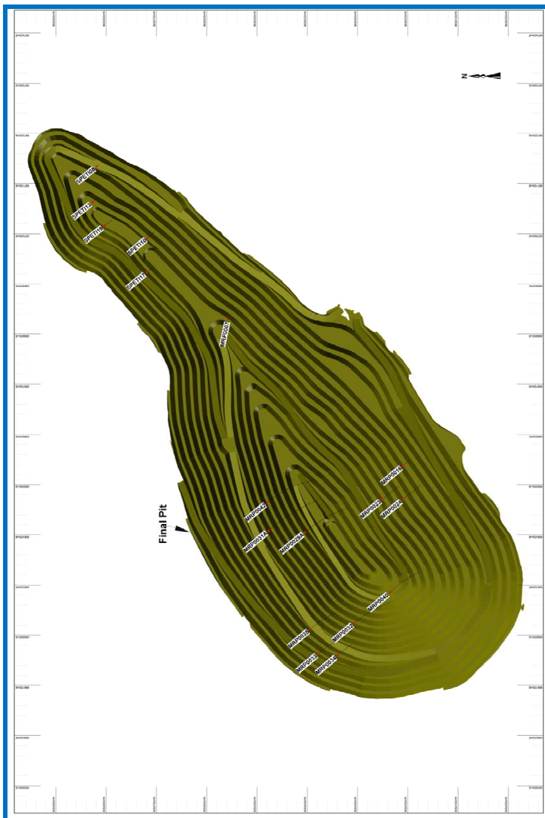


Figure 13-1: Master Composite Sample Locations

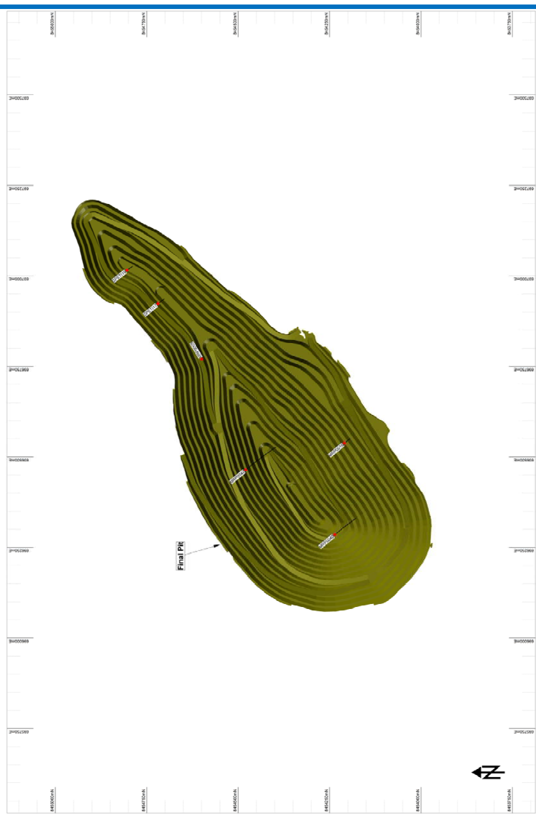


Figure 13-2: Low Grade Composite Sample Locations

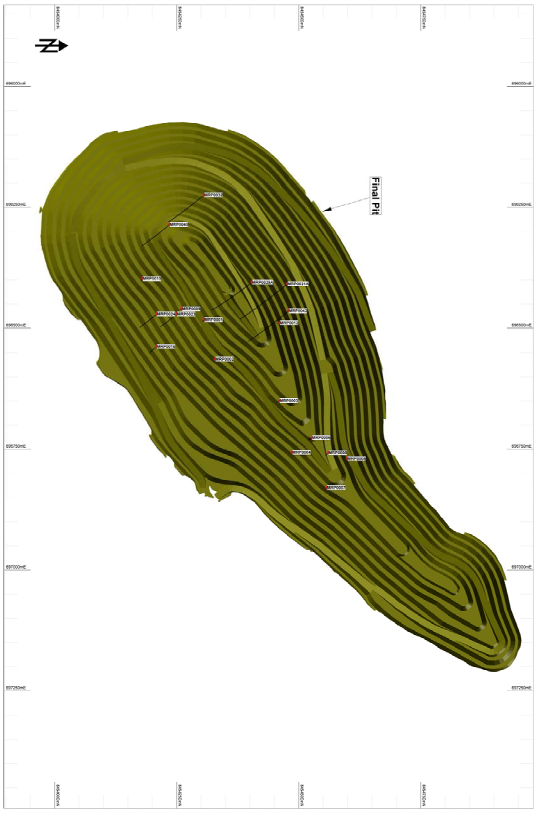


Figure 13-3: Medium Grade Composite Sample Locations

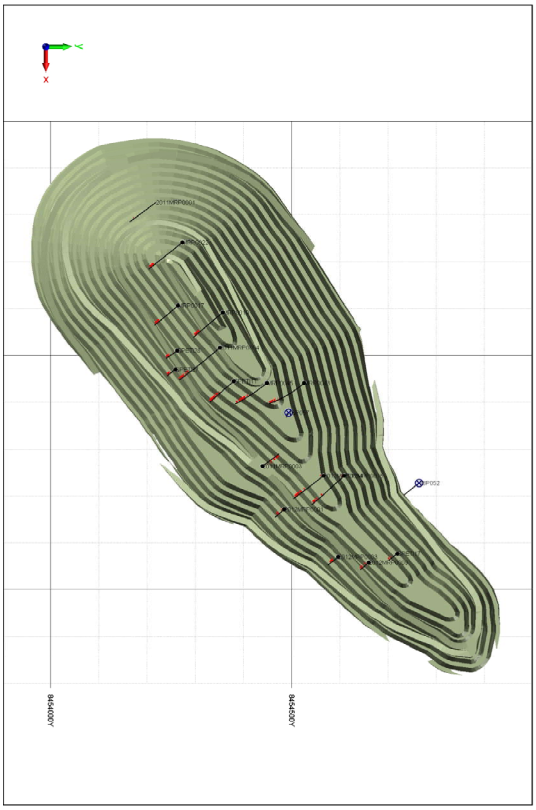


Figure 13-4: Locality Composite Sample Hole Locations 1

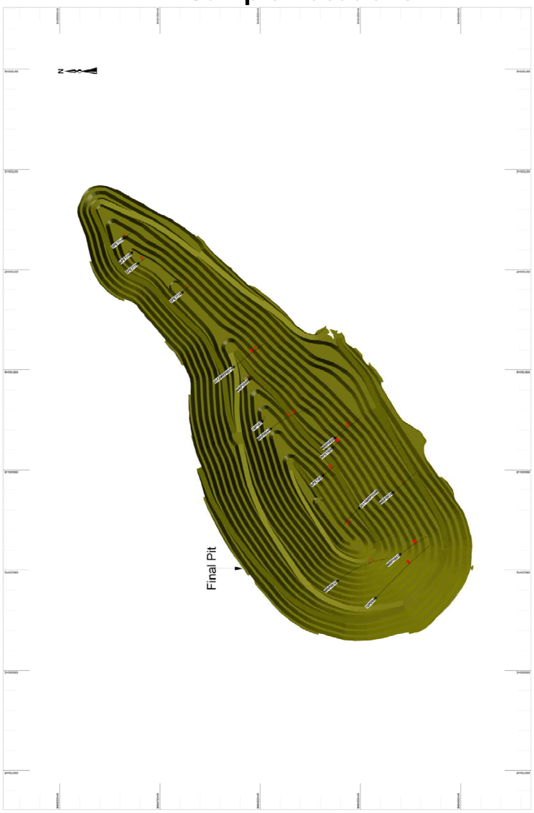


Figure 13-5: SGS Filtration Composite Sample Drill Hole Locations

13.4 Previous Test Work

13.4.1 Early Test Work

Test work conducted prior to the 2017 test work programs have been reported in prior NI43-101 submissions and is not reviewed herein with the exception of some comment regarding the Coffey Mining test work which was influential in establishing the flowsheet. The Coffey Mining work had also provided a selection of samples carried forward into the 2017 and later programs.

13.4.2 Coffey Mining Test Work Comment

Salient points resulting from the Coffey work are made as they are relevant to the later work and also highlight some of the ore characteristics.

The 2011 test work results for the Main, HW and FW composite samples all indicated that a leach residue of nominally 0.10g/tAu could be readily achieved via a process flowsheet that included grinding to a P80 of 45µm, pre-oxidation over a period of 12 hours at a pH of 12 and leaching at conventional cyanide concentrations for a period of 24 to 36 hours. This program highlighted there was little difference in behaviour between the three domains and this was a conclusion stated by Coffey.

In the case of the Main and HW domains, lower leach residues of 0.06g/tAu were achieved when the pre-oxidation and cyanidation leach times were extended to 72h and 48h respectively.

The FW composite sample did not have extended pre-oxidation and cyanidation leach time test work performed due to the good results achieved at lesser times and the lower contribution of this domain to the overall deposit.

This program highlighted a need to explore the pre-oxidation conditions, and as limited work was conducted at the 53µm and 75µm grind sizes, there was potential to further evaluate grind sensitivity.

The 2011 Coffey work also provided moderate work index values (Rod WI low 13kWh/t values and Ball WI of around 13kWh/t) but did suggest moderately higher Abrasion Index values of 0.34 for the Main and HW Composites.

Coffey managed a second program utilising the Master Composite. Various comminution tests were conducted as part of this program and presented tougher ore characteristics than had been observed in the original 2011 work. Given a larger sample base was used, this suggested there was some difference in the mechanical characteristics of the ores even though a larger population of sample may have been considered to smooth out variation. It is worth noting the ratio of Main had increased and there was little FW material present in the Master Composite.

The program revealed little benefit in gravity concentration and similarly, that flotation was not a cost-effective flowsheet option as the tails losses were still high and the final recovery was low. Importantly, the program showed that elevated dissolved oxygen levels were needed to accelerate telluride oxidation and air alone was unsatisfactory.

The program did not address grind sensitivity or additional pre-oxygenation options.

Coffey managed a pilot trial using the proprietary oxygenation device known as a Hyperjet. This test work was inconclusive as to if this device provided any benefit. Conventional oxygenation methods have been retained in the more recent test work. It being appreciated that achieving high dissolved oxygen levels in the laboratory is typically less efficient than can be achieved at full-scale. There remains the opportunity to utilise a more elaborate oxygenation system and plant design should allow for full-scale trials of units such as the Hyperjet, Filblast, Aachen Shear Reactor, Hypersparge or other proprietary devices.

13.5 Test Work Discussion

13.5.1 ALS Program A18001

Aim

This program was focussed on developing the fundamental flowsheet for processing the Mara Rosa material. Whilst the earlier programs had defined a grind, gravity, pre-oxygenation, whole of ore leach flowsheet, there remained questions regarding the most effective pH, if gravity and pre-oxygenation were required, if the grind was supported economically and what the most cost-effective residence time would be.

Samples

Please refer to Table 13-1 regarding sample details which includes a listing of the samples recovered from earlier programs and utilized in program A18001.

The supply of the original Master Composite used in earlier test work was exhausted part of the way through this A18001 program. A new "***master composite***" recipe was prepared by Amarillo using remnant coarse crushed Master Composite sample and re-combination of sized samples and SMC samples. Test work continued with this new composite referred to as Master Composite A. Details are provided per ALS report A18001.

Master Composite A was similarly exhausted with time, and consequently a low grade and a medium grade composite recipe was provided by Amarillo. These composites were prepared for comparative work.

These new low and medium grade composites (LG Composite and MG Composite respectively) comprised a lot of common sample intervals as had been used in the Master Composite and Master Composite A, as well as a minor number of new intervals that had been sent to ALS in preparation for variability work (program A19476 described in Section 13.5.3). The MG Composite was effectively a replacement "master composite" compiled based on updated ratios of the Main, HW and FW zones relevant at the time of compilation.

Program

The test work program included:

- Investigations regarding pre-oxygenation duration and benefits at a P80 45µm grind (Master Composite);
- Comparative tests at a P80 53µm grind (Master Composite);
- Grind sensitivity testing at P80 of 106µm, 75µm, 53µm and 45µm with and without pre-oxygenation as well as size by size head assays at each grind for gold and tellurium (Master Composite A);
- An ultra-fine grind range leach test at nominally P80 20µm (Master Composite A);
- pH sensitivity work to cover off the range pH 11.5, 12.0 and 12.5 (Master Composite A) with and without pre-oxygenation;
- Evaluate the influence of temperature at a P80 of 53µm and 40 degrees Celsius (Master Composite A);
- Prepare a Low Grade ("**LG**") and Medium Grade ("**MG**") composite for Bond Ball Mill Work Index ("**BBWI**") determination at various closing screens;
- Grind sensitivity of the MG composite to leaching, oxygen uptake determination and Weak Acid Dissociable ("**WAD**") cyanide determination of leach solution;
- Grind and temperature sensitivity and viscosity test work pre and post leach (combinations

of MG and LG composite);

- Preg-robbing test on MG composite;
- Bulk leach tests on MG composite followed by carbon characterization (adsorption kinetics and equilibrium) and cyanide detoxification test work. The final detoxified slurry was then used by Outotec for thickening test work (this work consigned prior to the decision to include tailings filtration in the flowsheet);
- P80 75µm 40 degree Celsius leach tests of LG and MG composites; and
- Investigations of cyanide concentration sensitivity (MG composite).

Key Observations

As program A18001 progressed, two key process variables were identified that had not previously been well controlled or explored.

One was pH and the variability of pH as the tests were conducted. It was found that very tight pH control was required to provide comparative results. The typical variation in pH from test work determination to determination normally tolerated in leach testing was resulting in variable outcomes.

The second process variable was temperature. It was found well into the program that temperature and control thereof was also contributing to not only differences in extraction, but also in pH modifier consumption. As a result, early program results are not directly comparable to results achieved later in the program when process variables were more closely monitored and controlled.

Results – Head Grade Deportment

A number of size-by-size assay head determinations were made at differing grind sizes on the Master Composite. They all showed similar trends where the higher grades of both gold and tellurium were found in the finer size fractions and the mass deportment was similarly biased toward the finer size fractions as a consequence. The Master Composite gold and tellurium head grades being nominally 1.5g/t and 3.4g/t respectively.

Two grind size distributions are presented by Figure 13-6 to Figure 13-9 for a P80 of 106µm and 53µm. These distribution characteristics are considered typical of the data collected.

Figure 13-6: Mass and Gold Distribution, P80 106µm

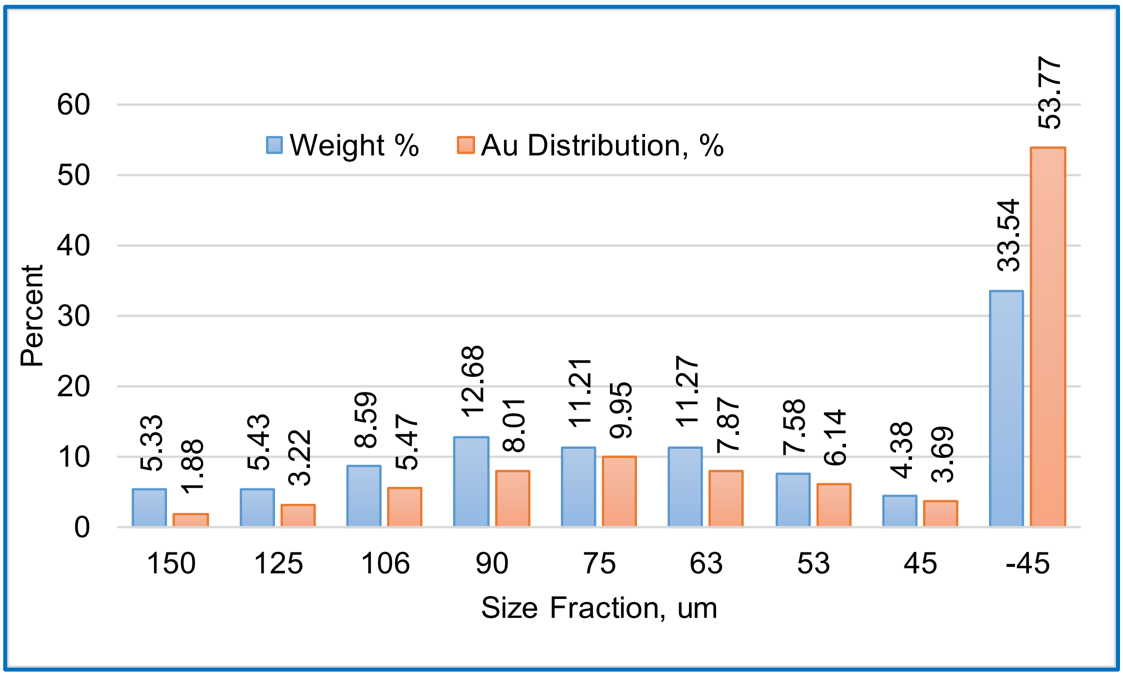


Figure 13-6 shows there is a tendency for the finer fractions to be dominated by the slimes (-45µm fraction). Note that the slimes fraction has a much higher gold deportment compared to the mass deportment.

Figure 13-7 highlights the tellurium grades tends to track the gold grade by fraction. The gold grade of the slimes fraction exceeds the back-calculated head grade of 1.79g/t. Similarly, the tellurium grade of the slimes fraction exceeds the back-calculated head grade of 3.35g/t.

Figure 13-7: Mass and Gold Distribution, P80 53µm

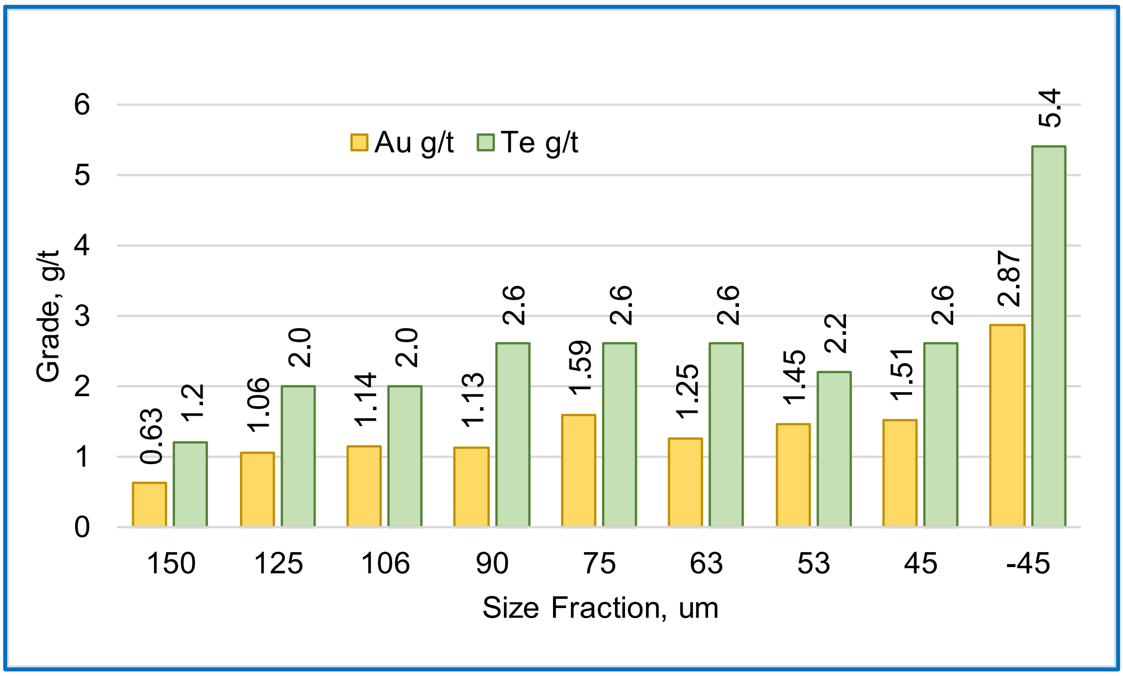


Figure 13-8 shows that at the finer grinds, the same trends appear with regard to gold deportment exceeding the mass deportment in the slimes fraction. The +90µm fractions have very low grade, probably as a result of this material comprising a higher proportion of lower grade yet tenacious silicates.

Figure 13-8: Mass and Gold Distribution, P80 53µm

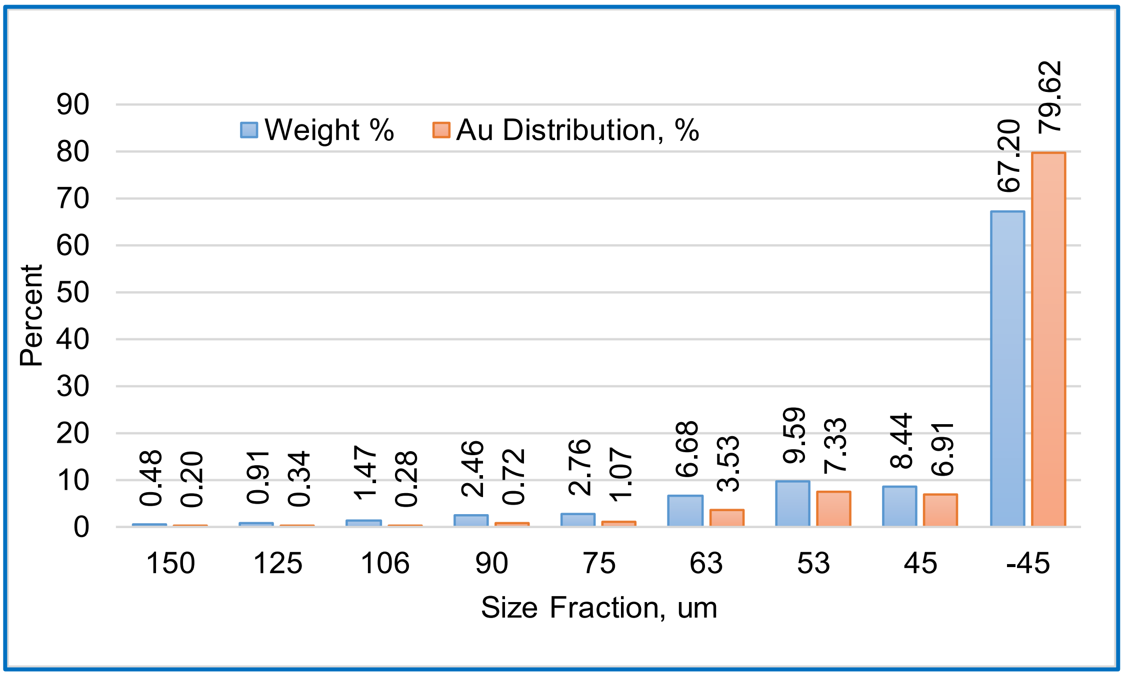
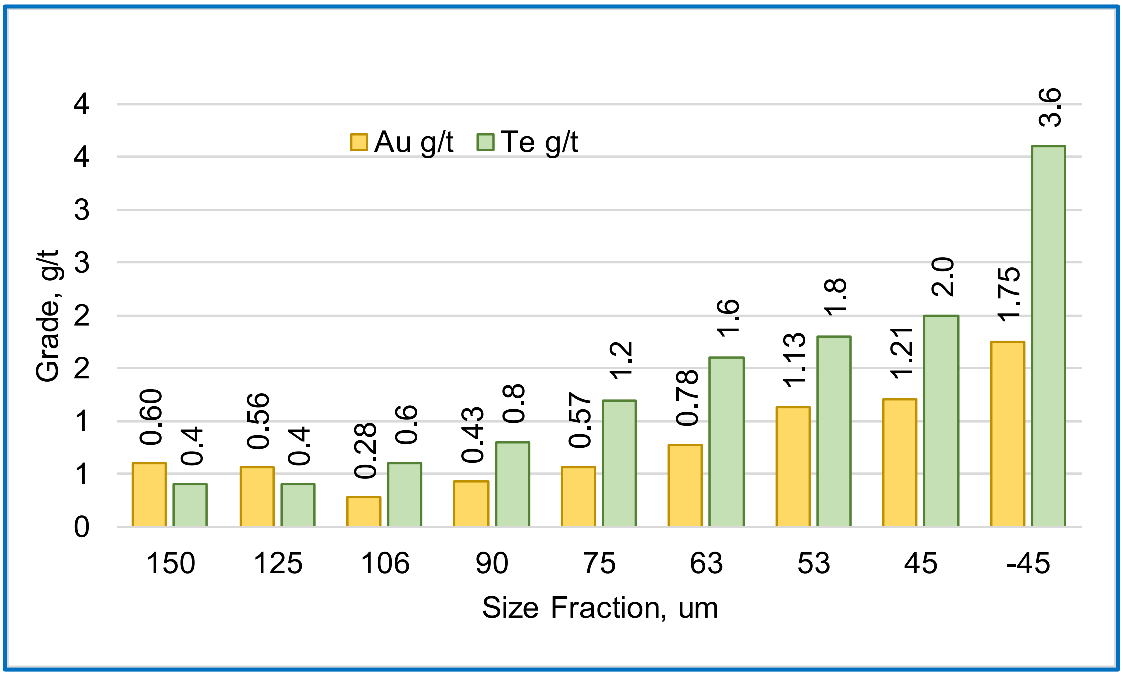


Figure 13-9 presents some beneficial liberation characteristics. At the finer grind, the tellurium assays of the coarse fractions are considerably lower, and this would align with these minerals being preferentially ground and reporting to the finer fractions, notably the slimes. The tellurium deportment as a distribution basically followed the gold distribution for the coarser 106µm grind. For this 53µm grind it is more biased toward the fines than the gold. The back-calculated head grades were 1.48g/t and 3.93g/t for gold and tellurium respectively. Somewhat lower than the values back-calculated for the 106µm sizing.

Figure 13-9: Size Fraction Grade, Au and Te, P80 53µm



This work shows that selective grinding of the tellurides is likely. In the full-scale operation, it is probable this effect will be amplified by the bias of higher specific gravity particles to the hydrocyclone underflow in the comminution circuit. This selective finer grinding of the "problematical" tellurides at plant scale is advantageous and will assist in maintaining high leach extractions.

Results – Work Index Determinations

Samples of the LG and MG Composite were submitted for Bond Ball Mill Work Index ("**BBWi**") determinations and differing grinds. Table 13-2 summarises the results.

Table 13-2: Comminution Results LG and MG Composites

Sample	Closing Screen (µm)	F_{80} (µm)	P_{80} (µm)	BBWi (kWh/t)
LG COMP	75	2,700	65	17.0
MG COMP	75	2,597	64	16.3
MG COMP	63	2,602	52	18.4
MG COMP	53	2,607	41	18.3

The results indicate the LG Composite will require slightly more grinding energy than the MG Composite and the MG Composite does show increased toughness at finer grinds. Additional comment is made regarding comminution parameters in Section 13.6 where key design criteria are discussed.

Results – Leach Evaluation

Investigation into the benefit of pre-oxygenation was conducted on the original Master Composite sub-samples at a P80 grind of 45µm. This work showed that there was no perceptible benefit from pre-oxygenation and no recognisable benefit from extending the leach for 24 hours to 32 hours.

What was indicated was a potential reduction in sodium cyanide ("**NaCN**") consumption with extended oxygenation. This is a common observation with sulphide ores and so could be expected as an outcome.

Results are presented per Table 13-3.

The term "Lime" is applied by ALS to describe hydrated lime of nominally 65% to 68% active CaO content. ALS test their reagent periodically to confirm activity. The use of the term "lime" is potentially confusing given there are a number of lime reagents such as hydrated lime, quicklime and even agricultural lime. Consequently, there is a need to specify purity when quoting reagent consumption.

The actual CaO demands of the Mara Rosa samples are detailed out per the reagent consumption estimate and key criteria sections of this report (Section 13.5.3 and Section 13.6) with consideration of the ALS reagent purity and that of commercial quicklime.

The term "lime" is used throughout Section 13 to be consistent with the ALS nomenclature unless described otherwise.

Table 13-3: Pre-oxygenation Benefit

Sample ID	Test Variation	Grind P_{80} µm	Residue grade Au g/t	Final Extrn %	Calc Head Au g/t	Assay Head Au g/t	NaCN Cons kg/t	Lime Cons kg/t
MASTER COMPOSITE	24 hr leach with 8 hr Pre Oxygenation	45	0.10	92.53	1.34	1.47 /1.51	0.10	5.12
MASTER COMPOSITE	24 hr leach with 12 hr Pre Oxygenation	45	0.10	93.00	1.43	1.47 /1.51	0.10	5.56
MASTER COMPOSITE	24 hr leach with oxygen sparge	45	0.10	93.04	1.36	1.47 /1.51	0.14	4.96
MASTER COMPOSITE	32 hr leach with oxygen sparge	45	0.10	93.12	1.38	1.47 /1.51	0.14	5.00

As the Master Composite was exhausted, two sources of sample remained from previous composite construction. 3.35mm reserves of the Master Composite and 25mm reserves of the same components used to build the Master Composite. These remnants combined to generate Master Composite A. The two Master Composite and Master Composite A sources were leached to explore as to if they would provide consistent outcomes. Both sample sources were leached at a P80 grind of 53µm and leached for 48 hours to test ultimate extraction. The results are presented per Table 13-4.

Similar leach performance was observed. However, the composite resulting from the coarser 25mm source and 3.35mm reserves provided a lower assay head grade but elevated calculated head grade. 48-hour extractions were similar on a percentage basis. The higher-grade sample gave a slightly higher residue which is typical of other leach test observations. The conclusion was the samples are very similar, but not identical. Consequently, results for the different sample origins are not comparable from the earlier part of the test program to the latter part for "**Master Composite**" samples.

Given the issue of key process variables noted above (pH and temperature), the early program results are not considered comparable to later program outcomes where controls were improved. That is, with time temperature and pH control improved and correspondingly the results considered more reliable. Point being that some care is required when comparing results for the various master composites depending on compositions and time of actual testing.

Table 13-4: Sample Comparison Leaching

Sample ID	Test Variation	Grind P_{80} µm	Residue grade Au g/t	Final Extrn % 48 h	Calc Head Au g/t	Assay Head Au g/t	NaCN Cons kg/t	Lime Cons kg/t
MASTER COMPOSITE A	48 hr leach , 02 sparge, ex 25mm reserves	53	0.21	89.68	2.03	1.38/ 1.57	0.25	4.80
MASTER COMPOSITE	48 hr leach , 02 sparge, ex 3.35mm reserves	53	0.18	89.34	1.69	1.47/ 1.51	0.19	6.37

Test work progressed using Master Composite A. A grind sensitivity exercise at pH 12.5 and 25°C was conducted. This was to explore project economics given the minor benefit identified for the finer grinds and lower head grades. It being considered opportunity may exist for a coarser grind. Results are presented per Table 13-5.

Table 13-5: Master Composite A Grind Sensitivity

Sample ID	Test Variation	Grind P_{80} µm	Residue grade Au g/t	Extn % 24 h	Final Extrn %, 48 h	Calc Head Au g/t	Assay Head Au g/t	NaCN Cons kg/t
MASTER COMPOSITE A	Grind Sensitivity with PreOX (ex A13954 25mm reserves)	106	0.36	68.1	84.3	2.26	1.38/ 1.57	0.17
MASTER COMPOSITE A	Grind Sensitivity with PreOX (ex A13954 25mm reserves)	75	0.24	79.3	85.7	1.68	1.38/ 1.57	0.18
MASTER COMPOSITE A	Grind Sensitivity with PreOX (ex A13954 25mm reserves)	53	0.18	85.5	89.5	1.71	1.38/ 1.57	0.20
MASTER COMPOSITE A	Grind Sensitivity with PreOX (ex A13954 25mm reserves)	45	0.15	83.9	91.3	1.73	1.38/ 1.57	0.21

The Master Composite A grind sensitivity work showed significant benefit in reducing the grind size from 106 down to 53µm. The benefit from 53 to 45µm greatly reduced. It also being noted that the leach rate for the finer grinds was much improved suggesting reduced residence time potential for the finer grinds. Finer grinds did present elevated NaCN consumption as an off-set. It should be noted the lime consumption was similar for all tests (not reported here).

High level assessment of the economics of a finer grind suggested opportunity existed for a coarser grind. As a consequence, a follow-up program including an intermediate grind size was conducted in the P80 75 to 45µm range. To provide data on even finer grinds, a sample was also subjected to an ultra-fine grind (UFG) and leaching. Leaching conducted at pH of 12.5 and leach temperature of 25°C. Tests were also monitored at 24, 36 and 48 hours to provide data regarding leach kinetics as is discussed in latter sections of this report.

Results are summarised by Table 13-6.

Table 13-6: Master Composite A Grind Sensitivity – Additional Grind Series

Sample ID	Test Variation	Grind P_{80} µm	Residue grade Au g/t	Final Extrn % 48 h	Calc Head Au g/t	Assay Head Au g/t	NaCN Cons kg/t	Lime Cons kg/t
MASTER COMPOSITE A	GRIND SENSITIVITY AT PH 12.5	75	0.31	82.54	1.78	1.38/ 1.57	0.20	9.56
MASTER COMPOSITE A	GRIND SENSITIVITY AT PH 12.5	60	0.22	87.44	1.75	1.38/ 1.57	0.21	9.78
MASTER COMPOSITE A	GRIND SENSITIVITY AT PH 12.5	53	0.16	91.46	1.87	1.38/ 1.57	0.21	6.17
MASTER COMPOSITE A	GRIND SENSITIVITY AT PH 12.5	45	0.15	92.36	1.90	1.38/ 1.57	0.20	5.61
MASTER COMPOSITE A	GRIND SENSITIVITY AT PH 12.5	<20	0.07	95.83	1.68	1.38/ 1.57	0.27	8.00

The results of this additional grind sensitivity program aligned with the earlier work. There was some scatter in the calculated head grades, with it being notable that the sample ground to a P80 of 45µm had the highest calculate head, thereby potentially escalating the final residue grade. Due to the head grade/residue relationship present, this clouds outcomes to a degree. Higher grades reporting higher residues all other things equal.

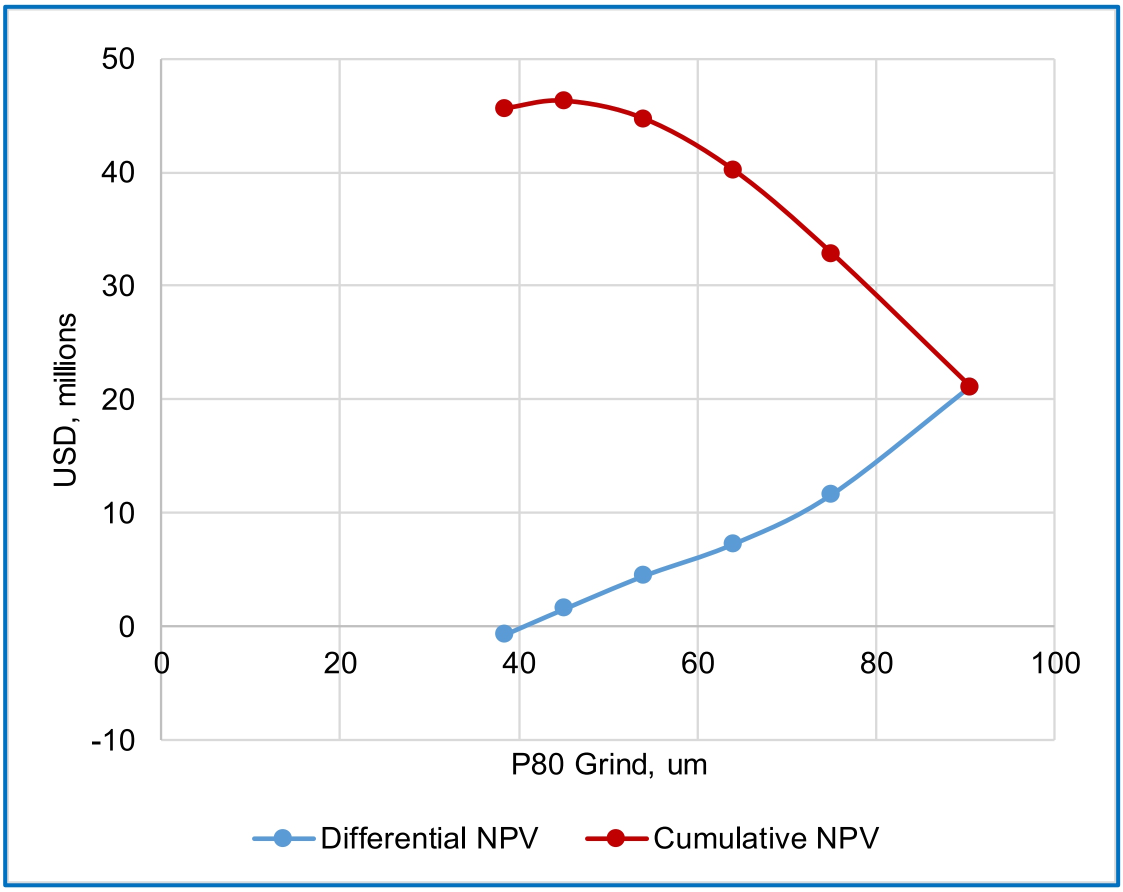
Be that as it may, the results again suggested that coarser grinds than 53µm result in a rapid increase in residue grade/loss in extraction. Importantly, the difference between a 53 and 45 µm grind is minor suggesting coarsening the grind to 53µm may well be the most economic option.

Aurifex undertook a high-level financial assessment looking at 3 year NPV returns at a discount of 10% and a gold price of US$1,200/oz. This was also based on a plant throughput of 2.5 Mtpa. The results are presented as Figure 13-10.

This analysis suggested there was a case to retain a P80 45µm grind. However, as the cumulative plot benefit (red line) was flattening out at grinds finer than 45µm, and if entry capital is considered, then it was quite plausible a 53µm P80 may be more economic. A decision was taken to focus on the 53µm grind size but also retain work at the 45µm grind size for comparative purposes.

It should be noted that the financial conditions applied in this analysis may be somewhat different to what would be applied if such an assessment were to be undertaken Q1 2022.

Figure 13-10: High Level NPV Assessment (2018)



Test work progressed to explore the influence of temperature and pH.

Tests at a pH of 11.5, 12.0 and 12.5 were conducted at 25°C. The pH 11.5 test was conducted with pre-oxidation to establish if similar extractions could be achieved at the lower pH to save reagent and a comparative test at pH 12.0 was run with pre-oxidation.

Tests at a pH of 12.0 and 12.5 were run without pre-oxidation to test the value of increased pH, and a test run at pH 12.5 at 40°C was run to explore the effect of temperature. Results of these tests are presented as Table 13-7.

All tests achieved consistent dissolved oxygen levels typically in the 20mg/L to 25mg/L range.

Table 13-7: pH and Temperature Influence

Sample ID	Test Variation	Grind P_{80} µm	Residue grade Au g/t	Final Extrn %	Calc Head Au g/t	Assay Head Au g/t	NaCN Cons kg/t	Lime Cons kg/t
MASTER COMPOSITE A	Pre-Ox and Water Bath at 25ºC and pH 12.0	53	0.21	88.29	1.79	1.38/ 1.57	0.23	3.38
MASTER COMPOSITE A	Pre-Ox and Water Bath at 25ºC and pH 11.5	53	0.21	88.75	1.87	1.38/ 1.57	0.26	2.73
MASTER COMPOSITE A	Water Bath at 25ºC and pH 12.5	53	0.22	88.27	1.88	1.38/ 1.57	0.21	4.58
MASTER COMPOSITE A	Water Bath at 25ºC and pH 12.0	53	0.18	90.46	1.89	1.38/ 1.57	0.21	2.24
MASTER COMPOSITE A	Water Bath at 40ºC and pH 12.5	53	0.18	89.49	1.71	1.38/ 1.57	0.42	29.67

The elevated temperature of 40°C was selected based on the likelihood that achieving a fine grind in a climate such as found at the Mara Rosa site could quite easily generate slurry temperatures exiting the grinding circuit of this order. Given the oxidation rate of tellurides is a function of pH and could be expected to be accelerated by temperature, elevated extractions and/or improved kinetics could be anticipated. In conflict to these aspects, the equilibrium dissolved oxygen level is reduced as temperature increases. Therefore, there are both positive and negative aspects of elevated temperature when processing auriferous telluride ore.

The work showed that the extractions are effectively the same within experimental error. There being a case to reduce pH with pre-oxidation.

Of interest is that the elevated temperature test produced an excessive dose of lime as well as elevated NaCN consumption. The log sheets for this test reported the pH did not reach 12 and it is suggested the pH meter was unable to accurately record the pH at the elevated temperature. This test is therefore considered non-representative, as has been supported by later tests.

The results of the pre-oxidation at pH 11.5 and 12.0 as well as tests at pH 12.0 and 12.5 are represented graphically by Figure 13 11 to Figure 13 14. Whist the ultimate extractions at 48 hours are similar, the kinetics of the tests presented some key observations.

Figure 13-11 represents the pre-oxidation test leached at pH 11.5. The kinetics are consistent for the gold leach and show that from 24 hours onwards the leach rate flattens out. It is possible leaching is continuing at 48 hours, and this could be due to a “fixed” rate of telluride degradation continuing to occur.

Figure 13-11: Pre-ox and pH 11.5 at 25°C

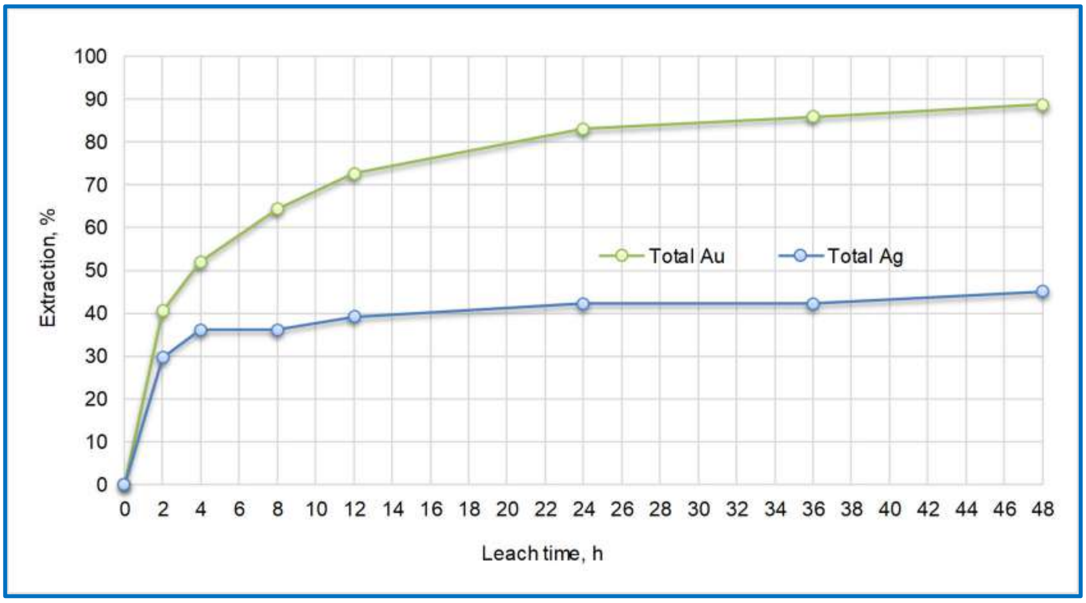


Figure 13-12 presents the pre-oxidation and leach at pH 12.0. The kinetics here are improved at the start of the leach but from 24 hours the curve is very similar to that of the pH 11.5 curve including continued leaching potential at 48 hours. In isolation, comparing these two curves would suggest there is little benefit in elevating the pH from 11.5 to 12.0.

Figure 13-13 presents a leach at pH 12.0 without pre-oxidation. This test presents a number of points:

- It suggests when observed in isolation that there is little value if any in pre-oxidation;
- The test work variability is such that the comparisons between these tests is considered to fall within experimental error. It being noted lower grade samples are influenced to a greater degree by a small yet disproportionate component of free or even high-grade mineral phases;
- That this test suggests little benefit in exceeding a 24 hour residence time given the

incremental extraction between 24 and 36 then 36 to 48 hours is minor. Somewhat in conflict with the previous two figures (Figure 13-11, Figure 13-12); and

- Test work controls need to be consistent and repeat work is necessary.

Figure 13-12: Pre-ox and pH 12.0 at 25°C

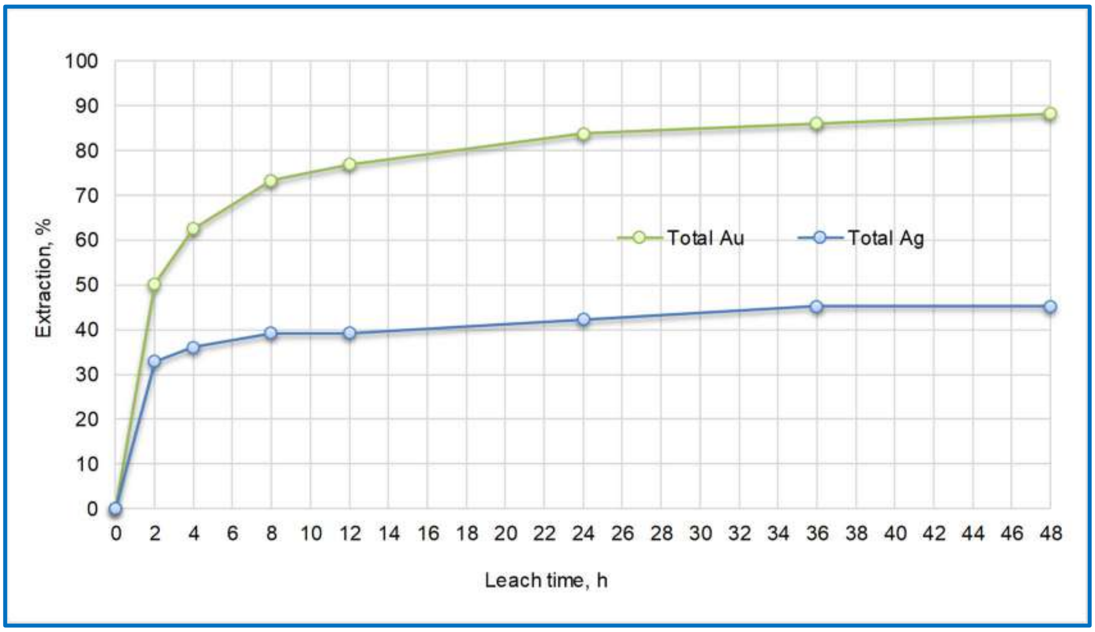


Figure 13-13: No Pre-ox and pH 12.0 at 25°C

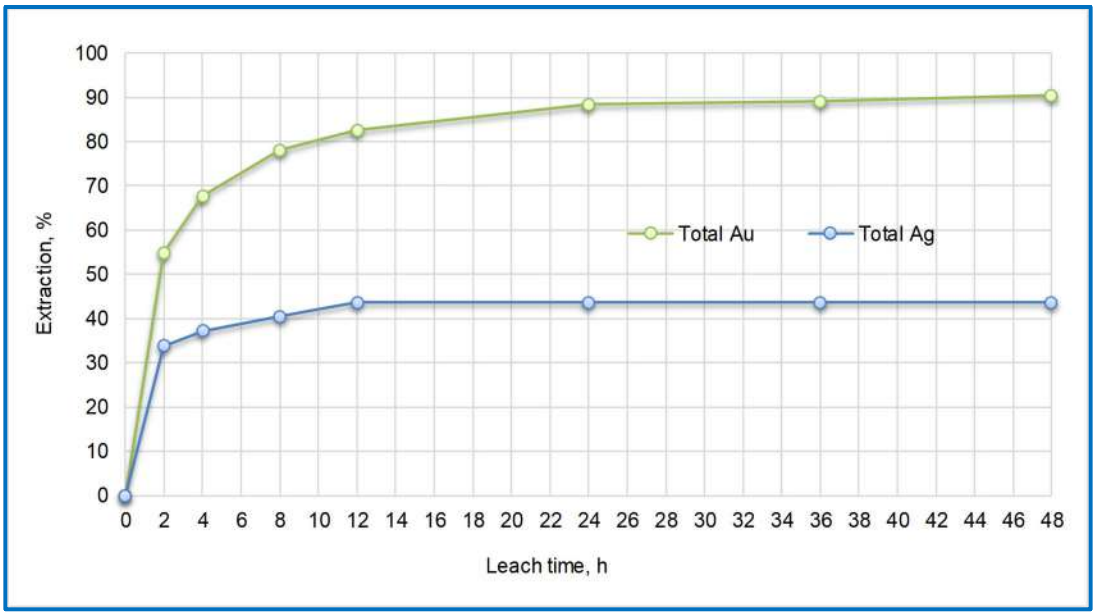
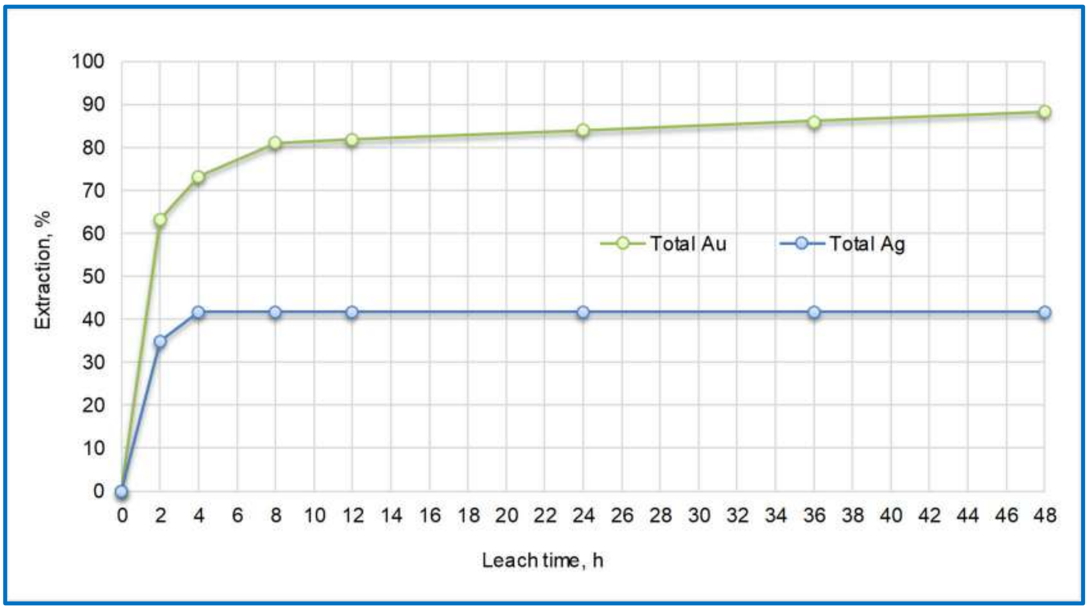


Figure 13-14 is directly comparable to Figure 13-13 to understand the potential benefit of pH influence. Figure 13-14 presents the kinetics for the leach at pH 12.5 without pre-oxygenation. The plot reveals the most rapid leach kinetics of all of the tests presented by the four graphs. However, the long leach tail as observed in the pre-oxygenation tests remains with a slow but consistent leach rate presented between 12 to 48 hours.

The long leach tail is a function of the solution grade increasing by an amount of 0.01g/m^3 to 0.02g/m^3. This is inside the accuracy of the sampling and assay. Yet this test and preceding tests show a pattern of this long leach tail suggesting that whilst inside the sample-assay error, it is real.

This in turn means that to establish the leach time for a full-scale plant, the incremental capital and operating costs for these minor additional extractions has to be analysed.

Figure 13-14: No Pre-ox and pH 12.5 at 25°C



Included in any analysis of leaching conditions is a need to consider reagent demands. Whilst pre-oxidation has earlier shown a reduction in NaCN use, these tests detailed in Table 13-7 suggested the lower pH tests resulted in elevated NaCN consumption at reduced lime consumption. The lime consumption increase with pH would be expected to exceed the value of the NaCN saved, but there is the consideration of cyanide detoxification costs. In addition, it was noted that the leach tests were using excess NaCN to retain consistency of outcomes, and NaCN dose had yet to be optimised.

The MG Composite was subjected to a round of grind sensitivity testing. Results are presented as Table 13-8. Tests were conducted at 25°C and a comparative test with Master Composite A was included for comparison.

Table 13-8: MG Composite Grind Sensitivity

Sample ID	Test Variation	Grind P_{80} µm	Residue grade Au g/t	Final Extrn %	Calc Head Au g/t	Assay Head Au g/t	NaCN Cons kg/t	Lime Cons kg/t
MG COMPOSITE	GRIND SENSITIVITY AT PH 12.5	75	0.19	87.73	1.51	1.36, 1.32, 1.31	0.19	3.68
MG COMPOSITE	GRIND SENSITIVITY AT PH 12.5	53	0.12	90.91	1.32	1.36, 1.32, 1.31	0.17	3.20
MG COMPOSITE	GRIND SENSITIVITY AT PH 12.5	45	0.18	87.93	1.45	1.36, 1.32, 1.31	0.17	4.09
MASTER COMPOSITE A	DIRECT LEACH IN WATER BATH AT 25ºC	45	0.15	90.78	1.57	1.36, 1.32, 1.31	0.20	4.57

These grind sensitivity tests for the MG Composite produced an inconsistent set of results. There are at least two contributing factors observable when the detailed test work log sheets are reviewed.

- Firstly, the pH control was not robust. The kinetic curves show a jump when the pH had been found to have dropped below 12.5 and additional reagent was added. These observations highlight the need to maintain pH inside a tight band if tests are to be comparable and also the need in a full-scale plant to have multiple pH monitoring and dosing points to retain leach kinetics and ensure final extractions are maximised; and

- Second, the dissolved oxygen levels were variable and the 45 µm grind test presented lower levels than the other tests. At times excessively high dissolved oxygen levels were present which can actually retard leaching due to gold surface passivation. This observation also highlights a need for close dissolved oxygen control including full-scale plant requirements.

Out of the MG Composite tests, the best controlled was the P80 75µm grind and the worst the 45µm grind, which is considered to go some way in explaining why the 45µm test does not sit where logic would anticipate.

These points noted, what are considered positive outcomes are:

- Even with these inconsistencies in test work control, high extractions are maintained;
- It is more difficult to control the pH and dissolved oxygen level in a lab situation. The full-scale plant can be expected to achieve better control and therefore more consistent and elevated extractions; and
- The flowsheet provides what are considered to be consistent outcomes.

To further evaluate the effect of elevated temperature, a program of work using the LG and MG composites was undertaken and included some grind sensitivity work for additional LG Composite. Results of which are summarised by Table 13-9.

The results show there is little difference in behaviour between 35°C and 40°C for the MG Composite. The indication is that the higher temperature provides an improved extraction, but this interpretation is inside the range of anticipated error of the assay determinations and clouded by the head grade influence. The higher temperature does seem to have used marginally more NaCN and has used significantly more lime.

Table 13-9: Temperature and Grind Sensitivity, Round 1

Sample ID	Test Variation	Grind P_{80} µm	Residue grade Au g/t	Final Extrn % 48 h	Calc Head Au g/t	Assay Head Au g/t	NaCN Cons kg/t	Lime Cons kg/t
MG Composite	Water Bath at 40 C and pH 12.0 target	53	0.095	93.2	1.40	1.36, 1.32, 1.31	0.33	5.48
MG Composite	Water Bath at 35 C and pH 12.0 target	53	0.105	93.1	1.53	1.36, 1.32, 1.31	0.29	4.85
LG Composite	Water Bath at 25 C and pH 12.0 target	53	0.095	86.9	0.73	0.67 / 0.73 / 0.61	0.23	2.79
LG Composite	Water Bath at 25 C and pH 12.0 target	45	0.13	87.4	1.04	0.67 / 0.73 / 0.61	0.23	2.64
LG Composite	Water Bath at 40C and pH 12.0 target	45	0.04	96.0	0.99	0.67 / 0.73 / 0.61	0.27	6.35

The LG Composite tests showed compromised results when compare the 53µm and 45µm grind sensitivity. In part clouded by the 45µm sample reporting a high calculated head.

When comparing the test at 40°C with those at 25°C, it does appear these is a significant improvement in extraction at the elevated temperature. Again, the elevated temperature test on the LG Composite has presented elevated NaCN and lime consumption.

A second round of temperature and grind sensitivity work was conducted, again on the MG and LG Composites. Results are presented per Table 13-10.

Tests on the MG Composite included a repeat of the previous test at 35°C and a test at an even higher temperature of 45°C. The repeat at 35°C gave similar and confirmatory results for the earlier test (see Table 13-9). The test at 45°C gave a similar extraction but resulted in a greatly elevated lime consumption. These tests continue to suggest there is some sensitivity in temperature and reagent consumption, particularly the lime consumption. They also present elevated extractions with temperature.

Table 13-10: Temperature and Grind Sensitivity, Round 2

Sample ID	Test Variation	Grind P_{80} µm	Residue grade Au g/t	Final Extrn % 48 h	Calc Head Au g/t	Assay Head Au g/t	NaCN Cons kg/t	Lime Cons kg/t
MG Composite	Water Bath at 45 C and pH 12.0 target	53	0.105	92.5	1.39	1.36, 1.32, 1.31	0.28	13.3
MG Composite	Water Bath at 35 C and pH 12.0 target	53	0.1	93.8	1.60	1.36, 1.32, 1.31	0.22	5.3
LG Composite	Water Bath at 45 C and pH 12.0 target	53	0.06	91.1	0.67	0.67 / 0.73 / 0.61	0.29	9.29
LG Composite	Water Bath at 35 C and pH 12.0 target	53	0.06	93.1	0.87	0.67 / 0.73 / 0.61	0.32	6.72
MG Composite	Water Bath at 40 C and pH 12.0 target	75	0.2	87.5	1.61	1.36, 1.32, 1.31	0.28	5.95
LG Composite	Water Bath at 40 C and pH 12.0 target	75	0.105	84.5	0.68	0.67 / 0.73 / 0.66	0.38	5.34

Given the elevated temperature was observed to provide elevated extractions, two tests were performed at a coarser grind of P80 75µm to establish if the temperature would allow a relaxation of grind. One test on each of the MG and LG Composites presented a loss of extraction of the order of 5 to 7%, thereby suggesting liberation at a 75µm grind was inadequate to provide effective telluride oxidation.

Previous test work had been conducted with an excess of NaCN. This technique ensuring leach performance is not retarded by inadequate reagent and at the same time, effectively reducing one of the test work variables whilst other variables (grind, kinetics, temperature, pH) are explored.

To understand NaCN sensitivity and to optimise reagent addition and associated operating costs, a series of tests was conducted on the MG Composite. These results are summarised by Table 13-11.

Table 13-11: MG Composite NaCN Sensitivity

Sample ID	Test Variation	Grind P_{80} µm	Residue grade Au g/t	Final Extrn % 48 h	Calc Head Au g/t	Assay Head Au g/t	NaCN Cons kg/t	Lime Cons kg/t
MG Composite	Water Bath at 40 C and pH 12.0 target, 300ppm NaCN	53	0.095	92.5	1.26	1.36, 1.32, 1.31	0.32	4.2
MG Composite	Water Bath at 40 C and pH 12.0 target, 370ppm NaCN	53	0.105	92.5	1.40	1.36, 1.32, 1.31	0.30	4.45
MG Composite	Water Bath at 40 C and pH 12.0 target, 270ppm NaCN	53	0.1	92.4	1.31	1.36, 1.32, 1.31	0.34	4.61
MG Composite	Water Bath at 40 C and pH 12.0 target, 230ppm NaCN	53	0.095	92.9	1.34	1.36, 1.32, 1.31	0.33	4.55
MG Composite	Water Bath at 35 C and pH 12.0 target, 270ppm NaCN	53	0.095	93.5	1.47	1.36, 1.32, 1.31	0.24	3.77
MG Composite	Water Bath at 35 C and pH 12.0 target, 230ppm NaCN	53	0.09	93.7	1.43	1.36, 1.32, 1.31	0.23	3.55

Based on the previous test work, flowsheet conditions of a P80 of 53µm, pH 12.0 and a temperature of 35°C and 40°C were selected. A leach time of 36 hours was anticipated to be the basis of design, but the leach time was kept at 48 hours to assess if the reduced NaCN levels retarded kinetics.

The test work results show no reduction in leach extraction or kinetics as the NaCN concentration was reduced from previous values of 500mg/L down to as low as 230mg/L. Extractions in excess of 92% were achieved in all tests and review of the kinetic curves show little benefit in extending leach times from 36h to 48h. That is, kinetics were not impacted.

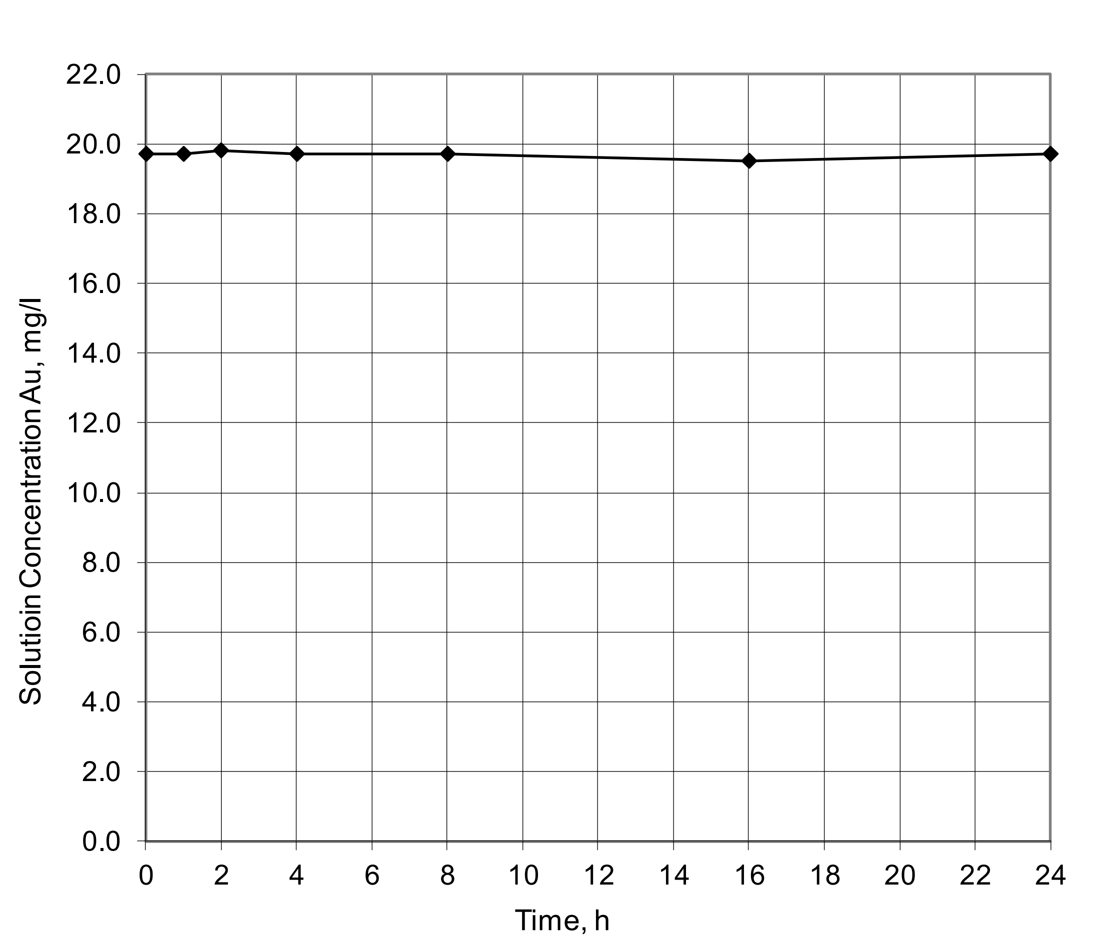
Actual NaCN consumption on a kg/t basis was similar to when higher concentrations were applied in earlier work. The value in the reduced concentration of NaCN is that the losses to tails are reduced as is the operating cost and demands on downstream cyanide detoxification.

This work suggests NaCN consumptions of 0.2kg/t to 0.25kg/t are possible, and that downstream cyanide detoxification system can be designed at a reduced kg/h NaCN and WAD CN load.

Results – Preg Robbing

A sample of the MG Composite was subjected to a preg-robbing test. Results are presented by Figure 13-15. The Mara Rose ores do not present any carbonaceous material, and so the zero preg-robbing result achieved is as expected. Preg-robbing is not a consideration with regard to full-scale plant design.

Figure 13-15: Preg-robbing Test, MG Composite



Results – Carbon Adsorption Characterization

Carbon characterization requires some 20 litres of slurry for testing. The opportunity was taken to undertake two 20 kg leaches of MG Composite at a P80 grind of 53µm and 45µm to compare performance at a large sample size. The leach tests were conducted at ambient temperature due to restrictions in equipment availability and for a duration of 24 hours. The leach test results are presented per Table 13-12.

Table 13-12: MG Composite Bulk Leach Tests

Grind P_{80} µm	Residue grade Au g/t	Extn, 2 h %	Extn, 4 h %	Extn, 8 h %	Extn, 12 h %	Final Extrn 24 h %	Calc Head Au g/t	Assay Head Au g/t	NaCN dose kg/t	NaCN Cons kg/t	Lime Cons kg/t
53	0.23	40.5	50.6	68.8	71.3	84.3	1.43	1.38, 1.57	0.61	0.08	2.25
45	0.36	31.3	42.1	53.2	60.2	75.8	1.47	1.38, 1.57	0.61	0.12	1.84

Neither test performed in line with the smaller leach tests in that the kinetics were slow and the extraction at 24 hours was lower than expected. In fact, the extraction at 24 hours was lower than may have been expected at 12 hours.

Review of the detailed log sheets showed there was adequate reagent and dissolved oxygen present. However, the NaCN consumptions (see Table 13-12) were very low as was the lime consumption.

Dissolved oxygen levels were nominally 20mg/L or lower, being lower than most of the smaller scale tests and a possible contributor to slow kinetics. No pre-oxygenation was undertaken.

The reason for these slow kinetics has not been established. However, it would seem from the NaCN characteristics that there was some issue with either NaCN purity, concentration and/or measurement as the test results suggest NaCN starvation. Additionally, a temperature effect can be expected given the ambient temperature applied.

As the samples were retained for a period of time prior to being used for carbon characterization, additional leaching was able to result. The contact solution grade for the carbon characterization was determined at 1.09mg/LAu. This equates to a final extraction at extended time of 92.4% for the P80 53µm test as applied.

The extended leach time behaviour shows the liberation and typical leach extraction characteristics remained, but the kinetics were impacted for some unproven yet suspected reasons. It was not practical to repeat these bulk tests due to limitations on sample availability.

The slurry was used for carbon characterization test work. Kinetic testing and equilibrium testing values achieved being:

- **Kinetic:** ΔAuc = k[Aus]tn where:
  - ΔAuc = change in carbon loading in g/m3.
  - k = constant (intercept) determined to be 139.5 h-1 from the test work.
  - Aus = solution concentration of gold, g/m3.
  - T = time, h
  - n = constant (slope) determined to be 0.735 from the test work.
- **Equilibrium:** log X/M = m log C + log K where:
  - X/M = mg of gold absorbed per g of carbon (at equilibrium).
  - C = gold remaining in solution g/m3.
  - m = constant (slope) determined to be 0.393 from the test work.
  - K = constant (intercept) where log K determined to be 3.731 from the test work.

Figure 13-17 and Figure 13-17 present the test work results for the kinetic and equilibrium testing.

Figure 13-16: Carbon Kinetics, 6 x 12 mesh

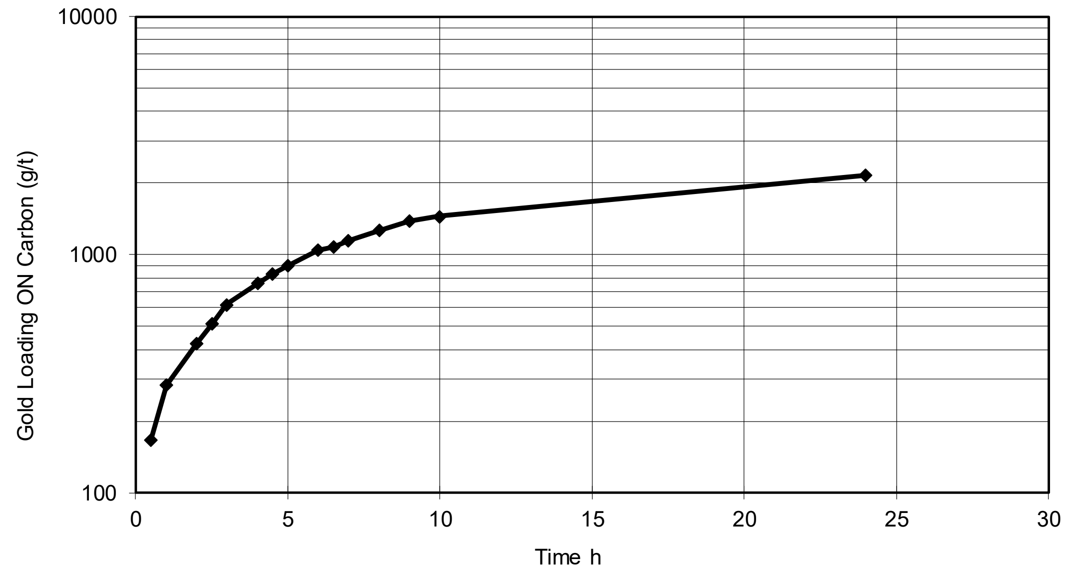
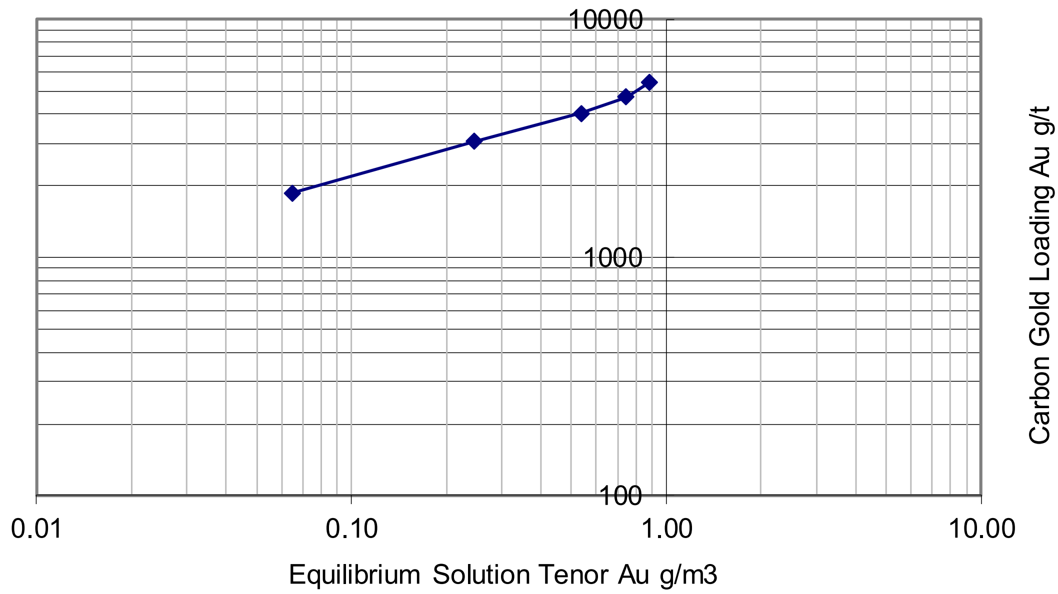


Figure 13-17: Carbon Equilibrium, 6 x 12 mesh



Results – Oxygen Uptake

An oxygen uptake test was conducted on the MG Composite bulk samples having a P80 of 53µm and 45µm. The tests conducted at a pH of nominally 12 and used oxygen as the sparging gas. The tests were not conducted at elevated temperature.

A number of the hourly readings reported elevated oxygen demands that do not align with the adjacent readings. Determining oxygen demands of supercharged slurries (oxygen concentrations higher than equilibrium) can produce such outcomes and therefore some consideration as to the issues of such testing must be allowed for.

Results are summarised by Table 13-13 and Table 13-14 including summary results of the demand calculations.

Table 13-13: MG Composite 45 µm Grind Oxygen Demand(1)

mins	@ 0 h	@ 1 h	@ 2 h	@ 3 h	@ 4 h	@ 5 h	@ 6 h	@ 24 h	@ 32 h	@ 48 h
0	7.27	39.04	36.43	40.99	42.84	41.76	47.05	43.61	40.04	39.14
1	7.1	39.02	36.35	40.99	42.82	41.7	46.78	44.57	39.99	39.09
2	7.05	38.97	36.3	40.99	42.82	41.25	46.11	44.53	39.98	39.06
3	7.01	38.96	36.23	40.99	42.8	41.25	46.03	44.46	39.96	39.06
4	6.96	38.83	36.16	40.93	42.8	41.23	45.72	44.43	39.95	39.05
5	6.92	38.76	36.09	40.93	42.79	41.17	45.63	44.04	39.91	39.02
6	6.87	38.65	36.02	40.88	42.76	41.06	44.81	43.37	39.86	39
7	6.83	38.42	35.94	40.85	42.7	40.59	44.67	43.21	39.83	38.98
8	6.8	38.42	35.86	40.82	42.65	40.49	44.16	42.87	39.8	38.95
9	6.76	38.33	35.69	40.73	42.63	40.37	44.07	42.74	39.77	38.93
10	6.73	38.28	35.62	40.6	42.57	40.26	43.98	42.69	39.71	38.92
11	6.64	38.19	35.56	40.52	42.56	40.18	43.55	42.63	39.68	38.9
12	6.6	38.07	35.5	40.48	42.52	39.54	43.39	42.4	39.63	38.89
13	6.57	37.99	35.42	40.43	42.49	39.19	43.16	42.38	39.6	38.87
14	6.56	37.97	35.34	40.35	42.43	39.03	42.96	42.27	39.58	38.86
15	6.53	37.92	35.26	40.29	42.42	39	42.84	42.24	39.54	38.84
mg/L/min	-0.045	-0.084	-0.080	-0.051	-0.030	-0.189	-0.287	-0.168	-0.034	-0.019

(1) Averages: All values -0.099. First 6 hours -0.109. +6 hours -0.074 mg/L/min. Excluding high values (yellow), average of remaining tests is -0.049 mg/L/min

Typically, the highest oxygen demands are found in the earliest time frames. As the oxygen consumers are sated, the oxygen demand falls away with time. The 48 hours demand is probably mostly due to oxygen concentration equilibrating with the atmosphere, and this value should be considered as a discount when assessing the design oxygen demand.

Table 13-13 presents early oxygen demands in the order of -0.08mg/L/m for the first two hours and a characteristic decay up until t = 5 h. The 5, 6 and 24 hour demands present values that appear atypical.

Table 13-14 presents lower demands that shown by Table 13-13. This may be due to less reactive sulphide surface area being present at the coarser grind or could be due to lower starting concentrations of oxygen which means atmospheric losses are reduced or sulphide oxidation rates are lower.

As these tests were not undertaken at temperature, the rate of sulphide oxidation is lower that what would be found in a full-scale plant. The equilibrium oxygen levels will be lower at elevated temperature which lower the sulphide oxidation rate in a full-scale plant. As such, there are conflicting considerations when establishing a design oxygen demand.

The demands are not high by industry standards and given the lack of reactive sulphides observed in the ore, this is to be expected. Designing for an oxygen demand of -0.1mg/L/min for $0<t<6$h and -0.05mg/L/min for $t>6$ would be expected to be conservative. However, additional comment is provided in Section 13.5.3 and revised criteria provided per Table 13-33.

Table 13-14: MG Composite 53 µm Grind Oxygen Demand

mins	@ 0 h	@ 1 h	@ 2 h	@ 3 h	@ 4 h	@ 5 h	@ 6 h	@ 24 h	@ 32 h	@ 48 h
0	7.13	31.95	32.76	33.33	42.04	40.89	44.61	41.03	40.11	38.45
1	6.87	31.95	32.76	33.33	42.02	40.87	44.5	40.92	39.89	38.47
2	6.32	31.95	32.75	33.33	42.01	40.77	44.41	40.98	39.89	38.44
3	6.1	31.95	32.75	33.24	42.01	40.77	44.37	40.98	39.87	38.43
4	5.95	31.95	32.75	33.2	41.99	40.76	44.27	40.98	39.86	38.41
5	5.86	31.95	32.74	32.95	41.89	40.76	44.16	40.98	39.8	38.38
6	5.79	31.95	32.73	32.8	41.86	40.75	44.09	40.98	39.82	38.37
7	5.72	31.95	32.71	32.56	41.83	40.73	44	40.98	39.8	38.35
8	5.65	31.94	32.68	32.8	41.82	40.65	43.86	40.98	39.77	38.33
9	5.62	31.93	32.63	32.24	41.8	40.61	43.33	40.97	39.77	38.31
10	5.58	31.93	32.57	32.2	41.79	40.56	42.99	40.97	39.76	38.29
11	5.54	31.93	32.53	32.11	41.76	40.53	42.58	40.95	39.76	38.26
12	5.5	31.92	32.44	32	41.72	40.46	41.96	40.83	39.73	38.25
13	5.47	31.91	32.37	31.94	41.72	40.41	41.9	40.78	39.7	38.23
14	5.44	31.91	32.29	31.84	41.67	40.37	41.52	40.7	39.68	38.21
15	5.41	31.9	32.25	31.27	41.63	40.29	41.31	40.67	39.64	38.2
mg/L/min	-0.094	-0.003	-0.034	-0.132	-0.028	-0.037	-0.232	-0.018	-0.021	-0.019

(1) Averages: All values -0.062. First 6 hours -0.080. +6 hours -0.019 mg/L/min. Excluding high values (yellow), average of remaining tests is -0.032 mg/L/min

Results – Rheology

Slurry rheology testing was undertaken on the MG Composite at two different grind sizes and at different pulp densities for a range of shear rates. All tests were conducted at a pH of nominally 12.0 and at ambient temperature. Tests were conducted pre-leach and post-leach.

Of note, the viscosity of the slurries increased post-leaching which suggests reaction by-products may be contributing to rheological behaviour.

Viscosity increases with a finer grind and increased pulp density. It being noted that the slurries are presented a spread of both shear thinning at lower shear rates and then minor increased viscosity at the highest shear rates.

The determinations suggest there should be no issues in agitating or pumping such slurries with conventional equipment over the ranges of viscosity and at the shear rates anticipated relevant to the selected flowsheet. Over-thickened slurries may require dilution facilities to be provided at pump suctions.

Results are summarised by Table 13-15.

Table 13-15: MG Composite Viscosity Data

Composite ID	Solids %	Viscosity (cPs) at Nominated Shear Rate (sec^{-1}) 4.2	7.4	13.1	21.9	38.9	67.4	119.2	209.5
Pre-Leach									
Medium Grade P80: 53µm, pH 12	60	0	149	144	136	141	156	167	198
	50	0	0	0	0	36	40	51	85
Medium Grade P80: 45µm, pH 12	60	674	425	300	237	190	180	202	218
	50	0	0	0	0	28	33	49	75
Post-Leach									
Medium Grade, P80: 53µm, pH 12	60	1085	680	444	330	226	152	158	177
	50	0	0	0	0	40	40	53	79
Medium Grade, P80: 45µm, pH 12	60	1647	998	648	452	307	208	189	194
	50	0	0	0	50	40	49	57	79

Results – Cyanide Detoxification

A sample of the MG Composite post bulk leach was subjected to continuous SO_2/air cyanide detoxification. Three SO_2:WAD CN ratios were tested at 4.89, 4.48 and 3.56 grams of SO_2/g WAD CN. Only the highest dose was found to provide a consistent outcome.

The pH of the sample was initially reduced to nominally 8.5 as the feed pH of around 12 would not allow the detoxification process to proceed. Once the testing was underway, hydrated lime slurry had to be dosed to counter the acid associated by dosing sodium meta-bisulphite, the SO_2 source. In the full-scale operation, the incoming new feed would replace some of the hydrated lime demand.

Detail of conditions and outcomes from detoxification test D1 are presented as follows:

- Operating pH: 8.54;
- Retention time: 56 minutes;
- SO_2 dose: 4.89kgSO_2/kg WAD CN;
- Copper dose as Cu: 109g/m^3 solution;
- Hydrated lime 60% CaO equiv: 0.91kg/kgSO_2;
- Free CN- in feed: 187mg/L;
- WAD CN in feed: 261mg/L;
- Cu in feed solution: 57.7mg/L (present as WAD CN);
- Other metals: Insignificant – not to be considered; and
- Effluent CN WAD <5mg/L.

There are a number of considerations with regards to this work:

- The free cyanide concentration was high. This has resulted in a high copper dose being required, and a value of double what would be considered high by industry norms. The sample was prepared with a high free cyanide dose, so this needs to be considered in full-scale design and when estimating operating costs. Particularly when it has been shown that much lower cyanide doses will provide high leach extractions.

- Only one pH range was tested. In the full-scale plant, there will be opportunity to experiment with a range of pH. The author is familiar with another project hosting auriferous tellurides where the operable pH range for SO2/air was found to be around 10; and
- These tests were not run at elevated temperature which may impact kinetics as well as reduce the dissolved oxygen concentration. This aspect being addressed by subsequent testing reported below.

13.5.2 Summary A18001

The A18001 program has provided support for a flowsheet that can consistently provide high leach extractions and has provided a number of design criteria that can be taken forward to full-scale plant design.

Several points have been raised in the foregoing regarding test conditions, shortcomings and also positive outcomes. A number of the compromises/shortcomings observed per program A18001 have been addressed in later work, namely, program A19476 discussed below.

13.5.3 ALS Program A19476

Aim

This program was focussed on understanding variability of the deposit and also providing data to allow the development of an extraction or recovery algorithm to be applied for reserve estimation and financial modelling.

Other work included work index determinations, oxygen uptake testing, cyanide detoxification, carbon characterization and filtration required to allow process design to be implemented.

Sample Condition

Previous programs had relied on historical drill core samples and these same sample sources were proposed for this program. It was decided that given the age of the core and remnant samples, that there needed to be evaluation of the core condition.

Samples were taken from drill holes SPETI-28, MRP0045, MRP0009, MRP 0014 and MRP0001 and were subjected to water permeability observation and optical observation of oxidation.

The water permeability work was used to give a qualitative indication of vugginess and interconnection of pores that might offer paths for oxidative solutions to pass. The SPETI-28 sample showed true permeability whereas the other samples did not show significant permeability apart from where vugs and veining was noted.

The more important investigation of the conditions of the sulphides present showed that there was very little oxidation. Some tarnish was observed on exposed surfaces, but fresh sulphides exposed by cutting the core showed no weathering related oxidation.

The work suggested the samples remained in good condition and would be suitable as a basis for this ongoing work.

Samples

To understand the variability of the deposit required the building of a number of Locality Composites. These were generated from core originating from eighteen (18) different drill holes. Twenty-seven (27) Locality Composites were made up. Some of the drill holes providing more than one composite. Some being contiguous, others representing different zones/lenses if a break in grade was present in the same hole.

Details of the Locality Composites are presented by Table 13-16. Note ALS refer to these composites as Variability Composites in their report.

Table 13-16: Details of Locality Composites

Drill Hole	Start interval m down hole	Finish interval m down hole	Composite name	Zone(1)
SPETI 028	86.0	90.0	VAR01	HW
SPETI 027	73.0	83.0	VAR02 A	M
SPETI 027	83.0	93.0	VAR02 B	M
MRP-017	130.0	136.0	VAR03 A	M
MRP-017	136.0	141.0	VAR 03B	M
MRP004	128.0	133.0	VAR04	HW
MRP004	145.0	157.0	VAR05	M
MRP-019	203.0	215.0	VAR06	M
SPETI 011	110.0	116.0	VAR07	HW
MRP 045	181.0	190.0	VAR08	FW
MRP003	95.6	104.0	VAR09	HW
MRP003	115.0	125.0	VAR10	M
SPETI 017	88.0	91.0	VAR11	HW
MRP 009	54.0	63.0	VAR12A	M
MRP 009	63.0	71.0	VAR12B	HW
MRP015	5.6	9.3	VAR13	M
MRP015	140.0	149.0	VAR14	M
MRP014	106.0	113.0	VAR15A	M
MRP014	113.0	121.0	VAR15B	M
MRP 022	223.0	234.0	VAR16	M
MRP003	23.0	31.0	VAR17A	M
MRP003	31.0	39.0	VAR17B	M
MRP001	32.0	41.0	VAR18	M
18P052	206.0	211.0	VAR19	FW
MRP001	196.0	201.0	VAR20	M
MRP001	233.0	241.0	VAR21	M
MRP043	208.0	216.0	VAR22	M

(1) HW = Hanging Wall Zone; FW = Foot Wall Zone; Main = Main Zone

In addition to the Locality Composites, a composite was made up to be subjected to additional cyanide detoxification and carbon characterization test work. This composite is referred to in the ALS work as the Detox Composite. This same sample was also dispatched to Outotec for additional thickening and filtration test work.

Program

The test work program included:

- Head assay/ICP analysis of the Locality Composites;
- Additional BBWi test work on locality samples;
- Cyanidation leaching of Locality Composites including some variation in leach conditions. This work to assist in deriving an extraction/recovery algorithm;
- Some grind sensitivity work to understand if there was variability in performance across the deposit given earlier grind sensitivity work was conducted on wide ranging composites;
- Preparation of a composite made up of variability sample remnants for cyanide detoxification, carbon characterization and filtration test work;
- Viscosity testing;
- Oxygen uptake testing; and
- Vendor (Outotec) thickening and filtration testing.

Results – Head Assays

To understand the variability that might be present and displayed by the samples, detailed head assays and SG determinations were undertaken.

A summary of the results is presented by Table 13-17. Results shown as NA mean the determination was below detection limits.

It will be noted the range of the values is quite limited and shows the variability of the samples, at least on elemental basis, is very low. The silver, copper, zinc and other cyanide soluble elements are consistent and suggest the influences of these elements regarding process design implications will be minor.

The sulphide sulphur assays range is similarly low suggesting consistency in sulphide content, at least by primary gold ore expectations. The Te assay follows the sulphide assay parameters closely. The tellurium grade can be expected to influence the metallurgical responses.

Table 13-17: Locality Composite Head Assays and SG Ranges

ANALYTE	Min	Max	Average	Median
Ag(ppm)	0.3	1.2	0.7	0.6
Al(%)	7.2	9.2	8.1	8.1
As(ppm)	NA	NA	NA	NA
Au(ppm)	0.5	3.8	1.5	1.2
Au(ppm)_rpt1	0.5	3.8	1.5	1.1
Ba(ppm)	405	1000	648	620
Be(ppm)	NA	NA	NA	NA
Bi(ppm)	NA	NA	NA	NA
C(%)	0.1	0.6	0.3	0.3
C org(%)	NA	NA	NA	NA
Ca(%)	1.4	4.1	2.3	2.2
Cd(ppm)	NA	NA	NA	NA
Co(ppm)	5.0	25.0	11.3	10.0
Cr(ppm)	10.0	110.0	43.7	40.0
Cu(ppm)	114	568	268	232
Fe(%)	2.0	5.2	3.1	2.9
K(%)	1.6	3.6	2.6	2.6
Li(ppm)	5.0	10.0	7.9	10.0
Mg(%)	0.4	2.2	1.0	0.9
Mn(ppm)	500	1300	867	900
Mo(ppm)	5.0	60.0	21.7	20.0
Na(%)	1.0	4.3	3.2	3.4
Ni(ppm)	5.0	40.0	17.5	15.0
P(ppm)	300	1000	544	500
Pb(ppm)	5.0	30.0	14.1	15.0
S(%)	0.7	3.1	1.5	1.6
S-2(%)	0.6	3.1	1.3	1.3
SiO2(%)	55.6	69.4	62.2	62.0
Sr(ppm)	184	346	253	236
Te(ppm)	1.2	7.0	3.2	2.8
Ti(ppm)	2000	5400	3378	3400
V(ppm)	28.0	140.0	65.8	60.0
Y(ppm)	NA	NA	NA	NA
Zn(ppm)	52.0	322.0	91.3	70.0
SG	2.7	3.0	2.8	2.8

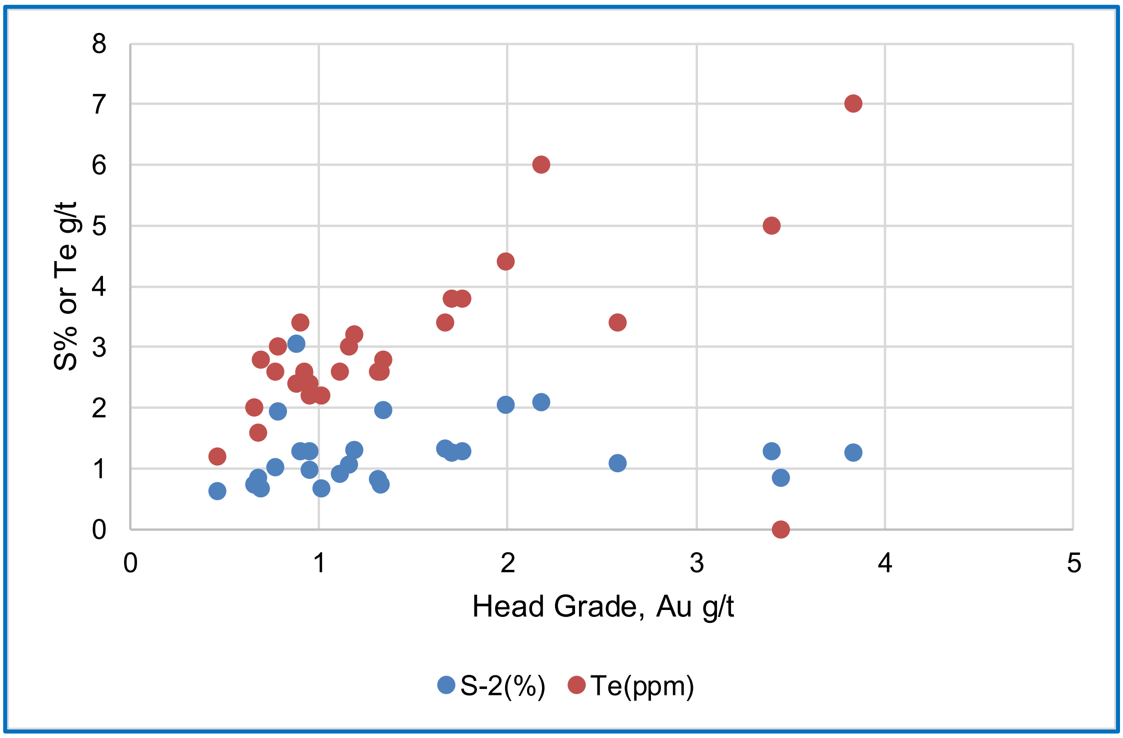
Table 13-18: Locality Composite Head Assays and SG Ranges presents a summary of assay data by Hanging Wall and Foot Wall ore types. This table again shows little elemental difference between these two categories, supporting the comments made about the lack of variability noted in Section 13.3.2 when considering metallurgical leach responses. This is not to say that the mineralogical deportment and/or physical characteristics are not different, but that with regard to deleterious elements or those elements that might be expected to drive the metallurgy, there is little variation.

Table 13-18: Locality Composite Head Assays and SG Ranges

Type	HW				FW			
Parameter	Average	Min	Max	Median	Average	Min	Max	Median
Ag(ppm)	0.6	0.3	1.2	0.6	0.7	0.3	1.2	0.6
Au(ppm)	1.1	0.5	3.8	1.1	1.7	0.7	3.8	1.3
Au(ppm)_rpt1	1.0	0.5	3.8	1.0	1.7	0.6	3.8	1.4
Co(ppm)	10	10	25	10	12	5	25	10
Cu(ppm)	243	118	568	225	274	114	568	232
Pb(ppm)	15	5	20	15	14	5	20	15
S(%)	1.3	0.7	3.1	1.3	1.6	0.8	3.1	1.6
S^{-2}(%)	1.1	0.6	3.1	1.0	1.3	0.7	3.1	1.3
SiO2(%)	62.3	58.8	69.4	62.2	62.3	55.6	69.4	62.2
Te(ppm)	2.4	1.2	7.0	2.5	3.5	2.0	7.0	3.0
Zn(ppm)	127	54	168	73	83	52	168	74
SG	2.8	2.8	3.0	2.8	2.8	2.7	3.0	2.8

Figure 13-18 presents the relationship between gold head grade and the sulphide and tellurium (telluride) head grades. There does appear to be a relationship present for the gold-tellurium couple, yet the gold-sulphide relationship is considered poor based on this data.

Figure 13-18: Carbon Kinetics, 6 x 12 mesh



Results – BBWi Determinations

A selection of the Locality Composites was subjected to BBWi determinations using a 75µm closing screen. The resultant grinds produced a P80 of nominally 63µm. As the grind size anticipated for the Project was 53µm, a selection of the tests were repeated with a 63µm closing screen to provide nominally 53µm P80 products and the associated BBWi results.

The two sets of results are presented per Table 13-19.

Table 13-19: Locality Composite BBWi Results

Composite ID	P_{80} µm	BBWI kWh/t	P_{80} µm	BBWI kWh/t
Var02A02B	65	17.1	52	18.8
Var03A03B	64	16.7	52	18.9
Var5	63	16.9	52	18.9
Var06	66	16.8	52	18.7
Var09	65	16.0	54	17.5
Var10	67	15.4	53	17.2
Var12A12B	58	16.2	53	18.0
Var14	64	17.4	53	19.7
Var15A15B	64	15.5		
Var16	62	19.8	51	22.9
Var17A17B	66	16.6		
Var18	65	18.4	52	20.3
Var22	65	15.7		

The results presented show that to achieve the finer grind results in a significant increase in the BBWi value.

Histograms comparing the distributions are presented as Table 13-19 and Figure 13 20 to represent the nominal 63µm and 53µm grind outcomes. These plots suggest that the 85th percentile BBWi for a 63µm grind is nominally 17.5kWh/t and for a 53µm grind, 20.5kWh/t.

Figure 13-19: Carbon Kinetics, 6 x 12 mesh

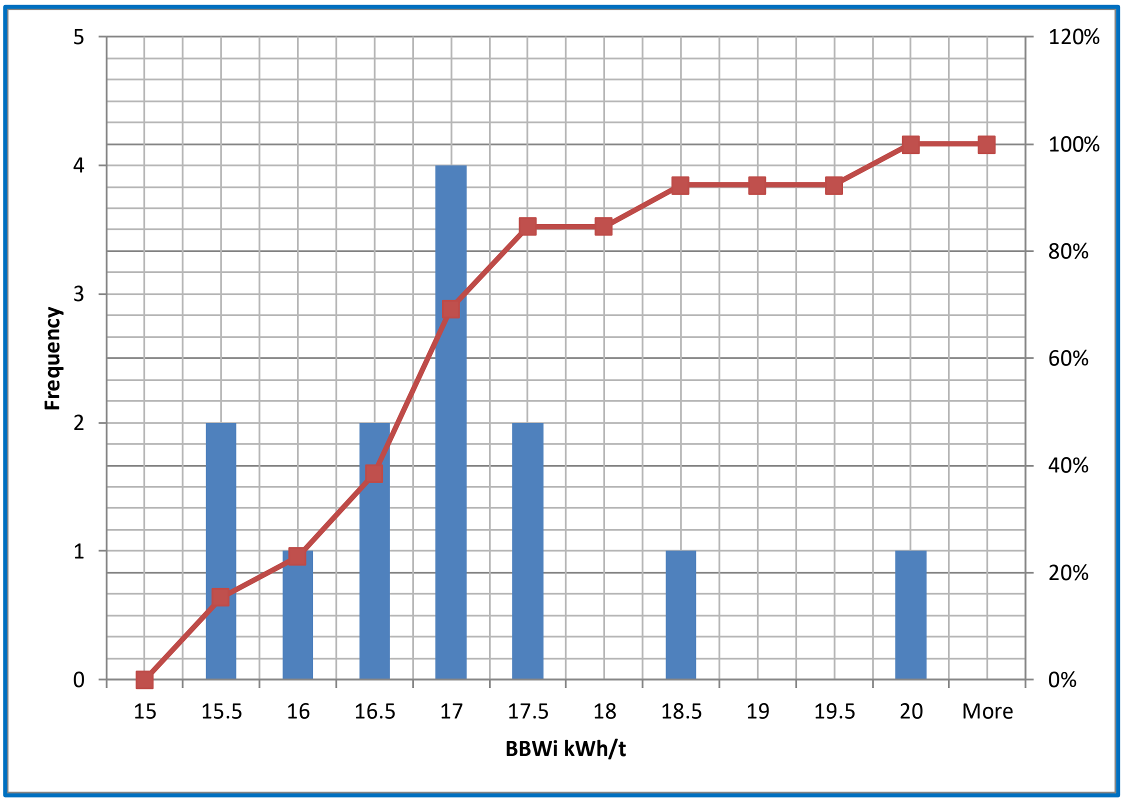


Figure 13-20: Locality Composite BBWi at 53µm P80 Product

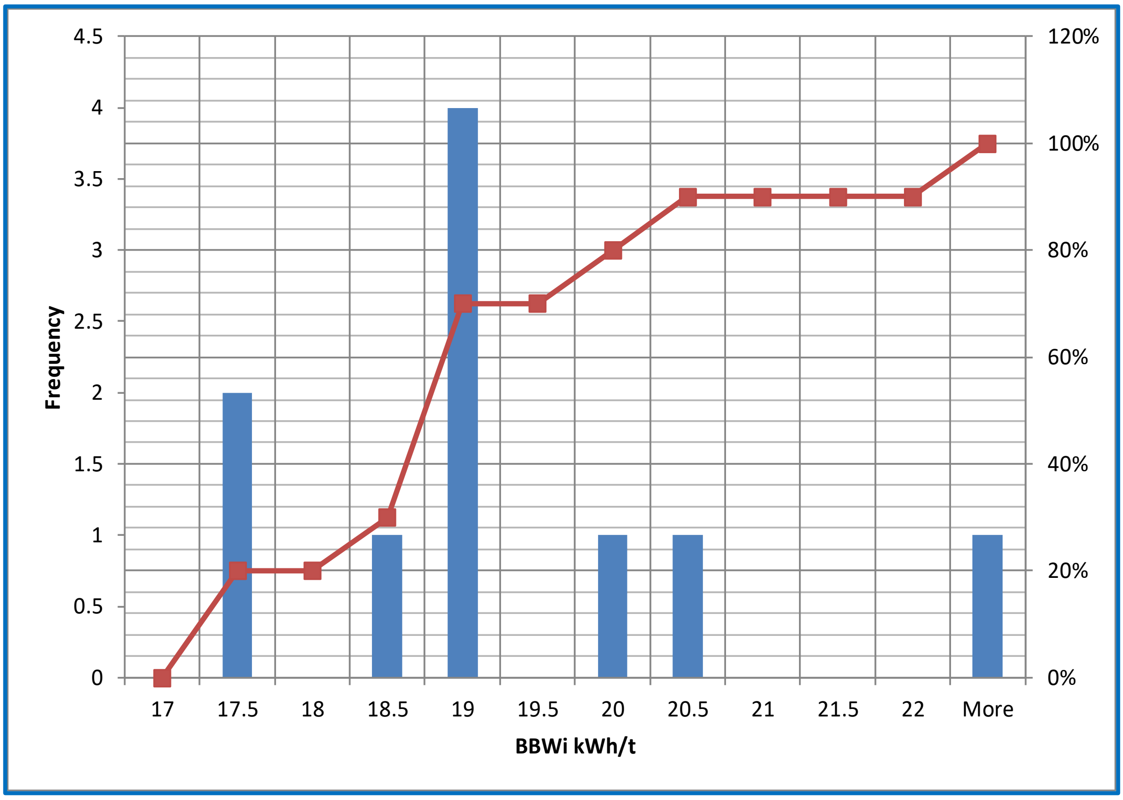
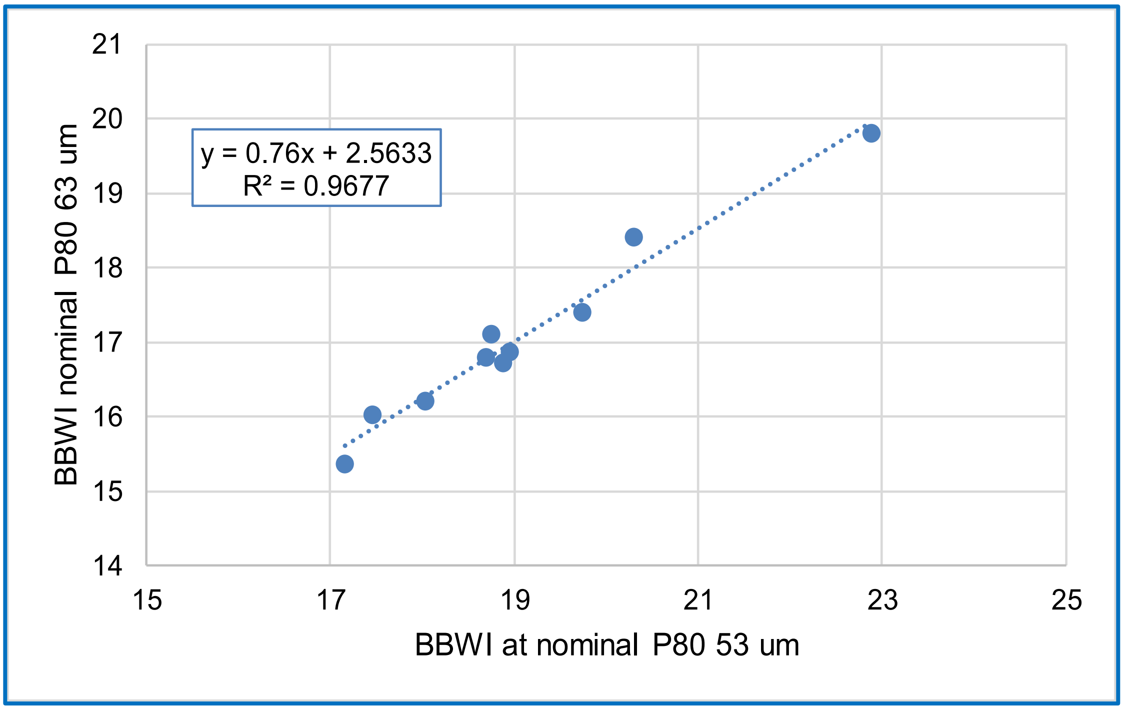


Figure 13 21 presents a plot of the BBWi determinations at the different product sizes. A strong relationship is shown. When attempting to estimate the 50th percentile from the histograms, the histograms require interpolation. It can be interpreted the 50th percentile values for the

BBWi at 63µm is around 16.75kWh/t and for 53µm around 18.75kWh/t. These values align with the relationship presented per Figure 13-21.

Figure 13-21: Locality Composite BBWi P80 63 and 53µm



When the data for the MG Composite BBWi (refer Table 13-20) is subjected to the relationship presented per Figure 13-21, the 53µm and 63µm closing screen results align.

This information suggests the various samples have some similar lithological characteristics being indicated by similar comminution aspects. The LG and MG Composites present BBWi values of 17kWh/t and 16.3kWh/t for a 63µm product, both values being close to the 50th percentile value presented for the Locality Composites. As the LG and MG Composites comprise a wider spread of components compared to the Locality Composites, such relationship alignment should be expected if these broader range composites are representative.

Results – Leach Tests

The test work has identified a number of test work parameters that influence the final leach extractions and reagent demands. Whilst some of these parameters are typical of free milling ores, such as grind size and residence time, the Mara Rosa material shows variability as a function of temperature and pH, and at times, aeration/oxygenation. Not all of these parameters influence typical free milling ores.

In the full-scale operation, there will be a need to understand the leach conditions to be applied as a function of the characteristics of the ore presenting to the mill. This is a typical practice for an operational plant. In the Mara Rosa case, there will be a need to cover off additional variables such as to establish pH and reagent consumption balances against potential enhanced leach extraction opportunities.

Leach tests were performed on the Locality Composites at a pH of 12.0 and at 35°C for 36 hours having a residual NaCN concentration of 150mg/L maintained. These conditions considered to be contained within the likely operating range for a full-scale plant. It is likely that elevated temperatures will present over the warmer months at full-scale and will provide elevated extractions. Similarly, the full-scale operation will have the ability to increase the lime addition and improve extractions if found to be cost effective. However, for understanding the variability of the ores and to provide a basis for extraction/recovery prediction, a fixed set of leaching conditions need to be applied. In the example of Mara Rosa, these conditions may not be the optimum for each sample tested.

Two grind sizes were tested. Eight (8) of the composites were tested at a P80 of 75µm as at the time of the program initiation, Amarillo and their engineering consultant were evaluating if coarsening the grind would be cost effective. The remaining twenty-eight (28) tests were conducted at a P80 of 53µm, being the grind that was taken forward for process design.

These leach tests were also used to provide an extraction/recovery algorithm which is discussed in more detail in Section 13.7.

Table 13-20: Locality Composite BBWi Results

Sample ID	Grind P_{80} µm	Residue grade Au g/t	Final Extrn % 36 h	Calc Head Au g/t	Assay Head Au g/t	NaCN Cons kg/t	Lime Cons kg/t	Assay Head Te g/t
VAR - 14	53	0.30	88.5	2.61	2.18 / 1.75	0.33	3.57	6.0
VAR - 22	53	0.20	91.1	2.24	2.58 / 2.13	0.35	4.59	3.4
VAR - 02B	53	0.27	91.8	3.27	3.40 / 3.66	0.19	3.12	5.0
VAR - 17A	53	0.21	94.2	3.52	3.45 / 3.26	0.20	2.33	0.2
VAR - 05	53	0.35	90.6	3.71	3.83 / 3.81	0.41	2.56	7.0
VAR - 09	75	0.33	74.8	1.31	0.46 / 0.47	0.10	1.98	1.2
VAR - 10	75	0.15	79.8	0.72	0.66 / 0.61	0.13	2.32	2.0
VAR - 07	75	0.07	91.1	0.73	0.68 / 0.69	0.14	1.79	1.6
VAR - 15A	75	0.33	82.7	1.91	0.69 / 0.99	0.13	2.10	2.8
VAR - 03A	75	0.11	86.9	0.80	0.77 / 0.73	0.16	2.27	2.6
VAR - 08	75	0.11	84.4	0.67	0.46 / 0.47	0.22	2.18	3.0
VAR - 16	75	0.10	91.1	1.07	0.88 / 2.82	0.10	2.29	2.4
VAR - 15B	75	0.19	83.2	1.13	0.90 / 0.97	0.21	2.36	3.4
VAR - 04	53	0.09	91.1	0.96	0.95 / 0.91	0.29	4.43	2.2
VAR - 06	53	0.13	93.5	1.92	1.67 / 1.73	0.37	4.40	3.4
VAR - 11	53	0.10	95.6	2.16	1.34 /1.34	0.22	4.35	2.8
VAR - 04	53	0.08	92.0	0.94	0.95 / 0.91	0.21	3.37	2.2
VAR - 12A	53	0.10	89.4	0.94	1.11 / 1.00	0.27	3.75	2.6
VAR - 13	53	0.09	93.8	1.46	1.33 / 1.40	0.15	3.35	2.6
VAR - 17B	53	0.17	91.8	2.00	1.99 / 1.80	0.42	4.05	4.4
VAR - 18	53	0.09	92.4	1.19	1.19 / 1.13	0.21	3.70	3.2
VAR - 19	53	0.11	91.6	1.25	1.01 / 1.13	0.21	2.92	2.2
VAR - 20	53	0.18	88.1	1.52	1.31 / 1.30	0.19	4.29	2.6
VAR - 21	53	0.09	90.6	0.96	0.92 / 0.72	0.19	3.49	2.6
VAR - 06	53	0.13	92.6	1.77	1.67 / 1.73	0.32	4.45	3.4
VAR - 11	53	0.09	93.6	1.32	1.34 /1.34	0.14	2.76	2.8
VAR - 12A	53	0.09	91.0	0.94	1.11 / 1.00	0.21	3.54	2.6
VAR - 19	53	0.12	90.5	1.21	1.01 / 1.13	0.23	3.60	2.2
VAR - 21	53	0.10	89.6	0.92	0.92 / 0.72	0.21	4.19	2.6
VAR - 09	53	0.05	89.8	0.49	0.46 / 0.47	0.17	3.76	1.2
VAR - 15A	53	0.05	95.6	1.14	0.69 / 0.99	0.20	3.52	2.8
VAR - 16	53	0.09	92.6	1.15	0.88 / 2.82	0.14	3.35	2.4
VAR - 15B	53	0.07	93.2	1.02	0.90 / 0.97	0.21	2.84	3.4
VAR - 12B	53	0.12	93.8	1.95	1.76 /1.48	0.31	4.35	3.8
VAR - 12B	53	0.14	92.6	1.83	1.76 /1.48	0.49	3.92	3.8
VAR - 12B	53	0.11	94.4	1.88	1.76 /1.48	0.47	3.58	3.8

Table 13-20 presents:

- A spread of head grades for both gold and tellurium;
- It shows relatively consistent and high extractions are achieved suggesting consistent metallurgical behaviour;
- As the samples originated from different depths and along strike, as well as various ore "types" of Hanging Wall, Foot Wall and Main, there appears to be little leaching behaviour difference across the deposit;
- High extractions continue to be achieved even at the higher tellurium head grades. This suggests the telluride oxidation remains effective as tellurium grade increases;
- A number of samples have presented higher calculated head grades than the assay heads suggesting there is some nugget effect. This needs to be considered when assessing the data; and
- Sodium cyanide consumption is consistent whereas there is more variation in the lime consumption.

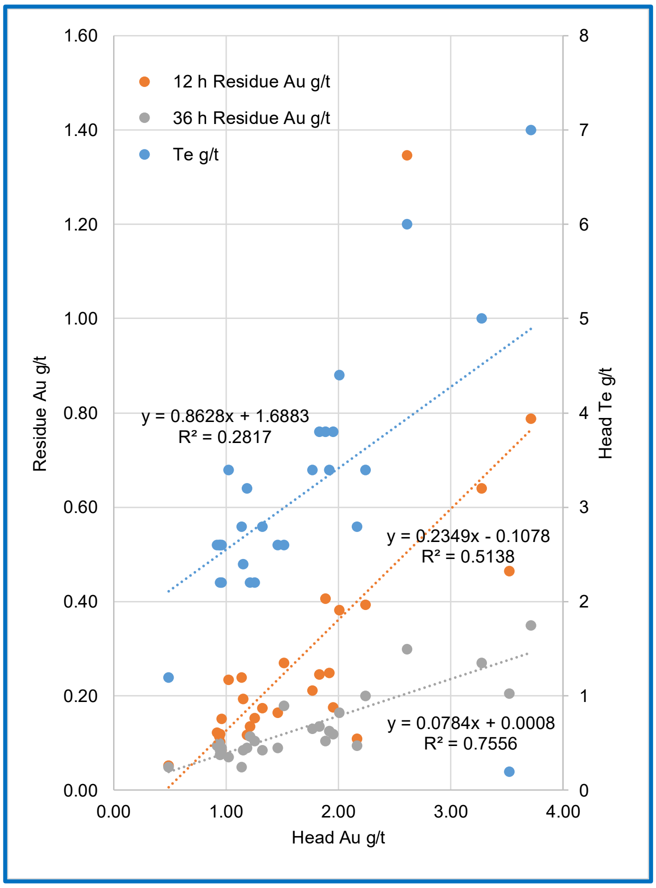
Figure 13-22 and Figure 13-23 present leach test behaviour for the 53µm tests reported per

Table 13-20.

Figure 13-22 shows the residue grade at 12 hours leach time and 36 hours leach time plotted against the gold head grade. The plot also shows the tellurium head grade as a function of gold head grade. The scatter in the results gives poor correlation when linear lines of best fit are applied. It should be noted here that the author also experimented with other forms of line of best fit including linear through the origin, and results were not as robust.

The plot does suggest the higher the gold grade, the higher the tellurium grade as has been presented above (Figure 13-18). What is also suggested is that the leach extraction is lower at the higher tellurium grades for the 12h data (determined from leach solution grades) compared to the 36h data given the 12h plot (blue) is steeper than the 36h data. That is, at 12h leach time there is a suggestion that oxidation of the tellurides is incomplete and additional time is required to oxidise/degrade them. This is logical and to be expected based on how other telluride containing ores behave.

Figure 13-22: Locality Composite Behaviour at 53 µm – 1



It is unclear if the elevated residue grades at the higher head grades are due to a lock up of gold simply as a function of gold head grade or if it is due to lock-up in tellurides. Nor is it possible to ascertain if the telluride lock up is due to a consistent proportion of non-digested telluride or some other gold-telluride association that is a function of tellurium head grade. More detailed analytical work would be needed to assess this. However, the point is somewhat moot given the high extractions achieved and the knowledge that to increase extractions above what is being achieved will probably be uneconomic.

The likelihood is the residual gold is locked up as a function of both gold and tellurium head grades.

Figure 13-23 explores the tellurium head grade association with the 12h and 36h residue grades. Again, the author evaluated various forms of equation and those presented gave the highest correlation. The plot shows the same potential lack of telluride digestion at 12h compared to 36 h as was suggested by Figure 13-22.

Figure 13-23: Locality Composite Behaviour at 53 µm – 2

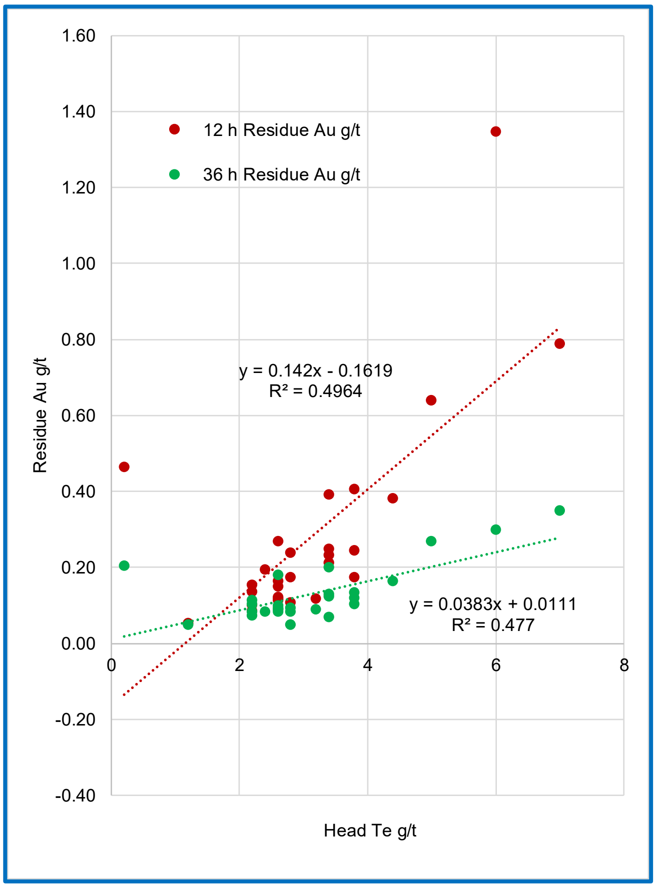


Figure 13 23 suggests the tellurium grade is not as effective in describing leach residue as the gold head grade is. This conclusion is presented based on the higher correlation coefficient for the gold relationship compared to the tellurium relationship.

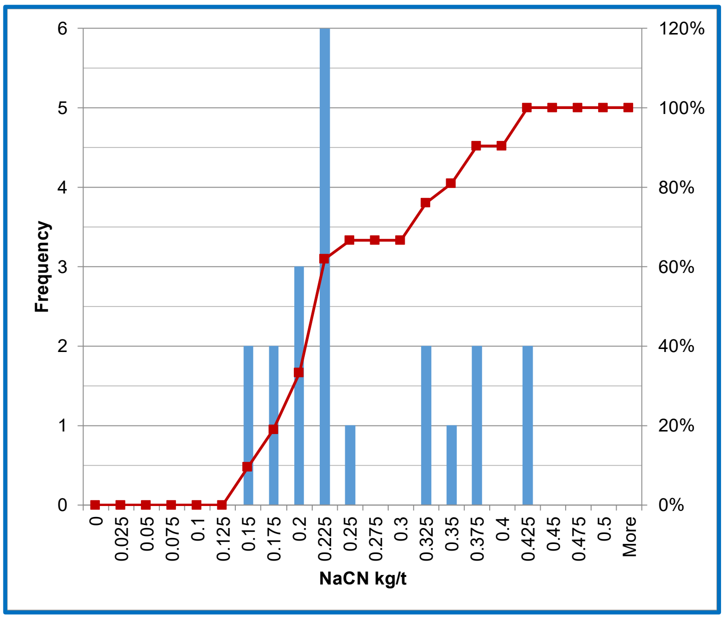
There are a number of repeat tests present in the data set. Repeats were conducted if it was observed there was poor reagent control (pH control) or assay results that did not reflect expectations. Some tests were repeated due to kinetic curves being atypical. The repeat tests gave consistent final leach extractions and generally similar reagent consumption outcomes as their partner tests. There are cases where the repeats provided more typical kinetic curve shapes and similar leach characteristics when variables such as pH were better controlled.

Results – Reagent Consumption of NaCN and Lime

Reagent consumption for the Locality Composites is summarised by the histograms Figure 13 24 and Figure 13 25. The data is drawn from twenty-one (21) of the Locality Composite tests in that it excludes duplicates.

Figure 13 24 presents a range of 0.15kg/tNaCN to 0.425kg/tNaCN consumption with the 50% value being between 0.2kg/t and 0.225kg/t. A value of 0.215kg/t can be taken as the average for operating cost purposes.

Figure 13-24: Locality Composite NaCN Consumption



There were a number of tests where no additional NaCN was dosed during the test. This suggests that in the full-scale operation, NaCN consumption can be expected to be lower. Similarly, full-scale plant often presents lower NaCN consumption than test work due to the volume : area ratio of the slurry being so much greater in full-scale than in test work conditions, and so gassing losses are reduced. Be that as it may, operating cost values should be based on the 0.215kg/t value presented above.

In addition to the actual consumption estimated from the test work is a need to consider the residual NaCN exiting the CIL/CIP. The target residual NaCN is 150mg/L and so this value is an additional consumption less any NaCN that could potentially be returned.

The proposed flowsheet requires cyanide detoxification prior to filtration, and consequently, there is no cyanide credit available from thickening or filtration recycles. The final estimate of NaCN consumption will need to be estimated as a function of the final water balance around the tailings circuit.

Figure 13-25: Locality Composite Lime Consumption

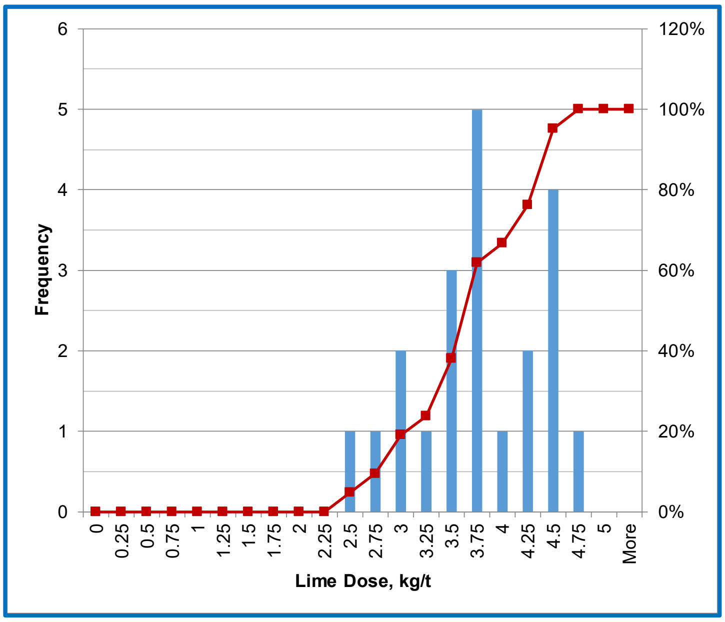


Figure 13 25 presents the lime consumption data.

As has been highlighted above, the term “Lime” applied by ALS refers to hydrated lime ($Ca(OH)2$) consisting of nominally 65%CaO to 68%CaO. Therefore, whilst these results are providing estimates of lime consumption, the consumption of the actual reagent to be dosed in a full-scale operation and the associated costs of such a reagent will need to be adjusted according to the purity of “quicklime” or “hydrated lime” supplied to site. Noting that commercial “quicklime” may typically be of the range of 80%CaO to 85%CaO equivalent, but low-quality suppliers may provide reagent at 65% CaO equivalent and good suppliers +90% (rare).

The lime consumption data from the Locality Composites ranges between 2.5kg/t and 4.75kg/t. The average is between 3.5kg/t and 3.75kg/t, so a value of 3.65kg/t average is considered applicable for operating cost estimates.

Results – Carbon Characterization

A sample of the Detox Composite (made up of a number of the same intervals used for Locality Composites) was leached and the slurry used for carbon characterization test work using Haycarb YAO 6 x 12 mesh carbon.

Figure 13-26 and Figure 13 27 present the results in graphical form.

Figure 13-26: Detox Composite Carbon Kinetics 6 x 12 mesh

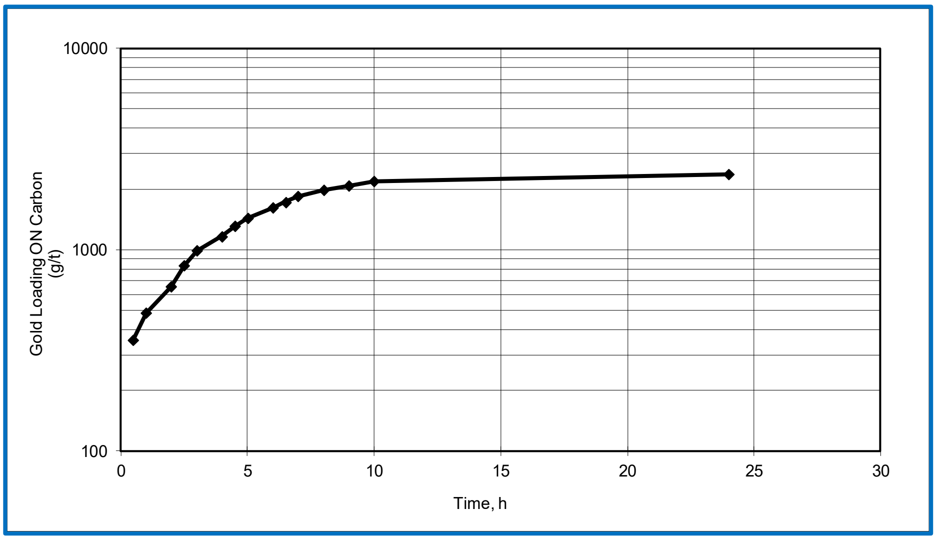
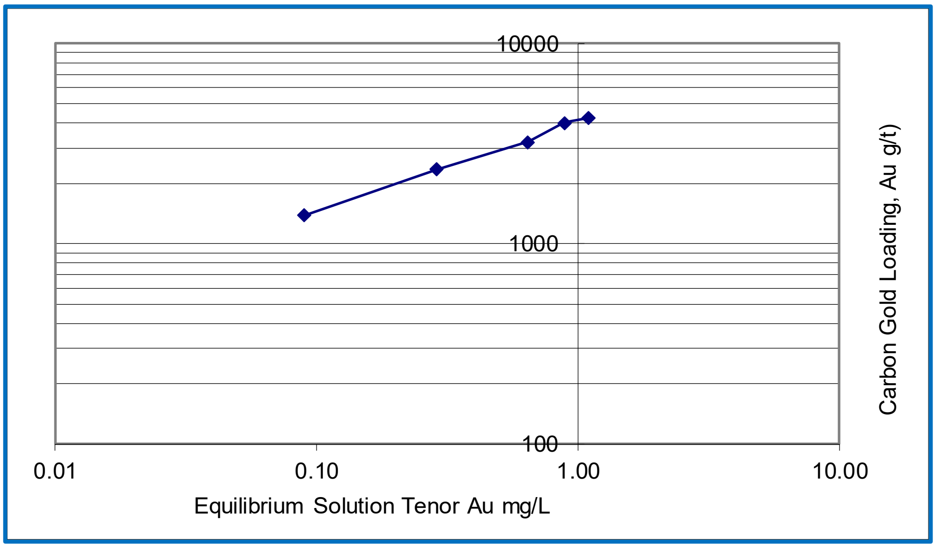


Figure 13-27: Detox Composite Carbon Equilibrium 6 x 12 mesh



Kinetic and equilibrium parameters as determined from the test work are presented as:

- **Kinetic:** k = constant (intercept) determined to be 261 h-1 from the test work.
  - n = constant (slope) determined to be 0.623 from the test work.
- **Equilibrium:** m = constant (slope) determined to be 0.451 from the test work.
  - K = constant (intercept) where log K determined to be 3.614 from the test work.

This data suggests better kinetics than the carbon characterization work reported in ALS report A18001, but poorer equilibrium values.

A possible reason for the improved kinetic values is potentially that the contact solution grade in the work presented per Figure 12-26 and Figure 12-27 was 1.38 mg/LAu, whereas the A18011 work, the contact grade was 1.09 mg/L Au. The kinetic curve presents by Figure 13 26 flattens out and this suggests approaching equilibrium, which depresses the "n" value.

Reason as to why the equilibrium values are not as good as the earlier carbon characterization test work per program A18001 are not apparent. Comparative solution assays are not available to ascertain if there are active species present that may have impacted carbon loading.

With regards to design, it is considered appropriate to use the slowest kinetics and the lowest equilibrium loading values when Table 13-21 undertaking process design.

Results – Oxygen Uptake

Oxygen uptake tests were undertaken as part of ALS program A18001 and were summarised as Table 13-13 and Table 13-14 above. This work was undertaken on the MG Composite and at ambient temperature. To address the issue of temperature impact and assess the impact of supercharged dissolved oxygen levels (values higher than equilibrium due to oxygen sparging), a further series of tests were run.

A number of Locality Composites were blended to represent the originally defined ore types of Main, HW and FW. Tests were run at 50% solids and pH 12 with both air and oxygen for the Main and HW composites, and air alone for the FW composite due to a lack of sample. Results are summarised per Table 13-21.

Table 13-21: Locality Composite BBWi Results

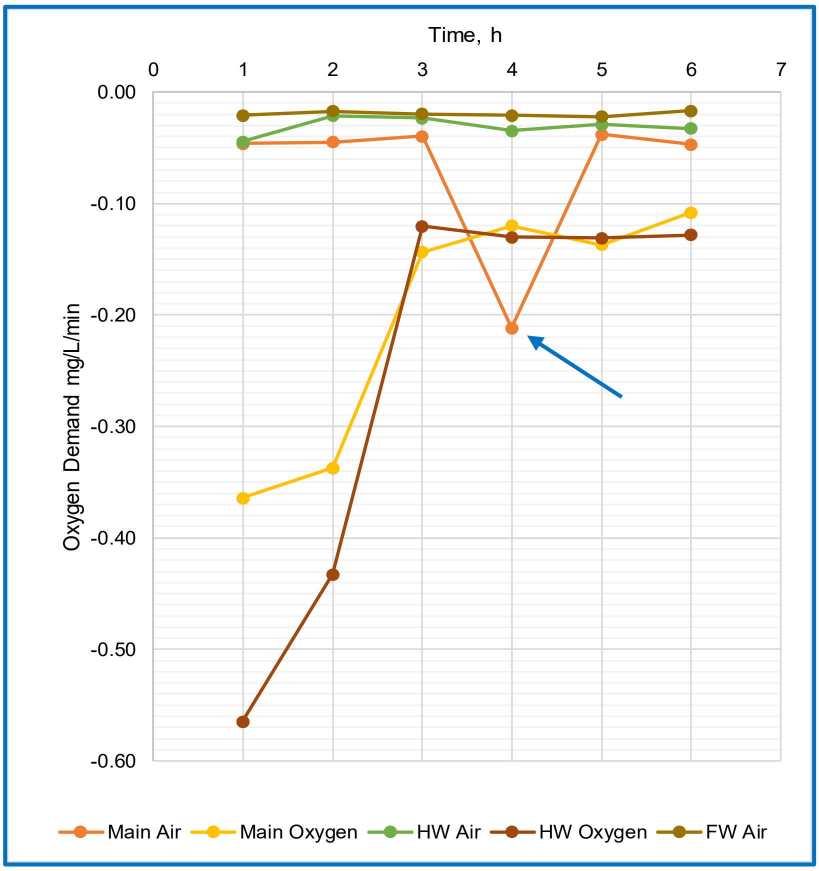
Time (hours)	Oxygen Uptake Rate (mg/L/min)				
	Main (Air)	Main (Oxygen)	HW (Air)	HW (Oxygen)	FW (Air)
0	-0.1077	-0.0836	-0.0874	-0.0579	-0.1061
1	-0.0463	-0.3640	-0.0444	-0.5646	-0.0210
2	-0.0448	-0.3371	-0.0213	-0.4328	-0.0172
3	-0.0397	-0.1437	-0.0235	-0.1205	-0.0195
4	-0.2117	-0.1200	-0.0349	-0.1300	-0.0209
5	-0.0379	-0.1372	-0.0290	-0.1309	-0.0220
6	-0.0469	-0.1080	-0.0327	-0.1281	-0.0169
24	-0.0174	-0.3048	-0.0092	-0.2883	-0.0042

The results from the A18001 program lead to the recommendation of designing for an oxygen demand of -0.1mg/L/min for 0<t< 6h and -0.05mg/L/min for t>6h and it would be expected to be conservative. It will be noted when evaluating Table 13-21 that for those cases where air has been sparged, these previous values are applicable. However, when oxygen has been applied, the oxygen demands are considerably higher when compared to air or the previous A18001 oxygen sparging data.

The data from Table 13-21 was plotted and is presented as Figure 13-28 for the first six (6) hours of sparging. There is one spurious result shown by the blue arrow that should be ignored.

The graph shows very high oxygen demand in the first two (2) hours which then flattens out for the next four (4) hours at around -0.12 to -0.14 mg/L/min. The air sparged tests present comparative oxygen consumptions of -0.02 to -0.04 mg/L/min.

Figure 13-28: Oxygen Demand Main, HW and FW



There are a number of factors that will drive the oxygen demand including:

- Reaction with solids, the rate of which may increase at elevated temperature and/or elevated dissolved oxygen concentrations;
- Loss to the atmosphere, which would be anticipated to be minor for air sparged tests but increased for tests that use oxygen as the dissolved oxygen levels are higher than equilibrium values with the atmosphere; and
- Loss to the atmosphere will also be increased with elevated temperature as the equilibrium dissolved oxygen concentration decreases with temperature.

It is not possible to identify the various mechanisms at play here without more complex testing apparatus. However, if it is assumed that after three (3) hours the consumption of oxygen by the solids is mostly sated, this would then indicate the loss to atmosphere for the oxygen sparged tests is the difference in the ore demand between the oxygen and air sparged tests. This value being nominally 0.1mg/L/min. Furthermore, the reality of the test work is that the surface area to volume ratio is around 30 to 40 times greater for the laboratory system compared to the full-scale plant tankage. Losses to atmosphere in the full-scale plant are much less for supercharged systems.

The true oxygen demand of the ore is therefore difficult to determine. It is suggested that the oxygen demand for oxygen sparged systems be discounted to address atmospheric losses. It is also noted that:

- Oxygen demands are not high by industry standards; and
- The three ore types of Main, HW and FW seem to have little difference in oxygen demand.

It is further suggested that:

- An oxygen demand be evaluated based on the design tank residence times by applying a mean of the oxygen demand over the same time period as presented by Figure 13-28. A discount of 0.1mg/L/min then be applied to address excess oxygen losses to atmosphere experienced by the test work. For example, a 4-hour tank residence for the first tank in circuit would apply the average oxygen demand for the 4-hour period for a design value of -0.312 + 0.1 = -0.212mg/L/min for the worst case HW results;
- All tanks post the first tank would apply a value of -0.05mg/L/min for t>4 hours which is considered to be aligned with the A18001 test work; and
- Full-scale plant design to include facility to increase oxygen sparging volumes, if necessary, by including additional sparge points and/or facility to add proprietary sparging systems.

Results – Viscosity Testing

The same Main, HW and FW composites built from Locality Composites and used for oxygen uptake work were subjected to viscosity test work at pH 12. Results of the tests conducted at various pulp densities and shear rate are summarised as Table 13-22.

Table 13-22: Locality Composite BBWi Results

Composite ID	Solids %	Viscosity (cP) at Nominated Shear Rate (sec ')							
		4.2	7.4	13.1	21.9	38.9	67.4	119.2	209.5
Main	60	3743	2974	2292	1931	1475	956	636	443
	55	561	361	264	179	129	98	96	142
	50	0	0	96	79	57	54	59	87
	45	0	0	0	50	44	42	49	65
	40	0	0	0	0	36	30	33	53
HW	60	2470	1806	1200	861	525	380	282	266
	55	101 1	680	444	301	206	152	138	140
	50	412	255	168	115	81	75	75	90
	45	0	0	0	50	48	44	50	65
	40	0	0	0	0	32	33	37	53
FW	60	1273	807	612	467	331	247	21 2	217
	55	449	319	216	158	113	100	99	107
	50	0	149	96	72	61	56	61	80
	45	0	0	0	0	36	35	42	56
	40	0	0	0	0	0	26	36	44

The samples present results with similar characteristics to earlier viscosity test work. The values present shear thinning slurries at lower shear rates followed by some shear thickening at the highest shear rates. The values are not considered problematical for pumping or agitation over the range of pulp densities that would be typical of the proposed flowsheet. Although it is noted at the higher pulp densities viscosity could become problematical for centrifugal pumping. If trends continued at pulp densities higher than tested such as excessively high pulp density from atypical plant operations, then these could be troublesome. An example being overly high pulp density thickener underflow.

It is noted that the three different samples do show different rheological properties. The Main material being more viscous than the HW which is in turn more viscous than the FW. As the Main material is the dominant ore type, design will need to consider the exposure to overly high pulp density resulting from atypical operations.

These samples were selected based on what has been noted to be a superseded lithological basis. However, these results suggest that whilst the test work on leaching presents little if any difference in behaviour, there is a likelihood that the lithology may influence some physical behaviour.

Results – Cyanide Detoxification

Earlier cyanide detoxification test work (program A18001) used a feed sample with high free cyanide concentration and was run at a temperature of nominally 22°C. As program A19746 progressed and cyanide concentrations were reduced, it was deemed appropriate to repeat the detoxification test work at the lower feed CN WAD concentrations. Similarly, testing the influence of temperature was addressed.

A sample of the Detox Composite post bulk leach was subjected to continuous SO_2/air cyanide detoxification at 22°C with a one-hour residence time. Three SO_2:WAD CN ratios were tested at 5.5 (stable at 5.2), 4.8 (actual 5.17 to 4.5) and nominally 3.7 grams of SO_2/g WAD CN. The highest two doses were found to provide a consistent outcome even though the 4.8 grams of SO_2/g WAD CN ratio was initially less stable when the SO2 dose was reduced to 4.5 grams of SO_2/g WAD CN. Dissolved oxygen levels of around 8mg/L were retained. These results align with the previous work.

The pH of the sample was initially reduced to nominally 8.7 – 8.9 as the feed pH of around 12 would not allow the detoxification process to proceed. Hydrated lime slurry was not required to be dosed to counter the acid associated by dosing sodium metabisulphite. This was a marked difference from the earlier A18001 test work program where, whilst the SO_2 ratio was effectively the same, the actual volume of reagent dosed per volume of sample was around one third in this later work. Less SO_2, less acid generated, less pH modifier required.

As the pH had equilibrated under the test conditions, and assuming the same conditions in the full-scale plant, the full-scale plant would not be expected to require continuous acid dosing. However, to initiate the detox reaction, either a shock dose of acid will be required or overdosing with sodium metabisulphite will be necessary to lower the pH of fresh feed.

The latter option presents a risk of de-oxygenating the slurry, which could be expected to be problematical. It does remain a practical solution to the issue and is a low capital approach. This would be a rare event and only necessary if the detoxification reactor has been drained or partially drained for maintenance or extended shut-down of the plant. To address the oxygen issue, a simple approach would be to install an oxygen sparge to supplement oxygen demand prior to start-up of the reactor but post the sodium metabisulphite “overdose” for pH control.

The option of allowing for an acid dosing system should be considered but with due consideration of ensuring there is no significant release of HCN during pH adjustment.

A second round of tests was undertaken at nominally 35°C. These tests were undertaken at three SO2:WAD CN ratios of 4.56, 3.67 and 2.76 grams of SO_2/g WAD CN. Note that the last ratio of 2.76 is not much greater than stoichiometric.

Dissolved oxygen levels of nominally 6g/L were maintained in the warmer slurry. This is a key observation as it was a concern that the elevated temperature may result in dissolved oxygen levels that would be too low to facilitate the reaction.

The first test at 4.56 grams of SO2/g WAD CN was initially unstable as the pH dropped below 8.5. On increasing pH to 8.5 the detoxification was effective. This suggests a pH of >8.5 is required.

The test at 3.67 grams of SO_2/g WAD CN provided good performance and maintained low WAD CN levels, whereas the test at 2.76 grams of SO_2/g WAD CN was unstable and performed inconsistently.

The work at elevated temperature suggests there may well be a need for hydrated lime dosing and that SO_2 doses may be able to be reduced. This observation is conflicting as lower SO_2 doses would suggest pH would remain elevated. Given the observation in the test work that the lime is consumed, this may well continue in the detoxification step. There will be a loss of calcium due to gypsum precipitation, but this does not explain the loss of hydroxide ion and therefore lowering of the pH. These mechanisms remain unexplained from the test work observations. However, this does not detract from having a basis of design.

Detoxification plant start-ups, particularly at low temperature (post extended shut-downs) will need a method of reducing pH to start the reactor off as described above. Oxygen supplementation will be a necessary facility, albeit one used rarely. Hydrated lime dosing will be necessary as well as the anticipated SMBS, copper sulphate and oxygen dispersion systems.

The following criteria are presented as ranges as interpreted from the detoxification test work.

- Operating pH: 8.5 – 8.9;
- Retention time: 60 minutes minimum;
- SO2 dose: 4.0 – 5.2 kg SO_2/kg WAD CN;
- Copper dose as Cu: 54g/m^3 solution (dose will be a function of feed concentration WAD CN);
- Ca(OH)2 dose: 0 – 6.3kg/kgSO_2;
- Free CN- in feed: 67mg/L;
- WAD CN in feed: 127mg/L;
- Cu in feed solution: 51mg/L (present as WAD CN);
- Other metals: Minor influence and can be ignored; and
- Effluent CN WAD <5mg/L.

13.5.4 Summary A19476

This program has provided confidence that high leach extractions can be achieved for a number of samples representing various areas in the deposit. The consistency of outcomes suggests little variability in metallurgy when the P80 53µm and pH 12.0 operating conditions at high dissolved oxygen concentrations (nominally 30mg/L) are applied. In addition, the ores do not appear to be overly NaCN sensitive at residual NaCN concentrations of greater than 150mg/L to 180mg/L.

Bulk samples subjected to carbon characterization and cyanide detoxification have provided design criteria in alignment with typical values and confirmed the SO_2/air cyanide detoxification process is suited. It is noted that the cyanide detoxification process may require acid dosing under some conditions to achieve the operating pH necessary for the reaction to continue.

13.5.5 Outotec Thickening Test Work Nov 2018

A sample of the P80 45µm bulk sample prepared from MG Composite was dispatched to Outotec in November of 2018 for flocculant screening and dynamic thickening work. This work is summarised by Outotec report S2103.

Outotec used a laser sizer to determine the P80 of the sample and measure a value of 66µm. This was in conflict with the value of 45µm as presented by ALS. Following repeat sizing work, the value of P80 45µm was confirmed as correct.

The sample Outotec tested was leach tailings at a pH of 12. Given rheology test work showed the pre-leach samples behaved differently (less viscous) to post-leach samples, the application of this post leach data generated by Outotec must be applied with some caution to pre-leach duties. It is not possible to say categorically if the pre-leach performance would be better or worse without testing. However, the expectation would be the pre-leach performance would be improved given the lower viscosity.

Outotec tested four flocculants and selected Magnafloc 10 as the reagent to take forward. This provided the best clarity and high settling rates. Various flux rates were trialled at differing flocculant doses. Results are summarised by Table 13-23.

Table 13-23: Locality Composite BBWi Results

Run No.	Feed Flux (t/(m^{2}h))	Liquor RR (m/h)	Flocculant Type	Dose (g/t)	Underflow Meas. Solids (% (w/w))	YS (Pa)	Overflow Solids (mg/L)
1	0.50	3.02	M10	20	62.8	47	<100
2	0.50	3.02	M10	10	62.7	36	<100
3	0.50	3.02	M10	5	62.7	26	<100
4	0.25	1.51	M10	20	64.9	71	<100
5	1.00	6.03	M10	20	58.9	34	<100
6	1.50	9.05	M10	20	56.5	28	<100

Performance is very good even at low flocculant dose rates in the 5g/t to 10g/t range and at low flux rates of 0.50t/m^2.h. Increasing the flux rate to 1.0t/m^2.h presents a drop in performance and even at 20g/t flocculant dose rates, an underflow pulp density of less than 60% solids results.

There is a trade-off here to compare capital cost of the thickener with operating cost associated with the flocculant and understanding the benefit of high underflow pulp densities. Consideration should also be given to the downstream cost of cyanide detoxification/cyanide management.

13.5.6 Outotec Filtration Test Work Feb 2019

A sample of the P80 45µm bulk sample prepared from the MG Composite was subjected to filtration test work by Outotec in February of 2019. This work is summarised by Outotec report 318437.

As described in the previous section, Outotec used a laser sizer to determine the P80 of the sample and measure a value of 66µm. This was in conflict with the value of 45µm as presented by ALS. Following repeat sizing work and confirmation, the value of P80 45µm was confirmed as correct.

Filtration tests were conducted a differing feed pulp densities and a differing feed rates using belt filter, fast operating filter press and chamber filter press methods. Testing was conducted at an ambient temperature of 23°C and a pH of 11.9. No washing and no filtration aids were used.

Results of the test work are summarised per Table 13-24.

Table 13-24: Outotec Filtration Test Work Results Feb 2019

Parameter	Horizontal Vacuum Belt	Fast Operating Filter Press	Chamber Filter Press
Test Filtration Rate kgDS/m^2/h	531	324	329
Cake Moisture Content, %	17.5	12.7	14.4
Cake Thickness, mm	15	57	58
Solids in Filtrate, mg/L	1220	610	610
Cake Density, kg/L		~ 2	~ 2
Average Drying Air Consumption, l/min		10	18
Total Cycle Time, min		7	6.75

The results show all three technologies are capable of filtering the tailings at 45µm.

The ability to handle and transfer the horizontal belt filter cake is questioned for a consistent long term operation. However, the 17.5% moisture cake produced appeared visually to be handleable. Also, as the sample has a P80 of 45µm and the project basis is to grind to 53µm, a horizontal belt filter may still be a viable option.

This fast operating or chamber filter press technologies are preferred on a process basis with regards to moisture content and handling. The comparative costs of these two technologies will need to be established considering the lower filtration rates and also the cost of air drying. Note that both of these press technologies have similar cycle times.

Outotec have reported the filter cake density to only one significant figure. This leaves some interpretation open as to application of a design value.

13.5.7 SMC Test Report – SGS Geosol Brasil

Samples of Mara Rosa material were sent to SGS Geosol Brasil for SMC testing in September of 2019. The drill holes used are detailed per Table 13-1. The intervals were composited as instructed by Amarillo and the composites subjected to SMC tests to establish Drop Weight and Mi parameters to allow derivation of A, b, ta and SCSE values.

Three composite samples were derived so as to represent a mix of mineralised lithologies. A high grade (Alto Teor), medium grade (Teor Medio) and low grade sample (Baixo Teor) were generated by Amarillo from previously mined rock available on site.

Table 13-25 and Table 13-26 provide a summary of results.

The DWi values obtained are in the mid to mid-high range according to the SMC data base. The Baixo Teor sample being considered in the hard range.

The respective Mi parameters all fall in the third quartile of the SMC data base and in the case of the Baixo Teor sample, the fourth quartile of the SMC data base suggesting ores are in the medium hard to hard ranges.

The product of the parameters A x b is a measure of the resistance to breakage. The A x b values for the samples tested again fall in the upper two quartiles, with the Baixo Teor sample at 85% in the JK data base.

These results all suggesting the sample tested are harder and tougher than the averages of the ores in the SMC/JK data bases.

Table 13-25: DWI and Mi Parameters

Sample Designation	DWi (kWh/m^3)	DWi (%)	Mi Parameters (kWh/t)			SG
			Mia	Mih	Mic	
Alto Teor	7.41	60	21	15.8	8.2	2.72
Baixo Teor	9.25	81	24.6	19.4	10.1	2.76
Teor Medio	6.93	54	19.8	14.7	7.6	2.73

Table 13-26: Mill Design Parameters

Sample Designation	A	b	t_a	SCSE (kWh/t)
Alto Teor	78.3	0.47	0.35	10.3
Baixo Teor	74.6	0.40	0.28	11.52
Teor Medio	70.5	0.56	0.37	9.99

13.5.8 Abrasion and Crushing Work Indices – SGS Geosol Brasil

The same three samples subjected to SMC testing described per Section 13.5.7 were subjected to Abrasion Index (Ai) testing and Crushing Work Index (CWi) determinations by SGS.

The results show the samples have high to very high abrasion characteristics and high wear of liners and grinding media can be expected. Results are summarised per Table 13-27.

Table 13-27: Abrasion Indices

Sample Designation	Ai g
Alto Teor	0.3593
Baixo Teor	0.3956
Teor Medio	0.3507

13.5.9 Outotec Thickening and Filtration Test Work March 2020

Outotec were requested to undertake filtration testing of a sample of cyanide detoxification tailings as had been prepared as part of ALS program A19475. Refer to Section 13.5.3 for detail of the sample and conditions.

Outotec had previously conducted filtration test work on a P80 45µm sample (refer Section 13.5.5) whereas the sample submitted and discussed herein was a P80 53µm sample aligned with the proposed flowsheet. Previous work had explored Horizontal Belt, Fast Operating Filter Press and Chamber Filter Press options whereas for this this work, Outotec were requested to only test the Fast Operating and Chamber Filter Press option at various pulp densities.

Results are presented per Table 13-28.

Table 13-28: Abrasion Indices

Parameter	45% solids Fast Operating Filter Press	Chamber Filter Press	55% solids Fast Operating Filter Press	Chamber Filter Press	60% solids Fast Operating Filter Press	Chamber Filter Press
Test Filtration Rate kgDS/m^2/h	299	337	316	340	314	334
Cake Moisture Content, %	14.7	14.7	13.5	13.5	13.6	14.2
Cake Thickness, mm	57	60	59	60	59	60
Solids in Filtrate, mg/L	<100	150	<100	150	<100	150
Cake Density, kg/L	~ 2	~ 2	~ 2	~ 2	~ 2	~ 2
Average Drying Air Consumption, l/min	8	9	9	10	10	14
Total Cycle Time, min	7.25	6.75	7.0	6.75	7.0	6.75

These results suggest there is little benefit to be had in thickening the filter feed (as far as filtering alone goes and ignoring detoxification and recycle of values benefits) apart from reducing the feed system capacity and any stock tank volume.

The capacities recorded are similar to the earlier work on the finer P80 45µm sample. Moisture values are similar in the final cakes produced. The cake densities are again reported at a value of nominally 2. This is consistent with the earlier work but again only provides one significant figure. Consequently, some scope remains undefined with regard to establishing a firm design value based on the Outotec data.

The filtrate clarity was improved in this round of work, with values at 150mg/L of solids present or less.

All filter cakes produced were presented by Outotec as suitable for handling suggesting suitable for transport to a dry stack tailings facility.

It is noted here that geotechnical work undertaken by Amarillo's tailings consultants have nominated a moisture content of less than 19% is required for tailings compaction meaning either technology tested is suited.

Outotec report 326264 presents a range of air blow times and moistures. It is apparent that the two filter types can be operated to generate overlapping conditions and outcomes. The Chamber Filter Press does require a 12-bar feed system compared to the Fast-Operating Filter Press at 6 Bar. Selection of either technology on a pure process basis is supported. Capital and operating cost analysis is required to provide a preferred technology.

13.5.10 ANDRITZ Filtration Test Work

Samples of Mara Rosa material were sent to SGS Geosol Brasil in September of 2019. The drill holes used are detailed per Table 13-1. The intervals were composited as instructed by Amarillo and the single composite prepared by SGS as a 53µm slurry. This slurry was then forwarded to ANDRITZ for filtration testing.

ANDRITZ undertook a series of tests at 40% solids and 50% solids feed pulp density. The intent being to establish if filtration would be effective with the potential to exclude a tailings thickener from the flowsheet. Results are presented per Table 13-29 and Table 13-30 respectively.

The results show pressure filtration can achieve moisture levels of nominally 13% at either 40% or 50% feed pulp density, with a slight benefit at the higher pulp density and with membrane pressing. The membrane press achieving lower moisture levels at lower air blow rates.

The tests were conducted at ambient temperature. Slightly better filtration rates and lower cake moistures may be found at elevated temperatures. Therefore, the data presented herein may well be slightly conservative.

No cake density values were provided per the test work report but have been relayed via correspondence from Andritz to Amarillo as having a wet cake value of 2.01kg/m^3 – 2.02kg/m^3. These values align with the Outotec test work.

Table 13-29: ANDRITZ Filtration Results – 40% solids feed

Test	1	2	3
PULP TEMPERATURE (°C)	Ambient	Ambient	Ambient
pH	12	12	12
% SOLIDS PULP:	40	40	40
PACKAGE TYPE	Recess	Recess	Recess
CHAMBER THICKNESS (mm)	50	50	50
FEED PRESSURE (kgf/cm^2)	6	6	6
MEMBRANE PRESSURE (kgf/cm^2)	0	0	15
CAKE MOISTURE (%)	13.70	13.00	12.80
AIR RATE (Nm^3/h.m^2)	300	500	300
FILTER MEDIA TYPE	249	249	249
FINAL CAKE THICKNESS (mm)	50.00	50.00	48.00
COMPRESSION FACTOR	1.00	1.00	0.96
CAKE DISCHARGE	Easy	Easy	Easy

Table 13-30: ANDRITZ Filtration Results – 50% solids feed

Test	4	5	6
PULP TEMPERATURE (°C)	Ambient	Ambient	Ambient
pH	12	12	12
% SOLIDS PULP:	50	50	50
PACKAGE TYPE	Recess	Recess	Recess
CHAMBER THICKNESS (mm)	50	50	50
FEED PRESSURE (kgf/cm^2)	6	6	6
MEMBRANE PRESSURE (kgf/cm^2)	0	0	15
CAKE MOISTURE (%)	13.50	12.90	12.70
AIR RATE (Nm^3/h.m^2)	300	500	300
FILTER MEDIA TYPE	249	249	249
FINAL CAKE THICKNESS (mm)	50.00	50.00	48.00
COMPRESSION FACTOR	1.00	1.00	0.96

13.5.11 TEFSA Filtration Test Work

A sample of the same P80 53µm slurry material prepared by SGS Geosol as had been sent to ANDRITZ was dispatched to TEFSA's laboratories in October of 2019 and subjected to filtration testing.

The sample pulp density and pH were adjusted to 40% solids and pH 12 prior to being subjected to membrane filter press testing. A moisture content of 18.8% was achieved after air blow. Given the press has a 30mm chamber and a membrane pressure of 16 Bar was applied, the results are not as good as had been achieved by ANDRITZ.

The final moisture content is close to the nominal 19% maximum required to be achieved for tailings compaction in the tailings stack. The filter cake density was also low at 1.39kg/m^3. This work would need to be repeated if TEFSA were to be considered as a vendor.

Results are summarised by Table 13-31.

Table 13-31: TEFSA Filtration Result Summary

Test	4
PULP TEMPERATURE (°C)	Ambient
pH	12
% SOLIDS PULP:	40
PACKAGE TYPE	Recess
CHAMBER THICKNESS (mm)	30
FEED PRESSURE (bar)	6
MEMBRANE PRESSURE (bar)	16
CAKE MOISTURE (%)	18.8
FILTER MEDIA TYPE	P-297
CAKE DENSITY (g/cm^3)	1.39
FINAL CAKE THICKNESS (mm)	23 - 26
FEED TIME (min)	7
SQUEEZE TIME (min)	2
BLOW TIME (min)	2

13.5.12 Brasfelt Filtration Test Work

A third cut of the SGS Geosol P80 53µm slurry sample was sent to Brasfelt for membrane press testing. Brasfelt did not provide a formal report but did conduct eight tests at various feed pulp densities. It is not clear as to if the samples were filtered at a pH of 12, but they were filtered at ambient temperature. Table 13-32 summarises the results.

Table 13-32: Brasfelt Filtration Result Summary

Feed Pulp Density, % solids	Pressure, bar	Moisture, %	Moisture post blow, %	Membrane pressure, bar	Moisture power squeeze, %	Moisture post squeeze and blow, %
40	6	33.2	11.8	12	16.8	9.6
50	6	20.2	11.5	12	11.6	9.8

Feed Pulp Density, % solids	Pressure, bar	Moisture, %	Moisture post blow, %	Membrane pressure, bar	Moisture power squeeze, %	Moisture post squeeze and blow, %
55	6	22.8	12.6	12	12.5	10.9
60	5	20	12.9	12	13.8	11.5

The data presented lacks some key details including volume of air blow. This makes it difficult to assess and compare the results with the other filtration test work. However, assuming the tests were conducted under a comparative basis, it is noted that the feed pulp density can be relaxed and low moisture content in the final cake achieved.

Brasfelt did provide additional information per their equipment supply proposal. This included a similar number of plates/cake volume as presented by ANDRITZ suggesting a similar cake density was achieved. Further detail can be found in Section 17 referencing the filtration design criteria applied.

13.6 Key Criteria

Key criteria derived from the test work summarised by this Section 13 is summarised per Table 13-33. The design engineer will take this data and previous test work results along with production schedule considerations and ascertain the necessary ranges of these parameters and others to generate the project Process Design Criteria ("**PDC**"). Consequently, the values presented within Table 13-33 can be expected to deviate from the PDC.

Table 13-33: Brasfelt Filtration Result Summary

Variable	Unit	Value
Comminution		
JK Axb, min, max toughness		39.5, 29.8
Bond CWi, min, ave, max	kWh/t	11.0, 18.9, 34.3
Bond BWi, range	kWh/t	17.2 - 22.9
Abrasion index	g	0.35 – 0.40
Rheology		
Viscosity at 50% solids, shear rate 4.2 s^{-1}	cP	0
Viscosity at 50% solids, shear rate 119 s^{-1}	cP	51 – 75
Viscosity at 60% solids, shear rate 4.2 s^{-1}	cP	0 – 3,743
Viscosity at 60% solids, shear rate 119 s^{-1}	cP	158 – 636
Grinding		
Product P80	µm	53
Pre-leach thickening		
Unit area	$t/m^2.h$	0.50 – 0.75 ore specific
Underflow density, design	% w/w	60
Leaching		
Oxygen uptake, 0<t<4 h	mg/L/min	-0.21
Oxygen uptake, +4 h	mg/L/min	-0.05
Maximum dissolved oxygen concentration	mg/L	30
Operating pH, nominal		12.0
Operating pH, maximum		12.5
Operating temperature range	ºC	35 - 45
Leach residence time	h	36
NaCN minimum concentration	mg/L	150
Carbon Characteristics		
Kinetic (Nicol Fleming) k	h^{-1}	139.5
Kinetic (Nicol Fleming) n		0.623
Equilibrium K (Freundlich)		3.614
Equilibrium m (Freundlich)		0.451
Sizing	mesh	6 x 12
Tailings Thickening		
Unit area	$t/m^2.h$	0.50 – 0.75 ore specific
Underflow density, design	% w/w	60
Cyanide Detoxification		
Residence time, test work	h	1.0
Residence time, design minimum	h	1.5
Operating pH, nominal		8.5 – 9.0
SO_2/WAD CN ratio, design	g/g	4 – 5.5
Target WAD CN concentration post detox	mg/L	<5
Maximum WAD CN discharge post detox	mg/L	<20
Copper, Cu2+	g/g CN free	0.85
Reagents		
NaCN consumption – leach, average	kg/t	0.215
60% CaO equivalent "lime", average	kg/t	3.65#
Pre-leach flocculant, nominal	g/t	20
Tailings flocculant, nominal	g/t	20
Tails Filtration(1)		
Feed pulp density	% w/w	>40
Filtration rate(1)	$kgDS/m^2 \cdot h$	110 - 300
Residual filter cake moisture	%	<15
Maximum filter cake moisture placed in stack	%	19
Filter cake density, wet	kg/m^3	1.9 – 2.0
Solids in filtrate	mg/L	<150

(1) Criteria dependent on technology selected and feed conditions.

(2) Assumed commercial quicklime purity 60%. Will depend on supplier.

13.7 Extraction Prediction

13.7.1 Base Model

A number of extraction models had been generated to assess the different extractions as a function of leach time and head grade. Whilst these models provided insight into the differential extraction as a function of time, they did not provide as robust a prediction of the residue grade at the 36 hour leach time, being the final design leach residence.

To simplify and provide a more robust algorithm, the residue data at 36 hours leach time was used and plotted against the head grade of the samples. Sixteen (16) of the locality sample tests were used to generate the relationship. Those tests which were duplicates or were conducted at grinds other than the P80 53µm were removed from the set.

The samples retained for analysis and the 36h leach residue grades determined from the leach tests are summarised by Table 13-34.

Table 13-34: Brasfelt Filtration Result Summary

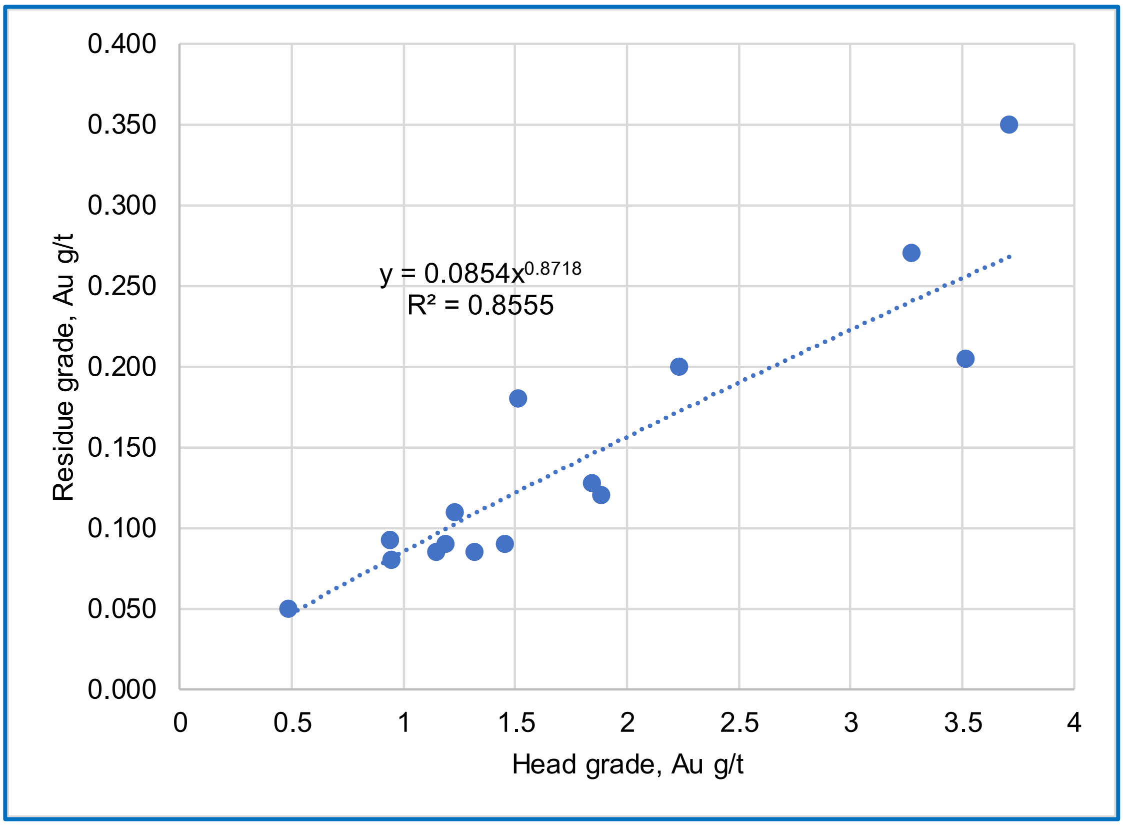
Sample	Au head g/t	36 h Residue Au g/t	Te head g/t
VAR - 09	0.49	0.05	1.2
VAR - 21	0.94	0.09	2.6
VAR - 12A	0.94	0.09	2.6
VAR - 04	0.95	0.08	2.2
VAR - 16	1.15	0.09	2.4
VAR - 18	1.19	0.09	3.2
VAR - 19	1.23	0.11	2.2
VAR - 11	1.32	0.08	2.8
VAR - 13	1.46	0.09	2.6
VAR - 20	1.52	0.18	2.6
VAR - 06	1.84	0.13	3.4
VAR - 12B	1.89	0.12	3.8
VAR - 22	2.24	0.20	3.4
VAR - 02B	3.27	0.27	5.0
VAR - 17A	3.52	0.21	0.1*
VAR - 05	3.71	0.35	7.0

(1) Value assumed to be half of limit of assay resolution.

These results were plotted, and a line of best fit was applied. A power function giving the best correlation coefficient (R2 = 0.86). The plot is presented as Figure 13-29. The form of the equation (line of best fit) being

- Residue grade = 0.0854 x Au0.8718 where:
  - Residue grade = Grade of the 36-hour leach residue, Au g/t
  - Au = Gold head grade of sample.

Figure 13-29: Residue Grade versus Head Grade, 36 hours



The actual residue grades were then plotted against the residue grades predicted by the algorithm, presented by Figure 13-30. As is often found with such algorithms, there is an off-set or skew in the predictions in that the line of best fit should be linear and should have a gradient of one (1). Per Figure 13-30 it will be noted that the gradient (blue line) is 0.92 meaning the algorithm is biasing the residue estimate some 8% low. This is even though the average of the actual residues and the predicted residues are both equal to 0.14g/t.

To address this off-set, an adjustment can be applied. In this instance, if a value of 0.015g/t is added to the algorithm output, the line of best fit now had a gradient of 0.9996 (orange line). This lifts the average adjusted residue grade to 0.15g/t, or 0.01g/t higher than the average of the actuals.

When evaluating the range from 1 < gold head grade < 2g/t where the bulk of the resource grades sit, the adjusted actual residue grade versus the algorithm grade is found to have a gradient of one (1). The average actual residue grade is 0.11g/t compared to the average algorithm (unadjusted) value of 0.12g/t. If the algorithm is to reflect the bulk of the grades more accurately, then the algorithm should remain unadjusted. It will overstate the residue (understate the extraction) of this key grade range.

The decision was made to progress with the non-adjusted version of the algorithm.

The algorithm presented above predicts the residue grade of the solids from the leach. In addition to this loss, in a full-scale plant there are also losses associated with solution loss to tails and carbon losses due to degradation/abrasion of the carbon.

A typical solution loss is 0.01 to 0.015 g/m3 solution tails. When allowances are made for pulp density and the associated solution to solids ratio, this soluble loss approximates to conservatively 0.015g/t of ore for a well-designed circuit.

A typical carbon loss is 20g of carbon per tonne of solids processed. Much of the loss is due to carbon handling during elution and reactivation, then pumping carbon back to the plant. There is a minor loss in the tanks themselves caused by agitation and inter-tank transfers. If it is assumed the average grade of the carbon loss is 400/t gold per tonne of carbon, the loss to tails is 0.08g/t of ore processed.

The soluble and carbon loss is therefore of the order of 0.023g/t of ore processed.

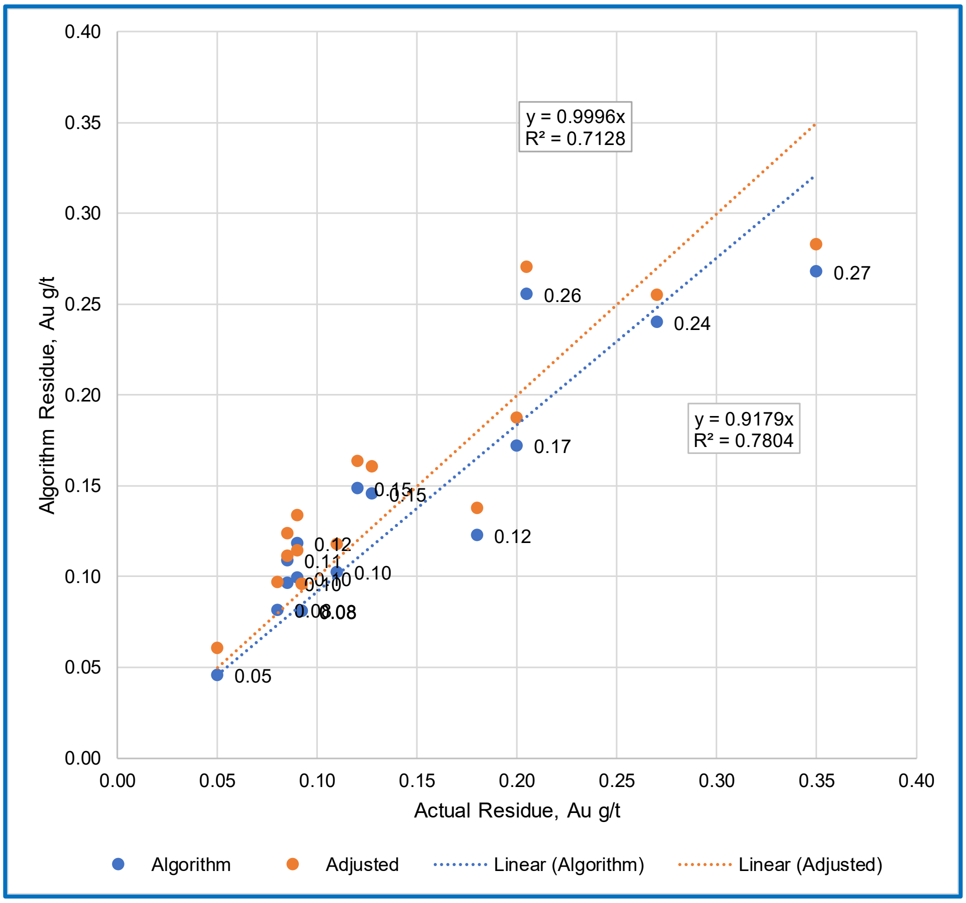
The recovery of gold equals the extraction fewer other losses. The extraction is the difference in the head grade less the residue grade. Therefore, the recovery can be described as:

- **Recovery** = Au - 0.0854 x Au0.8718 – 0.023 where:
  - Recovery = grams of gold recovered per tonne of ore feed.

Alternatively:

- **Recovery %** = [(Au - 0.0854 x Au0.8718 – 0.023) / Au] x 100% where:
  - Recovery % = the percentage recovery.

Figure 13-30: Residue Grade versus Head Grade, 36 hours



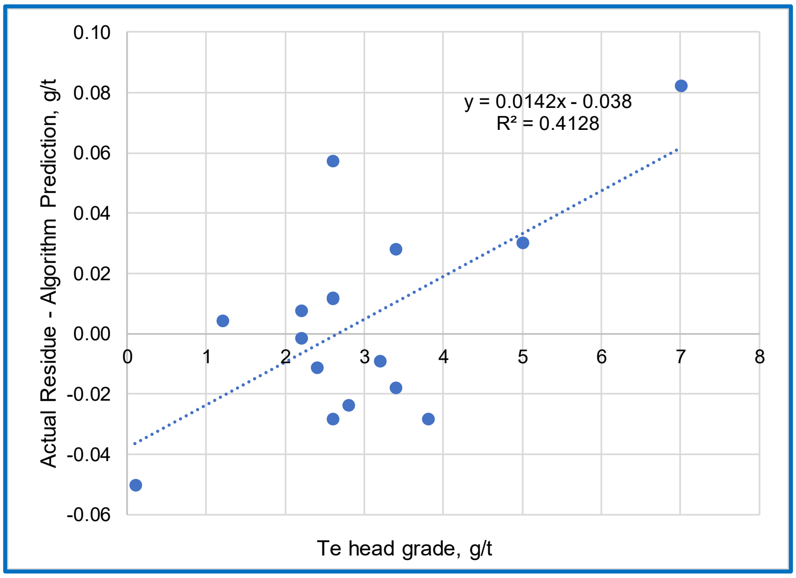
13.7.2 Influence of Tellurides

Figure 13-18 in Section ALS Program A19476 (discussion on program A194756) presented an association with gold and tellurium grade. The extraction algorithm described in the previous section has been based on gold alone. Given there is an apparent relationship between gold and tellurium, then it can be expected tellurium influences are inherently included in the recovery relationship described in the previous section.

An exercise was undertaken where the difference in the actual 36h residue grade and the algorithm estimate of residue grade was plotted against the tellurium head grade. The results are presented as Figure 13-31.

The graph presents a lot of scatter, particularly in the 2g/tTe to 4g/tTe range. The linear line of best fit is heavily influenced by the lowest tellurium assay and the highest assay values. If it were not for these two points, the plot would suggest no apparent or at least no robust relationship.

Figure 13-31: Residue Grade versus Head Grade, 36 hours

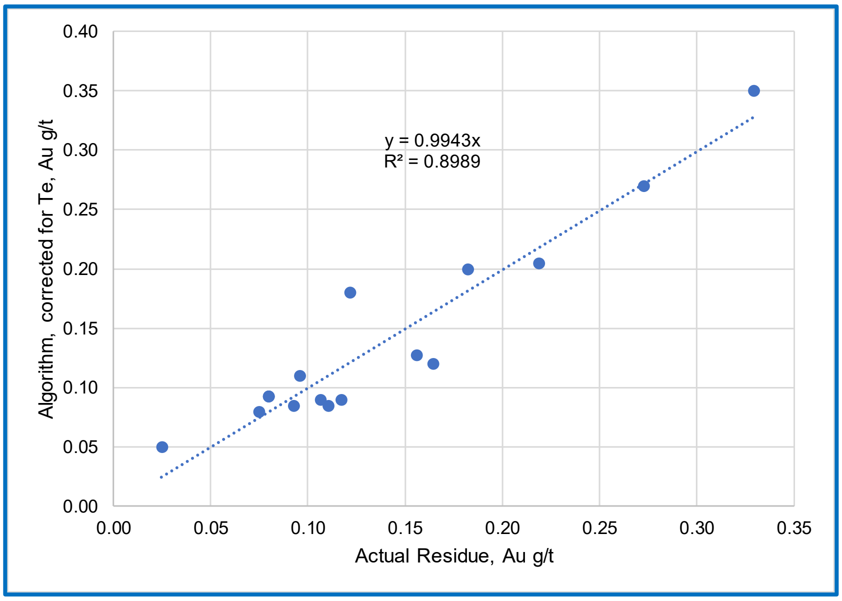


Influences considered; the line of best fit was applied as a "correction" to the residue algorithm. The tellurium head grade applied, and the "correction" added to the residue grade predicted by the gold head grade algorithm

- Residue grade = 0.0854 x Au0.8718 + 0.0142 x Te – 0.038 where:
  – Te = Tellurium head grade, g/t

When the "corrected" residue grade is plotted against the actual residue grade at 36 h, the plot presented as Figure 13-32 results.

Figure 13-32: Residue Grade versus Head Grade, 36 hours



This plot presents an improved correlation coefficient compared to the gold only algorithm and presents a gradient of one (1) which is what is anticipated if the relationship is strong. This suggests there is merit in having an algorithm that includes tellurium head grade to predict residue grade of gold. It also being noted the average of the residues is 0.14g/t, the same as the average of the actuals.

However, the important grade range of 1 < gold head grade < 2g/t is not well represented. The gradient of this range is 0.9 meaning the residues are understated (extraction overstated) and the correlation coefficient (R2) of this select data set alone is only 0.05.

Whilst there is a case to use the tellurium head grade as an adjustment or correction to the residue algorithm, there are a number of issues:

- There is not a robust tellurium data base present in the block model, and so tellurium grades would need to be estimated to align with the gold assay frequency block by block. This will invoke undefined error;
- In the most dominant grade range of gold, the extraction will be overestimated; and
- The relationship of the tellurium influence is overly dependent on one low and one high assay. Whilst the relationship presented per Figure 13-32 seems very robust, the dependency on these two assays may provide an outcome that is, at least in part, coincidental.

Consequently, the recommended extraction/recovery algorithms are those based on gold head grade only.

The data does suggest that more work could be undertaken in this area, with the opportunity to improve day to day operating expectations in the full-scale plant. If for example grade control drilling indicated plant feed with elevated tellurium grades, there may be a case to increase pH to increase telluride oxidation and improve gold extraction. Tellurium grade and deportment potentially being an important parameter for day-to-day plant operations.

13.8 Conclusion

The test work described herein has provided support for the proposed flowsheet to be applied at Mara Rosa and is considered adequate to take into process design. The flowsheet being to crush, grind, leach at 53 µm for 36 hours at a pH of 12.0 at anticipated temperatures of +35°C generated as a consequence of grinding effort. The work has shown the carbon characteristics remain in the range typical of the industry, even though elevated pH is present. The work has also shown that SO_2/air cyanide detoxification is applicable using reagent doses and residence times again typical of the gold industry.

To reduce capital cost, the decision to take the tailings thickener out of the flowsheet has been made. Filtration testing at a nominal pulp density of 40% and 50% solids has shown filtered solids can be generated at moisture contents that will allow handling and placement. Press type filter technologies appearing the most appropriate.

The samples used in the test work have been sourced from a large number of drill holes and from varying depths along strike. The basic work (both earlier work by Coffey and latter work managed by Amarillo directly) to define the flowsheet has been conducted on a number of composites suggesting “average” or “typical” performance will provide high leach extractions in the 90% range. As the test work programs have progressed, and as test work control has improved, the Locality Composites tested have provided very consistent results in both extraction outcomes and reagent demands. This lack of variability suggests the Mara Rosa material can be expected to provide consistent leach extractions in the 90% range and also supports adequate coverage of the deposit by the samples selected. That is sensitivity to sample location is minor and is not a key driver with regard to the metallurgical responses.

There do not appear to be any deleterious elements or compounds present. An exception may be considered to be the presence of auriferous tellurides themselves. However, as the flowsheet has provided high leach extractions, these tellurides are no longer considered deleterious. The extractions achieved are high even by typical free milling ores in this head grade range.

14 MINERAL RESOURCE ESTIMATES

The Mineral Resource estimate for Posse has been updated by Gregory Keith Whitehouse, BSc (Geology), MAusIMM (CP) of Australian Exploration Field Services Pty Ltd (AEFS), to reflect all drilling into the target area to end 2019 and updated topographic data. This Resource Estimate supersedes all previous estimates.

The 2021 Mineral Resource Estimate which was reviewed and confirmed as part of this CPR is summarised in Table 14 1.

Table 14-1: 31 December 2021 Mineral Resource Estimate(1)

Category	Tonnes (Mt)	Au grade (g/t)	Troy Ounces (koz)
Measured Mineral Resource	14	1.2	510
Indicated Mineral Resource	19	1.1	640
Total of Measured and Indicated Mineral Resource	**32**	**1.1**	**1,200**
Inferred Mineral Resource	0.10	0.52	1.7

(1) Note that Tonnes, Grade and Ounces in the 2020 Resource Estimate summarised in Table 14-1 have been reported to 2 significant figures only to reflect the uncertainty inherent in any Mineral Resource Estimate. A cut-off grade of 0.35g/tAu has been used for the Mineral Resource Estimate. The Mineral Resource is quoted inclusive of Mineral Reserves.

The Posse Deposit, which is the focus of Amarillo's Posse Gold Project, is situated near the town of Mara Rosa in Goiás State, Brazil. Australian Exploration Field Services Pty Ltd ("**AEFS**") was retained by Amarillo to visit the site, review procedures, validate and where appropriate correct the drillhole database for the Posse Deposit. Subsequently, the mineral deposit was modelled by Mr Whitehouse and an estimate of the gold mineral resource was made. Mr Whitehouse has been responsible for Mineral Resource Estimates on the Posse Deposit since 2010, those estimates have been reported in NI 43- 101 compliant technical reports listed in Section 2 of this report.

The historic data used in this mineral resource estimate was detailed in:

- Drilling pre 2010, NI 43-101 report, dated 30 June 2010, titled Independent Mineral Resource Estimate and Preliminary Economic Assessment, authored by Hoogvliet Geological Services and AEFS (HCS & AEFS, 2010);
- Drilling conducted by Amarillo between November 2010 and March 2011 was detailed in the NI 43–101 report, dated 30 July 2011, Report on Independent Site Visit and Resource Estimate, authored by Hoogvliet Geological Services and AEFS (HCS & AEFS, 2011); and
- Results from a further series of 59 diamond drill holes completed between June 2011 and December 2012 were discussed the NI 43-101 report, dated 21 July 2016, Posse Deposit, Mara Rosa, Brazil, Mineral Resource Update 21 (AEFS, 2016).

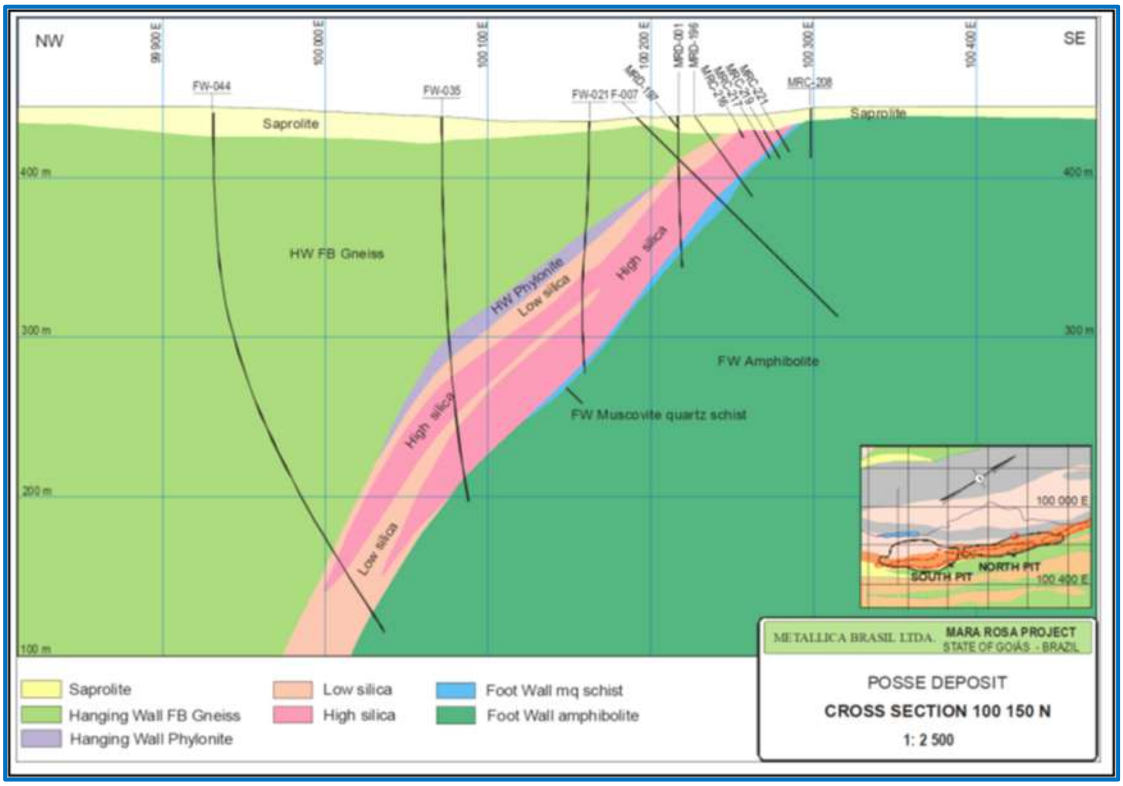
A site visit and review of site procedures, on behalf of Mr. Whitehouse, to support this report was carried out in October and November of 2018 by Mr. John Watts of AEFS.

There was no resource drilling from the end of 2012 until May of 2018. The mineral resource definition drilling which was started in May 2018 and finished in February 2019 has been discussed in this report under Section 10.2, Drilling 2018 – 2019. The updated topography which was used as an input to the resource estimation process is discussed in this report under Section 9 of this CPR. The validation and verification of data which informed the resource estimation process is discussed under Section 12. Note that an additional 10 drillholes were drilled into specifically targeted areas of the resource in 2021. An analysis of this data, Section

12.8, has shown that the data in these holes does not make a material change to the Mineral Resource and accordingly the Mineral Resource has not been updated from that previously reported. Similarly, an extensive risk analysis program, and associated stress testing of the model, Section 12.7, has provided additional support for the modelling procedures adopted. Additional modelling, Section 12.9 also serves to confirm the 2020 Mineral Resource.

The current estimate as reported in the 2020 Statements is based on a complete re interpretation of the Posse mineralisation. The reinterpretation resolves issues in previous work with relationships between higher grade zones which appeared to exist in the Hanging Wall and the Main part of the orebody. The new interpretation based on grade zones within the predominantly schistose lithology defines a higher-grade core within a diffuse lower grade zone of mineralization. In this respect the current interpretation is similar to the model outlined by MINCON in 2003, Figure 14-1.

Figure 14-1: Geology Model, MINCON, 2003(1)



(1) The section corresponds to Amarillo's Section 10 and AEFS' Section 79. The high silica zone closely corresponds to the higher-grade portion of the mineralization as interpreted in the current mineral resource.

14.1 Historical Reports

The mineral resource estimate in this report represents the eighth Independent Mineral Resource Estimate completed on behalf of Amarillo for the Posse Deposit. The historic reports were:

- **Estimate 1:** Estimation of an Inferred Mineral Resource Estimate in March 2007 by CCIC;
- **Estimate 2:** An updated resource estimate which complied with the *"Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards on Mineral Resources and Mineral Reserves Definitions Guidelines"* was provided to Amarillo by CCIC in February 2008;
- **Estimate 3:** HCS and AEFS completed Mineral Resource Estimates in 2010 and 2011. The 2011 resource estimate was used as the basis of the Pre–Feasibility study conducted by Coffey Mining in 2011.

- **Estimate 4:** An update to the Mineral Resource which incorporated additional drilling and improved downhole surveys and surface topography was completed by AEFS in July 2016.
- **Estimate 5:** An update to the 2016 Resource which incorporated improved delineation of the oxidation zones within the resource and a reduced cut-off grade of 0.35g/tAu gold based on the results of pit optimization work was completed by AEFS in January 2017 and incorporated into an Update to the Pre-Feasibility Study by SRK Consulting Brazil in April 2017;
- **Estimate 6:** An update to the 2016 Resource which recognized the presence of backfill material in the model and which further reduced the cut-off grade to 0.2g/tAu as the result of optimization work by Whittle Consulting was completed by AEFS in November 2018 and incorporated in the Technical Update on the Pose Gold Project by SRK Consulting Australia in November 2018; and
- **Estimate 7:** The original report of this resource conducted as part of the 2020 DFS study for the Posse project. Details are outlined in this report.

All resource reporting by HCS and AEFS has been in accordance with the CIM Definition Standards current at the time the reports were written. This report of the 2020 model has been termed the eighth model as while the resource reported is identical to that reported with the 2020 DFS the model has been extensively reviewed and stress tested as part of that review process, additionally two further models constructed independently and using different approaches to modelling have confirmed the results of the 2020 model.

14.2 2020 Mineral Resource

14.2.1 Data Utilized

Data used for modelling consisted of a set of drill collars, downhole surveys and assay data for gold together with wireframe boundaries, which defined a series of grade zones, geology and backfill.

While the PFS study in 2018, (SRK Consulting, 2018) had recommended modelling other elements such as Te which could affect metallurgical recovery at the time the 2020 Resource was completed there were insufficient assay results available to allow meaningful estimates to be made. Samples from the 2019 drill program were analysed by both fire assay and ALS method ME-MS61 for multiple elements. Subsequently samples analysed as part of the re assay program described in Section 12.7.1 were also tested with the same analysis methods as have a wide range of historic sample pulps from past Amarillo drilling. Samples from the 2021 drilling program were also tested by fire assay for Au and for multiple elements. Future modelling work should make use of this new data.

The grade boundaries, used to build the mineralisation wireframes, at 0.1g/tAu , 0.5g/tAu and 1.0g/tAu were based on grade composites of the assay data. The 0.1g/tAu grade composite generally sits within the schistose zone which sits between a Biotite Gneiss Hanging Wall and an Amphibolite Footwall. There is some local minor intrusion of the mineralization into both the Biotite Gneiss Hanging wall and the Amphibolite Footwall lithologies. The geological wireframes were from models of the Biotite Gneiss, Schist and Amphibolite units together with a Mafic Dyke which crosses the deposit. These were overlain by Soil and Saprolite units developed on top of the fresh rock geology. The backfill was modelled from differences between the current topography and the post mining topography as shown in archival data from WMC who mined two pits in the 1990’s.

The geological and backfill wireframes were used to assign density values to blocks in the model. The density values were in turn derived from gamma ray density logs obtained in 2011 during a geophysical survey of 28 holes at Posse, (AEFS, 2016). The density values are supported by dry weight / wet weight density values from 129 samples commissioned by Amarillo at the request of SRK in 2019, see Section 12.4 of this CPR. Further work on the variation of density was completed in 2020 and 2021 as discussed in Sections 12.7.2. The addition of this data will not materially change the 2020 resource estimate as re-reported in the 2021 Statements.

The 2018 Technical Update, (SRK Consulting, 2018) recommended the acquisition of a LIDAR based topographic surface over the Posse Gold Project area together with a bathymetric survey of the flooded portions of the Project pits. This work was carried out and the results of the surveys have been used as a part of the current resource calculation to define elevations for all drillholes drilled since 1996 when mining stopped. The 2019 topographic surface has also allowed the reconstruction of the pre mining surface, and the elevation of holes drilled from this surface to 1993. The 2019 topography has also been merged with the post mining survey made by WMC to calculate the volumes of backfill in the pits together with the location and volume of borrow material. This information has also allowed the elevation of holes drilled while mining was in progress to be confirmed. The LDAR and Hydrographic survey etc. is discussed in Section 9.1 of this CPR.

A raw block model was generated in the WGS84, UTM Zone 22 S coordinate system with primary blocks of 5m (E), 10m (N) and 5m (RL) rotated by 50 degrees to the east around the RL (vertical) axis to align with the strike of the mineralization. Primary blocks were sub-blocked to a minimum of 1m x 1m x 1m to fit the mineralized domain, topography and other wireframes.

Model Dimensions were as shown in Table 14-2.

Table 14-2: 31 December 2021 Mineral Resource Estimate

Item	Min Centre	Block Size	Max Centre	Blocks	Min Subblock Size	Rotation
East	696,000	5	697,500	188	1	0
North	8,453,800	10	8,455,200	200	1	0
RL	-100	5	500	121	1	50

Prior to grade interpolation:

- The raw block model was sub-blocked and coded with reference to appropriate wireframes to indicate which blocks were above or below the topography, the base of Soil, the base of Saprolite and whether the block was in backfill or borrow. This information was then used to code the lithology and associated density field in the model;
- The mineralization wireframes were used to sub-block and code the block model to indicate which of three grade zones, Z1, Z2 or Z3 a block related to;
- The wireframes for the primary geology Biotite Gneiss, the lithologic Hanging Wall; Amphibolite, the lithologic Footwall; and Mafic Dyke were used to write lithology codes and densities into the model. Blocks in the model not coded with other Lithology types between the Hanging and Footwalls were coded as Schist. The majority of the mineralisation occurs within this Schist unit, generally referred to as the Posse Schist; and
- Drillhole density, the basis, together with an evaluation of potential for economic extraction and cut-off, for classifying the resource, was modelled on an inclined long section. Polygons representing each of the potential classifications were converted into wireframes which were rotated back into real world coordinates. This information was then written into the block model.

Grade interpolation used Ordinary Kriging applied in four passes to each of the grade domains Z1, Z2 and Z3. The fourth pass was a scavenger run, to ensure all blocks were interpolated. Blocks interpolated in Run 4 (R4) were only classified as part of the resource to maintain continuity of resource classification.

After grade interpolation the model was passed to the pit optimization routine within Micromine to generate a series of nested pits based on nominal mining and processing and selling costs with a US$1,500/oz selling price and a revenue factor (RF) of 1.2. A RF of 1.2 equates to a US$1,800/oz gold price.

Blocks inside the pit with a RF of 1.2 were considered to have met the test of reasonable prospects for economic extraction necessary to classify material under one of the resource categories Measured, Indicated or Inferred which had been pre-coded into the block model. The categories Measured, Indicated and Inferred have the same meaning as in CIM Definition Standards.

Grade tonnage curves were then drawn, and a cut-off grade of 0.35g/tAu was selected for resource reporting purposes. The steps taken to generate the final model are discussed in more detail below.

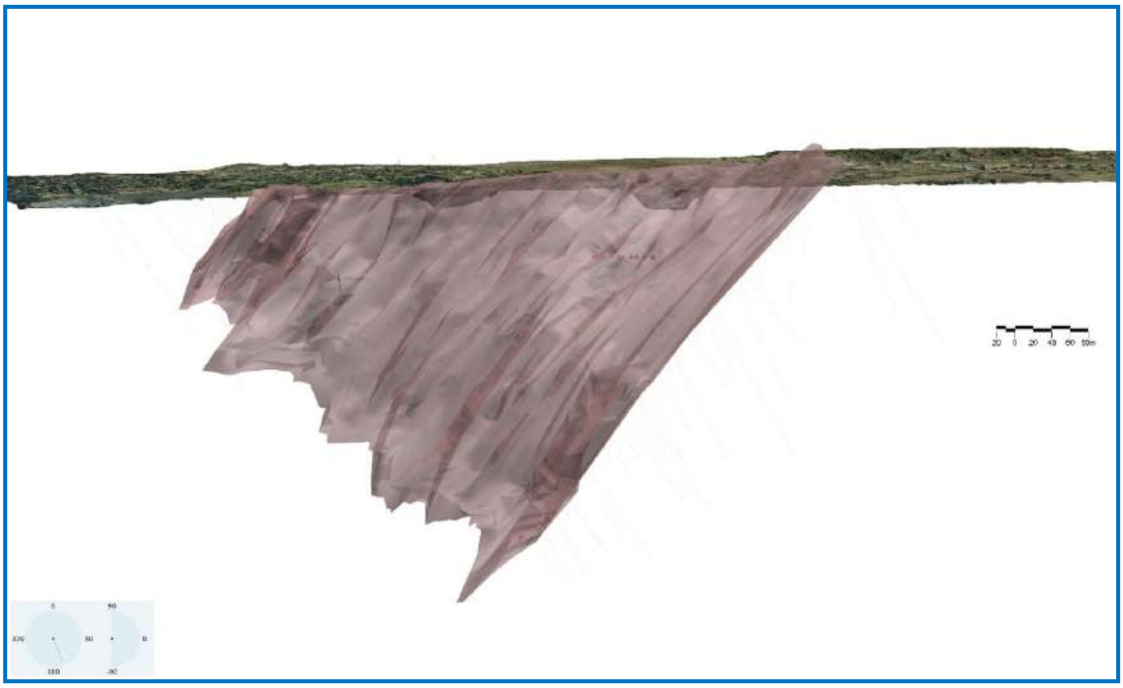
14.2.2 Mineralization Wireframing

Initial modelling work consisted of defining a set of consistent sections to be used in the development of wireframes based on drillhole intersections of geology and grade. The sections were oriented at an azimuth of 50 degrees to the UTM coordinate system grid north so as to be at right angles to the strike of the mineralization.

In historic modelling from 2010 to 2018, three mineralization domains, equating to grade zones were recognized, Hanging Wall, Main and Footwall. The grade in both the Hanging Wall and Footwall was lower than the grade in the Main Zone and neither the Hanging Wall nor Footwall zones were continuous. Occasional splays of Main Zone into the Hanging Wall were incorporated in the Hanging Wall domain as they were not well defined by drilling.

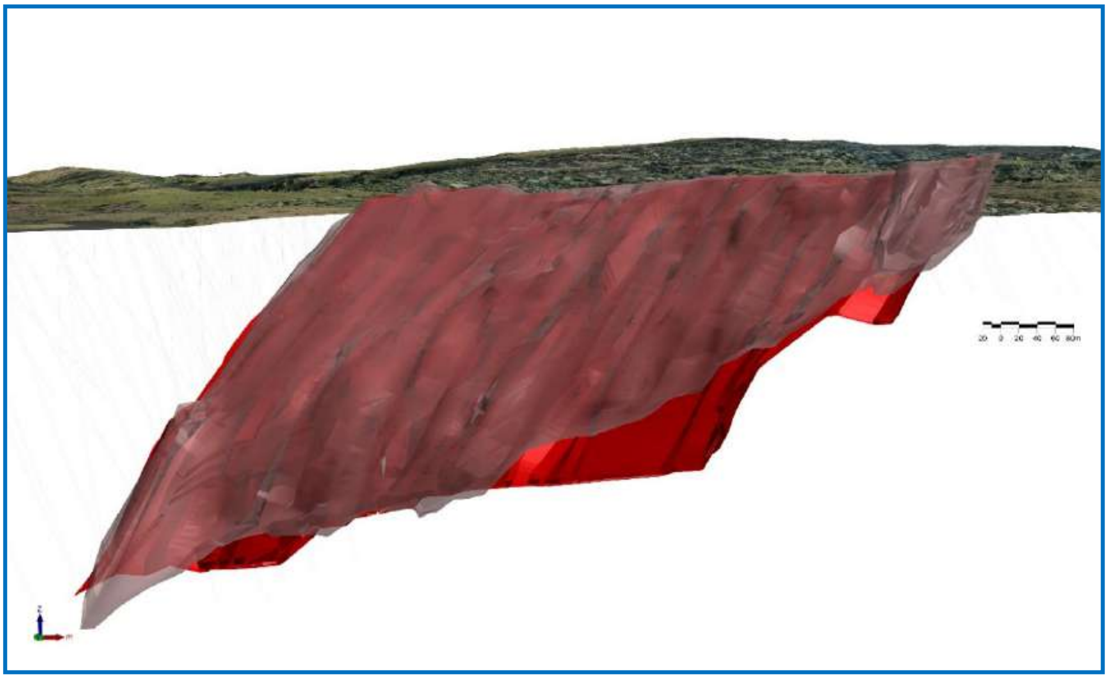
This approach was abandoned for the current round of modelling as an examination of the assays and the geology suggested that there was a broad mineralized zone which broadly matched the boundary of the schistose zone at Posse. This broad mineralized zone was defined by a grade composite of at least 5m of material with a grade over 0.1g/tAu, containing no more than 4m below 0.1g/tAu, with a maximum length of any interval under 0.1g/tAu of 2m. This compositing process defined intervals ranging from 5m (the minimum acceptable length to 125m with an average length of 26m. The boundaries indicated by the 0.1g/tAu composite (the Z3 domain) were interpreted as polygons snapped to drillholes on 25m sections, and then wireframed, Figure 14 2.

Figure 14-2: Wireframed boundary, Z3 Domain (View looking to the south east)



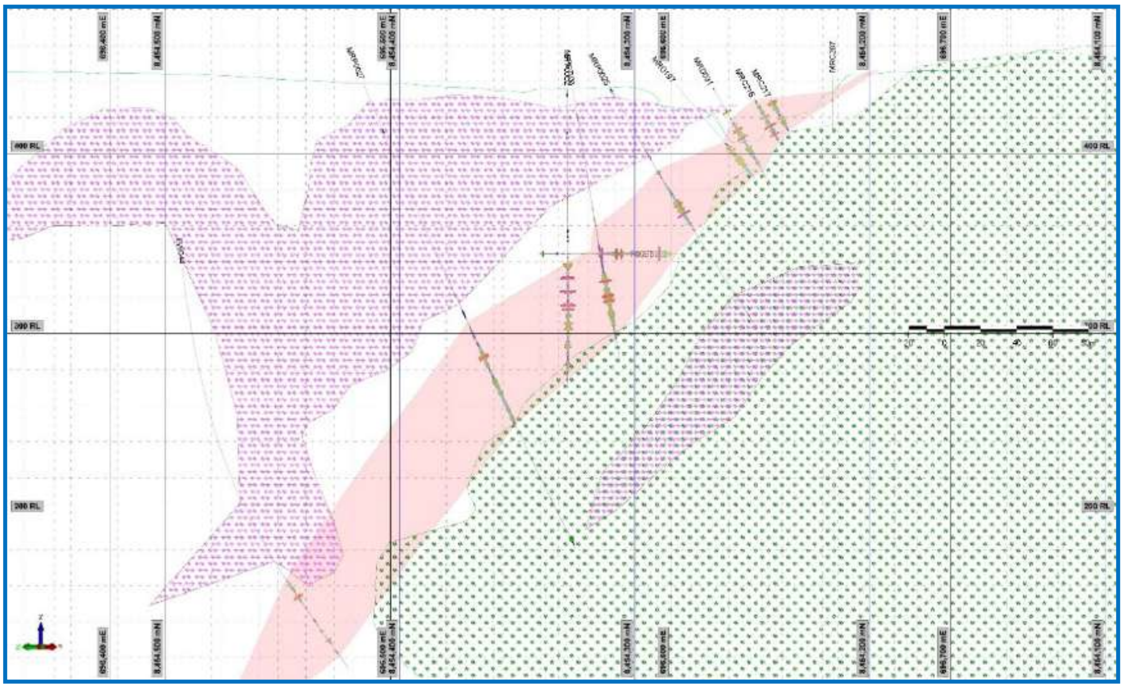
The Z3 domain defined by the 0.1 grade composite used 64 new drillholes from drilling in 2018 and 2019, in addition to historical drilling. Most historical drillholes had changes in elevation after referencing to the 2019 LIDAR topography. Nevertheless, the Z3 domain was not dissimilar to the shape and volume formed by the combined Hanging Wall, Main and Footwall domains used in historic modelling work, Figure 14-3. In the more northern parts of the model the historic wireframes extended deeper than the current Z3 domain boundary. This area has not been tested by drilling and was previously either classified as Inferred or not classified. A land clearance restriction which will be lifted if the Project proceeds to mine development has restricted the siting of drill rigs to target this area.

Figure 14-3: Z3 domain and the historic ore zone (View looking north west, historic ore zone in red and current interpretation in transparent light brown)



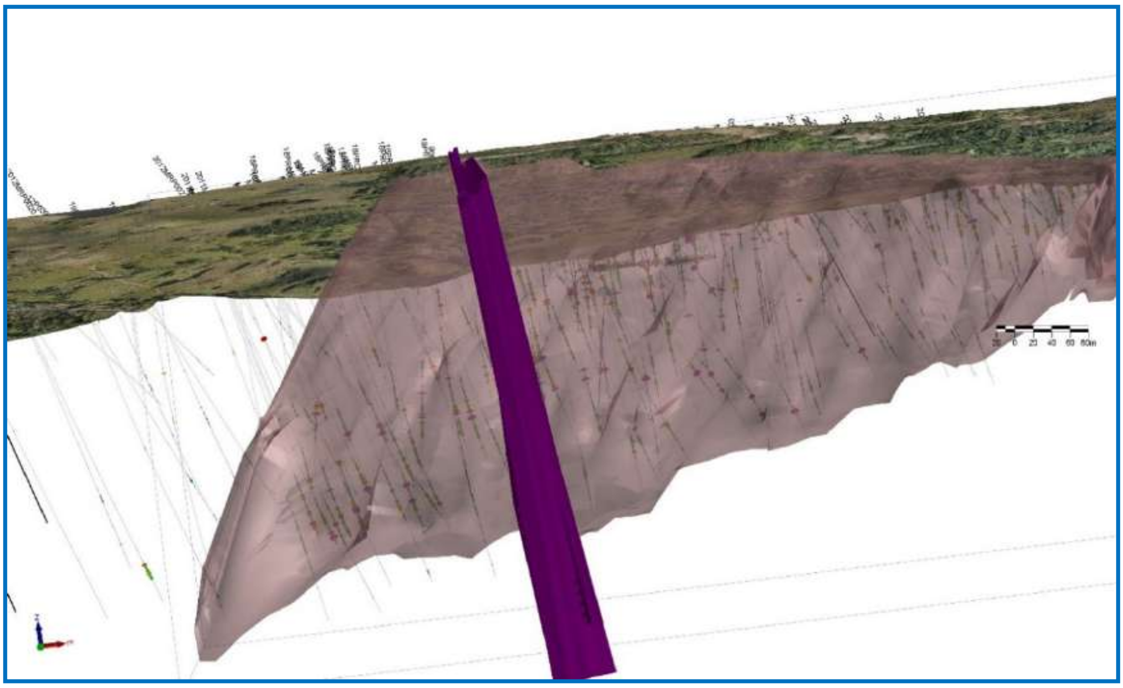
The mineralization is largely confined to the schistose zone which is situated between a Biotite Gneiss hanging wall and the Amphibolite footwall, Figure 14 4, drill results indicate that the hanging wall and footwall units are discontinuous and there appears to be potential for repeats of these structures.

Figure 14-4: Mineralized zone between Biotite Gneiss hanging wall and Amphibolite footwall (The schist unit which largely hosts the mineralization is in white. Biotite Gneiss, purple; Amphibolite, green; mineralized zone, pink)



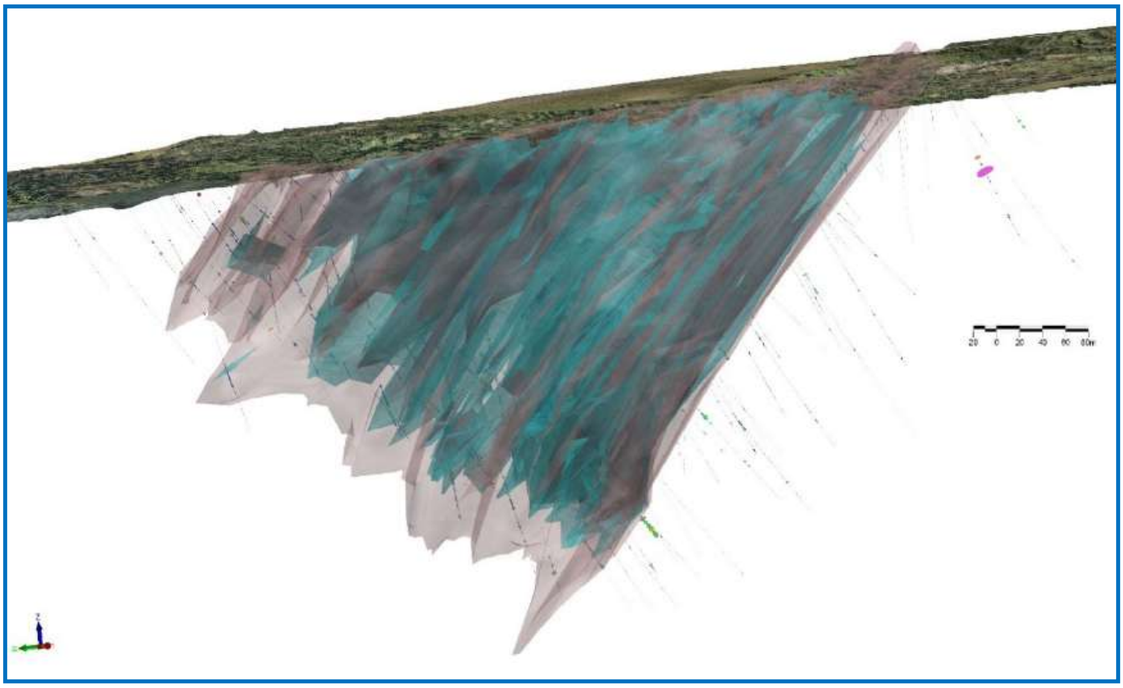
A late, un-mineralized basaltic dyke, which had been encountered in a number of drillholes, was modelled and incorporated as a separate wireframe, which transects the Posse mineralization. In earlier models this dyke was not well defined and there were concerns about its effect of the volume of mineralisation. Current modelling indicates that this dyke, Figure 14-5, is thin and has negligible effect on the mineralized volume.

Figure 14-5: Basaltic Dyke



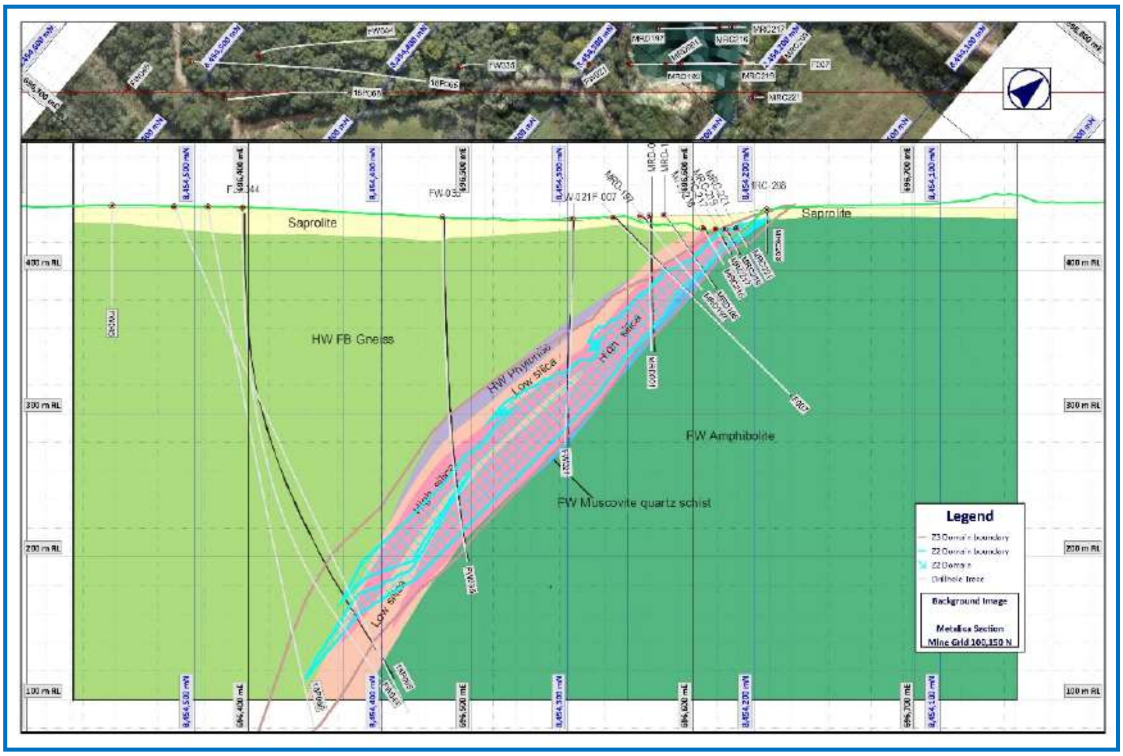
Within the Z3 mineralized domain it was possible to trace near continuous nested zones of higher-grade mineralization. These were defined by the same process as that adopted for the Z3 mineralized domain. For this model grade zones representing a 0.5g/tAu (Z2 domain) and a 1.0g/tAu (Z1 domain) were generated as shown in Figure 14-6.

Figure 14-6: Nested Z3, Z2 & Z1 Domains (Z3 domain in brown, Z2 domain in blue, Z1 domain in dark brown)



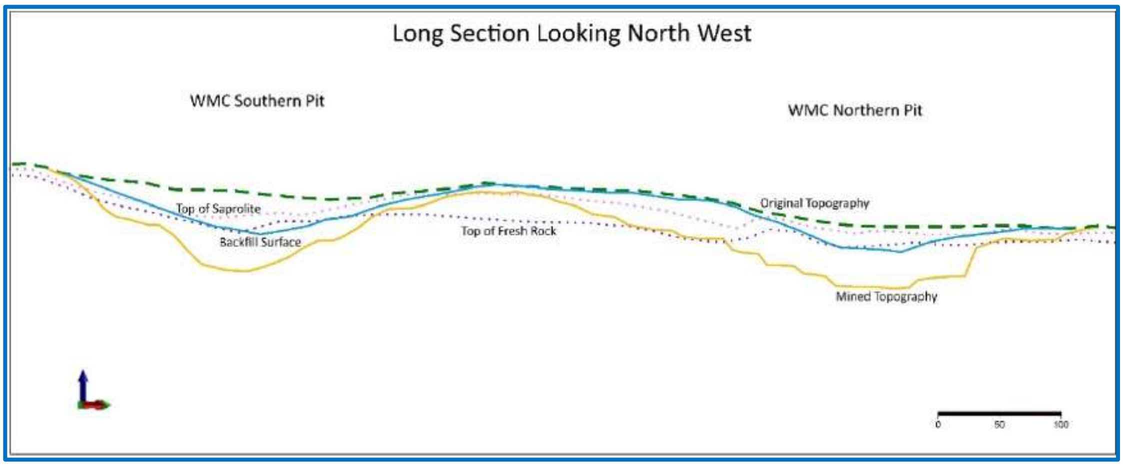
Based on the current geological information in the drillhole database, which contains limited information, there is no obvious geological basis for these domains. However, work by MINCON for Amarillo in 2003 using geological interpretations developed by MBL noted higher grades associated with more intense shearing and more importantly increased silicification and larger amounts of sulphide material, Figure 14-1 above. MINCON relied on a model developed by MBL in 1997 and as part of this modelling work 30 cross sections through the mineralization were made. Unfortunately, a copy of these has not been located. The one cross section presented in the MINCON report has been extracted and compared with the current interpretation, Figure 14-7. The MINCON interpretation is shown in solid colour, the current AEFS interpretation is shown by the red polygon and the blue hatched polygon. The change in interpretation is largely due to the addition of more recent drillhole information.

Figure 14-7: Comparison of current interpretation to MBL interpretation (Gold grade is largely confined to the schistose unit with grade controlled by degree of silicification)



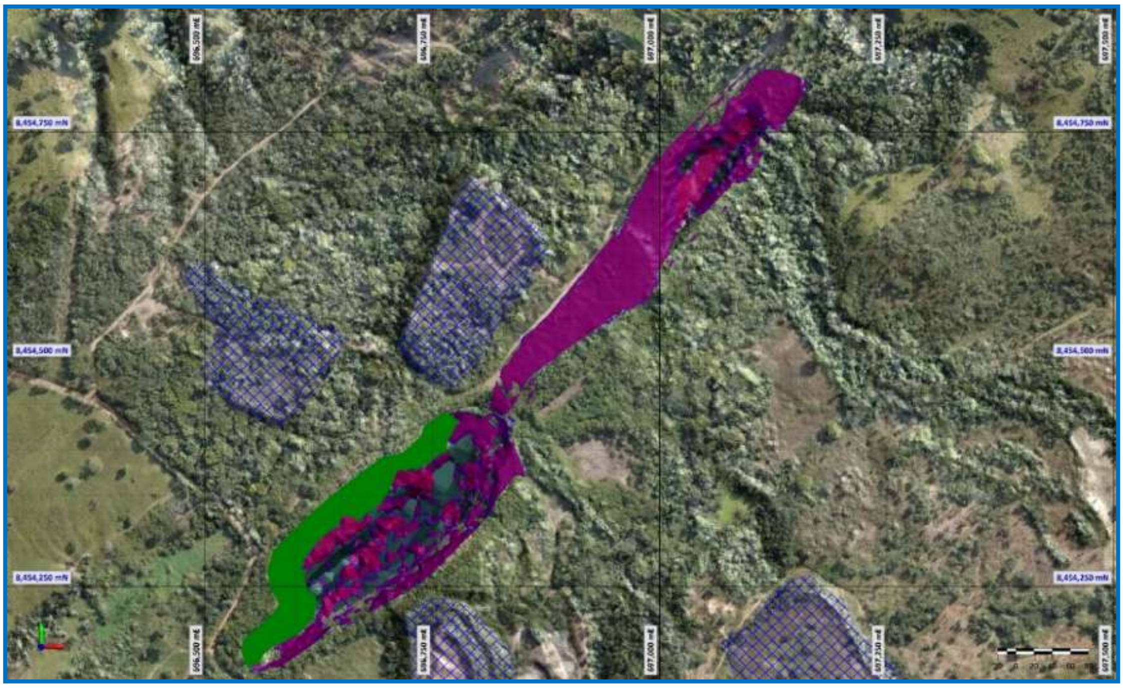
Additional wireframes representing the top of saprolite and top of fresh rock were constructed based on information provided by Amarillo in the drillhole database. These surfaces generally sit one below the other and under the topography. However, in some areas they coalesce so that there are areas with no soil and saprolite at surface, other areas have a deeper soil profile and a thinner saprolite profile. With information from the post mining survey and the LIDAR and bathymetric surveys the mined surface and the backfilled surface were reconstructed, and wireframe surfaces constructed to record this information. These wireframes allowed the block model to be coded to indicate which zone blocks belong to, these codes were then used to control rock densities and resource reporting. A long section showing the various surfaces is shown in Figure 14-8.

Figure 14-8: Surfaces used to control zonation in the block model



The volume of backfill identified is shown in purple in Figure 14-9. The southern pit appears to have been partially filled, in part by using material immediately adjacent to the pit on the northwestern side. The northern pit has been extensively backfilled.

Figure 14-9: Backfill, Borrow and Dumps (Areas of backfill in purple, areas which supplied backfill material in green and historic dumps in blue)



Comparison of the current surface and the mined surface suggests there is 163,000m^3 (326,000t) of backfill in the old pits. The source of this is thought to be a mixture of material from the old dumps and local borrow. The local borrow volume has been estimated at 38,000m^3 (72,000t). Given the sources the fill is likely to be mineralised, but it has been excluded from the resource due to limited drilling information. An analysis of the drillhole database shows that only three holes drilled post mining and placement of fill intersect the fill with a combined intersection length of 22m. The average grade of the intervals is 0.37g/tAu (Sichel's T estimator, Mean 0.38g/tAu and Median 0.27g/tAu).

Should mining proceed it is recommended that this material be drilled to ascertain the grade as it has potential to form a low-grade stockpile or to be used as part of mill commissioning.

14.2.3 Statistics and Compositing

Prior to modelling assay methods in the drillhole database were ranked, Table 14-3, so that a set of preferred assays could be generated using the most appropriate assay when a sample had been assayed by multiple methods.

Table 14-3: Assay Method Ranking [1]

Assay Method	Rank	Comment
Au_ICP (ppb)	5	Used for approximately 700 samples
Au_FA (ppb)	4	Used for approximately 3740 samples, the majority were also measured using ICP
Au_FA (ppm)	3	The majority of assays, 32,695, were measured using this method
Au_GRA21 (ppm)	2	Used for over range, > 10g/t Au on the most recent drill program; 9 records only
Au_FA_Reanalysis (ppm)	1	Used for over range and check assay work on earlier drilling, there are no samples with results for both Au_GRA21 and Au_FA_Reanalysis; 58 records only

[1] The lowest (best) ranked assay for an interval was used where there were multiple competing assays

Table 14-4: Summary statistics for all raw assays

Normal Statistics	All Zones
Minimum	0.0001
Maximum	175.40
No of points	38,132
Mean	0.435
Variance	3.064
Standard deviation	1.750
Median	0.070
Coefficient of variation	4.028
Outliers	4,993
Sichel's T-Estimator	0.459

The gold assay data was then coded according to mineralized domain and was then composited to 1 m downhole intervals for modelling. Statistics for the coded raw assays are shown in Table 14-5 and for the 1m composites in Table 14-6.

Table 14-5: Summary statistics for raw assays by domain

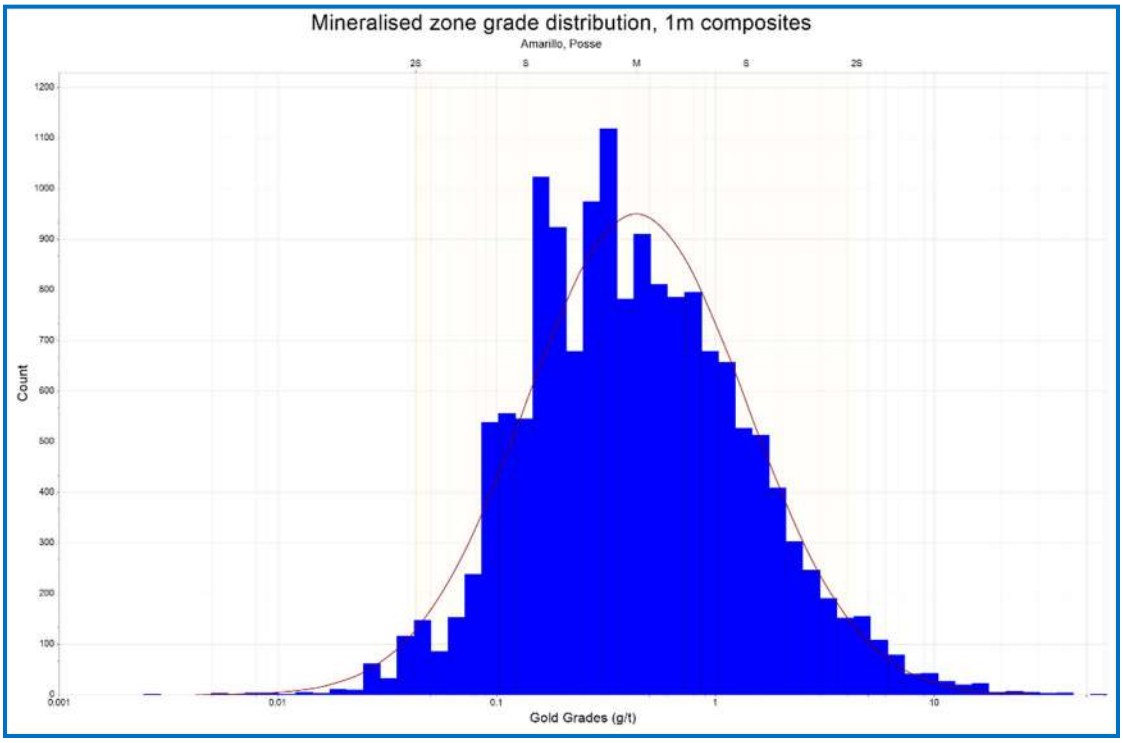
NORMAL STATISTICS	All Zones	Z3	Z2	Z1
Minimum	0.0025	0.0025	0.023	0.013
Maximum	90.000	33.700	41.17	90.000
No of points	15,592	8,406	3,353	3,856
Mean	0.961	0.322	0.986	2.338
Variance	4.923	0.443	3.448	13.816
Standard deviation	2.219	0.666	1.857	3.631
Median	0.400	0.200	0.650	1.420
Coefficient of variation	2.309	2.071	1.883	1.553
Outliers	1,504	599	337	370
Sichel's T-Estimator	0.883	0.300	0.890	2.240

Table 14-6: Summary statistics for 1m composites by domain

NORMAL STATISTICS	All Zones	Z3	Z2	Z1
Minimum	0.0025	0.0025	0.023	0.013
Maximum	61.100	16.800	41.170	61.100
No of points	15,639	8,763	3,222	3,677
Mean	0.917	0.311	0.983	2.307
Variance	3.461	0.262	2.819	8.848
Standard deviation	1.860	0.511	1.679	2.975
Median	0.400	0.210	0.660	1.494
Coefficient of variation	2.029	1.643	1.709	1.289
Outliers	1,474	544	321	354
Sichel's T-Estimator	0.853	0.295	0.895	2.223

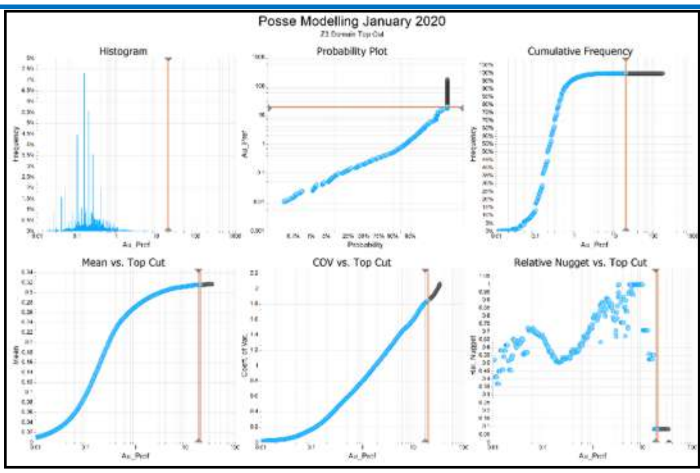
The statistics suggest a log normal distribution of data, Figure 14-10.

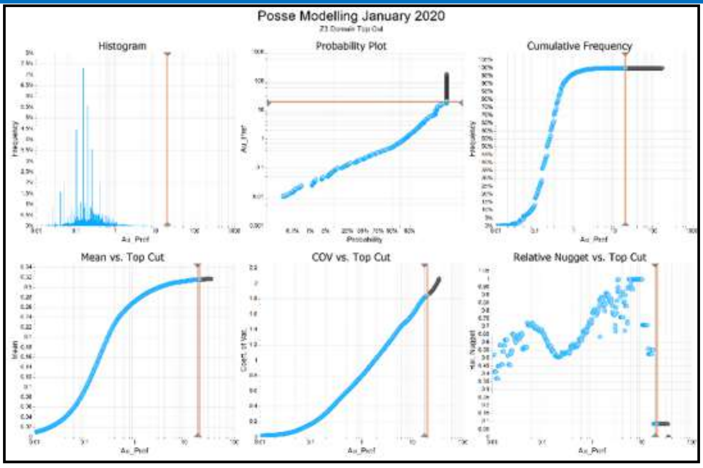
Figure 14-10: Histogram 1m composites

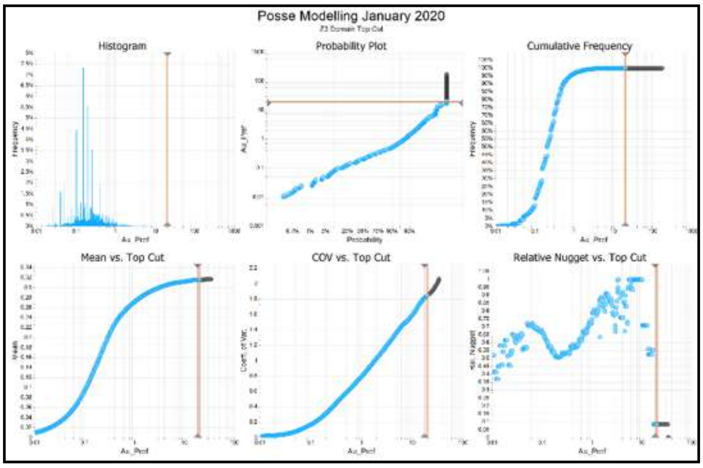


For precious metal deposits being modelled by Ordinary Kriging a coefficient of variation below 1.8 is recommended and each of the three domains meet this requirement, however in order to further reduce the effect of outliers the data was analysed to find a suitable top cut for each of the three domains, Figure 14-11.

Figure 14-11: Top cut analysis domains Z3, Z2 & Z1







Top cut values chosen are listed in Table 14-7.

Table 14-7: Calculated Top Cut values

Domain	Top Cut	Description
Z3	20	Low grade halo, defined by grade composites > 0.1 g/t Au
Z2	30	Medium grade zone, defined by grade composites > 0.5 g/t Au
Z1	40	High grade zone, defined by grade composites > 1.0 g/t Au

There has been further work carried out on the top cut or capping values to be applied to data as part of the ITE consultants risk review. The only reason for cutting or capping high grades is to limit the effect of high-grade outliers in the data distributions which describe the grade in a mineralised domain. A useful guide to the need for capping is the Coefficient of Variation ("**COV**"), calculated as SD/Mean, which measures the relative dispersion (spread) of data points about the mean. A high COV indicates more variability in the dataset and hence more outliers. In practical terms a COV of less than 1.8 indicates little need for top cutting data particularly when dealing with log normal datasets. If data is not capped a very few extreme values in a dataset can seriously distort the mean of the dataset, on the opposite side of the coin, capping which is too aggressive will reduce the mean and result in data estimates which fail to capture the full range of data in a mineralised domain.

As part of the modelling work carried out during the risk review a range of different cut values were imposed on the data set. This stress testing has shown that a wide range of top cut values have little effect on the model.

14.2.4 Variography

Variography for gold using the top cut gold values was completed by Conarco Consulting using Snowden's Supervisor software. For variography the Z3 and Z2 domains were combined. Data from the two domains have then been used for continuity modelling using a normal scores transform. All data reported are the results from the back-transform of the normal scores.

To determine the nugget value, a downhole variogram with a 1 m lag has been used. The result of the nugget value was then fitted to a nested two structure spherical model. This resulted in well-constructed variograms, see Figure 14 12 for the variography for the Z2 & Z3 domains and Figure 14 13 for the variography for the Z1 domain. The variogram model parameters for the Z2 & Z3 domains are set out in Table 14 8 and in Table 14 9 for the Z1 domain.

Figure 14-12: Semi variogram modelling for the Z2 & Z3 domains

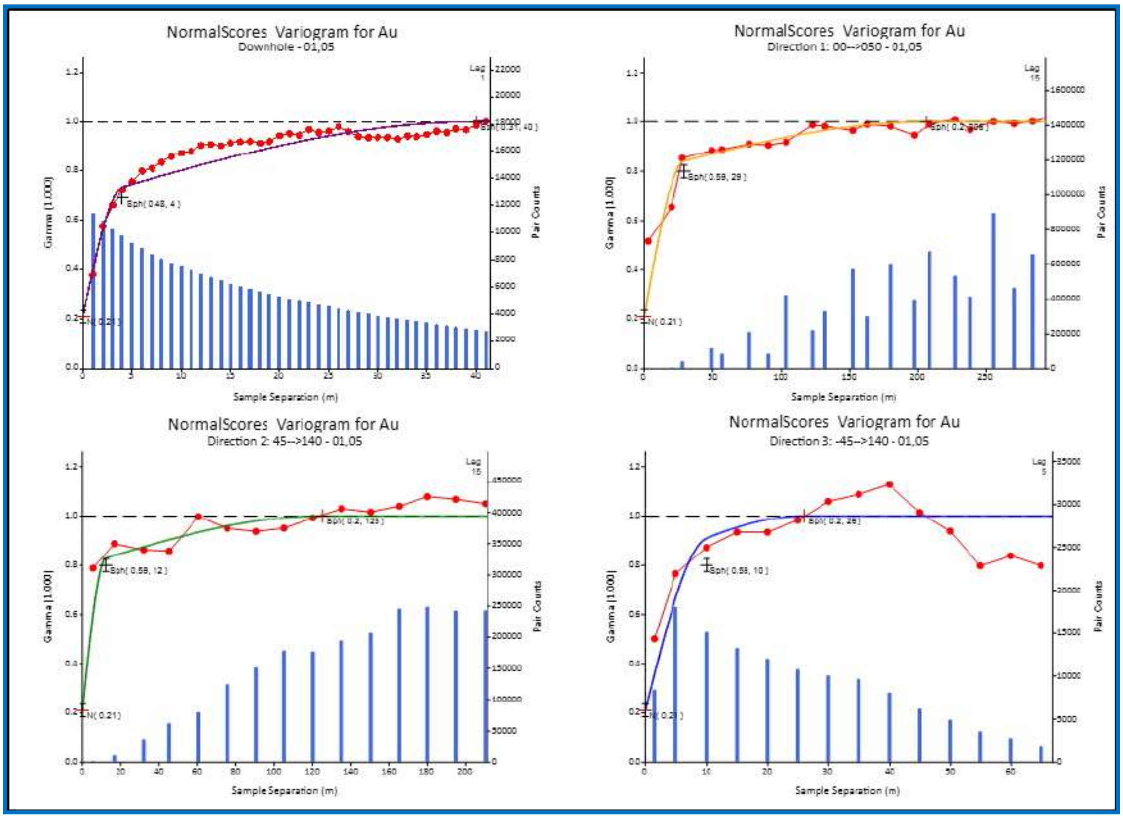


Figure 14-13: Semi variogram modelling for the Z1 domain

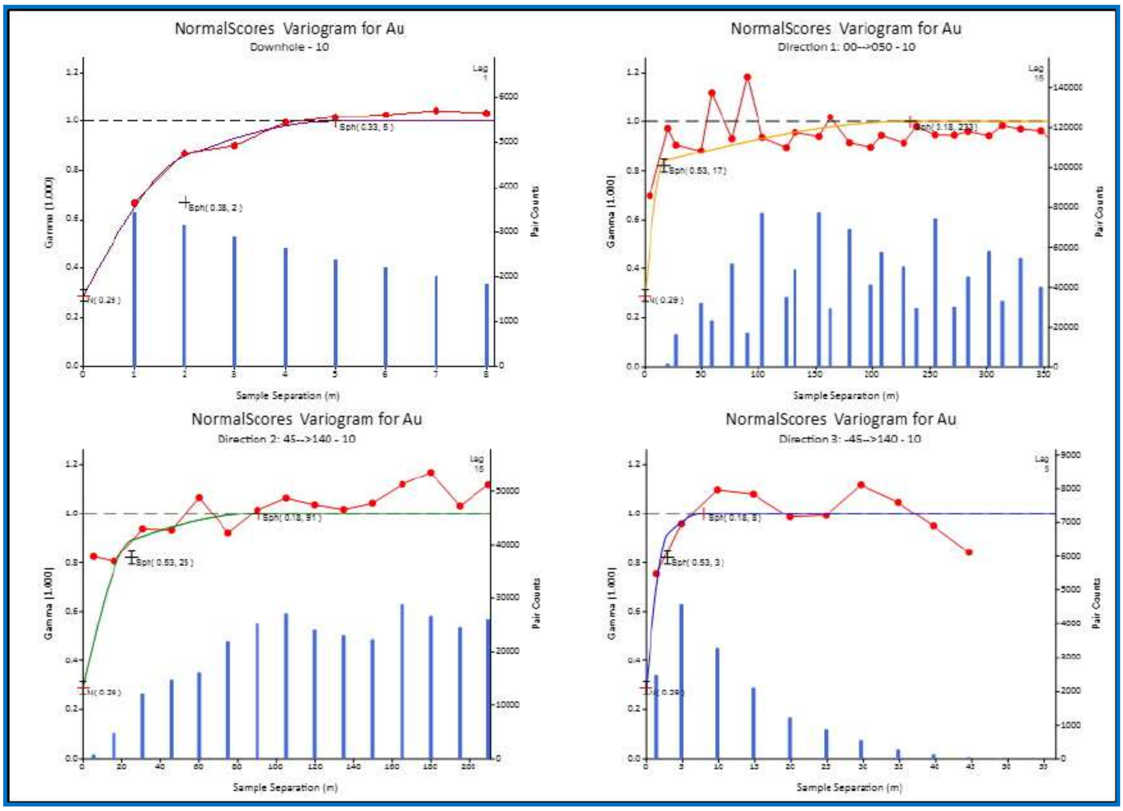


Table 14-8: Semi variogram model parameters for the Z2 & Z3 domains

Rotation1	C0	C1	A1	C2	A2
50	0.4	0.5	29	0.1	206
0	0.4	0.5	12	0.1	125
-135	0.4	0.5	10	01.	26

Table 14-9: Semi variogram model parameters for the Z1 domain

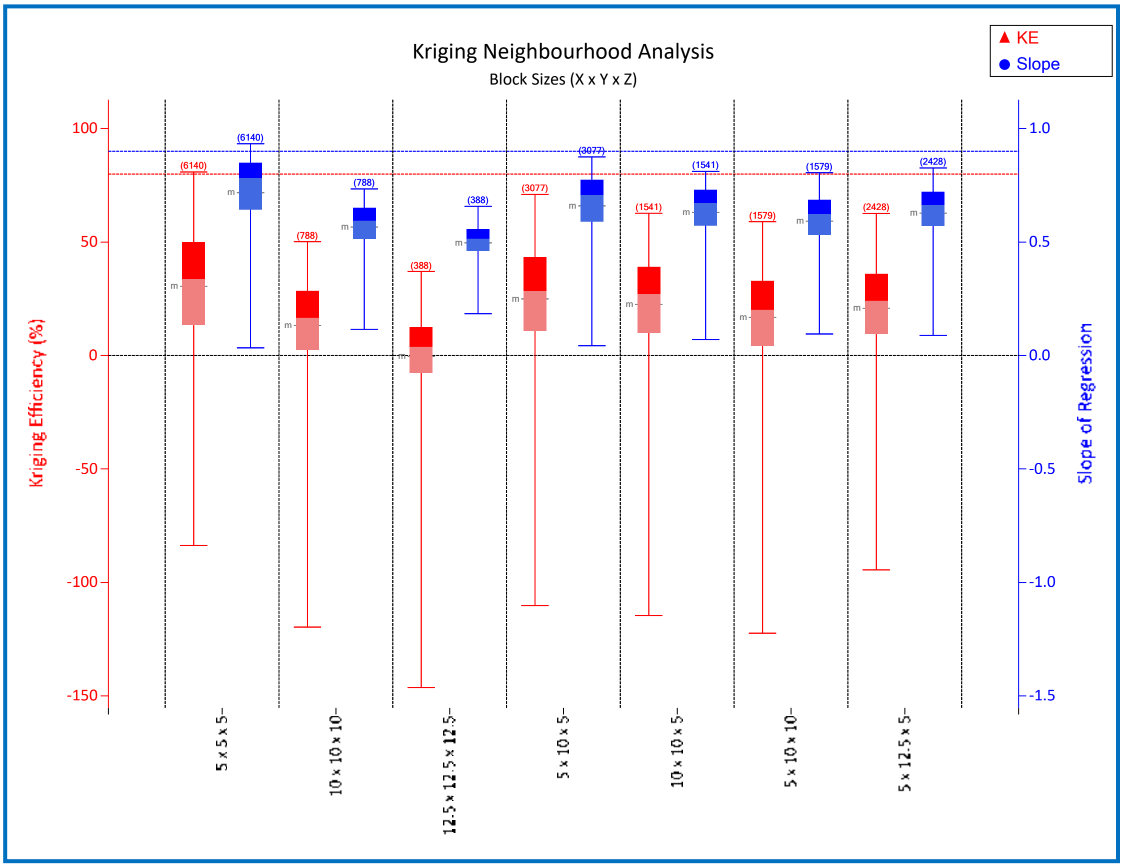
Rotation1	C0	C1	A1	C2	A2
50	0.4	0.5	17	0.1	233
0	0.4	0.5	25	0.1	91
-135	0.4	0.5	3	01.	8

14.2.5 Kriging Neighbourhood Analysis

A multi-block kriging neighbourhood analysis (“**KNA**”) was completed for the High-Grade, Z1, domain to determine the optimum block size as well as the appropriate minimum and maximum number of samples to be used in estimating. This was achieved by estimating a given point at a range of block sizes, with differing number of samples, maximum samples per drill hole (set to 4), differing search ranges determined by the variography and discretization steps.

A parent block size of 5 (X) x 10 (Y) x 5 (Z) has been selected as appropriate based on the average drill spacing and also, Figure 14-14, KNA was used to select a block size with the best overall kriging efficiency, slope of regression and minimal negative kriging weights. Note that there are differences in the block size between the resource model with primary blocks of 5 x 10 x 5 and the reserve model which used the resource model re-blocked to have blocks of 5 x 5 x 5. From a resource estimation perspective 5 x 10 x 5 with extensive sub blocking to 1 x 1 x1 is an appropriate size. Similar granularity is achieved using a regular block size of 5 x 5 x 5 for the reserve which necessary uses blocks suited to the equipment used for mining which will determine the selectivity with which ore and waste can be delineated.

Figure 14-14: Z1 domain, kriging efficiencies and slope of regression (Kriging efficiency in red, slope of regression in blue)



The minimum (8) and maximum (30) numbers of samples for the estimation have been determined from KNA, Figure 14-15, flattening of the curves for kriging efficiencies and slope of regression suggest that there is little benefit in using a greater number of samples. A check for negative kriging weights, Figure 14-16, supported these choices.

Figure 14-15: Change in kriging efficiencies and slope of regression with sample size

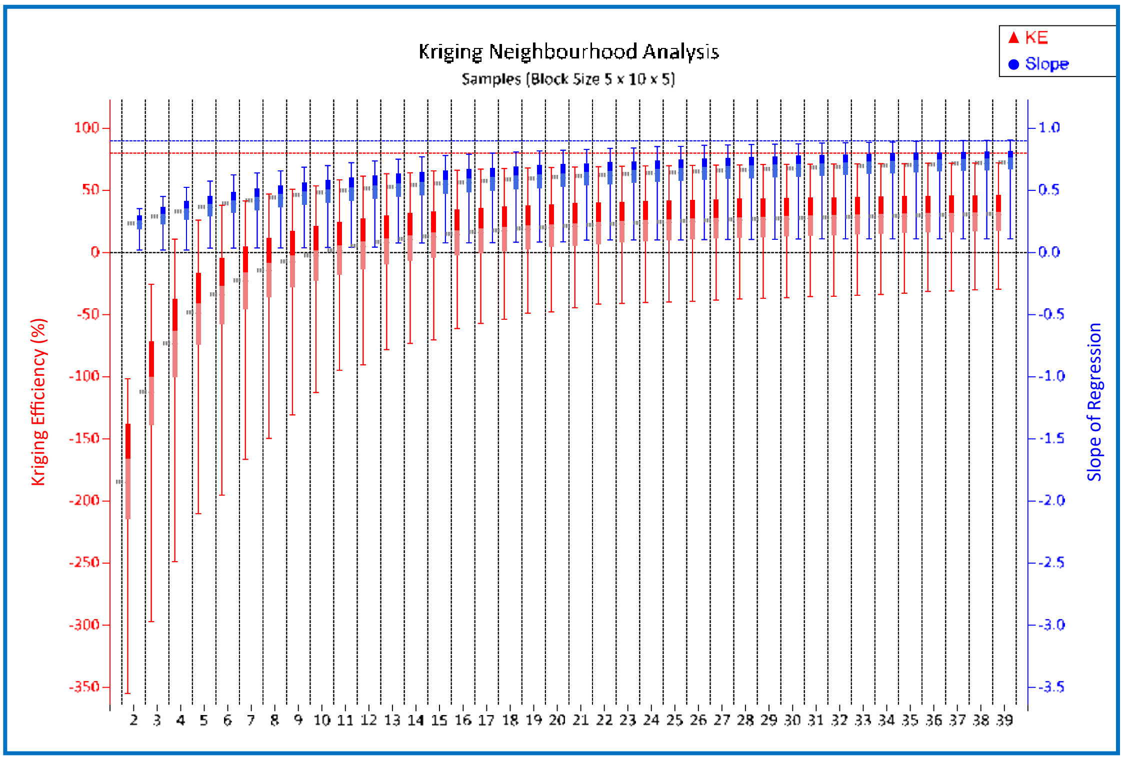
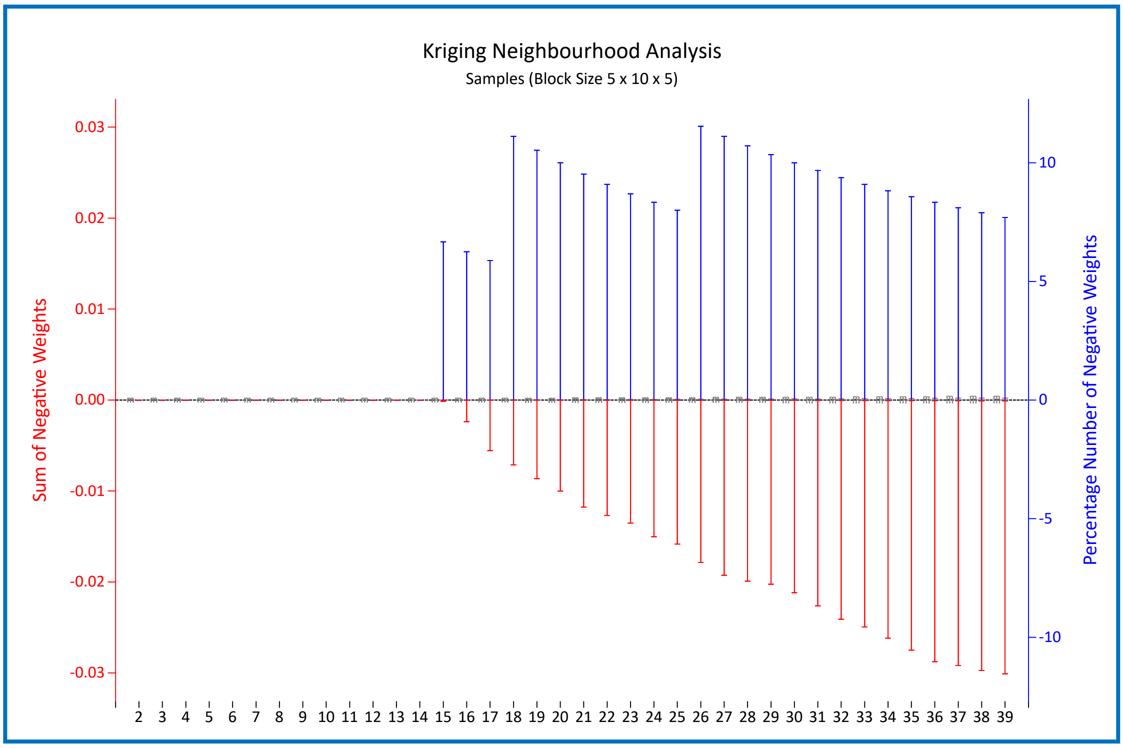
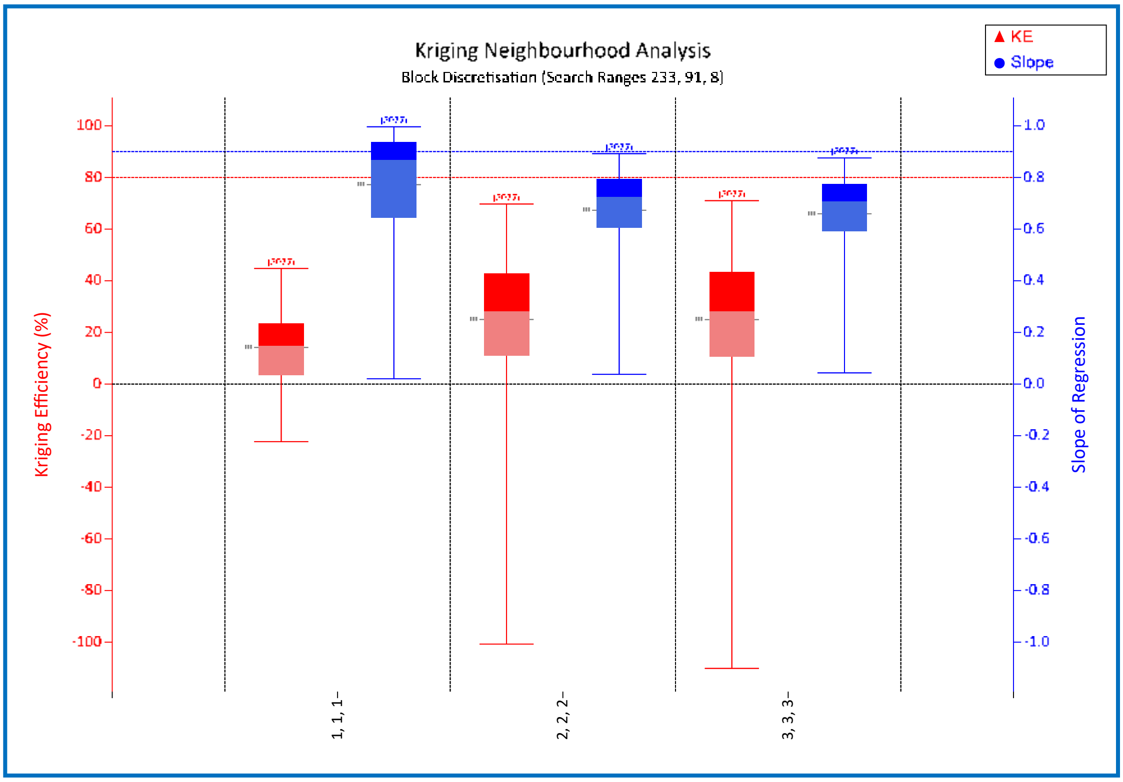


Figure 14-16: Change in negative kriging weights with sample size



Using the above results, a comparison of the discretization steps, Figure 14 17, showed a 3(X) x 3(Y) x 3(Z) regime provided the best outcome.

Figure 14-17: Kriging efficiencies and slope of regression change with numbers of discretization points



These results, summarized in Table 14-10, were used as the basis for modelling parameters selected for all domains when estimating grades into the block model.

Table 14-10: Semi variogram model parameters for the Z1 domain

Block Size			Samples		Search			Discretization
X	Y	Z	Min	Max	Major	S-Major	Min	
5	10	5	8	30	233	91	8	3x3x3

Cross validation routines were used to compare kriged estimates based on the modelled semi variograms and KNA with input data for each domain. The output from the cross-validation routines for each of the semi variogram models were used to construct QQ Plots which compared the shape of the input distribution with the modelled data distribution. The results which demonstrate the closeness of fit between the model estimates and the original input values (composites) are shown in Figure 14-18 and statistics comparing the estimate with the actual values are shown in Table 14-11. For all domains the match between estimated values and actual values is very good with the graph plotting a near straight line indicating the form of the distributions is the same, the rotation of the plotted points from the background line (red), which indicates both data distributions are the same, is a result of the decrease in variance in the set of estimated values vs the actual values. The decrease in variance is to be expected in the estimated values as the estimate is made using multiple points which do not include the point being estimated. In all cases the mean value for the set of estimated values is very similar to the mean value for the set of actual values.

Figure 14-18: QQ Plots estimates vs actual domains Z3, Z2 & Z1

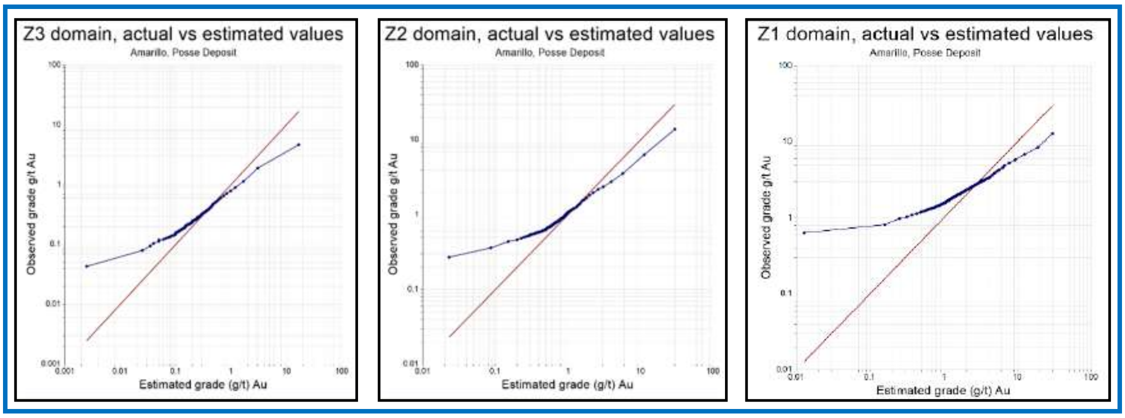


Table 14-11: Estimated compared with actual values for Domains Z3, Z2 & Z1

Zone	Z3	Z2	Z1
Source	Estimate	Estimate	Estimate
No Points	8619	6247	3604
Mean	0.31	1.54	2.31
Variance	0.07	0.86	1.57
S Deviation	0.27	0.93	1.25
Source	Composites	Composites	Composites
No Points	8640	6306	3618
Mean	0.31	1.54	2.30
Variance	0.26	4.83	7.62
S Deviation	0.51	2.2	2.76
CC	**0.961**	**0.957**	**0.981**
X Var / Y Var	3.72	5.63	4.84

14.2.6 Block Model

The block model was based on 5m x 10m x 5m blocks rotated 50 degrees around the Z axis to fit the blocks to the wireframe boundaries. The model extents and rotations are shown in Table 14 2. Various wireframes as outlined in Section 14.2.1 were then assigned to fields in the model to code the model prior to grade interpolation. The coding process sub-blocked the primary 5m x 10m x 5m to a minimum sub-block size of 1m x 1m x 1m. Further detail on the coding and the model is shown in Table 14-2.

Table 14-12: Model constraints

Model Field	Codes	Sub-blocked	Comments
MR2019Topo	Abv / Bel	Yes	Blocks coded Abv are air blocks
MR1996Mined	Abv / Bel	Yes	There are areas where this surface is above the current surface, these areas represent borrow areas for pit backfill.
Soil	Abv / Bel	Yes	Soil is not continuous across the modelled area
SAP	Abv / Bel	Yes	Saprolite is not continuous across the modelled area
Backfill	Fill / Cut	Yes	Code fill represents backfill in the pits, approximately 163,000m^3. Cut represents borrow used for backfill in the pits, approximately 38,000m^3
Dyke	MAF	Yes	The one dyke modelled is thin and only forms a defined zone with the aid of sub-blocks
Density	various	No	Density values defined with reference to the Topo, Soil, SAP, Backfill and Dyke fields together with the lithological model developed for Biotite Schist, Amphibolite and Schist. See Table 12-5 for values. As described in Section 12.7.2 further work has since been carried out to determine the spatial variability of the density within the resource and should be considered in future work. The variability is however limited, and densities used in the 2020 resource are still considered appropriate.
Lith	various	No	As the density values were written into the model appropriate Lithological codes were written to the Lith field
01_WF	01	Yes	The Z3 domain, see Section 14.2.2
05_WF	05	Yes	The Z2 domain, see Section 14.2.2
10_WF	10	Yes	The Z1 domain, see Section 14.2.2
POV_Class	1-Meas, 2-Ind, 3-Inf	No	See Section 14.2.9

No provision was made in the 2020 resource model to exclude blocks which had been affected by the underground development carried out by Western Mining as part of their investigation of the Posse mineralisation because no proper records of this work had been located at that time. This was raised as part of the risk review and is discussed under Section Size of Historic Underground Development. Additional data has now been located and has been shown to have an insignificant effect on the 2020 resource model.

14.2.7 Interpolation

The Interpolation used Ordinary Kriging with variography as set out in Section Variography. Each of the three mineralization domains, Z1, Z2 & Z3, coded in the block model, was interpolated separately, using just the data points from inside that zone as the data source. All domains were modelled using 4 interpolation runs. The search parameters for each interpolation run are listed in Table 14-13 and Table 14 14.

Table 14-13: Common model parameters for interpolation runs

Long Axis	Azimuth	Plunge	Intermediate Axis	Rotation	Short Axis	Sectors
1	50	-11	0.4	45	variable	1

The search ellipsoids for each interpolation run varied in size, see Table 14-14. In all searches the minimum points for interpolation were set to 8 and the maximum to 30 with a minimum of 1 and a maximum of 4 points for source. The KNA had indicated that there was no need to run a multi sector search. All searches other than the 4th run for each of the 3 domains required a minimum of 2 sources to estimate a block. The 4th interpolation run for each domain was designed as a scavenger run to make sure all blocks in the domain were populated. When interpolating sub-blocks, the interpolations were run using the nominal master block for any related sub-blocks with all sub blocks receiving interpolated values for the nominal master block.

Table 14-14: Search parameters for each interpolation run

Domain	Run	Ellipsoid Axis	Intermediate Axis	Short Axis	Minimum Sources
Z1	1	78	31	8	2
Z1	2	156	62	10	2
Z1	3	206	93	11	2
Z1	4	206	93	11	1
Z2	1	69	28	7	2
Z2	2	138	55	11	2
Z2	3	233	93	15	2
Z2	4	233	93	15	1
Z3	1	69	28	7	2
Z3	2	138	55	11	2
Z3	3	233	93	15	2
Z3	4	233	93	15	1

During the interpolation process in addition to a grade estimate for each block, information on the run in which the block was interpolated, number of points and data sources, minimum and average distance to points contributing to the estimate, the value of the point closest to the block centre, kriging standard error and slope of regression were written to each block.

14.2.8 Model Validation

Once constructed the model, it was tested, using Swath and QQ plots. Swath plots were used to compare input gold and output gold grades across all domains in north, south, and elevation directions. These are shown in Figure 14-19 through Figure 14-21. Each of the swath plots shows a suitable level of smoothing during the estimation together with a generally good correlation between the input and output grades. This provides confidence that the grade estimation process is robust.

Figure 14-19: Swath plot, in the north direction (Number of samples as blue bars' input composites as purple line; estimated grades as orange line)

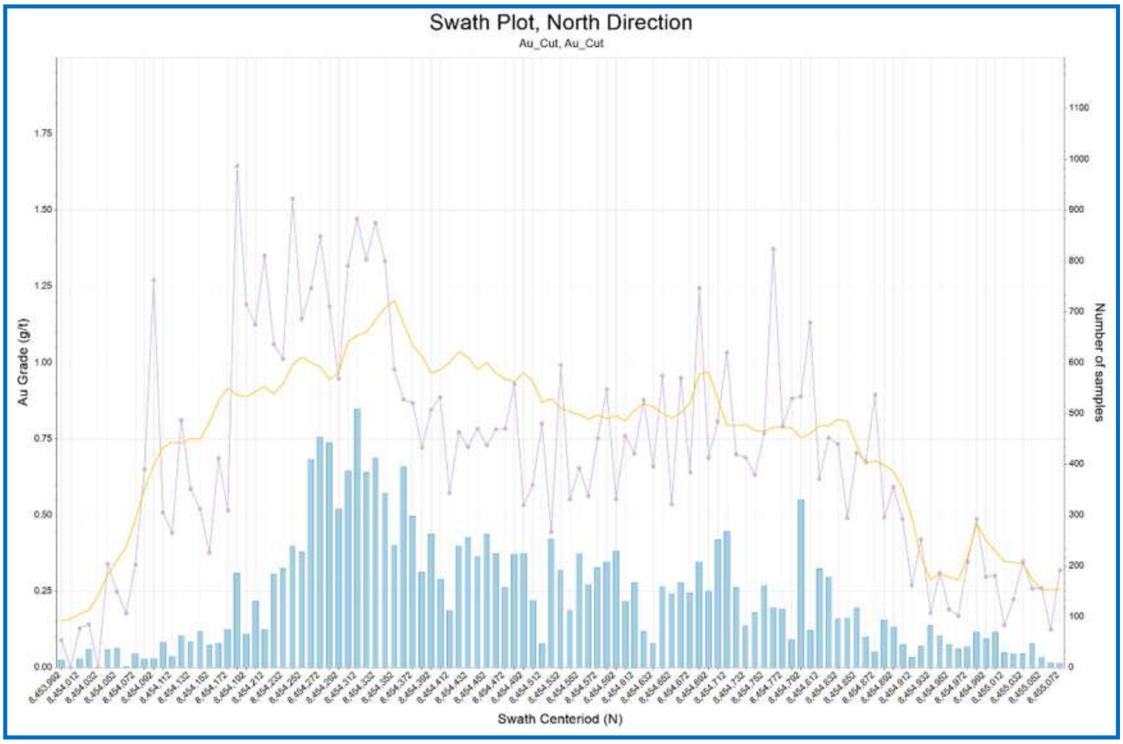


Figure 14-20: Swath plot, in the east direction (Number of samples as blue bars' input composites as purple line; estimated grades as orange line)

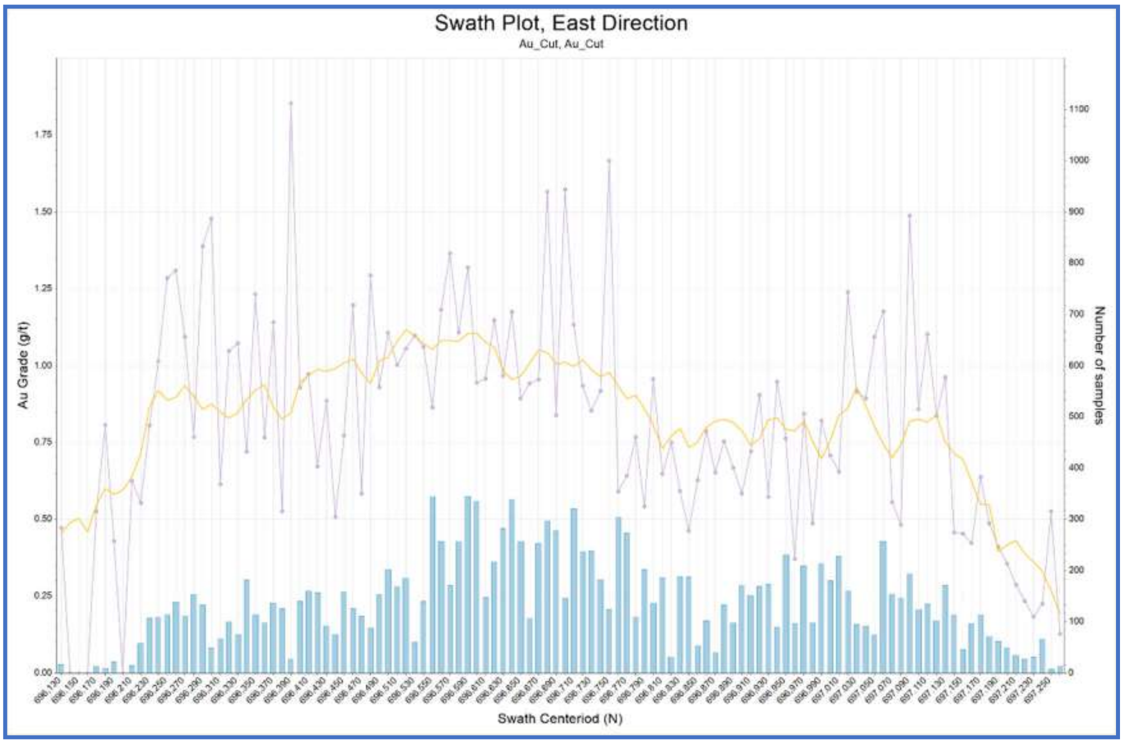
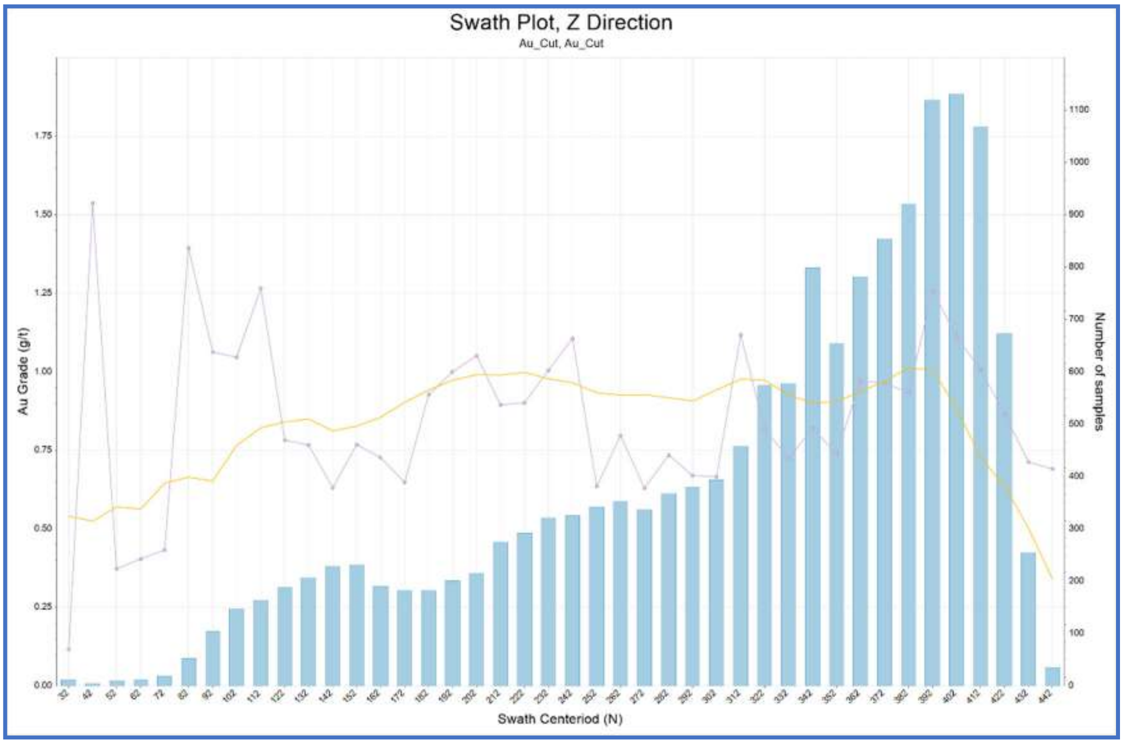
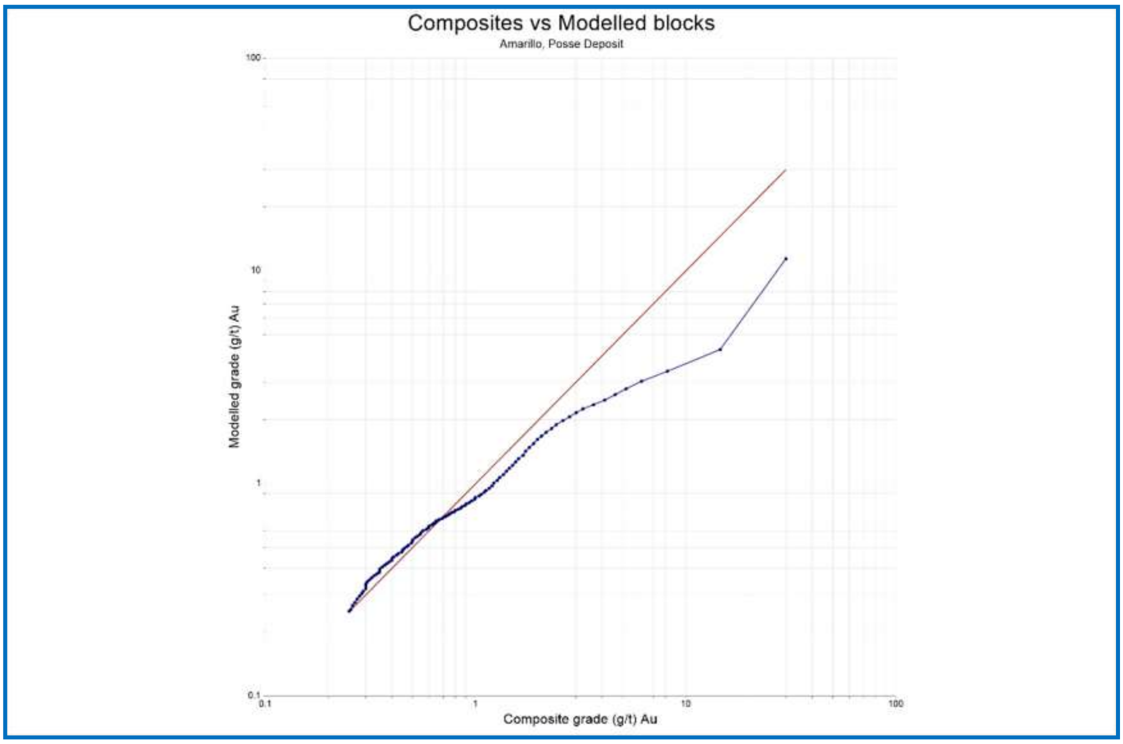


Figure 14-21: Swath plot in the elevation direction (Number of samples as blue bars' input composites as purple line; estimated grades as orange line)



The QQ plot, Figure 14 22, provides a simple way of comparing two data sets of very different sizes, 1,137,162 blocks in the model, and 10,200 composites. If the two data sets had the same distribution, they would plot along the red line. The modelling process involves a degree of smoothing of the data with the result that lower modelled grades are higher than the equivalent input grades, whereas the higher modelled grades are lower than the equivalent input grades. The mean of the input data set is 0.91 while that of the modelled dataset is 0.90 indicating a robust model.

Figure 14-22: QQ Plot composites vs modelled blocks



As expected, the model shows that grades are smoothed, nevertheless there is a high degree of correlation between the source and modelled data suggesting the model is a good fit to the bulk of the source data.

14.2.9 Model Classification and Reporting

The classification of blocks into Measured, Indicated and Inferred Mineral Resource blocks used zoning of data based on data density (drillhole pierce points) as determined in an inclined long section together with a test of potential for economic extraction using pit optimization. The test of potential economic extraction is not intended to imply that all the blocks passing the test will be economic to mine, rather, it indicates that there are reasonable grounds for considering the blocks as input to a reserve estimate.

Zoning

The zoning of the data was determined from an inclined long section with boundaries digitized over the section, based on the density of the drillholes. Holes in Zone 1 were generally spaced within 20 meters of each other. Those in Zone 2 were generally within 40m of each other and those in Zone 3 were within 80m of each other. The zone boundaries determined in long section were then expanded in and out of the section plane and wireframed to produce a set of solids. The wireframes were then rotated back into normal X, Y Z space and used to flag blocks in the model with appropriate Zone codes. The zones are shown in Figure 14-23 and in Figure 14-24 with the Z3 wireframe, the mineralization boundary, superimposed.

Figure 14-23: Resource Category Zones

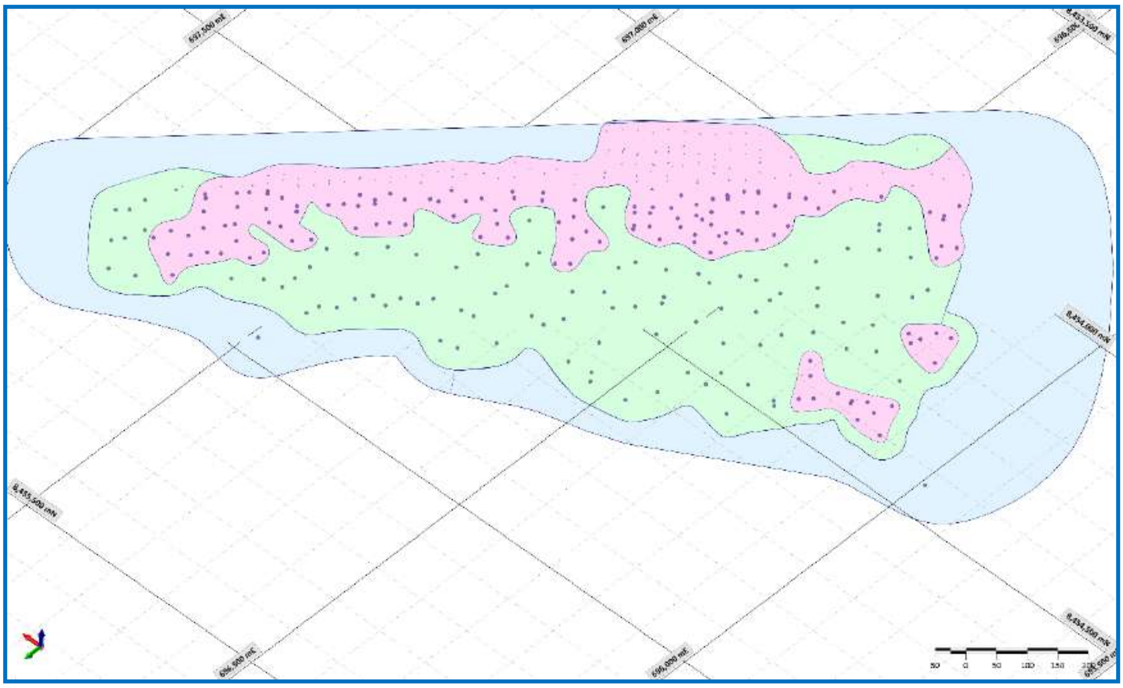
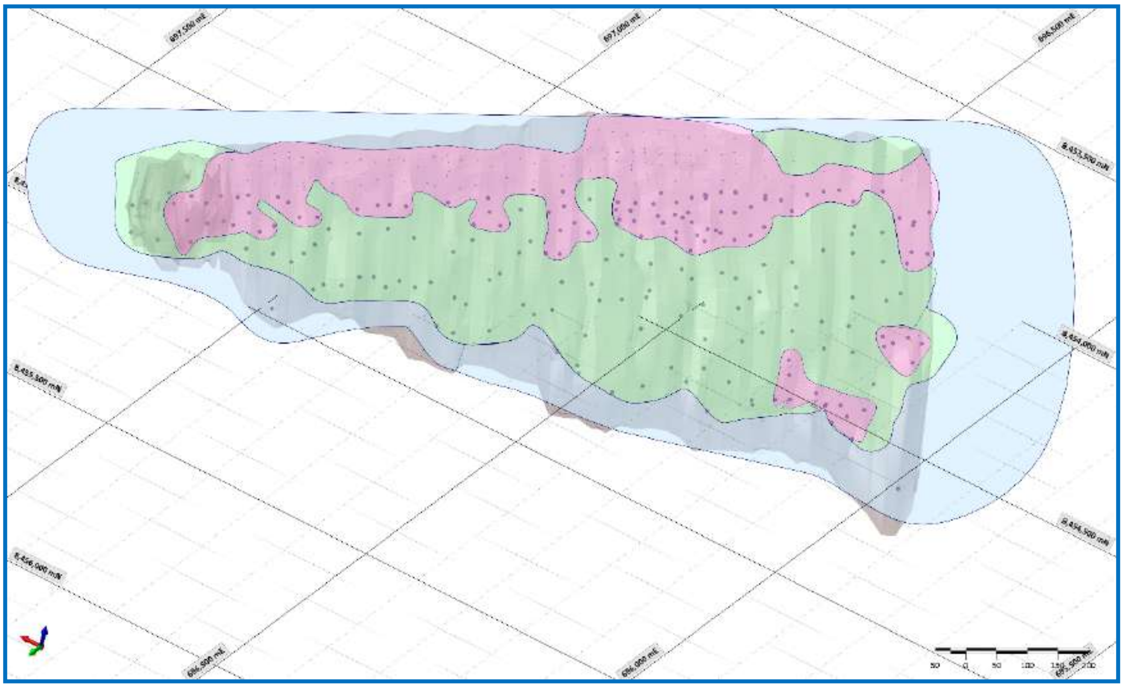


Figure 14-24: Resource zones with mineralization boundary superimposed



Those portions of the Z3 mineralization wireframe outside the resource category zones were not considered for potential classification as part of the resource.

The model was then tested using a pit optimization routine to generate a nominal pit shape that would meet a test of potentially economic to mine.

Pit generation inputs are set out in Table 14-15.

Table 14-15: Base pit generation parameters

Item	Value
Mining cost	US$1.97 per tonne
Recovery	90%
Processing Cost	US$ 15.60 per tonne
Selling / Royalty Cost	US$ 90.12 per ounce

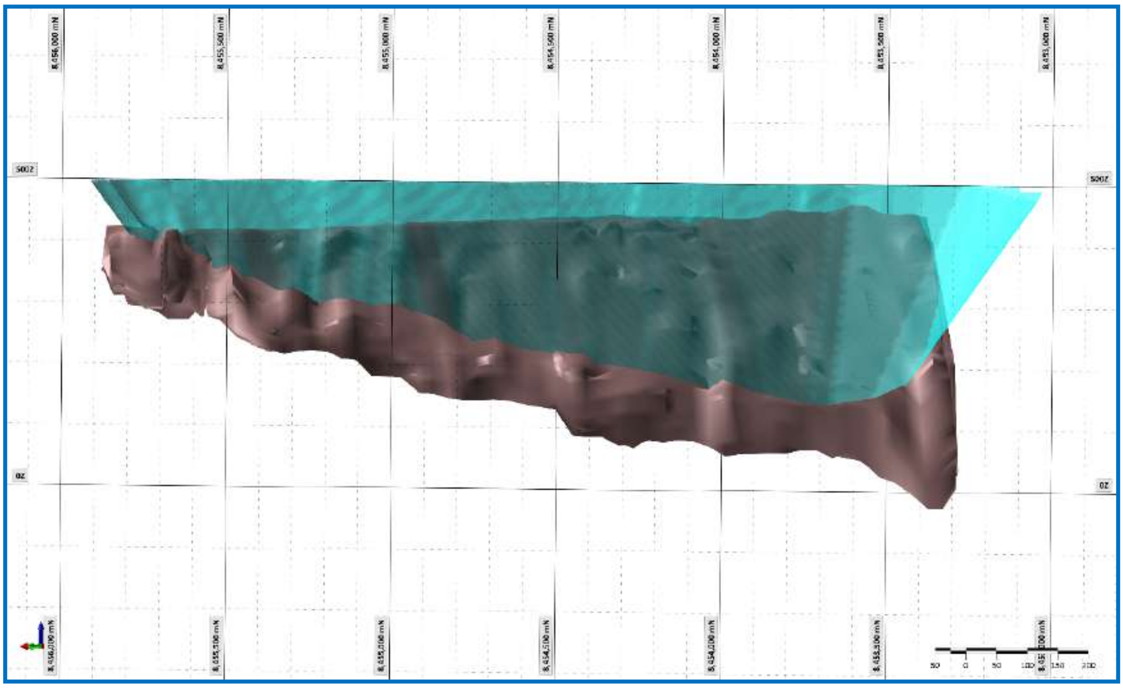
Item	Value
Gold Price	US$1,500 per ounce
Revenue adjustment factor	1.2

The pit generation parameters are in line with those used in the 2018 report, modified for current gold price and with the addition of potential upside in the form of the positive Revenue Factor (“**RF**”).

No attempt was made to analyses the results to determine if the resulting pit shape was the “best” possible shape. Rather it was assumed that blocks in a reasonably likely pit shape met the test in the CIM Definition Standards, (CIM Standing Committee on Reserve Definitions, 2014) that there are reasonable prospects for eventual economic extraction.

The resulting pit shape further constrained the set of blocks which could be classified as part of the resource. Only those blocks, inside the mineralization envelope, inside the nominal pit shape, Figure 14-25, were reported as part of the mineral resource. Note that economic modelling by SRK indicated that there were blocks in the footwall which economic modelling included but which were excluded by the nominal pit originally developed by AEFS. The AEFS nominal pit was therefore expanded slightly to include those footwall blocks. Additional constraints to ensure block were below the modelled ground surface and were not in areas of backfill, were placed on blocks reported as being part of the resource.

Figure 14-25: Portion of mineralization enclosed by a nominal pit



14.2.10 Mineral Resource

A Mineral Resource can only be declared for material which is considered to have potential for economic extraction at some point in the future. The cut-off at which a resource is reported should also meet this criterion, it should not include material which does not have reasonable potential to be mined and processed.

The definition of a Mineral Reserve on the other hand applies a specific set of economic parameters to a mineral resource to determine which portions of the Resource can be mined under those economic conditions.

In the case of the Posse Deposit economic modelling of the blocks in the model has indicated that the lowest grade block to be mined as ore has a grade of 0.37g/tAu. On this basis the cut-off grade for the mineral resource has been set at 0.35g/tAu. The Mineral Resource above a cut-off of 0.35g/tAu declared for the Posse Deposit is summarized in Table 14-16 and Table 14 17 while a grade tonnage curve for the deposit is shown in Figure 14 26. It should be noted that the Mineral Resource figures quoted are inclusive of any Mineral Reserves.

Table 14-16: 2021 Mineral Resource Statement(1)

Posse Gold Project – Mineral Resource 2021							
Above g/t	Volume Mm^3	Tonnes Mt	Density (t/m^3)	Au g/t	Au koz	Zone	Classification
0.35	1.4	3.9	2.80	2.40	290	Z1	Measured
0.35	1.5	4.1	2.80	2.20	290	Z1	Indicated
0.35	1.3	3.7	2.80	0.92	110	Z2	Measured
0.35	1.6	4.6	2.80	1.00	150	Z2	Indicated
0.35	2.2	6.0	2.80	0.56	110	Z3	Measured
0.35	3.7	10	2.80	0.60	190	Z3	Indicated
0.35	0.04	0.10	2.40	0.52	1.7	Z3	Inferred
0.35	4.9	14	2.80	1.20	510	All	Measured
0.35	6.8	19	2.80	1.10	640	All	Indicated
0.35	0.04	0.10	2.40	0.52	1.7	All	Inferred

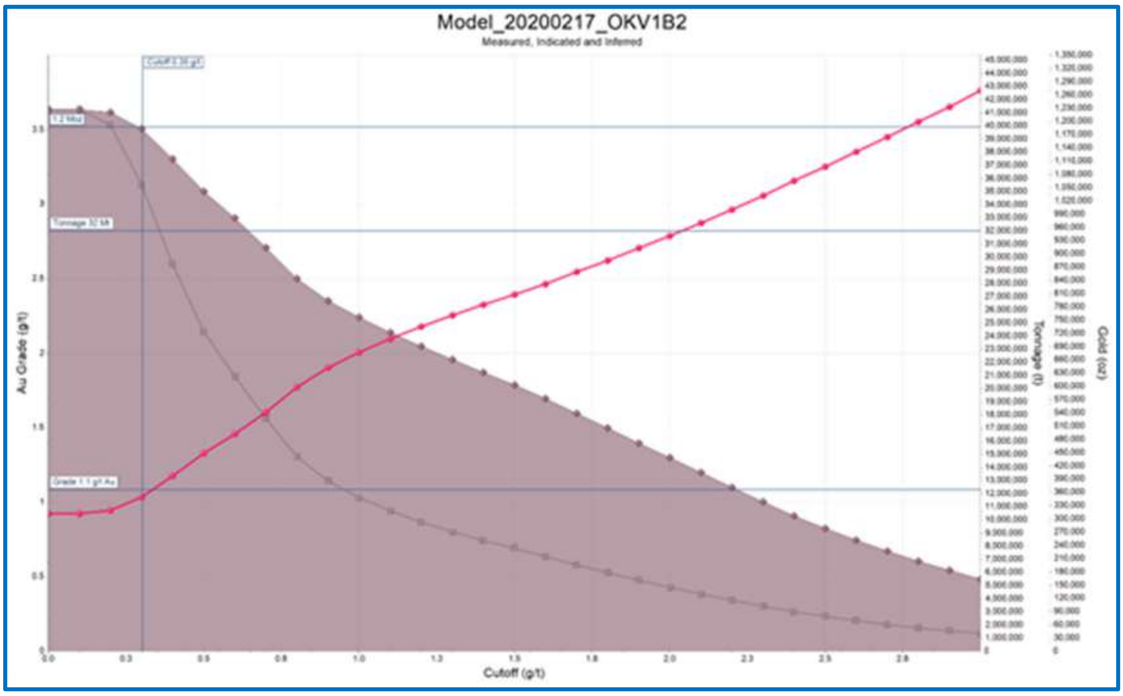
(1) All figures have been rounded to 2 significant figures. A cut-off grade of 0.35g/tAu has been used. Due to rounding numbers may not sum. The Mineral Resource is inclusive of Mineral Reserves

Table 14-17: 2021 Mineral Resource Statement: Measured and Indicated Mineral Resources(1)

Posse Gold Project - Mineral Resource Summary – Measured and Indicated only							
Above g/t	Volume Mm^3	Tonnes Mt	Density (t/m^3)	Au g/t	Au koz	Zone	Classification
0.35	1.4	3.9	2.80	2.40	290	Z1	Measured
0.35	1.5	4.1	2.80	2.20	290	Z1	Indicated
0.35	**2.9**	**8.0**	**2.80**	**2.30**	**590**	**Z1**	**M&I**
0.35	1.3	3.7	2.80	0.92	110	Z2	Measured
0.35	1.6	4.6	2.80	1.00	150	Z2	Indicated
0.35	**3.0**	**8.3**	**2.80**	**0.98**	**260**	**Z2**	**M&I**
0.35	2.2	6.0	2.80	0.56	110	Z3	Measured
0.35	3.7	10	2.80	0.60	190	Z3	Indicated
0.35	**5.8**	**16**	**2.80**	**0.58**	**300**	**Z3**	**M&I**
0.35	4.9	14	2.80	1.20	510	All	Measured
0.35	6.8	19	2.80	1.10	640	All	Indicated
0.35	**12**	**32**	**2.80**	**1.10**	**1,200**	**All**	**M&I**

(1) All figures have been rounded to 2 significant figures. A cut-off grade of 0.35g/tAu has been used. Due to rounding numbers may not sum. The Mineral Resource is inclusive of Mineral Reserves

Figure 14-26: Grade tonnage curve for Posse 2020 Mineral Resource



Mineral Resource Modifying Factors

No further modifying factors such as mining methods, metallurgy, environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other factors are required to support the current Mineral Resource Statements as reported herein.

14.3 Interpretation and Conclusions

The Posse Deposit is hosted by a mylonitic shear hosted zone in a high greenschist to low amphibolite metamorphic terrain. The ore body strikes NE-SW and dips about 50° to the NW. On average, the ore body is about 30m wide. Alteration is dominated by silicification, sericitization, K- feldspar flooding and pyritisation. Gold is positively correlated with the intensity of silicification and total sulphide content and occurs as 10-100 micron sized particles along the margins of silicates and in association with pyrite (FeS_2) and frohbergite ($FeTe_2$).

The Mineral Resource reported in this CPR, has established a mineral resource of around 32Mt containing around 1.2MozAu, at a grade of 1.10g/tAu, above a cut-off grade of 0.35g/tAu in the Measured and Indicated Resource categories. A further 100kt containing 1.7koz Au, at a grade of 0.52g/tAu, above a cut-off of 0.35g/tAu has been classified as an Inferred Mineral Resource.

The updated mineral resource includes Measured, Indicated and Inferred Mineral Resource categories. Drilling completed in 2019 and reported as part of this report has significantly increased the confidence in the current mineral resource estimate compared to that reported in 2018. The resource has been extensively tested in a risk review, Section Geological Risk Assessment, this work suggests it is still appropriate to use the resource reported as part of the 2020 DFS as the current mineral resource for the Posse deposit.

The opinion of AEFS is that the character of the Mara Rosa Property, the Posse Deposit and the Mineral Resource Estimate reported herein is appropriate to support the continued development of the Posse project and valuations which may be derived from the current knowledge of the project.

15 MINERAL RESERVE ESTIMATES

15.1 Summary

The Mineral Reserves is derived from Measured and Indicated Resources based on CIM guidelines. As described in Section 14, the 2020 Mineral Resources estimate remained unchanged and are reported in the 2021 Statements.

To convert Mineral Resources to Mineral Reserves, consideration was given to forecasts and estimates of gold price, metallurgical recovery, mining dilution and ore loss factors, royalties and costs associated to mining, processing, overhead, refining and logistics. After the completion of the DFS 2020, some of these parameters were updated to reflect more accurately the current economic conditions of the Project, including:

- Long term gold price;
- Processing operating costs;
- Mining operating costs;
- G&A costs; and
- Project implementation and mine schedules.

SRK verified the effect of these changes on the economic cut-off grade and pit design. No material impact was noted. Therefore, the Mineral Reserve estimated in the DFS 2020 remained unchanged. Specifically, the Mineral Reserve estimated in 2020 reached 23.8Mt (dry) at an average grade of 1.18g/tAu. The detailed breakdown of the Mineral Reserve is presented in Table 15 1. It is SRK's opinion that the Mineral Reserve estimation is compliant with CIM Definition Standards.

This Mineral Reserve is estimated on the basis of currently available information. The Reserve classification reflects the level of accuracy of the updated DFS.

Table 15-1: 31 December 2021 Mineral Reserve Estimate(1)

Mineral Reserve	Diluted tonnes (Mt dry)	Diluted grade (g/t Au)	Contained metal (koz Au)	Estimated recovery (%Au)	Recoverable metal (koz Au)
Proven	11.8	1.20	456	89.9%	410
Probable	12.0	1.16	446	89.8%	401
Total Mineral Reserve	23.8	1.18	902	89.9%	811

(1) A gold price of US$1,450/oz is assumed. An exchange rate of R$5.05 to US$1.00 is assumed. Mineral Reserves are based on Measured and Indicated Mineral Resources only. Mineral Reserves above an economic cut-off grade of 0.37g/tAu. The Mineral Reserve is included in the Mineral Resource quoted in Table 14-17.

15.2 Disclosure

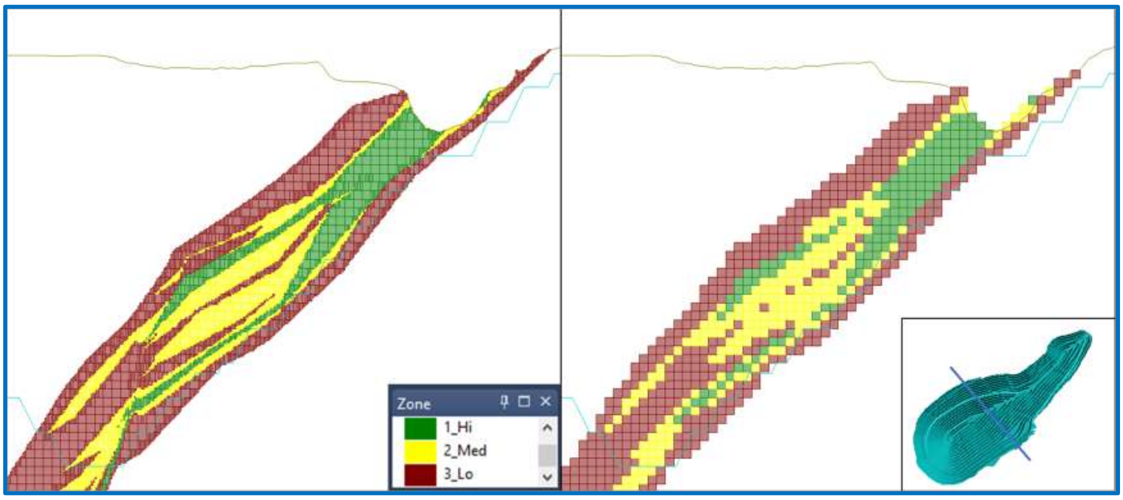
Mineral Reserves reported in Section 15 were based on the update of the DFS completed under the supervision of Mr Paulo Laymen who is a Qualified Person as defined in CIM Definition Standards, and an associate consultant of SRK which is independent of Amarillo Gold.

15.3 Mine Planning Model, Mining Dilution and Ore Losses

Mining inventories are particularly sensitive to Au grade and the manner in which the geological resource model blocks are manipulated to match the Selectivity Mining Unit ("**SMU**") for the mining method selected. In this study, SRK sought to produce a mine planning model adequate for bench mining with small excavators (4.5m^3 bucket) while maintaining the lithological domains present in the resource model.

To achieve this aim SRK performed a regularization process by which the 5m (W) x 10m (L) x 5m (H) sub-celled resource model was converted into a 5m (W) x 5m (L) x 5m (H) model. Figure 15 1 illustrates the effect of block model regularization. The mineralized domains described in Section 14.2, i.e., high (Z1), medium (Z2) and low grade (Z3), are represented in green, yellow and red, respectively.

Figure 15-1: Vertical section of resource model (left) and mining model (right)



As a result of this procedure, a 4% mining dilution and a 4% ore loss factor were estimated within the resource pit shell. The grade tonnage curves of the mine planning model against the resource model are shown in Figure 15-2.

A summary of the mine planning inventory within the resource pit is reported in Table 15-2.

Figure 15-2: Grade tonnage curve – comparison of resource model and mining model

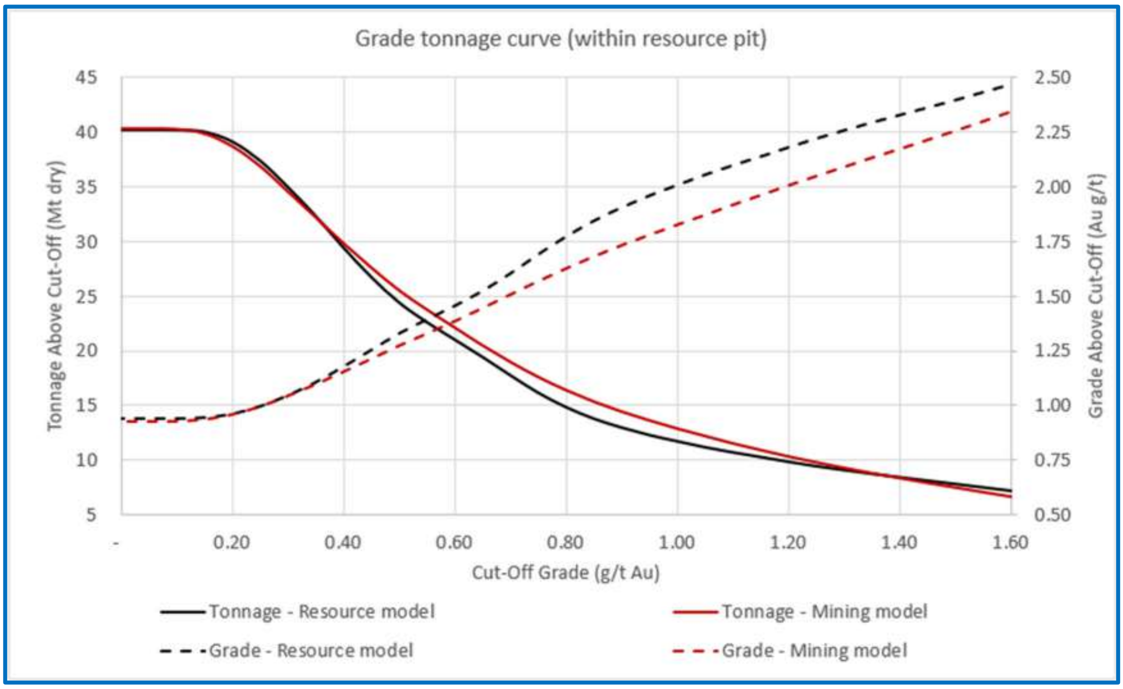


Table 15-2: Mining inventory within the resource pit

Above cut-off (g/t Au)	Volume (Mm^3)	Diluted tonnes (Mt dry)	Density (t/m^3)	Diluted grade (g/t Au)	Contained metal (koz Au)	Classification
0.35	4.9	13.6	2.77	1.16	507	Measured
0.35	6.7	18.5	2.77	1.05	624	Indicated
0.35	**11.6**	**32.0**	**2.77**	**1.10**	**1,131**	**M&I**
0.35	0.0	0.1	2.41	0.53	1	Inferred
0.35	**11.6**	**32.1**	**2.77**	**1.10**	**1,132**	**MI&I**

15.4 Geotechnical Pit Slopes

No additional geotechnical studies were undertaken since the completion of the DFS 2020. Therefore, the pit slope angles used by SRK were based the 'DFS Geotechnical Assessment 2013' prepared by Coffey. The geotechnical design parameters are presented in Table 15-3.

Table 15-3: Geotechnical design parameters[(1)]

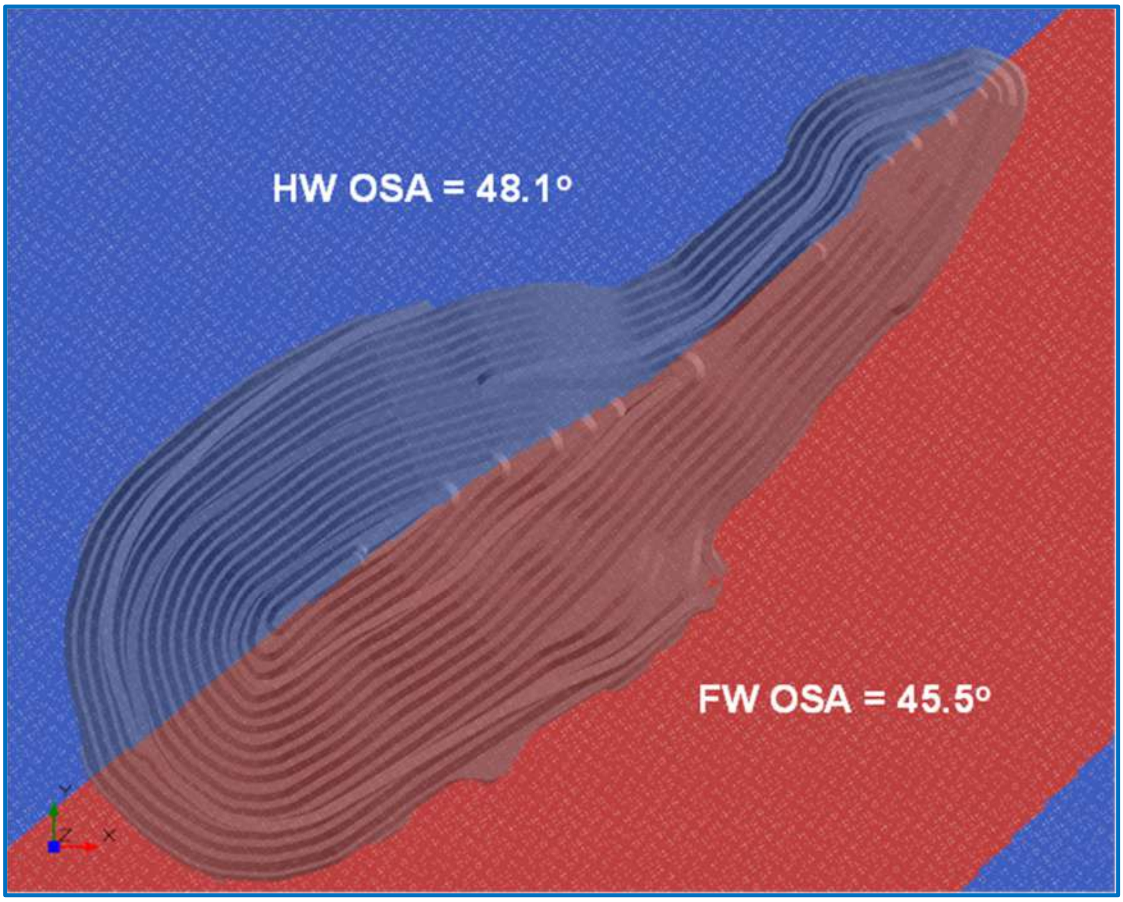
Domain	Design Sector	Slope dip direction (°) From / To	Weathering	BFA (°)	BW (m)	BH (m)	IRSA (°)	Design Options	IRSH (m)	OSH (m)
All	All	All	Weathered	55.0	5.0	10	40.0	1	20	
Schist	North	350/070	Fresh	65.0	6.0	20	52.5	1		
	South	190/250	Fresh	65.0	6.0	20	52.5	1		
Footwall Amphibolite	East	070/190	Fresh	65.0	14.5	20	40.0	1		280
				65.0	10.5	20	45.0	2	260	
				65.0	8.5	20	48.0	3		
Hanging wall Gneiss	West	250/350	Fresh	65.0	6.0	20	52.5	1		

(1) Abbreviations: BFA - Batter Face Angle; BW - Berm Width; BH - Batter Height; IRSA - Inter-Ramp Slope Angle (crest to crest); IRSH - Inter-Ramp Slope Height; OSH - Overall Slope Height. Source: (Coffey Mining, 2013).

These parameters were further adjusted to obtain the overall slope angles used for the pit optimization. Specifically, two main sectors, i.e., footwall and hanging wall, were created as shown in Figure 15-3. Besides, a preliminary design of in-pit haul road, which included three 13-m wide roads, was performed to anticipate their effect on the slope angles. As a result of it, the overall slope angles were:

- Footwall OSA = 45.5°; and
- Hanging wall OSA = 48.1°.

Figure 15-3: Geotechnical sectors for pit optimization



15.5 Pit Optimization

15.5.1 Pit optimization parameters

The input optimization parameters were agreed between Amarillo and SRK and are listed in Table 15-4. Prices and costs are in US$.

Mining costs were based on updated quotes obtained from mining contractors after the consolidation of the DFS 2020. Mining cost adjustment factors were applied to reflect specific conditions of drilling and blasting, variable haul distances and grade control.

Table 15-4: Pit optimization parameters

Parameters	Updated DFS	Basis
Included Mineral Resources	Measured & Indicated only	AEFS
Total material movement	20.0 Mtpa dry	Fleet and mine capacity
Ore processed	2.5 Mtpa dry	Plant capacity
Mining dilution and ore loss	Included in the mining model	SRK
Average moisture content	3%	SRK
Breakeven cut-off grade	0.37 g/t Au	Calculation
Geotechnical (OSA)	HW = 48.1° / FW = 45.5°	Coffey & SRK
Power cost	US$0.0565/kWh	Amarillo
Diesel cost(1)	US$0.92/L	Amarillo
Reference mining cost for waste	US$1.85/dmt	Amarillo & SRK
Reference mining cost for ore	US$2.24/dmt	Amarillo & SRK
Overall average mining cost	US$1.92/dmt	Amarillo & SRK
Detailed mining costs contractor for waste		
Drilling (fresh material)	US$0.22/dmt	Amarillo - Mine Contractor
Drilling (weathered material)(2)	US$0.08/dmt	Amarillo - Mine Contractor
Blasting (fresh material)	US$0.34/dmt	Amarillo - Mine Contractor
Blasting (weathered material)(2)	US$0.12/dmt	Amarillo - Mine Contractor
Detailed load, hauling and dump costs contractor for waste		
In-pit haul distance	Variable	SRK
Ex-pit haul distance	1,762 m	SRK
Haul distance up to 500 m	US$0.83/dmt	Amarillo - Mine Contractor
Haul distance 501 – 1,000 m	US$0.89/dmt	Amarillo - Mine Contractor
Haul distance 1,001 – 1,500 m	US$0.94/dmt	Amarillo - Mine Contractor
Haul distance 1,501 – 2,000 m	US$0.99/dmt	Amarillo - Mine Contractor
Haul distance 2,001 – 2,500 m	US$1.03/dmt	Amarillo - Mine Contractor
Haul distance 2,501 – 3,000 m	US$1.08/dmt	Amarillo - Mine Contractor
Haul distance 3,001 – 3,500 m	US$1.13/dmt	Amarillo - Mine Contractor
Haul distance 3,501 – 4,000 m	US$1.19/dmt	Amarillo - Mine Contractor
Haul distance 4,001 – 4,500 m	US$1.24/dmt	Amarillo - Mine Contractor
Haul distance 4,501 – 5,000 m	US$1.30/dmt	Amarillo - Mine Contractor
Haul distance 5,001 – 5,500 m	US$1.35/dmt	Amarillo - Mine Contractor
Haul distance 5,501 – 6,000 m	US$1.41/dmt	Amarillo - Mine Contractor
Owner mining costs		
Labour, dewatering, systems, and others	US$0.18/dmt	Amarillo & SRK
Additional mining costs for ore		
Grade control	+ US$0.25/dmt	SRK
Drilling (fresh material)	+ US$0.26/dmt	Amarillo - Mine Contractor
Hauling cost	- US$0.12/dmt	SRK
Total additional mining cost for ore	+ US$0.39/dmt	SRK
Processing recovery and costs		
Plant recovery (P_{80} 53 µm grind size)	(Au-(0.0854xAu$^{0.8718}$+0.023))/Au	Aurifex
Plant processing cost	US$11.32/dmt	Amarillo
Tailings haulage and disposal cost	US$1.00/dmt	Amarillo
G&A costs	US$0.77/dmt	Amarillo
Revenue and selling costs		
Gold price	US$1,450/oz	Amarillo
Refining, transportation, insurance, sales	US$12.00/oz	Amarillo
Gold price net of refining & transport	US$1,438/oz	Calculation
CFEM (1.5% of Gold price)	US$21.75/oz	Amarillo
Royalties (Royal Gold and Franco-Nevada)	US$53.93/oz	Amarillo
Net revenue	US$1,362.32/oz	Calculation
Discount rate	5%	Amarillo

(1) Diesel cost net of 9.25% Brazilian taxes (PIS and COFINS), as it is subject to tax recovery by Amarillo.
(2) Only 30% of the weathered material is drilled and blasted.

15.5.2 Pit optimization results

A number of nested pit shells were generated by Whittle software for a range of revenue factors on the gold price. Preliminary cash flows are estimated by the optimizer based on a 5% discount rate and a nominal gold price of US$1,450/oz.

Three optimization scenarios are automatically generated by Whittle software:

- Best case. It is based on an increasing pit shell extraction sequence;
- Worst case. It follows a bench-by-bench mining sequence; and
- Specified case. The extraction sequence is created upon predefined pushback geometries. This scenario is considered as closest to the operational and economic reality.

The pit optimization results are presented in Table 15-5 and Figure 15-4. At this stage, no stockpile provision was considered. The cash flow figures presented do not include capital costs.

The ultimate pit shell was selected through marginal analysis. Larger pit shells that have modest increases in NPV for large increases in rock tonnage were avoided. The 0.81 revenue factor (“**RF**”) pit shell (pit No 62) was selected as the basis for detailed designing of the ultimate pit and intermediate phases. This pit shell greatly coincides with the pit selected in the DFS

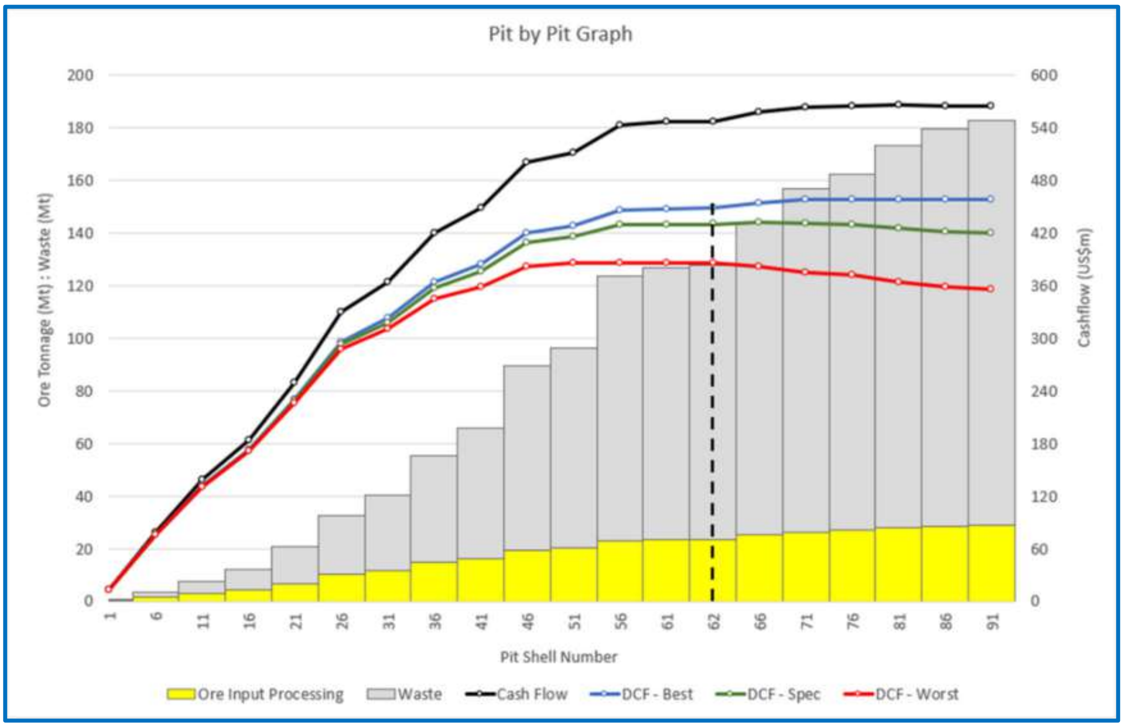
2020 which corresponded to 0.91RF.

Table 15-5: Pit optimization results and preliminary cash flows @1,450 US$/oz gold price

Pit by Pit Graph Scenario Hb			Ore Input Processing		Waste	Strip Ratio	Contained metal	Recovered metal	Cash Flow[(1)]	DCF Spec[(2)]
Pit	RF	US$/oz	Mt dry	g/t Au	Mt dry	t:t	koz Au	koz Au	US$m	US$m
1	0.2	290	0.2	1.75	0.1	0.56	14	13	13.2	13.1
6	0.25	362.5	1.6	1.69	1.9	1.19	86	78	78.8	76.4
11	0.3	435	3.1	1.58	4.4	1.43	156	142	138.1	131.1
16	0.35	507.5	4.6	1.48	7.4	1.61	217	196	184.2	172.1
21	0.4	580	6.6	1.45	14.1	2.15	305	276	249.6	228.7
26	0.45	652.5	10.3	1.30	22.4	2.18	430	388	330.0	292.8
31	0.5	725	11.9	1.28	28.3	2.38	487	439	364.1	318.0
36	0.55	797.5	14.7	1.24	40.8	2.77	588	530	420.0	358.0
41	0.6	870	16.4	1.22	49.3	3.00	646	581	448.6	377.0
46	0.65	942.5	19.5	1.21	70.0	3.58	760	684	500.7	409.5
51	0.7	1015	20.4	1.20	76.0	3.72	790	711	512.2	415.8
56	0.75	1087.5	23.1	1.19	100.4	4.34	888	798	543.2	429.1
61	0.8	1160	23.6	1.19	103.2	4.37	901	810	546.7	430.2
62	**0.81**	**1174.5**	**23.7**	**1.19**	**103.9**	**4.38**	**904**	**813**	**547.4**	**430.4**
66	0.85	1232.5	25.3	1.18	118.5	4.69	955	859	557.7	432.4
71	0.9	1305	26.4	1.17	130.7	4.95	994	893	563.9	431.1
76	0.95	1377.5	27.1	1.16	135.3	4.99	1,012	909	565.3	429.3
81	**1.0**	**1450**	**28.0**	**1.15**	**145.2**	**5.19**	**1,040**	**934**	**565.9**	**425.8**
86	1.05	1522.5	28.7	1.15	151.1	5.26	1,058	950	565.4	421.5
91	1.1	1595	29.0	1.14	154.1	5.32	1,065	956	564.7	419.6

(1) Undiscounted cash flow with no capital cost included.
(2) Discounted cash flow for the specified case do not include capital costs.

Figure 15-4: Pit by pit graph with preliminary cash flows @1,450 US$/oz gold price



15.5.3 Pit sensitivity analysis

SRK performed several pit optimization runs varying the mining and processing costs, metallurgical recovery and overall slope angles ("**OSA**") to verify the sensitivity of ultimate pits. The following scenarios were performed and analysed:

- Base case scenario presented in Section 15.5.2;
- Downside variation:
  - Mining cost 10% higher,
  - Processing cost 10% higher,
  - Metallurgical recovery 5% lower,
  - All above negative factors combined,

– Overall slope angles 5° gentler; and

- Upside variation:
  - – Mining cost 10% lower,
  - – Processing cost 10% lower,
  - – Metallurgical recovery 5% higher,
  - – All above positive factors combined,
  - – Overall slope angles 5° steeper.

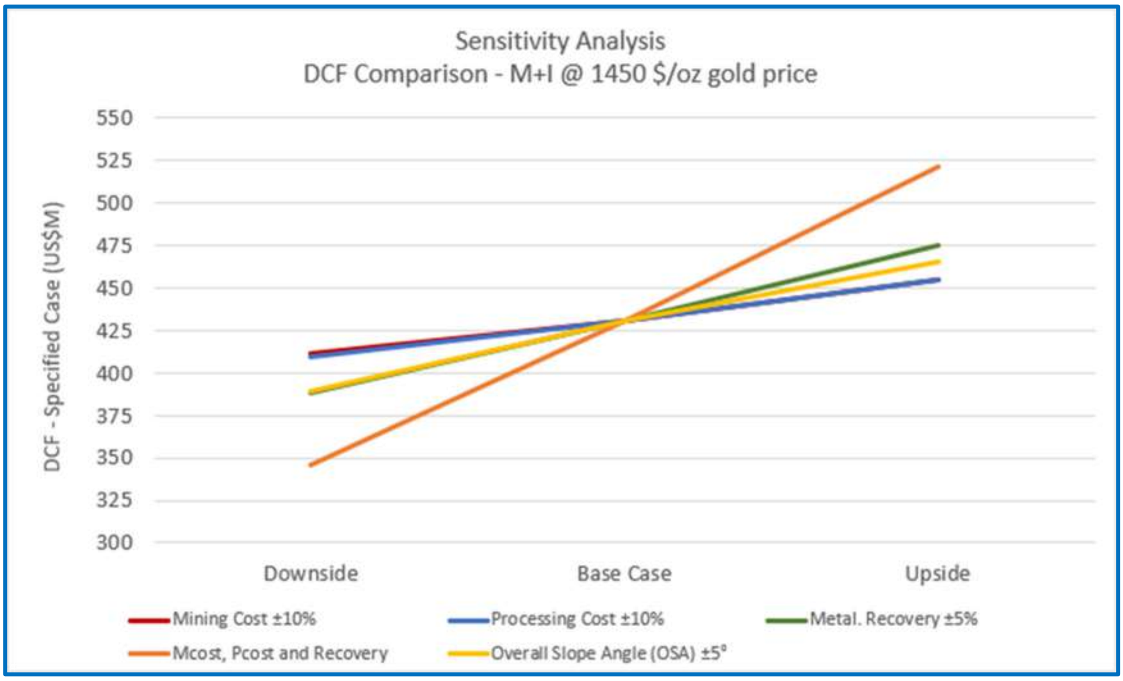
Table 15 6 summarizes the results for all scenarios at a 0.81 revenue factor pit shell. Figure 15 5 presents preliminary cash flows for a whole range of revenue factors.

The sensitivity analysis indicates that the pit shell is most sensitive to metallurgical recovery and overall slope angles, and least sensitive to mining and process costs. Variations on mining and process costs cause similar impact on the discounted cash flows as shown in Table 15-6 and Figure 15-5.

Table 15-6: Sensitivity results for the 0.81 revenue pit shells

0.81 Revenue Factor Pit Shells	Base Case	Downside variation					Upside variation				
		MCost +10%	PCost +10%	MRec. -5%	Costs & Rec.	OSA -5°	MCost -10%	PCost -10%	MRec. +5%	Costs & Rec.	OSA +5°
Ore processed (Mtdry)	23.7	23.3	22.4	22.8	19.2	21.4	25.9	26.1	25.8	28.7	25.8
Diluted grade (g/tAu)	1.19	1.19	1.23	1.21	1.26	1.18	1.18	1.14	1.16	1.11	1.19
Contained metal (kozAu)	904	893	883	888	777	812	980	957	962	1027	986
Recover. metal (kozAu)	813	803	794	758	665	730	881	859	907	968	887
Waste (Mtdry)	104	101	103	102	78	97	127	114	118	132	110
DCF Spec. Case (US$m)	430	411	410	388	346	389	455	454	475	522	465

Figure 15-5: Sensitivity results on the preliminary cash flows @1,450 US$/oz gold price



15.6 Ultimate Pit Design

In 2020, detailed designing was performed on the ultimate pit selected at that time (0.91 RF) including accesses and ramps. As the pit selected in the current work (0.81 RF) greatly coincided with the 2020 pit, it was decided to preserve the previous pit design.

The geometrical assumptions are listed below:

- Geometric parameters: derived from Table 15-3.
- Roads and ramps width: 13m; and
- Maximum ramp gradient: 10%.

Figure 15-6 shows the ultimate designed pit. A comparison of tonnage and grades of the optimized pit shell and the designed pit is presented in Table 15 7. It is SRK's opinion that the differences of tonnage and grades found are acceptable.

Figure 15-6: Ultimate designed pit

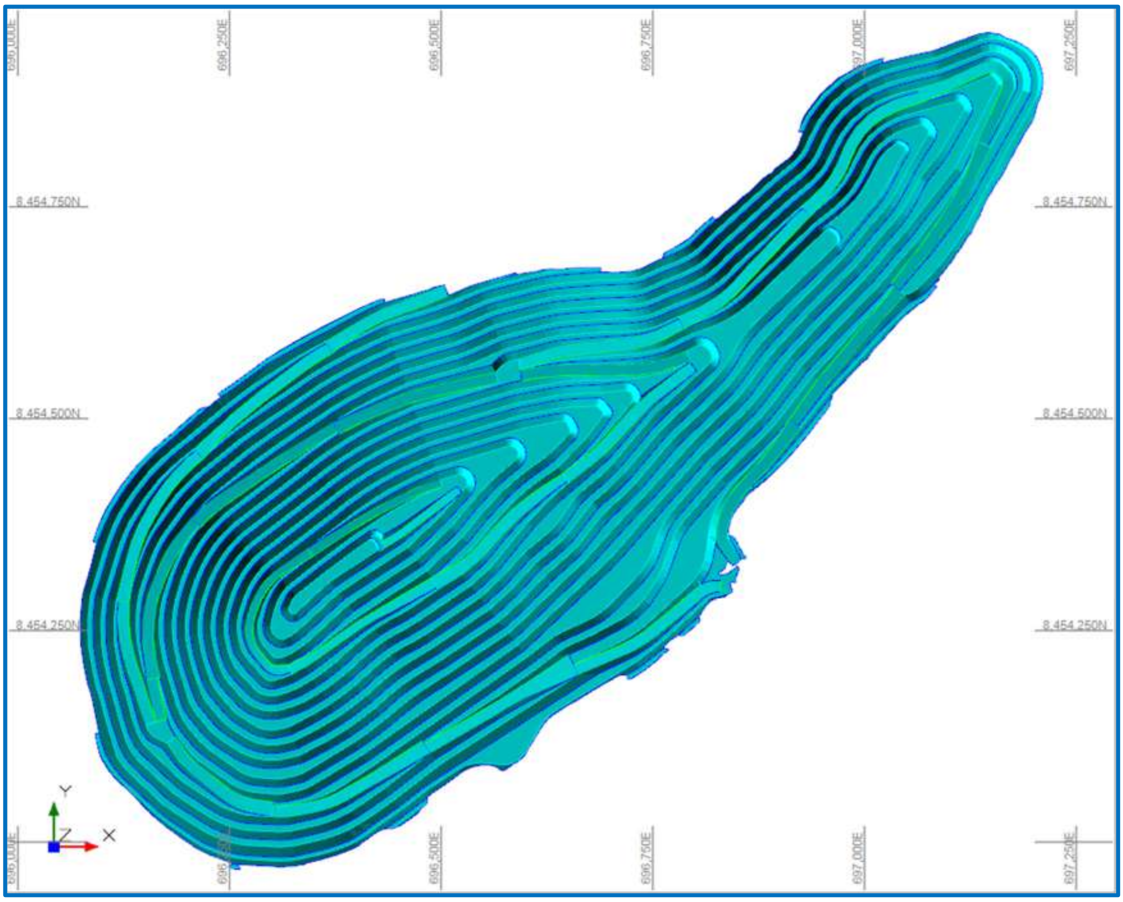


Table 15-7: Comparison between the optimized pit and the designed pit(1)

Pit comparison	Ore tonnes (Mt dry)	Diluted grade (g/t Au)	Contained metal (koz Au)	Waste tonnes (Mt dry)	Strip Ratio (t/t)
Whittle pit shell (RF 0.81)	23.6	1.19	903	104.0	4.41
Ultimate designed pit	23.8	1.18	902	105.7	4.44
Difference (%)	0.9%	-	-0.1%	1.6%	-

(1) Note: above a cut-off grade of 0.37 g/t Au.

15.7 Economic Cut-off for Mineral Reserve Definition

Table 15 5 in Section 15.5.1 presented the input parameters used for pit optimization. They were based on data and quotes collected after the completion of the DFS 2020. Specifically, the following parameters were updated:

- Long term gold price;
- Processing operating costs;
- Mining operating costs;
- G&A costs; and
- Project implementation and mine schedules.

SRK verified the effect of these changes on the economic cut-off grades and pit design. No material impact was noted. A comparison of the input parameters used for the calculation of the economic cut-off grade is presented in Table 15-8. A 0.37g/tAu cut-off grade was estimated for the purpose of the Mineral Reserve estimation.

Table 15-8: Comparison between the optimized pit and the designed pit(1)

Parameters	Unit	DFS Update 2021	DFS 2020
Exchange rate	R$:US$	5.05	4.20
Gold price	US$/oz	1,450	1,400
Selling cost	US$/oz	87.28	85.05
Waste mining cost	US$/t	1.85	1.63
Ore mining cost	US$/t	2.24	1.95
Processing cost	US$/t	12.32	11.49
G&A cost	US$/t	0.77	0.75
Metallurgical recovery	%	$(Au-(0.0854*Au^{0.8718}+0.023))/Au$	$(Au-(0.0854*Au^{0.8718}+0.023))/Au$
Mineral Reserve cut-off	g/t Au	0.37	0.37

Therefore, the Mineral Reserve estimated in the DFS 2020 remained unchanged. Specifically, the Mineral Reserve estimated reached 23.8Mt (dry) at an average grade of 1.18g/tAu. A Mineral Reserve of 23.8Mt (dry) at an average grade of 1.18g/tAu was reported as of 31 December 2021. The detailed breakdown of the Mineral Reserve is presented in Table 15 9. SRK believes that the reserve estimation is reasonable and meets the CIM Definition Standards.

Table 15-9: 31 December 2021 Mineral Reserve Estimate(1)

Mineral Reserve	Diluted tonnes (Mt dry)	Diluted grade (g/t Au)	Contained metal (koz Au)	Estimated recovery (%Au)	Recoverable metal (koz Au)
Proven	11.8	1.20	456	89.9%	410
Probable	12.0	1.16	446	89.8%	401
Total Mineral Reserve	23.8	1.18	902	89.9%	811

(1) A gold price of US$1,450/oz is assumed. An exchange rate of R$5.05 to US$1.00 is assumed. Mineral Reserves are based on Measured and Indicated Mineral Resources only. Mineral Reserves above an economic cut-off grade of 0.37g/tAu. The Mineral Reserve is included in the Mineral Resource quoted in Table 14-17.

16 MINING METHODS

16.1 Mining Operations

The Posse Gold Project is based on a mining concept that uses conventional drill, blast, load and haul techniques for all mining areas and rock types. One hundred per cent of the fresh rock and 30% of the saprolite will be blasted and loaded with excavators into 8x4 on-road trucks, and hauled to final destinations, i.e., primary crusher, low grade stockpiles or waste dumps. Direct mining will be applied to soft material such as soil and fill materials.

Specifically, primary mining will be undertaken by 74-t hydraulic excavators coupled with 45-t heavy tipper trucks.

A 15-month pre-stripping phase, between October 2022 and December 2023, was planned to ensure an initial supply of ore. During this stage the pit will require dewatering of the existing pit lakes. Water will be removed using a pumping system and directed to the planned water reservoir.

The ore and ore/waste contact materials will be mined in 5-m high benches for selectivity purposes, while double benches of 10-m high will be adopted for waste where there is no risk of dilution or ore loss.

The mining method will generate variable quantities of low grade that will require the use of stockpiles. Front-end loaders ("**FELs**") will provide RoM feed and stockpile re-handling.

The mined waste will be distributed into six waste dumps.

The mined materials will be transported along roads cut through the mining area to give a suitable gradient. Double-lane haul roads were designed with 13m width. Exceptionally, 10m wide roads were designed to reach the pit bottom. The maximum gradient used is 10%.

A loss of trafficability on haulage routes is anticipated to occur on a seasonal basis during the rainy season. Management plans and risk reduction strategies should be developed along with detailed procedures to recover haul road sheet and mining areas as soon as practicable.

Predominantly, the primary crusher will be directly fed by trucks and, occasionally, by a FEL. The ore stockpiled will feed the plant through a combination of FELs and trucks.

Grade control will be performed via drilling, sampling and assaying potential ore material within the pit boundaries.

The mine will operate 365 days, 24 hours in 3 shifts. The base case for the Project is a contractor operation.

16.2 Mine Layout

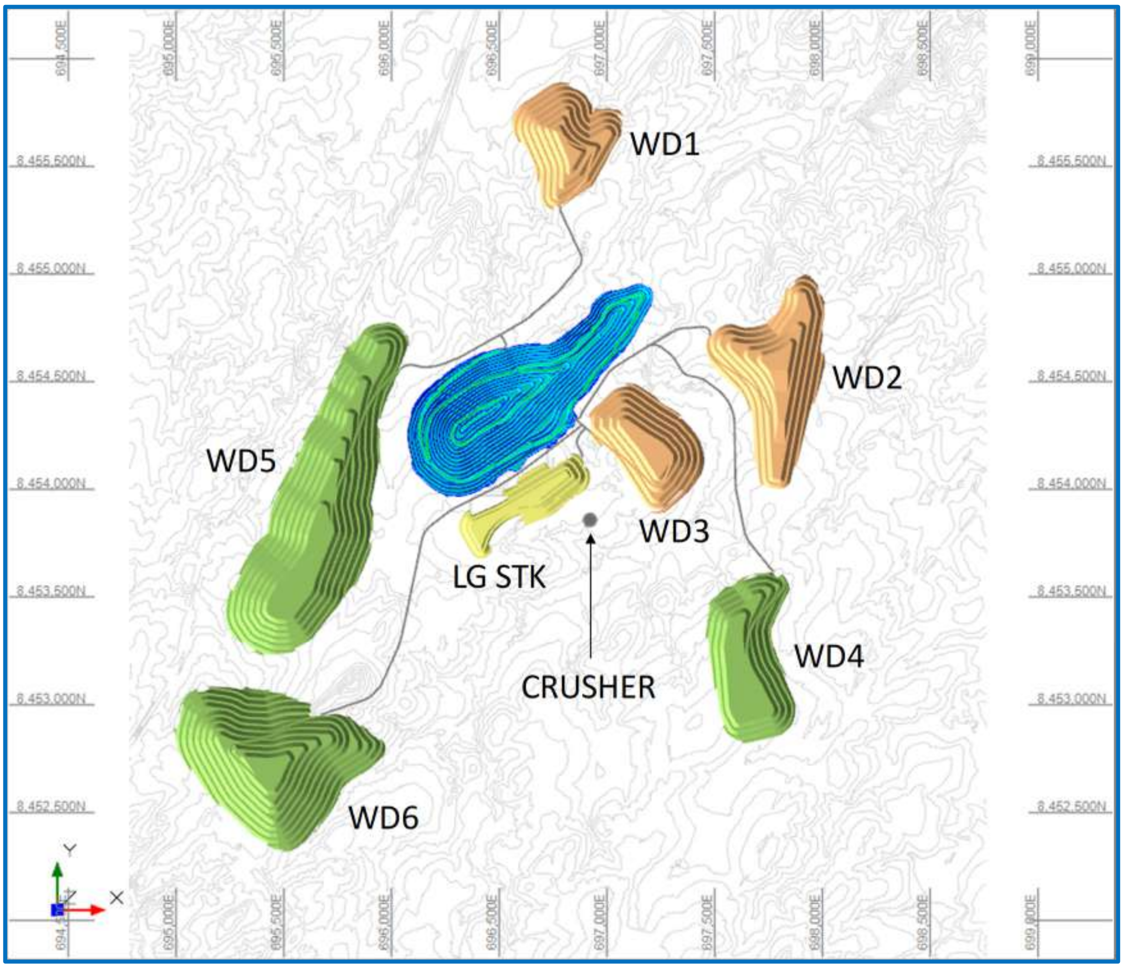
A general mine layout is presented in Figure 16-1 including all key components: pit, waste dumps, low grade stockpile, primary crusher and ex-pit haul roads. Other infrastructure items are described in Section 18.

The area selected for the installation of the primary crusher and process plant is at 600-m safety distance.

The waste dumps 1, 2, 3 and 4 account for 36% of total waste dumping capacity of the Project, i.e., sufficient to meet more than 3 years of operation, including the pre-stripping phase. These waste dumps have already been granted the Installation License (**"LI"**). Basic engineering projects were completed for waste dumps and the low-grade stockpile.

Additional areas were identified for future waste dumping and licensing (WD 5 and 6). SRK performed conceptual designs following similar geotechnical parameters as WD 1, 2, 3 and 4. Detailed engineering and licensing for these waste dumps will be undertaken when required by the mine schedule.

Figure 16-1: Mine layout



The volumetric capacity of the waste dumps and the low-grade stockpile are shown in Table 16-1 along with the average haul distances to the pit.

Table 16-1: Waste dumps and stockpile's capacities and ex-pit haul distances

Waste Dump	Capacity (Mm^3)	Contribution (%)	Haul distance (m)
WD1	4.0	8%	1,250
WD2	6.4	13%	1,400
WD3	3.7	7%	490
WD4	4.2	8%	2,400
WD5	16.9	34%	1,500
WD6	15.1	30%	2,400
Total WD	**50.2**	**100%**	**1,740**
Low-grade stockpile	1.5	100%	600

16.3 Mine Scheduling

16.3.1 Phases Design

Given that the pit design and the economic cut-off grade has not varied since 2020, SRK decided also to preserve the intermediate pushbacks designed at that time, which included accesses and ramps. These phases were designed using the same geometrical parameters as the ultimate pits.

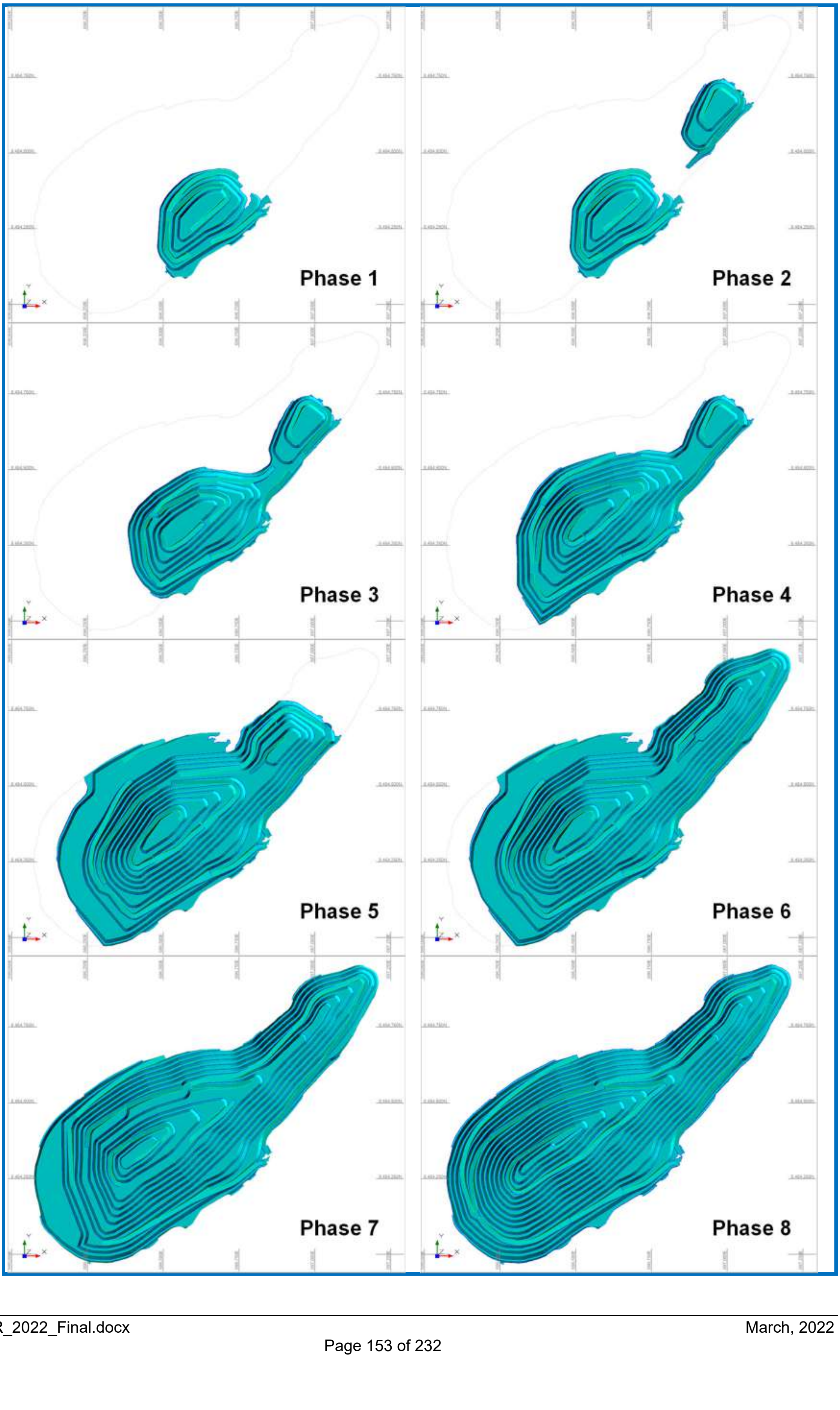
All basic geometrical parameters used are summarized below:

- Geometric parameters: derived from Table 15-3;
- Roads and ramps width: 13m;
- Maximum ramp gradient: 10%; and
- Minimum mining width: 50m. In specific situations, a minimum mining width of 25 was

assumed.

Figure 16-2 presents the designed phases.

Figure 16-2: Pit phases designs



16.3.2 Production Scheduling

The mine scheduling presented in the DFS 2020 was adjusted in accordance with the milestones of the new implementation schedule.

The objective of the production scheduling is to meet the production needs of the mill and maximize the NPV while maintaining adequate operational practices and a safely operated mine. Production scheduling involved the definition of mining phases, the cut-off and stockpiling strategy and the development of a mining schedule.

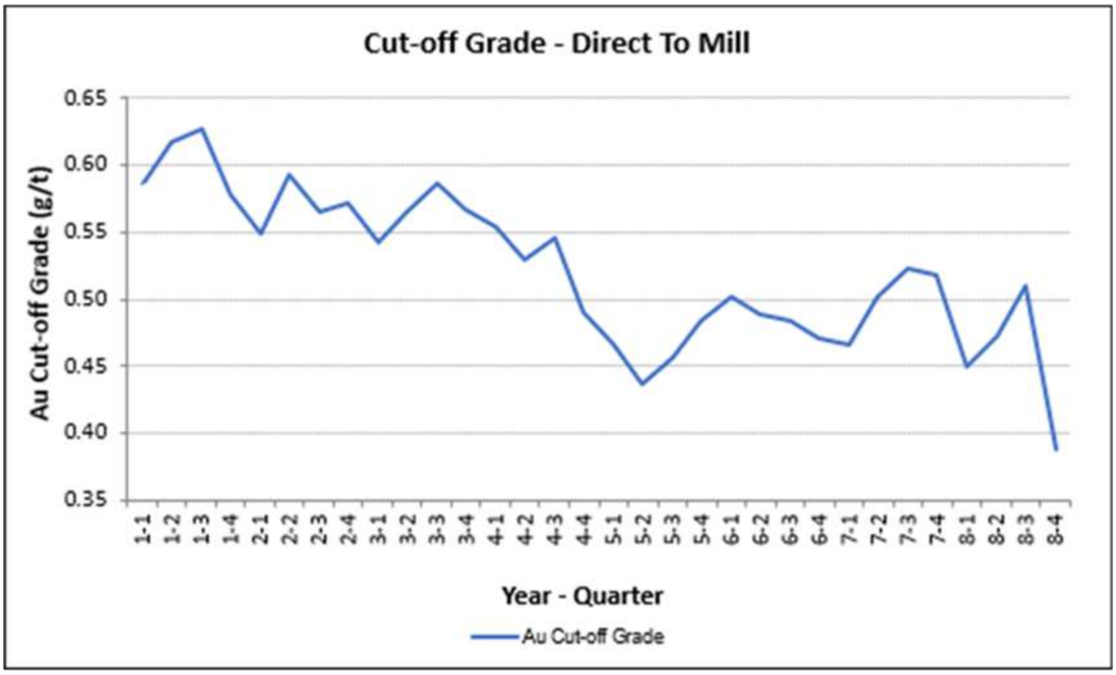
The mining sequence was performed on a quarterly basis with Prober, software developed by Whittle Consulting, based on the pushbacks defined previously. The sequence was further detailed by SRK to generate the definitive production schedule.

The following assumptions were made in undertaking the production scheduling work:

- Only blocks classified as Measured and Indicated Resources above the variable cut-off grade were scheduled and considered as ore;
- Blocks classified as Inferred or below a 0.37g/tAu cut-off grade were flagged as waste;
- A 15-month pre-stripping phase, between October 2022 and December 2023, was planned to ensure an initial supply of ore;
- Processing plant capacity of 2.5Mtpa;
  - Production start-up for January 2024.
  - Ramp-up: January 50% / February 75% / March 100%;
- Maximum annual rock movement of 20.0Mtpa;
- Maximum annual vertical advance rate per pushback: 60m;
- Maximum vertical lag between phases: 50m; and
- Mining at the north end of the pit only after the relocation of the Araras creek in Year 3 of production (2026).

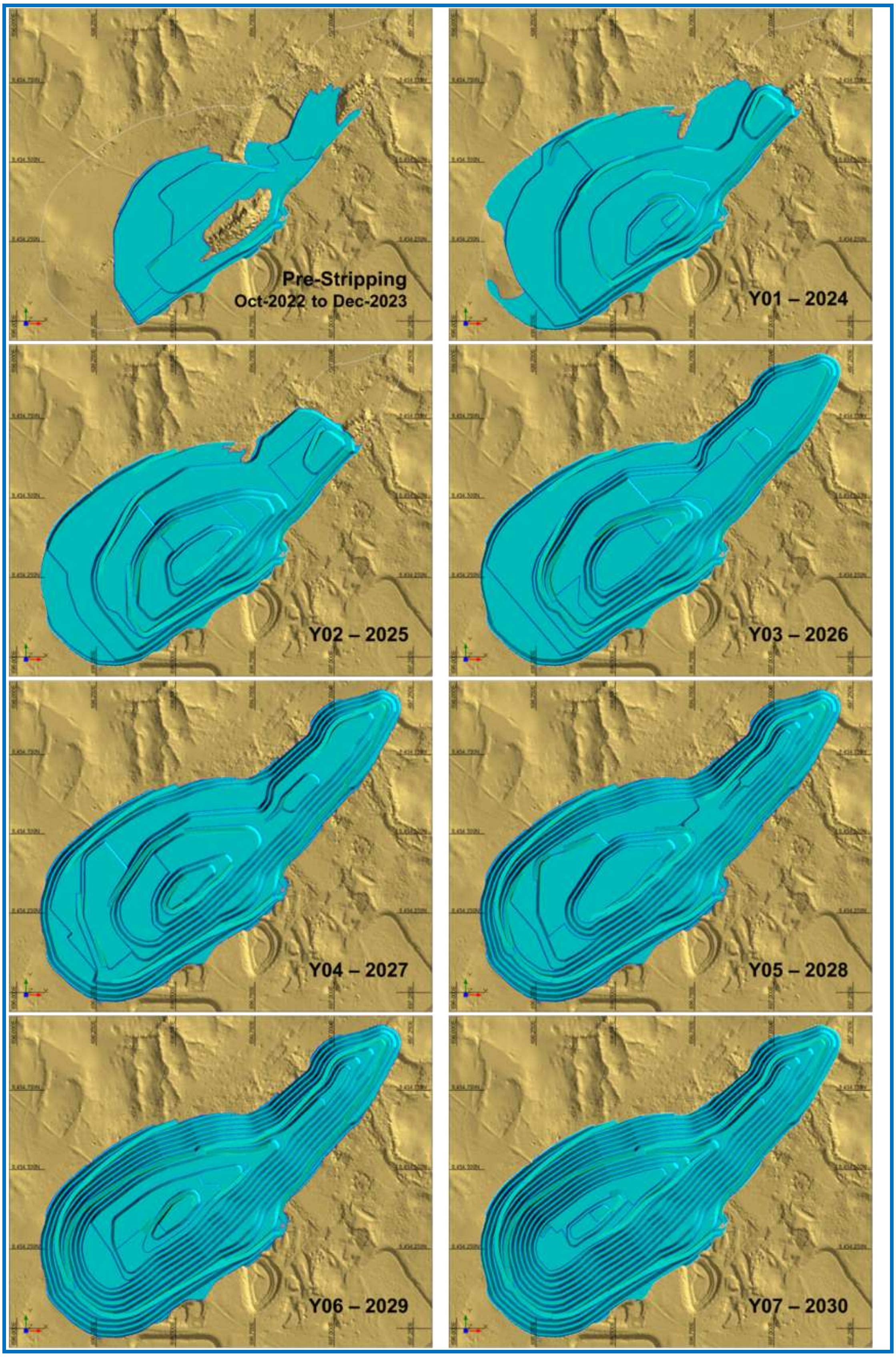
Figure 16-3 presents the variable cut-off grades throughout the LoM.

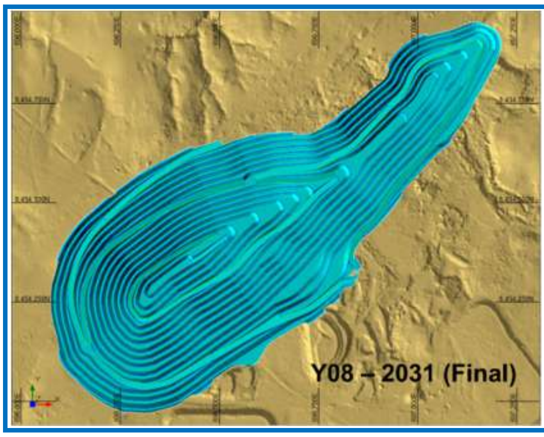
Figure 16-3: Variable cut-off grade



The end-of-period mine layouts are presented in Figure 16-5 and in Appendix A in greater detail.

Figure 16-4: End-of-Period Mine Designs





The results of the mining scheduling are shown in Figure 16-5, Table 16-2 and Table 16-3.

The dumping schedule and the average haul distances are shown in Figure 16 6. The dumping schedule graph indicates that the waste dumps (WD 1, WD 2, WD 3 and WD 4) have capacity until Year 3.5. These waste dumps have already been granted the Installation License (**“LI”**). Then, the Project will gradually add extra capacity with WD 5 and WD 6. All waste dumps are located within the existing tenement boundaries.

Figure 16-5: Mine scheduling results

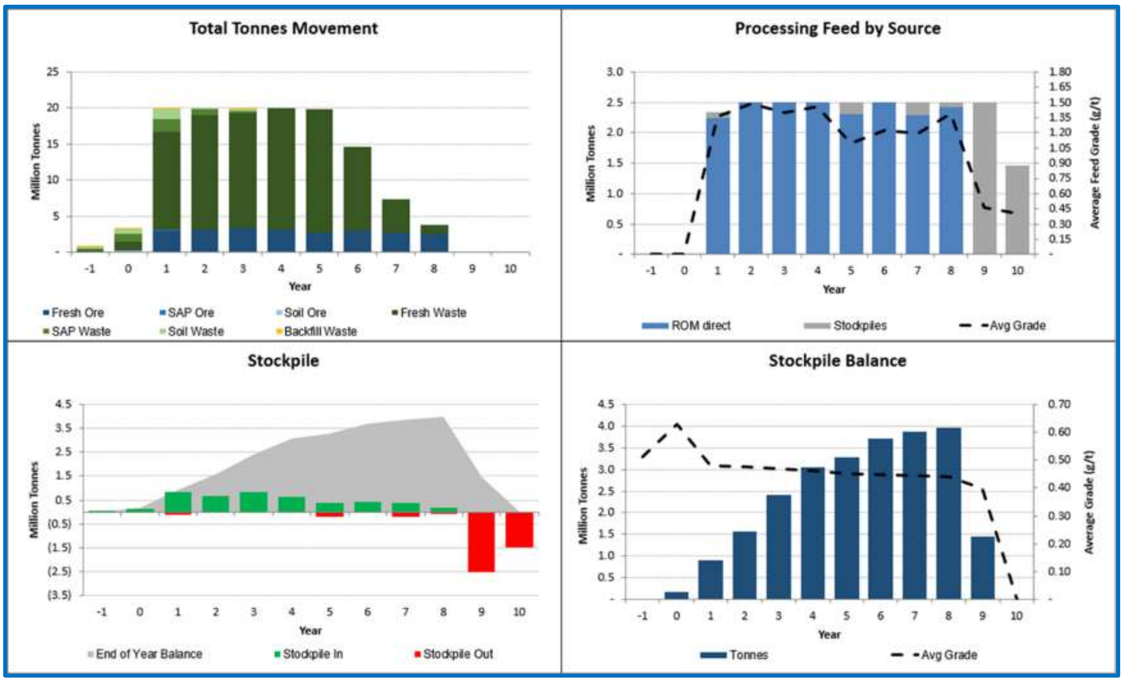


Table 16-2: Mining Physicals

Mining Physicals	Unit	LoM Totals	Year -1 2022	Year 0 2023	Year 1 2024	Year 2 2025	Year 3 2026	Year 4 2027	Year 5 2028	Year 6 2029	Year 7 2030	Year 8 2031	Year 9 2032	Year 10 2033
Pre-Production Tonnes Mined	**kt dry**	**4,103**	**718**	**3,386**	-	-	-	-	-	-	-	-	-	-
Soil Ore	kt dry	29	1	28	-	-	-	-	-	-	-	-	-	-
SAP Ore	kt dry	85	14	71	-	-	-	-	-	-	-	-	-	-
Fresh Ore	kt dry	60	5	56	-	-	-	-	-	-	-	-	-	-
Soil Waste	kt dry	852	187	665	-	-	-	-	-	-	-	-	-	-
SAP Waste	kt dry	1,466	274	1,192	-	-	-	-	-	-	-	-	-	-
Backfill Waste	kt dry	143	9	134	-	-	-	-	-	-	-	-	-	-
Fresh Waste	kt dry	1,468	227	1,240	-	-	-	-	-	-	-	-	-	-
Production Tonnes Mined	**kt dry**	**125,360**	-	-	**19,919**	**19,944**	**19,914**	**19,962**	**19,841**	**14,609**	**7,383**	**3,787**	-	-
Soil Ore	kt dry	25	-	-	6	-	19	-	-	-	-	-	-	-
SAP Ore	kt dry	80	-	-	69	-	11	-	-	-	-	-	-	-
Fresh Ore	kt dry	23,526	-	-	3,006	3,160	3,314	3,137	2,716	2,931	2,664	2,599	-	-
Soil Waste	kt dry	1,686	-	-	1,278	168	240	-	-	-	-	-	-	-
SAP Waste	kt dry	2,990	-	-	1,859	800	331	-	-	-	-	-	-	-
Backfill Waste	kt dry	185	-	-	126	-	60	-	-	-	-	-	-	-
Fresh Waste	kt dry	96,869	-	-	13,576	15,817	15,941	16,826	17,126	11,678	4,719	1,188	-	-
Total Tonnes Mined	**kt dry**	**129,464**	**718**	**3,386**	**19,919**	**19,944**	**19,914**	**19,962**	**19,841**	**14,609**	**7,383**	**3,787**	-	-

Mining Physicals	Unit	LoM Totals	Year -1 2022	Year 0 2023	Year 1 2024	Year 2 2025	Year 3 2026	Year 4 2027	Year 5 2028	Year 6 2029	Year 7 2030	Year 8 2031	Year 9 2032	Year 10 2033
Soil Ore	kt dry	54	1	28	6	-	19	-	-	-	-	-	-	-
SAP Ore	kt dry	165	14	71	69	-	11	-	-	-	-	-	-	-
Fresh Ore	kt dry	23,586	5	56	3,006	3,160	3,314	3,137	2,716	2,931	2,664	2,599	-	-
Soil Waste	kt dry	2,538	187	665	1,278	168	240	-	-	-	-	-	-	-
SAP Waste	kt dry	4,456	274	1,192	1,859	800	331	-	-	-	-	-	-	-
Backfill Waste	kt dry	329	9	134	126	-	60	-	-	-	-	-	-	-
Fresh Waste	kt dry	98,337	227	1,240	13,576	15,817	15,941	16,826	17,126	11,678	4,719	1,188	-	-
Total Tonnes Mined	**kt dry**	**129,464**	**718**	**3,386**	**19,919**	**19,944**	**19,914**	**19,962**	**19,841**	**14,609**	**7,383**	**3,787**	**-**	**-**
Ore	kt dry	23,805	20	154	3,081	3,160	3,343	3,137	2,716	2,931	2,664	2,599	-	-
Waste	kt dry	105,659	697	3,231	16,838	16,785	16,571	16,826	17,126	11,678	4,719	1,188	-	-
Strip Ratio (Waste / Ore)	**t / t**	**4.44**	**34.49**	**20.92**	**5.46**	**5.31**	**4.96**	**5.36**	**6.31**	**3.98**	**1.77**	**0.46**	**-**	**-**

Table 16-3: Ore Mining and Plant Feed

Ore Mining and Plant Feed	Unit	LoM Totals	Year -1 2022	Year 0 2023	Year 1 2024	Year 2 2025	Year 3 2026	Year 4 2027	Year 5 2028	Year 6 2029	Year 7 2030	Year 8 2031	Year 9 2032	Year 10 2033
Total Ore Mined by Res. Class	**kt dry**	**23,805**	**20**	**154**	**3,081**	**3,160**	**3,343**	**3,137**	**2,716**	**2,931**	**2,664**	**2,599**	**-**	**-**
Measured	kt dry	11,791	5	68	2,816	3,111	2,841	1,708	851	230	-	162	-	-
Indicated	kt dry	12,014	15	87	265	48	502	1,428	1,865	2,701	2,664	2,437	-	-
Total Ore Mined by Destination	**kt dry**	**23,805**	**20**	**154**	**3,081**	**3,160**	**3,343**	**3,137**	**2,716**	**2,931**	**2,664**	**2,599**	**-**	**-**
	Au g/t	**1.18**	**0.51**	**0.64**	**1.14**	**1.28**	**1.16**	**1.25**	**1.03**	**1.10**	**1.14**	**1.35**	**-**	**-**
Direct to Mill	kt dry	19,268	-	-	2,246	2,502	2,500	2,501	2,314	2,501	2,292	2,412	-	-
	Au g/t	1.35	-	-	1.39	1.49	1.40	1.46	1.14	1.22	1.25	1.42	-	-
Pit to Stockpile	kt dry	4,537	20	154	835	658	843	636	402	431	372	186	-	-
	Au g/t	0.46	0.51	0.64	0.48	0.47	0.46	0.44	0.42	0.43	0.44	0.43	-	-
Pit to Stockpile >= 0.55	kt dry	361	5	77	161	56	60	2	-	-	-	-	-	-
	Au g/t	0.63	0.69	0.84	0.58	0.57	0.56	0.55	-	-	-	-	-	-
Pit to Stockpile >= 0.43	kt dry	2,341	10	49	439	392	466	336	133	208	209	99	-	-
	Au g/t	0.48	0.47	0.48	0.49	0.49	0.49	0.48	0.45	0.46	0.47	0.46	-	-
Pit to Stockpile >= 0.37	kt dry	1,835	5	28	235	210	316	298	269	223	163	87	-	-
	Au g/t	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	-	-
Plant Feed by Source	**kt dry**	**23,805**	**-**	**-**	**2,341**	**2,502**	**2,500**	**2,501**	**2,500**	**2,501**	**2,500**	**2,501**	**2,500**	**1,460**
	Au g/t	**1.18**	**-**	**-**	**1.36**	**1.49**	**1.40**	**1.46**	**1.10**	**1.22**	**1.19**	**1.39**	**0.47**	**0.40**
RoM Direct	kt dry	19,268	-	-	2,246	2,502	2,500	2,501	2,314	2,501	2,292	2,412	-	-
	Au g/t	1.35	-	-	1.39	1.49	1.40	1.46	1.14	1.22	1.25	1.42	-	-
Stockpile to Plant	kt dry	4,537	-	-	94	-	-	-	186	-	209	88	2,500	1,460
	Au g/t	0.46	-	-	0.75	-	-	-	0.59	-	0.52	0.48	0.47	0.40
Contained Ounces	**Oz**	**902,434**	**-**	**-**	**102,364**	**119,800**	**112,738**	**117,469**	**88,463**	**98,119**	**95,866**	**111,417**	**37,455**	**18,744**
Processing Recovery	%	89.9%	-	-	90.2%	90.5%	90.3%	90.4%	89.6%	89.9%	89.9%	90.3%	85.7%	84.6%
Recovered Ounces	**Oz**	**811,023**	**-**	**-**	**92,357**	**108,424**	**101,815**	**106,203**	**79,264**	**88,225**	**86,158**	**100,635**	**32,080**	**15,864**

Figure 16-6: Waste dumping sequence and average haul distances

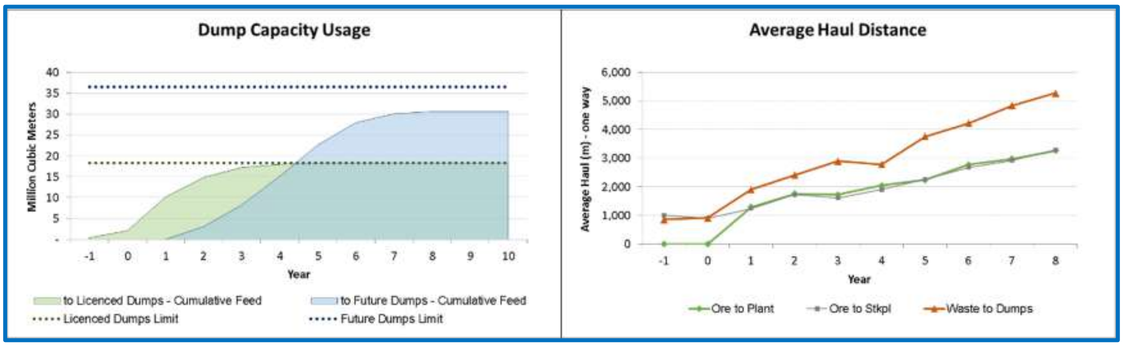


Table 16-4: Waste Sequence by Waste Dump

Waste Dumps Capacity Usage	Unit	LoM Totals	Year -1 2022	Year 0 2023	Year 1 2024	Year 2 2025	Year 3 2026	Year 4 2027	Year 5 2028	Year 6 2029	Year 7 2030	Year 8 2031	Year 9 2032	Year 10 2033
Total dumps required capacity	**m³x1000**	**48,795**	**375**	**1,712**	**8,047**	**7,713**	**7,577**	**7,629**	**7,775**	**5,295**	**2,137**	**534**	**-**	**-**
Licensed (LI) Dumps	**m³x1000**	**18,207**	**375**	**1712**	**8048**	**4673**	**2396**	**723**	**280**	**-**	**-**	**-**	**-**	**-**
Waste Dump 1	m³x1000	3,992	-	-	3,045	947	-	-	-	-	-	-	-	-
Waste Dump 2	m³x1000	6,379	-	-	3,392	2,987	-	-	-	-	-	-	-	-
Waste Dump 3	m³x1000	3,698	375	1712	1,611	-	-	-	-	-	-	-	-	-
Waste Dump 4	m³x1000	4,138	-	-	-	739	2396	723	280	-	-	-	-	-
Future dumps	**m³x1000**	**30,588**	**-**	**-**	**-**	**3040**	**5181**	**6906**	**7495**	**5295**	**2137**	**534**	**-**	**-**
Waste Dump 5	m³x1000	15,962	-	-	-	3,040	5,181	6,906	835	-	-	-	-	-
Waste Dump 6	m³x1000	14,626	-	-	-	-	-	-	6,660	5,295	2,137	534	-	-

Table 16-5: Average Haul Distances

Average Haulage Distances	Unit	LoM Totals	Year -1 2022	Year 0 2023	Year 1 2024	Year 2 2025	Year 3 2026	Year 4 2027	Year 5 2028	Year 6 2029	Year 7 2030	Year 8 2031	Year 9 2032	Year 10 2033
Direct to Mill														
Tonnes	kt dry	19,268	-	-	2,246	2,502	2,500	2,501	2,314	2,501	2,292	2,412	-	-
Volume in situ	m³x1000	6,939	-	-	814	900	902	900	832	899	824	867	-	-
Avg. Haulage Distance	m	2,256	-	-	1,287	1,748	1,727	2,043	2,237	2,773	2,970	3,256	-	-
Pit to Stockpile														
Tonnes	kt dry	4,537	20	154	835	658	843	636	402	431	372	186	-	-
Volume in situ	m³x1000	1,652	9	66	305	237	306	229	145	155	134	67	-	-
Avg. Haulage Distance	m	1,908	996	895	1,241	1,729	1,614	1,899	2,256	2,676	2,920	3,284	-	-
Pit to Waste Dumps														
Tonnes	kt dry	105,659	697	3,231	16,838	16,785	16,571	16,826	17,126	11,678	4,719	1,188	-	-

Average Haulage Distances	Unit	LoM Totals	Year -1 2022	Year 0 2023	Year 1 2024	Year 2 2025	Year 3 2026	Year 4 2027	Year 5 2028	Year 6 2029	Year 7 2030	Year 8 2031	Year 9 2032	Year 10 2033
Volume in situ	m^3x1000	39,510	304	1,386	6,516	6,246	6,135	6,177	6,296	4,287	1,730	432	-	-
Avg. Haulage Distance	m	2,966	856	907	1,900	2,404	2,894	2,782	3,757	4,218	4,837	5,272	-	-
Stockpile to Plant														
Tonnes	kt dry	4,537	-	-	94	-	-	-	186	-	209	88	2,500	1,460
Volume in situ	m^3x1000	1,652	-	-	34	-	-	-	68	-	76	32	910	532
Avg. Haulage Distance	m	750	-	-	750	-	-	-	750	-	750	750	750	750

16.4 Mining Equipment

The mining equipment selected for the Posse Gold Project consists of small hydraulic excavators and a combination of down-the-hole ("**DTH**") and top-hammer drill rigs.

Hard waste materials will be drilled by 6¾" diam. DTH rigs at 10-m high benches and at a 4.40m (Burden) x 6.50m (Spacing) pattern. A 760g/m^3 powder factor is estimated. The ore and ore/waste contact materials will be drilled by 4" diam. top-hammer drills at 5-m high benches and at a 2.50m (Burden) x 3.50m (Spacing) grid pattern. In this case, an 880g/m^3 powder factor is anticipated.

A hydraulic excavator with a back hoe configuration having an operating weight of 74t and equipped with a bucket of 4.5m^3 volume was selected to meet the required production rates and truck matching.

Haulage will be performed by 45-t heavy tipper class on-road trucks with a 25m^3 capacity. These trucks will be used for all rock movements.

Figure 16-7: Drilling Equipment: DTH for the waste (left); Top-hammer for the ore (right)

Figure 16-8: Loading Equipment: Backhoe Excavator for primary mining (left); FEL for re-handling (right)

Figure 16-9: Haulage Equipment

As part of the routine mining sequence Amarillo will complete a grade control program to monitor the mining production. Grade control will be conducted on a daily basis to delineate the orebody, obtain samples for testing, and to update and correlate the geological database to improve prediction of head grades. The samples will be collected basically from reverse circulation ("**RC**") drilling.

The drainage system will include ditches, diversions, sediment sumps, and sediment ponds equipped with pumps. Interceptor ditches will be excavated around the active mining areas to divert rain water away, and to minimize erosion and potential fines.

38-t and 22-t track dozers were selected for a variety of auxiliary applications in the mine including waste dumping, stripping activities, and roads and accesses construction.

A 20-t operating weight grader is planned for road maintenance.

Water required to wet the access and haul roads will be provided by water trucks with 30,000L capacity, keeping dust within acceptable levels.

The following ancillary equipment is also included as part of the mining fleet:

- Front-end loaders for re-handling;
- Breaker hammer;
- Backhoe loader;
- Low bed transporter truck;
- Refuelling and lube truck;
- Field maintenance truck;
- Portable lightning towers;
- Compactor;
- Tire handler; and
- Pick-ups.

A contractor fleet will be the base case for the Posse Gold Project. As part of the Posse DFS, Amarillo collected a number of quotes from selected mining contractors based on the DFS mine plan. The fleet numbers for the key mining equipment provided by contractors were estimated as follows:

- 5 hydraulic excavators;
- 1 front-end loader;
- 34 heavy tipper trucks;
- 3 DTH drills; and
- 2 top-hammer drills.

A maximum annual diesel consumption of 7,127,000 litres is estimated. Supply storage and handling of explosives will be the responsibility of a specialised outsourced company. An explosive emulsion will be pumped into the holes by a truck equipped with a 60-t tank.

16.5 Mining Labour Requirements

Managers, administrative workers and some technicians will work day shift only. Amarillo's mining workforce will include staff dedicated to the mine supervision as shown in Table 16-6.

Table 16-6: Amarillo's mining workforce

Item	Shifts	Crew	Worker/shift	Total
Mine Operation				
Mine Operation Manager	1	1	1	1
Mine Coordinator	1	1	1	1
Safety Technician	3	4	1	4
Shift Supervisor	3	4	1	4
Mine Planning				
Mine Planning Engineer	1	1	1	1
Junior Engineer	1	1	1	1
Mine Planning Technician	1	1	2	2
Geotechnical Engineer	1	1	1	1
Geotechnical Assistant	1	1	1	1
Surveyor	1	1	1	1
Surveyor Assistant	1	1	2	2
Geology				
Geologist	1	1	1	1
Junior Geologist	1	1	1	1
Grade Control Technician	3	4	1	4
Grade Control Assistant	3	4	2	8
Total				**33**

Equipment operators will work depending on the type of equipment as indicated in Table 16-7. The number of workers estimated for the mining contractors' operations are presented in Table 16-8. It is thus anticipated that the total workforce will reach 475 workers throughout the mine life.

Table 16-7: Mine equipment operation schedule

1 Shift	2 Shifts	3 Shifts
Explosive truck	Grader	Hydraulic shovel excavator
	Backhoe loader	Front-end loader
	Portable lightning tower	Off-highway truck
		Blast hole drill rigs
		Bulldozer
		Water truck
		Low bed transportation truck
		Fuel and lube truck
		Field mechanical truck
		Tire handler
		Light vehicle

Table 16-8: Contractor’s workforce

Area	Loading & Haulage	Drilling	Blasting
Operators	251	22	6
Maintenance	127	8	
Supervision and Admin.	21	5	2
Total number of Workers	**399**	**35**	**8**

17 RECOVERY METHODS

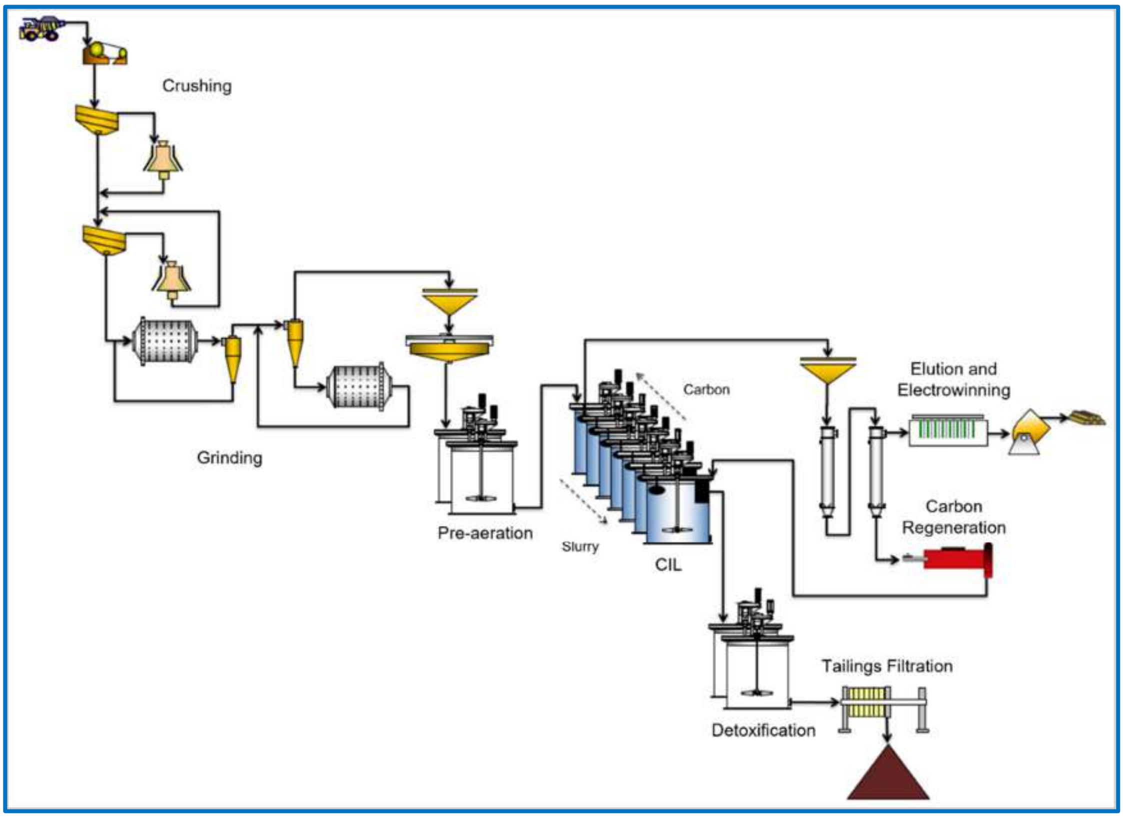
The unit operations used to achieve plant throughput and metallurgical performance are well proven in the gold/silver processing industry. The flowsheet incorporates the following major process operations:

- three-stage crushing and crushed ore stockpile;
- two stage ball mill grinding and classification;
- pre-leach thickening;
- pre-oxidation and carbon-in-leach (“**CIL**”) adsorption;
- desorption, regeneration and gold room;
- tailings detoxification, filtration and disposal;
- fresh and reclaim water supply; and
- reagents preparation and distribution.

The overall process plant flowsheet is shown in Figure 17-1. The solids throughputs for the following parts of the plant are:

- crushing plant is 6,849t/d or 439t/h at 65% availability;
- process plant is 6,849t/d or 317t/h at 90% availability; and
- tailings filtration plant is 6,849t/d or 402t/h at 85% availability.

Figure 17-1: Overall process flow diagram



17.1 Process Design Criteria

The process design criteria summary is provided in Table 17-1.

Table 17-1: Process design criteria summary

Parameter	Units	Value
Plant Throughput, Design	t/d	6,849
Ore Gold Grade, Design	g/t Au	1.51
Crushing Plant Availability	%	65
Crushing Plant Design Throughput	t/h	439
Mill Availability	%	90
Mill Design Throughput	t/h	317
Tailings Filtration Plant Availability	%	85
Tailings Filtration Plant Design Throughput	t/h	402
Primary Crusher		Jaw Crusher, 1300 mm x 1000 mm
Secondary Crusher		Cone Crusher
Tertiary Crushers		Cone Crusher
Fine Ore Stockpile Capacity	h	4.0
Primary Ball Mill Dimensions		5.5 m dia. x 8.5 m EGL
Primary Ball Mill Installed Power	MW	5.0
Secondary Ball Mill Dimensions		5.5 m x 8.5 m EGL
Secondary Ball Mill Installed Power	MW	5.0
Product Particle Size, at 80% passing (P_{80})	µm	53
Pre-leach Thickener Underflow Pulp Density	%wt solids	51
Pre-aeration + CIL Tanks	#	2 + 7
Pre-aeration Residence Time, per tank	h	4
Oxygen Uptake Pre-oxidation	kg/t	0.48
Leach Extraction	% Au	90.1
Leach pH Target		12-12.5
Oxygen Uptake Leaching	kg/t	0.48
Pre-aeration + CIL Residence Time	h	36
Desorption Process - type		Pressure Zadra
Elution Column Capacity	t carbon	6.0
Acid Wash and Elution Cycles per Day	#	1
Detoxification - Method		$O_2/SO_2/Cu^{2+}$
Detoxification Tanks	#	2
Detoxification Residence Time	h	1.0
Detoxification Discharge CN_{WAD}	mg/L	20
Tailings Specific Filtration Rate	kg/m²/h	112
Final Cake Moisture	%wt H_2O	15

17.2 Process Plant Description

17.2.1 Crushing and Stockpile

RoM ore is discharged to the RoM ore hopper by 50t dump trucks. The RoM ore hopper incorporates a horizontal static grizzly with 800mm x 800mm apertures to screen out lump oversize. The static grizzly oversize is reclaimed by a front-end loader and stockpiled for rock breaking. The static grizzly undersize discharges into the RoM ore hopper. Ore is reclaimed by a vibrating grizzly feeder with 115mm aperture.

The oversize from the vibrating grizzly enters the C130 1,300mm x 1,000mm primary jaw crusher powered by a 160kW motor set at a closed sized setting (CSS) of 125mm. The vibrating grizzly undersize and jaw crusher product combine and discharge onto the primary crusher transfer conveyor. A tramp magnet removes tramp steel from the primary crushed ore as it transfers by conveyor to the primary classification screen.

The primary classification screen is a double-deck screen, with 70mm and 12mm apertures operating in open circuit with the secondary crusher. Primary screen oversize from both decks feeds the secondary crusher.

Screen oversize from the primary classification screen feeds the secondary crusher directly. The secondary crusher is an HP500 cone crusher with 370kW motor. This crusher is in the secondary / tertiary crushing building, in the crushing area. The fine ore product from the Secondary Crusher with an 80% passing size (P80) of 33mm is conveyed to secondary screening for classification, combined with the product from the tertiary crushers and primary screen undersize. A weightometer measures the recirculating load.

This product is fed to two double screen secondary classification screens, with 25mm and 12mm apertures. These screens operate in closed circuit with the tertiary crushers, with one dedicated to each crusher. Secondary screen oversize from both decks feeds the tertiary crusher surge bins, while the undersize is transferred to the stockpile via conveyor.

The tertiary crushers are two HP500 cone crushers with CSS of 15mm and 370kW motors located in the secondary / tertiary crushing building. A surge bin with ten minutes retention time located above each crusher ensures choke feed conditions. A belt feeder extracts ore from each surge bin to feed each tertiary crusher. The fine ore product with a (P80) of 8mm is conveyed to secondary screening for classification, combined with the product from the secondary crushers and primary screen undersize as described previously.

The fine ore stockpile has a live capacity of 1,268t, providing 4 hours of mill feed. Three variable speed reclaim pan feeders provide three live draw-down pockets. The pan feeders are sized such that any single unit can handle nominal mill feed demand if the other units are unavailable.

17.2.2 Grinding and classification

Reclaimed fine ore is conveyed by the ball mill feed conveyor from the fine ore stockpile to the primary ball mill feed chute. A weightometer after the stockpile reclaim measures the amount of fresh mill feed ore entering the primary ball mill. Quick lime is metered to the ball mill feed conveyor using a rotatory valve or screw conveyor.

The grinding circuit consists of two single pinion 5.5m x 8.5m ball mills operating in series to achieve a final product with a target P80 of 53µm. Each ball mill operates in closed circuit with a hydrocyclone cluster. The primary ball mil circuit is in direct configuration where fresh feed reports to the mill directly and then goes to classification through the hydrocyclones. The secondary ball mill circuit is in reverse configuration, where fresh circuit feed reports to the classification hydrocyclone cluster first and only the underflow feeds the mill. The primary ball mill is equipped with a single 5MW low speed synchronous motor with liquid rheostat starter and will operate at 78% of critical speed. The secondary ball mill is equipped with a single 5MW low speed synchronous motor with liquid rheostat starter and a variable speed drive ("**VSD**") to operate between 60% and 80% of critical speed. Process water is added at a controlled rate into the feed chutes to achieve a nominal pulp density of 68% to 70% solids (w/w) at the mill discharge.

Grinding media (steel balls) for both mills is supplied in bags and are unloaded into ball kibbles with bag breakers. There is one dedicated kibble for each mill and each ball kibble is lifted by a 3-t davit crane (also used for cyclone maintenance) before discharging the balls into the mills.

Primary ball mill discharge product is screened by a mill trommel with 10 x 45mm slotted apertures. Trommel oversize is collected in a bin for disposal while the undersize gravitates to the primary classification cyclone feed pump box where the slurry is diluted with process water and pumped with a cyclone feed pump to the primary classification cyclone cluster. The primary cyclone cluster includes one operating and one stand-by cyclone. A density meter monitors slurry density and helps control the amount of process water added in the pump box to produce a target cyclone feed density of 57% (w/w) to achieve an 80% passing 210µm overflow product from the cyclones.

Secondary ball mill discharge is screened by a mill trommel with 10 x 45mm slotted apertures. Trommel oversize is collected in a bin for disposal while the undersize gravitates to the secondary classification cyclone feed pump box where the slurry is diluted with process water and pumped with a cyclone feed pump to the secondary classification cyclone cluster. The secondary cyclone cluster includes five operating cyclones and three stand-by cyclones. A density meter monitors slurry density and is used to control the amount of process water required to produce a target density of 55% solids (w/w) feed density to achieve an 80% passing 53µm overflow product from the cyclones.

The secondary cyclone overflow is sampled for product sizing before it gravitates to a linear trash screen with 800µm slots. Oversize material is removed, falling to a trash bin at ground level and underflow gravitates to the pre-leach thickener feed pump box. Hydrated lime slurry is added to the ground ore slurry in the pump box to raise pH to an operating range between 12.0 and 12.5 in the pre-oxidation stage.

Spillage in the grinding area is contained within a full concrete slab and bunded area. A single grinding area sump pump is located at the low point in the bunded area to return spillage to the primary hydrocyclone feed pump box.

Secondary cyclone overflow trash screen undersize is thickened from 28% to 51% solids (w/w) in a 29 m diameter high-rate pre-leach thickener to reduce slurry water content for downstream pre-oxidation and carbon-in-leach ("**CIL**"). Flocculant is metered to the thickener feedwell to aid with settling, producing a pre-leach thickener overflow clarity of <200mg/L suspended solids. Thickener overflow gravitates to the process water tank and is recycled for plant use while underflow slurry is pumped by a variable speed pump to the pre-oxidation circuit for slurry conditioning prior to leaching.

The thickener is located next to the process water tank, at an elevation lower than the grinding building and the leach area. There is a containment bund under the thickener surrounding the underflow pumps with a sump for a mobile pump to be used when required. Around the process water pumps, there is a separate bunded area, also with a sump for a mobile pump to be used when required. Both thickener and process water bunded areas are constructed with concrete adequate for containment of cyanide solutions.

17.2.3 Pre – Oxidation and CIL

Thickener underflow is pumped to the pre-oxidation tank feed box. Hydrated lime slurry is added to the feed box to increase the slurry pH as required to meet the range target of 12.0 to 12.5. The pre-oxidation/CIL circuit consists of two 1,833m^3 pre-oxidation tanks and seven

1,833m^3 CIL tanks. Total circuit live volume is 16,497m3, which allows 36 hours of residence time for a 317t/h solids feed rate at a 50% solids (w/w) feed density. The pre-oxidation tanks have 8 hours retention time and the CIL tanks have 28 hours retention time.

The pre-oxidation circuit is composed of two tanks equipped with dual impeller mechanical agitators to ensure uniform mixing of slurry and oxygen. The first pre-oxidation tank overflows in to the second tank. The circuit includes individual tank bypass valves and piping to allow single tank pre-aeration if required. Oxygen is added to the pre-aeration tanks via a slurry recirculation pump and a proprietary high shear mixer. The high shear mixers are designed to improve slurry oxygenation, allowing a dissolved oxygen ("**DO**") level of 30mg/L to be achieved. These dissolved oxygen concentrations are required to oxidize tellurium in the ore and minimize sulphide passivation that impacts gold recoveries. Dissolved oxygen is measured with probes in each of the pre-oxidation tanks and oxygen flow is controlled as required to ensure setpoint is being maintained.

Sodium cyanide solution is delivered to the second pre-oxidation tank as the primary cyanide addition point. Probes installed in the pre-oxidation tanks measure slurry pH and hydrated lime slurry is added as required to maintain a pH target of 12.0-12.5. An online leach cyanide analyser measures the free cyanide concentration and is used to control the cyanide addition to the second pre-oxidation tank. A hydrogen cyanide ("**HCN**") gas detector is used to monitor any hydrogen cyanide gas that may be generated in the pre-oxidation/CIL area.

The CIL circuit is comprised of seven adsorption tanks equipped with dual impeller mechanical agitators to ensure uniform mixing of slurry, oxygen and carbon. Oxygen is added to the CIL tanks via submerged distribution cones. As slurry flows continuously from the first to the last adsorption tank, carbon is pumped counter–currently in pre-set intervals from the last adsorption tank to the first. For operational flexibility, valves and piping allow slurry flow to bypass individual adsorption tanks if required and all carbon advance pumps are piped such that they can by-pass single adsorption tanks.

Loaded carbon is recovered from the first adsorption tank to be sent to elution while eluted/regenerated carbon is screened over a 1.2mm aperture sizing screen and is added to the adsorption circuit at the last tank. Fine carbon is transferred to the tailings pump box and are discarded. Target carbon concentration in each CIL tank is 15g/L.

The CIL tanks are arranged with the tops of each tank at the same elevation. Mechanically swept inter-stage pumping screens are used to move slurry to the next downstream tank, while retaining activated carbon. A top travelling 10t gantry crane in the pre-oxidation/CIL area is used to remove the interstage screens to a dedicated bay/stand area for maintenance and routine cleaning. A spare interstage screen is available to allow rapid screen changeover if a screen becomes plugged or damaged.

The pre-oxidation/CIL area is located outdoors to the southwest of the pre-leach thickener in a bunded area. All tanks are contained in the same bunded area with two sump pumps that return any spillage back to the pre-oxidation tank pump box.

17.2.4 Cyanide Detoxification

The cyanide detoxification circuit destroys weak acid dissociable cyanide ("**CNWAD**") to a target value of 20mg/L for disposal using the conventional O2/SO2/Cu2+ process. The cyanide detoxification circuit consists of two 454m^3 tanks flowing by gravity with total circuit retention time of 2 hours.

SO2 is supplied in the form of sodium metabisulphite dosed to the detoxification feed pump box. Copper sulphate acts as a catalyst for the reaction and is also dosed into the detoxification feed pump box. Acid generated as a by-product is neutralized with lime slurry, added to the detoxification feed pump box. Oxygen is supplied to each detoxification tank via a submerged dispersion cone. The tanks utilize high shear agitators to enhance oxygen dissolution in the slurry to meet the oxygen demand of the cyanide destruction process.

A CNWAD analyser automatically monitors slurry weak acid dissociable cyanide concentration and results are used to control detoxification reagent additions. Two-stage sampling is used to take a representative tailings sample after the slurry has been detoxified and prior to feeding the carbon safety screen.

Slurry from the detoxification tanks is gravity fed to a vibrating carbon safety screen to recover any carbon in the event of damage, wear or other issues with the final CIL interstage screen. Coarse carbon recovered from the 1.0mm square screen is collected in a bin that can be manually transferred for re-use in the CIL circuit. Tailings discharging from the carbon safety screen undersize gravitates to the tailings pump box where it is pumped to the tailings tank. From the tailings tank, the material is pumped by two pumps in series to the tailings filtration stock tank through a single pipeline.

The detox tanks, tailings pump box and pumps are all located on a higher elevation of the pre-oxidation/CIL bunded area. Spillage from the detox area reports to the pre-oxidation/CIL bund to be returned to the circuit.

17.2.5 Tailings Filtration

The tailings filtration system is comprised of three operational and one stand-by 2,000mm x 2,000mm horizontal plate and frame filter presses with a dedicated feed pump and a common stand-by pump. Each filter press has 180 chambers and are designed to achieve a target filtered tailings product of 85% solids (w/w).

The filter slurry feed line has a recirculation line back to the tailings filtration stock tank. Filtrate is collected in a tank and is pumped back to the process water tank. A separate pump box collects the filter wash water, which is pumped back to the pre-leach thickener feed by a dedicated pump.

Filtered tailings discharge onto the bottom floor level, from where they are reclaimed by front-end loaders onto dump trucks and transported to the tailings stockpile.

17.2.6 Carbon Desorption

The desorption circuit includes acid wash, elution and carbon regeneration. The circuit is fed loaded carbon from the CIL circuit. Slurry from the first CIL tank is pumped to the loaded carbon screen with the oversize discharging into the 6-t acid wash column. Screen undersize slurry gravitates back into the first CIL tank.

In the acid wash column, the loaded carbon is washed with hydrochloric acid (HCl), at a 3% (w/v) concentration that is diluted from a reagent grade concentration of (32% w/w) with an in-line mixer. After the 0.67 bed volume (BV) acid wash, the carbon is rinsed with 4 BV of water to remove residual acid. Rinse solution reports to the cyanide detoxification feed box. The loaded, washed carbon is then pumped to the elution column.

The area around the acid wash column has an individual containment bund to avoid contact between acids and spillage from potentially cyanide bearing process streams. This bund is acid resistant, and any spillage reports to the cyanide detox feed pump box via sump pump.

The elution circuit is a pressure Zadra circuit with a 6-t elution column. Elution consists of passing 2.0% w/v sodium hydroxide and 0.2% w/v sodium cyanide solution at 140°C through the column. Total duration of one elution cycle is 16 hours. The loaded solution from the elution column is pumped directly to the electrowinning cells and electrowinning tails return to the strip solution tank to be pumped to the elution column. After an elution cycle is complete, 3 BV of rinse water is pumped through the solution column before transferring the eluted carbon to the regeneration stage.

The area around the elution column has an individual containment bund to avoid contact between acids and spillage from potentially cyanide bearing process streams. Spillage reports to the cyanide detox feed pump box via sump pump.

17.2.7 Carbon Regeneration

After the elution rinse is complete, spent carbon is pumped to the carbon dewatering screen. Screen oversize is fed into the carbon regeneration kiln feed hopper. Dewatering screen undersize gravitates to the detox tank feed box. The carbon is then fed into the carbon regeneration kiln via a screw feeder. This propane fired kiln is a horizontal, rotary unit designed to regenerate 100% of the spent carbon and is designed for 70% utilization.

The kiln operates at a temperatures setpoint between 700°C to 750°C and the carbon is maintained at this temperature for 15 minutes to allow reactivation to occur. Pre-drying of carbon occurs in the feed end of the kiln and produces volatile organic compound gases through the regeneration process in the kiln. The gases are vented to atmosphere via the carbon regeneration kiln exhaust fan after being filtered. Regenerated carbon discharges from the kiln to a quench tank for cooling. The quench tank has make-up water added to it to keep the kiln discharge submerged and prevent carbon oxidation.

When required, fresh carbon can be added to the quench tank to make-up for losses of fine carbon due to attrition. From the carbon quench tank regenerated carbon and any make-up fresh carbon is pumped to the carbon sizing screen. The sizing screen oversize returns carbon to the last CIL tank in the train, while the quench water and fine carbon from the undersize are sent to the tailings pump box.

17.2.8 Gold Room

The gold room is used for electrowinning and smelting. Electrowinning is part of the Carbon Desorption circuit, as explained in Section 17.2.6.

When the elution cycle is completed, gold and silver sludge is removed from the two electrowinning cells using high-pressure washing to the sludge filter feed tank, to feed a plate and frame filter. Filtrate is recirculated back to the feed tank and sludge filter cake is dried in a drying oven to be prepared for the smelting process.

Sludge is mixed with flux reagents in a mixer and fed to the propane smelting furnace. The fluxes react with any impurities present to form a low viscosity, free flowing slag whilst gold and silver remain as molten metals. The furnace is then tipped to pour the melt into the moulds. The slag is separated and the cleaned, cooled doré is weighed, stamped, sampled and placed in the vault. Doré is securely shipped off site for final refining and sale.

17.2.9 Reagents and Consumables

Reagents are mixed in a separate building to the southwest end of the process plant. Separate bunded areas control any spillage. Tank storage capacity has been generally sized based on reagent consumption rates to supply the process without any interruption, or according to available delivery volumes.

Dry reagent storage will be housed in the separate building, to the west of the reagent mixing building. Reagents are transported by forklift to the reagent mixing area. Sodium cyanide dry storage is separate from the warehouse of the other reagents.

Reagent consumptions are based on project specific test work or industry operating practice.

A summary of the estimated reagents and steel media consumption rates are shown in Table 17-2.

Table 17-2: Estimated reagent and grinding media consumption

Reagents	Form	Unit	Specific Consumption
Quicklime (design)	Solid, pebbled or granulated	kg/t feed	6.5
Flocculant	Granular powder	g/t feed	20
Activated Carbon	Solid, granular, coconut	g/t feed	50
Leach Sodium Cyanide (design)	Solid, briquettes	kg/t feed	0.47
Leach Lime	Solid (fine powder)	kg/t feed	4.5
Pre-oxidation/Leach Oxygen	Gas	kg/t feed	0.63
Hydrochloric Acid for Acid Wash	Liquid (32% w/w), solution	m³/strip	0.7
Sodium Hydroxide for Elution	Liquid (20% w/w), solution	m³/elution	1.0
Sodium Cyanide for Elution	Liquid (20% w/w), solution	m³/elution	0.6
Detox SMBS	Solid, powder	kg/t feed	1.6
Detox Copper Sulphate	Crystalline granules solid	kg/t feed	0.09
Detox Oxygen	Gas	kg/t feed	0.85
Detox Lime	Solid	kg/t feed	1.5
Borax	Powder	kg/100kg concentrate	60
Silica	Powder	kg/100kg concentrate	30
Sodium Nitrate (Nitre)	Powder	kg/100kg concentrate	5.0
Sodium Carbonate	Powder	kg/100kg concentrate	5.0
Grinding Media	40-75 mm balls	kg/t feed	1.82

17.2.10 Services

Raw Water

Raw water is captured from the Rio do Ouro, feeding the raw water reservoir. From the reservoir, two raw water pumps (one duty, one stand-by) send the water to the raw water tank and the fire water tank. Fire water pumps are connected to the fire water tank and provide fire suppression water to the network in the plant.

Raw water is fed to the elution water tank by gravity. It is also used as process water make-up, feeding the process water tank by gravity. Three gland water pumps (two duty, one stand-by) supply water from the raw water tank to the final tailings pumps.

Two dust suppression pumps (one duty, one stand-by) pump water to the plant crushing area dust suppression system. Two service water pumps (one duty, one stand-by) are installed to provide water to the service points along the plant.

Process Water

The plant main process water source is the pre-leach thickener overflow. Other sources are filtrate from the tailings filtration process and drainage from the tailings dry stack decant reservoir. Process water can also be supplied from the raw water tank.

Three (two duty, one standby) process water pumps supply process water to the various consumers throughout the plant site, with the main consumer being the grinding circuit. The process water tank is constructed from mild steel and has a live volume ensuring 120 minutes of residence time.

Gland Water

Two (one duty, one standby) gland water pumps are fed from the process water tank. These pumps supply the gland water to the various slurry pumps throughout the plant site, except the tailings pumps, which have a dedicated gland water system described in the Raw Water section.

Oxygen

Oxygen will be supplied to the two pre-oxidation tanks, the seven leach tanks and the two detoxification tanks by a vendor-supplied pressure swing adsorption ("**PSA**") oxygen plant. There are two off-takes for oxygen supply; one to the pre-oxidation/CIL area and another to the detoxification circuit. The oxygen is reticulated at a pressure of 550kPag.

Power

The process plant is calculated to have installed power of 11.4MW, with a nominal operating demand of 11.6MW. G&A Services are expected to require an additional 0.3MW, with a nominal demand of 0.2MW. A breakdown of power consumption by area is shown below in Table 17-3.

Table 17-3: Estimated power consumption

Area	Installed (kW)	Nominal Demand (kW)
Crushing	1,600	1,295
Process Plant	12,529	10,148
Low Voltage Distribution	102	87
Raw Water	121	102
Total	14,352	11,632
Warehouse/Maintenance Workshops	128	108
Laboratory	64	54
Admin Buildings	96	81
Total	288	243

17.3 Process Plant Layout

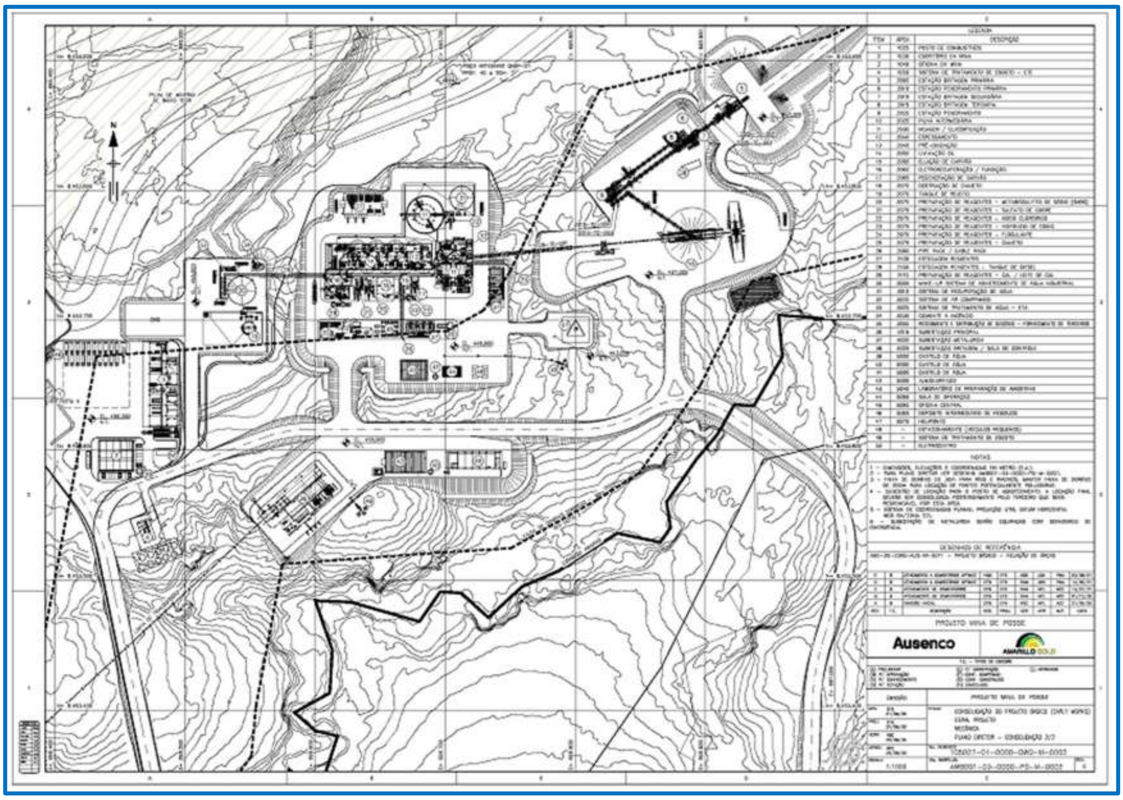
The process plant layout includes a separate area for crushing, which begins at the RoM pad and contains conveyors and buildings for the three stages of crushing and two stages of classification. The crusher area substation is between the crushing and classification buildings.

With exception of the pre-leach thickener and process water tank plateau and the reagents plateau, the mill is on one plateau, at 446masl. This area also includes the mill substation and laboratory.

The plant general arrangement is shown in Figure 17-2 and Figure 17-3.

Figure 17-2: Project layout

Figure 17-3: General arrangement



17.4 Major Process Equipment

The major process equipment is shown in Table 17-4.

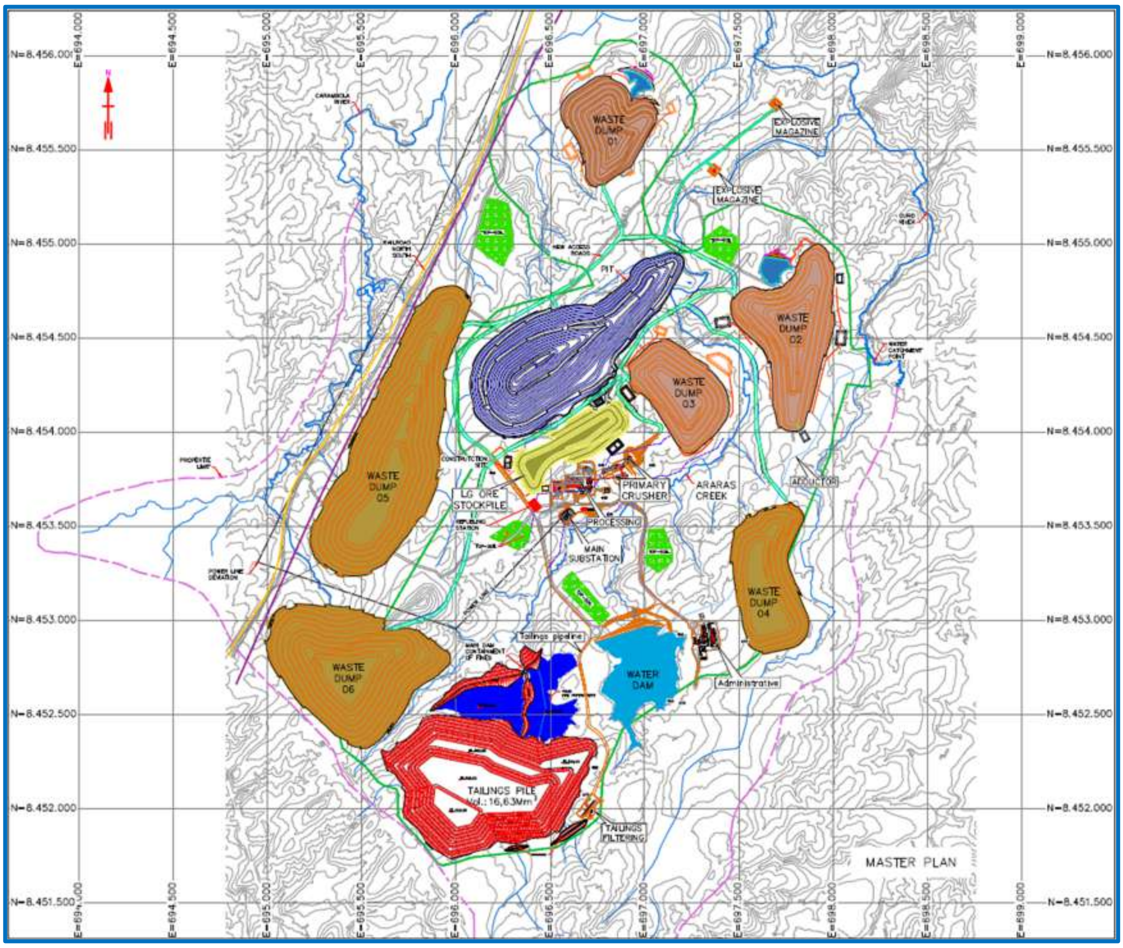
Table 17-4: Main process equipment

Area	Item	Quantity	Description
Primary crushing	Primary jaw crusher	1	C130, 1,300mm x 1,000mm, 160kW motor, CSS 125mm
Primary screening	Primary screen	1	Double-deck, banana type, 1.8m x 4.9m, 70mm and 12mm apertures
Secondary crushing	Secondary cone crusher	3	HP500, 370kW, CSS 35mm
Secondary screening	Secondary screen	2	Double-deck, banana type, 2.4m x 6.1m, 25mm and 12mm apertures
Tertiary crushing	Tertiary cone crusher	2	HP500, 370kW, CSS 15mm
Grinding	Primary ball mill	1	5.5m diameter x 8.5m EGL, 5MW
Grinding	Primary cyclone cluster	1	2 - 33" hydrocyclones (1 operating / 1 stand-by)
Grinding	Secondary ball mill	1	5.5m diameter x 8.5m EGL, 5MW
Grinding	Secondary cyclone cluster	1	8 - 20" hydrocyclones (5 operating/3 stand-by)
CIL	Pre-oxidation tank	2	1,833m^3 live volume with 90kW agitator
CIL	CIL tank	7	1,833m^3 live volume with 90kW agitator, intertank screen and carbon advance pumps
Elution	Acid wash and pressure Zadra system	1	6t batch type circuit, with one acid and one elution column and two electrowinning cells
Regeneration	Regeneration kiln	1	6t/day propane regeneration kiln
Detoxification	Detox tank	2	545m^3 live volume with 11kW agitator
Tailings filtration	Tailings filter presses	4	2,000mm x 2,000mm plate and frame filter presses with 180 chambers

18 PROJECT INFRASTRUCTURE

Figure 18-1 shows the mine site layout including all key infrastructures. The following sections describe each component of the mine infrastructure.

Figure 18-1: Mine Site Layout



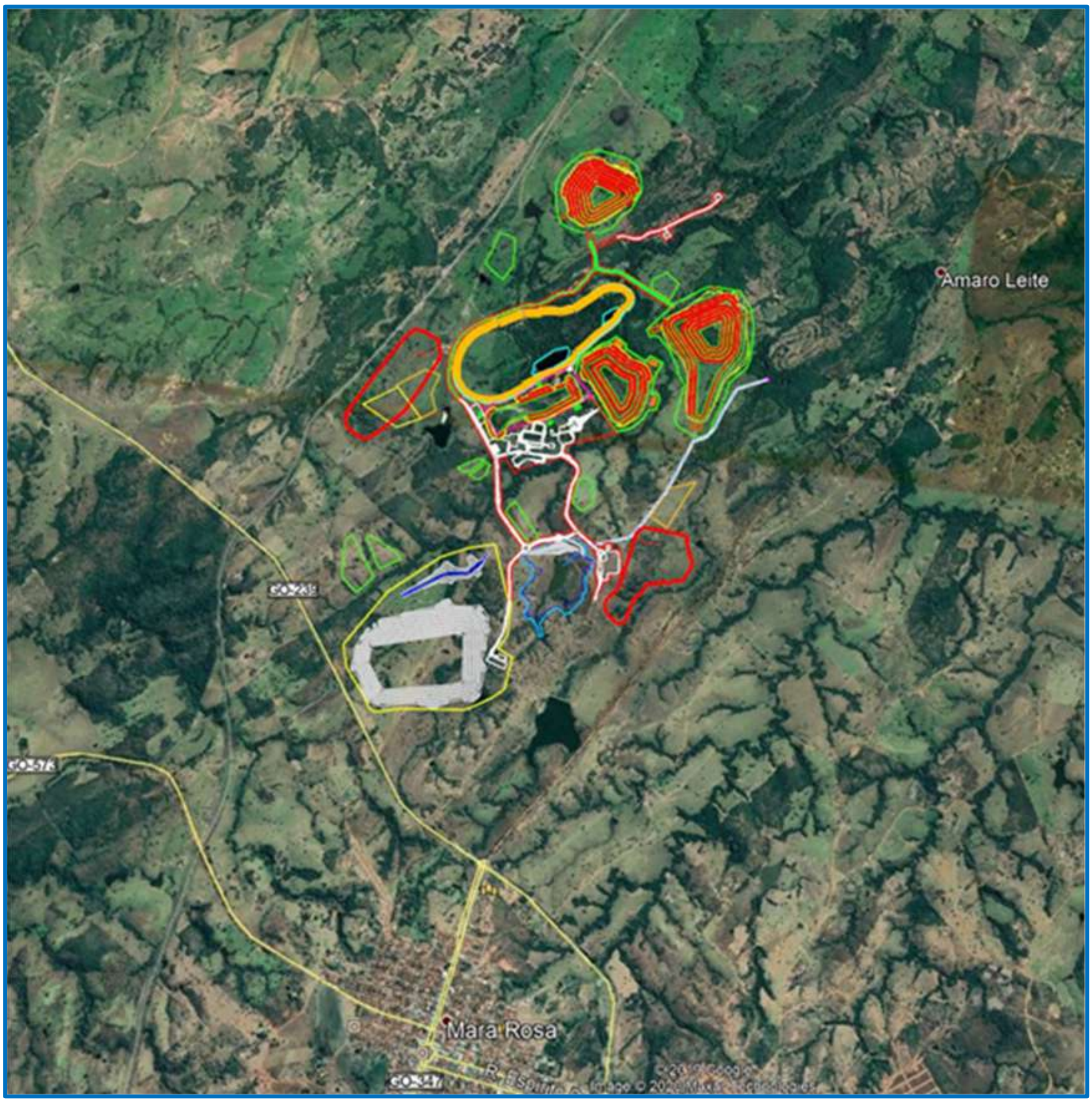
18.1 Site Selection and Site Earthworks

The Posse Gold Project is located approximately 6km north of the municipality of Mara Rosa, which is in the north of the state of Goiás, central Brazil. Figure 18 2 shows the approximate location of the process plant, associated administration offices and workshops.

The factors considered for layout and site selection are listed below:

- The Process Plant will be located more than 200m distance from the existing watercourse due to environmental requirements;
- Minimizing distance from mine to crushing and process plant;
- Topography favourable to reduce earthworks;
- Minimization of the impact of clearing vegetation;
- Separate heavy mine traffic from non-mining light vehicle traffic; and
- Utilize existing roads to reach the site and connect the various development sites.

Figure 18-2: Project Location



18.2 General Access and On-Site Roads

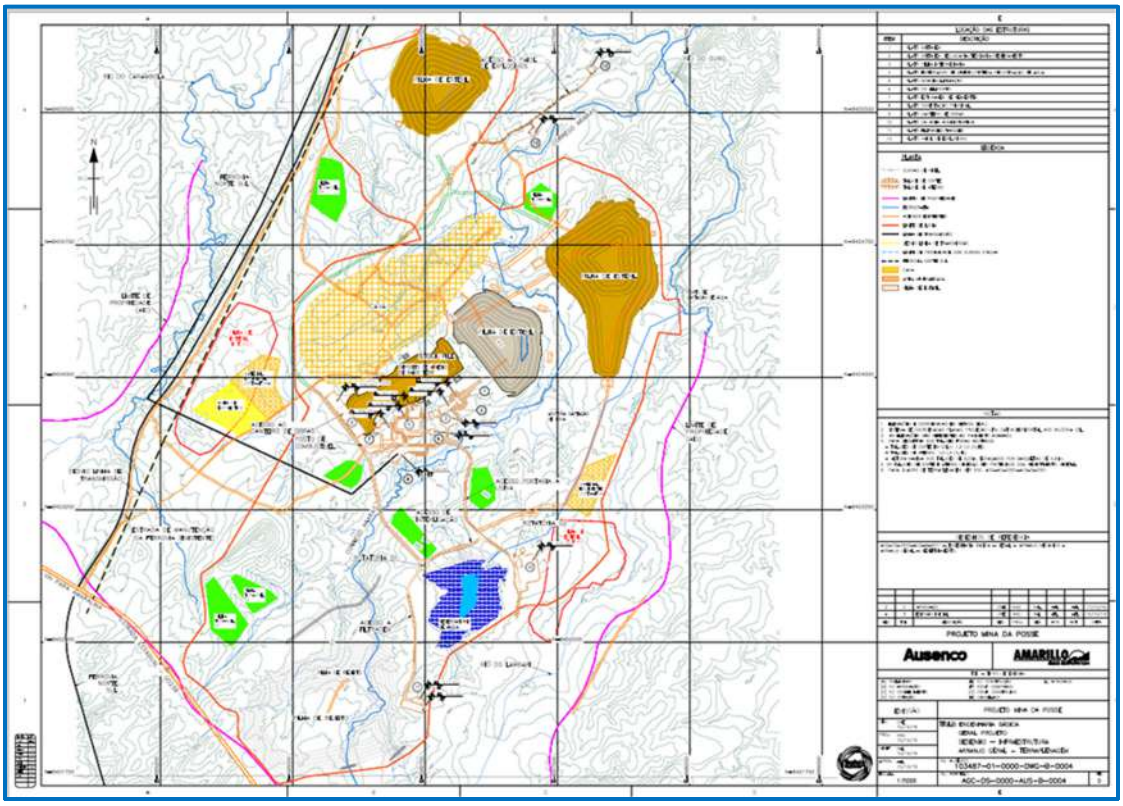
18.2.1 Public Roads

GO-239 is the public road that connects the town of Mara Rosa and the Posse mine. Figure 18 2 shows GO 239 road.

18.2.2 Access and Service Roads

The access and most of service roads are existing roads, minimizing earthworks and clearing of vegetation. Figure 18-3 shows the access and service roads for the Posse Gold Project.

Figure 18-3: Access and service roads



18.3 Power Supply and Distribution

18.3.1 Electrical Power Source

The power will be received through a 138kV transmission line with a simple circuit, which will supply the main substation. Figure 18 3 shows the transmission line that will be constructed covering 67km between Porangatu and the mine site. The main transformer will lower the receiving voltage from 138kV to 13.8kV which is the site distribution voltage.

The plant will have three secondary substations: Metallurgical process circuit, crushing circuit and Tailings Filtration circuit.

18.3.2 Power Demand Estimates

The estimated demand for feeding the plant is 21.56MVA that represents 97% of the loading power. Table 18-1 shows the estimated demand for the main and secondary substations.

Table 18-1: Estimated demand

		Demand (kVA)	Power Factor
Main Substation		21,556	0.97
Secondary Substations	Metallurgy	15,576	0.87
	Crushing	2,668	0.83
	Tailings Filtration	2,344	0.80

18.3.3 Main Substation

The main substation will be located south west to the process plant and will consist of an external concrete pad for high voltage equipment and a control room, comprising the electrical panel room and the cable room. The main transformer will lower the receiving voltage from 138kV to 13.8kV as noted.

The following systems will integrate the main substation:

- Grounding;
- Protection against lightning strikes;

- Limiting the ground fault current;
- Reliable voltage;
- Light and emergency lighting;
- Supervision, control and protection;
- Detection, alarm and firefighting with a clean suppression agent;
- Access control;
- Air conditioning and pressurization; and
- Safety signs and escape routes.

18.3.4 Site Power Distribution

The distribution on the plant between the main substation and the secondary substations at 13.8kV, will be overhead lines with protected or isolated circuits depending on the need due to the location of their installation.

18.3.5 Secondary Substations

The plant will have three secondary substations:

- **Process Plant:** provides power for the metallurgical process: grinding and classification, leaching, elution, carbon regeneration, electrodeposition, reagents, detox;
- **Crushing:** provides powering for the crushing plant; and
- **Tailings Filtration:** provides power for the tailings filtration plant.

The transformers installed in the secondary substations will have a step-down ratio of 13.8kV to 0.48kV in the case of supplying process and utility loads (well pumps, welding sockets, maintenance hoists, overhead cranes, etc.) or 480-127V in the case of powering the lighting system and general-purpose sockets, both in the industrial and administrative areas of the plant.

The power transformers will be designed in order to always respect the maximum limit of 2.5MVA, aiming not to increase the short-circuit power that passes through them.

18.3.6 Stand-by/Emergency Power Supply

Emergency power supply by diesel generator is provided for essential loads.

18.3.7 Grounding System and Protection Against Lightning Strikes

The main substation will have a grid to control gradients and equalize potential interconnected to the plant's industrial grounding grid. The medium voltage panels of the secondary substations will have surge arresters at the entrance and when there is no panel, in the cases where the transformer will be mounted directly at the pole, the surge arresters will be installed on the pole itself. The low voltage auxiliary power panels will have surge suppressors at the entrance.

The neutral of transformers whose low voltage side is 13.8kV or 0.48kV will be grounded by resistive impedance limiting the phase-to-ground fault current to 100A-10s and 3A-continuous respectively and will not be distributed. The neutral of transformers whose low voltage side is 220/127V will be solidly grounded and distributed. All equipment, gates, handrails, structures and other fixed metal parts subject to energization, even if accidental, must be grounded or equalized.

Where required by Brazilian standards, there will be a protection system against lightning strikes, including capture, descent and flow / grounding systems.

18.4 Plant Control System

18.4.1 Process Control System

The automation system will promote an efficient and safe control of the plant, reducing the decision-making time in order to promote an increase in quality and productivity. The process control system will be designed for a continuous operation regime and will have a stand-by power supply. The automation system will have an open architecture, which allows for integration with third party systems (mill, water treatment plant, filter, control panel for diesel pump, compressor control panel, etc.).

18.4.2 Field Instruments and Valves

Field instruments and valves will be wired either to the Process Control System ("**PLC**") or to the specialty PLC's.

18.4.3 Closed-Circuit Television

Process video cameras will be installed in twenty-three locations to assist the operator's view of the process. These cameras are viewed on a separate display in the main control room and in the security control room. The CCTV system will be unique and should share the same management and control platform with the property security system and the operational process, thus, it will have full integration as a single system.

18.5 Communication System

An integrated cabling system will be used, supported by hybrid network architecture, that is, optical fibre for the backbone and copper twisted pair cable for the local network. The single-mode fibre-optic cable will be used for low-loss and high-bandwidth optical systems. The fibre optic cable must be suitable and capable of operating with good performance and continuous operation in an industrial environment with a high level of suspended ore dust, high humidity and noisy areas.

18.6 Compressed Air System

The compressed air system at site is comprised of two 700kPa rotary screw compressors with integrated drier and filter (one duty, one standby) and distribution network for the various service and instrumentation points.

18.7 Buildings

18.7.1 Process Plant

The process plant at the Posse Gold Project will be separated into 6 buildings located west to the crushing building and south to the low-grade stockpile. These buildings are primary crushing, secondary and tertiary crushing, grinding, leaching, thickener and reagents.

The primary crushing building will be a 12.72m (long) x 5.75m (wide) x 12.4m (high) steel construction with cross-sectional frames to increase the stability. The building will house a jaw crusher.

The secondary and tertiary crushing complex will be divided in two structures. Secondary crushing will be in a 14.96m (long) x 4.46m (wide) x 14m (high) steel building, while tertiary crushing in a 9.7m (long) x 5.96m (wide) x 18m (high) steel structure. The building will house, apart from the three cone crushers (one secondary and two tertiaries), conveyors, screeners, feeders and silos.

The ore thickener tower will be a steel structure with dimensions of 4.13mØ x 8m (high). The grinding building has dimensions of 29.82m (long) x 40.34m (wide) x 16.35m (high). The building will be a steel structure and will house the two ball mills, feeders and cyclones.

The leaching building will be an 81.7m (long) x 27m (wide) x 21.95m (high) with 8 levels of platforms. The steel structured building will house agitators, samplers, pumps and a crane.

The reagents complex will include two warehouses and two process buildings. The warehouses size will be 16m (wide) x 24m (long) x 5.6m (high) and 16m (wide) x 15m (long) x 5.6m (high) and will be made of steel. The first process building dimensions will be 5.5m (long) x 12.2m (high) with two intermediate platforms, while the second building will be 18m (wide) x 5.5m (long) x 13.4m (high) with five intermediate platforms.

18.7.2 Industrial Support Buildings

Figure 18 4 shows the industrial area, where the Process Plant and Industrial Support Buildings are located.

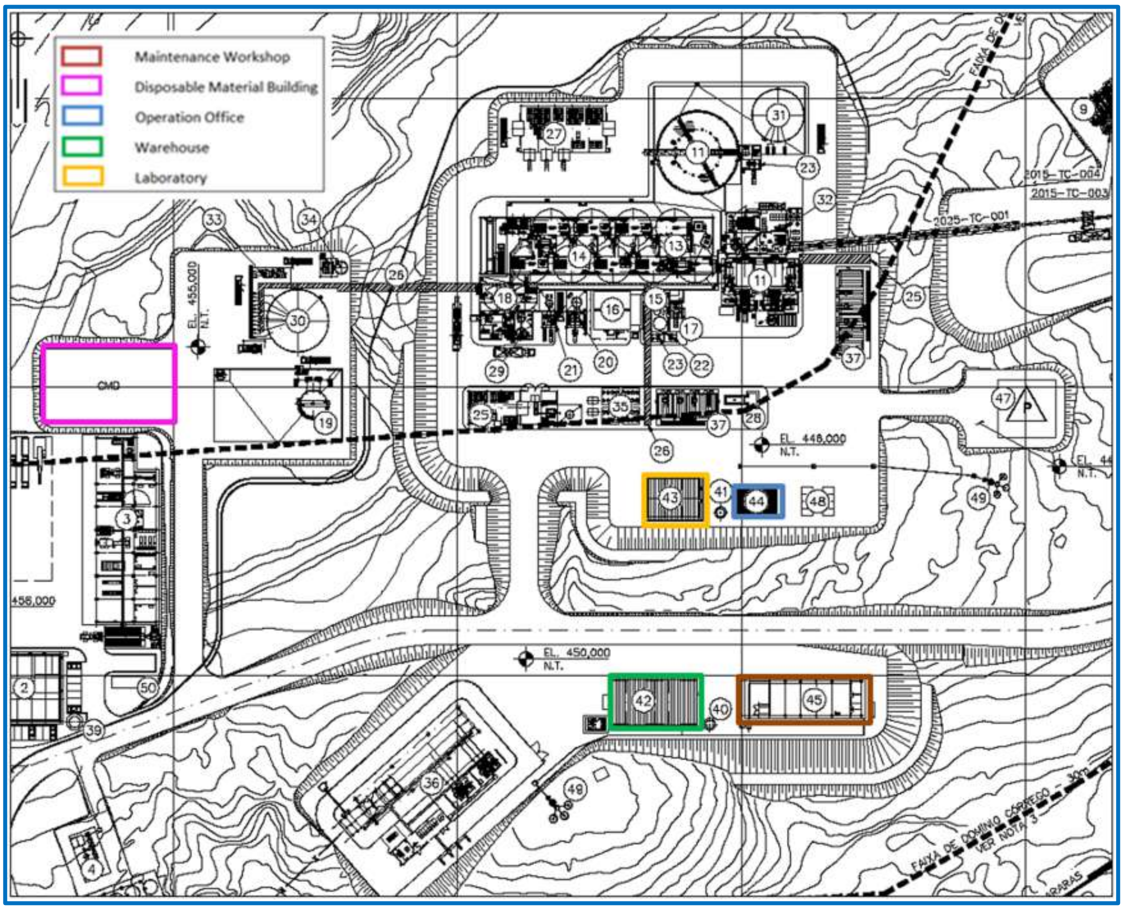
Maintenance Workshop

The Maintenance Workshop will be located south of the process plant in the industrial area next to the Warehouse. The masonry building will include a 36.1m (long) x 12.0m (wide) construction with 4 heavy equipment bays and one space for light vehicles. Next to the building is considered a roofless wash building of 12m (long) x 8m (wide). On the west side of the shop, a 12m (long) x 3.8m (wide) administrative office will be built. This two-store office will house a tool shop, washrooms, meeting room and one office for the supervisor.

Disposable Material Building

The Disposable Material Building will be located west of the process plant. The building is a construction for storage of disposable material that will consist of an open fenced yard and a shed for the storage of hazardous waste. The structure of 9m (long) x 5m (wide) will have masonry walls and a metallic roof. The shed will have the capacity to store two buckets with materials contaminated with oil, grease and paint, boxes for lamps and drums for electronic scraps, batteries. In the yard, metals, plastics, glass, rubber, wood and non-recyclable materials will be stored.

Figure 18-4: Industrial support buildings



Operations Office

The Crushing Operation Office will be located south of the process plant in the industrial area. The Crushing operation office will be a 13.5m (long) x 7.5m (wide) masonry building which will house offices, a meeting room, storage space, hall and washrooms.

Warehouse

The Warehouse will be a fabric structure of 30.0m (long) x 15.0m (wide) easily to assembly at site, with a ceiling height of 6m at its lowest point. The Warehouse will be located south of the process plant in the industrial area next to the Maintenance Workshop. There will be wind fans on the roof and a high-strength concrete floor. A prefabricated 6m (long) x 3.55m (wide) wooden building will be the warehouse support, comprising an office with 2 workstations, storage space and washrooms.

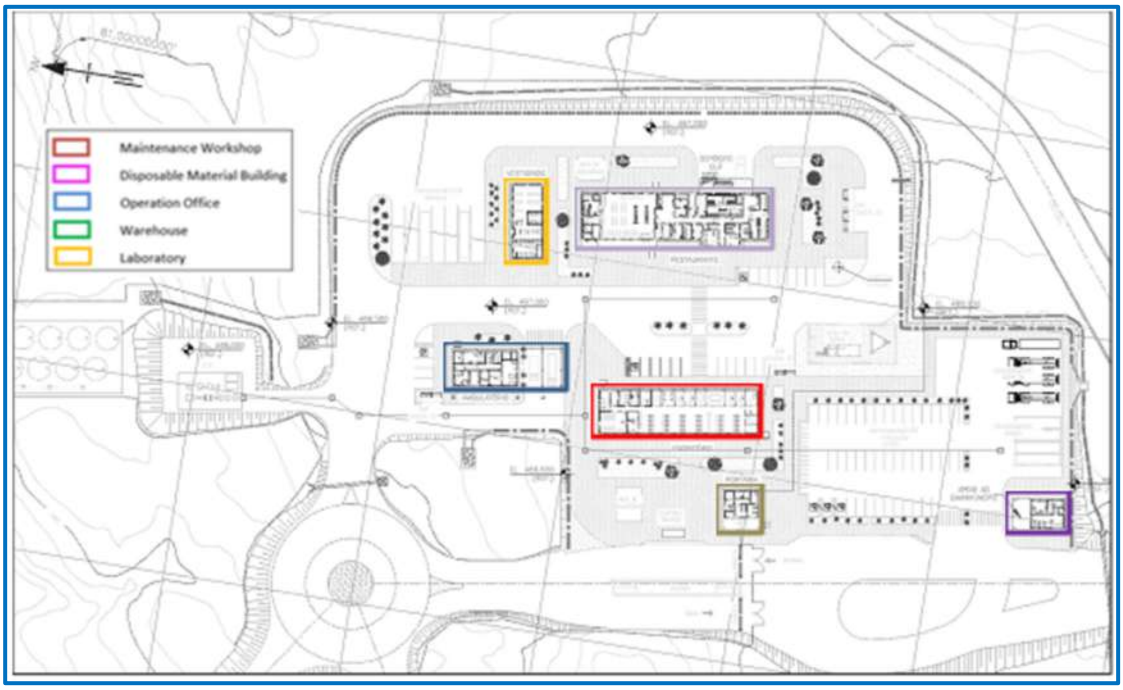
Laboratory

The Laboratory will be a 28m (long) x 15m (wide) prefabricated wooden construction, with space for different test rooms, offices, storage space, waste disposal and washrooms.

18.7.3 Administration Buildings

Figure 18 5 shows the administration area which is located south to the Process Plant.

Figure 18-5: Administration buildings



Gate House and Security Building

The Gate House and Security building will be located in the administration area, southeast of the process plant. The gate house building will be a prefabricated 11.6m (long) x 13m (wide) wooden building, that includes a reception with a training room for 8 people, weight control room, property security office, storage space and two toilets.

Truck Drivers Building

The Truck Drivers support building will be a prefabricated 12.9m (long) x 7.4m (wide) wooden construction. In its layout considers a balcony, an office room with one workstation, storage space, and washrooms.

Administration Office

The Administration Office will be located in the administration area, southeast of the process plant. It will be a 44.6m (long) x 14m (wide) prefabricated wooden and single-story building. The building will house offices, meeting rooms, workstations, washrooms and storage space.

First Aid and Fire Department

The First Aid and Fire Department will be located southeast of the process plant in the administrative area next to the Administration Office. It will be a 20.15m (long) x 11.75m (wide) prefabricated wooden and single store building. The construction will house a reception, nursery, pharmacy, offices and a fire brigade tool room with external access.

On the south side of the building will be space to park an ambulance and a fire truck.

Locker Room Building

The Locker Room building will be located southeast of the process plant in the administrative area next to the Cafeteria. The Locker Room will be a 10.82m (long) x 22m (wide) prefabricated wooden building, which will have changing rooms and washrooms.

Canteen

The Cafeteria will be a 103.6m (long) x 19.86m (wide) masonry building located southeast of the process plant in the administrative area in front of the Administration Office.

18.8 Water Systems

18.8.1 Fresh Water Supply and Distribution

A channel will discharge water from the Rio do Ouro to a well, where the fresh water supply will be pumped via two vertical centrifugal pumps (one duty, one standby) to a water reservoir through a 200mm HDPE pipeline. Two floating centrifugal pumps (one duty, one standby) will drain the water reservoir and pump the fresh water to a tank close to the process plant reservoir through a 180mm HDPE pipeline. In the tank, two pumps (one duty, one standby) will supply fresh water to various consumers.

18.8.2 Potable Water System

Potable water will be supplied to the process plant by treating filtered fresh water. A vendor-supplied water treatment plant will generate potable water. Physicochemical processes will be used to remove inorganic pollutants, heavy metals, oils and greases, colour, sediments, suspended solids, non-biodegradable organic materials and dissolved solids. Two pumps (one duty, one standby) will supply potable water to the leaching and reagent plants and to two tanks which they will provide water to various consumers.

Potable water will be supplied to the administration area by a water truck which will fill daily a tank near the site.

18.8.3 Reclaim Water System

The main source of process water will be reclaimed from tailings filtration and stored in the tailing filtration tank. Three (two duty, one standby) centrifugal pumps will be used to supply multiple consumers through a 16'' pipeline. Two pumps (one duty, one standby) will supply blend water to various consumers through a 4'' pipeline.

18.8.4 Sewage Collection

The sewage from the administration area will be collected via a network of buried PVC piping and conveyed by gravity to a sewage treatment plant located north the administration area.

The collected sewage from the industrial area will conveyed by buried PVC pipeline to a biodigester treatment plant located south of the process plant, between the main substation and warehouse buildings.

18.9 Fire Protection

A network of firefight hydrants will be placed in the industrial area considering a radius of 30m, which corresponds to the length of the hose, not counting the range of the water jet. The hydrants will be placed in free access points, preferably close to the streets and 15 metres away from the external wall of the buildings.

Three pumps will supply water to the fire hydrants network from a tank. The main pump and the jockey pump will have electric engines while the third one will be a diesel pump. The fire water tank will have a reserve capacity of 60m^3 of water, which correspond to 30min of capacity.

The main pump must be designed to meet the flow rate of 2 open hydrants. The jockey pump will be designed to maintain pressure in the line. The diesel engine pump will have similar characteristics as the main pump and will be used in case of a failure.

Automated fire detection will be installed in the industrial and administration buildings. All the buildings will have hand held fire extinguishers, emergency exits, emergency lights and signalling.

18.10 Accommodation

No accommodation on site is considered as the plant is located 6km north of the town of Mara Rosa.

18.11 Filtered Tailings Pile

Amarillo envisages the installation of a storage facility for tailings originated from a gold ore processing plant which will accommodate 23.8Mt (dry tonnes). The filtered tailings pile design was developed by GHT Engenharia ("**GHT**").

Geotechnical laboratory tests were performed on tailings samples. The maximum dry density measured through a Normal Proctor Test, as defined by Brazilian Standards NB-7182, was 1.52t/m^3. Using an efficiency of 95% during the pile construction process a calculated final density of 1.44t/m^3 was used as design criteria. Hence, to accommodate 23.8Mt, a minimum capacity of 16.53Mm^3 is required. The tailings disposal will be developed over a period of 11 years of the Project's estimated life. The disposal will be carried out in a staggered manner.

An executive project was developed for the first stage of the pile which will last for approximately 2 years. For the second stage, a design at a PFS level was created, supported by specific laboratory testing, to accommodate the total required volume for the LoM (16.53Mm^3). Table 18-2 shows the cumulated filtered tailings production per year.

Table 18-2: Tailings volume requirements

Period (Year)	Accumulated Volume (Mm³)
1	0.76
2	2.50
3	4.24
4	5.97
5	7.71
6	9.45
7	11.18
8	11.92
9	14.66
10	16.39
11	16.53

As the material from the processing is in the form of slurry, a filter pressing process will be implemented to remove the excess of water. Then, the tailings will be transported via on-road trucks to a storage pile. The pile will be equipped with a waterproofed foundation using a HDPE geomembrane to prevent the spillage of contaminant components contained in the tailings.

The pile will be constructed in a valley. Any upstream springs will be diverted through a specific drainage system underneath the pile through ARMCO type drainage pipes so that the water can be drained without being contaminated and released to the environment downstream. Contention dams (Dykes 01 and 02) upstream of the pile will be installed to enable this deviation.

The HDPE lined downstream reservoir is designed to allow sedimentation and prevent contact with the natural soil of the terrain. The water accumulated in this reservoir will be continually pumped and reused in the processing plant. This reservoir will be located between the tailings pile and main dyke as shown in Figure 18-6 and Figure 18-7.

To reduce initial capital costs, the main dyke showed in Figure 18-6, will be configured so as to enable the division of the total area, destined for the final pile, into two valleys. The valley closure located to the east allows formation of an initial smaller reservoir.

The stream diversion system for rainwater on the upstream sub-basins 01 and 02 consist of the installation of ARMCO type pipes with a diameter ranging between 1.5m and 2.3m, for a total extension of approximately 3km. The pipes will be installed after the conclusion of vegetation clearing and installed underneath the geomembrane. The diversion pipes will be installed under the tailings pile, under the reservoir, and will follow downstream of the main dike finally discharging into the Corrego de Araras creek.

Due to project constraints, waterproofing with HDPE is not limited to the pile foundation area. Both the bottom of the reservoir and the sides of the dikes are to be also waterproofed, ensuring total separation of the contaminated water from the environment.

By the end of the life of the mine, the slopes of the pile will be enveloped with a layer of clay, followed by mine closure procedures of revegetation.

Figure 18-6: Filtered Tailings Pile Area for approximately 2 years of operation

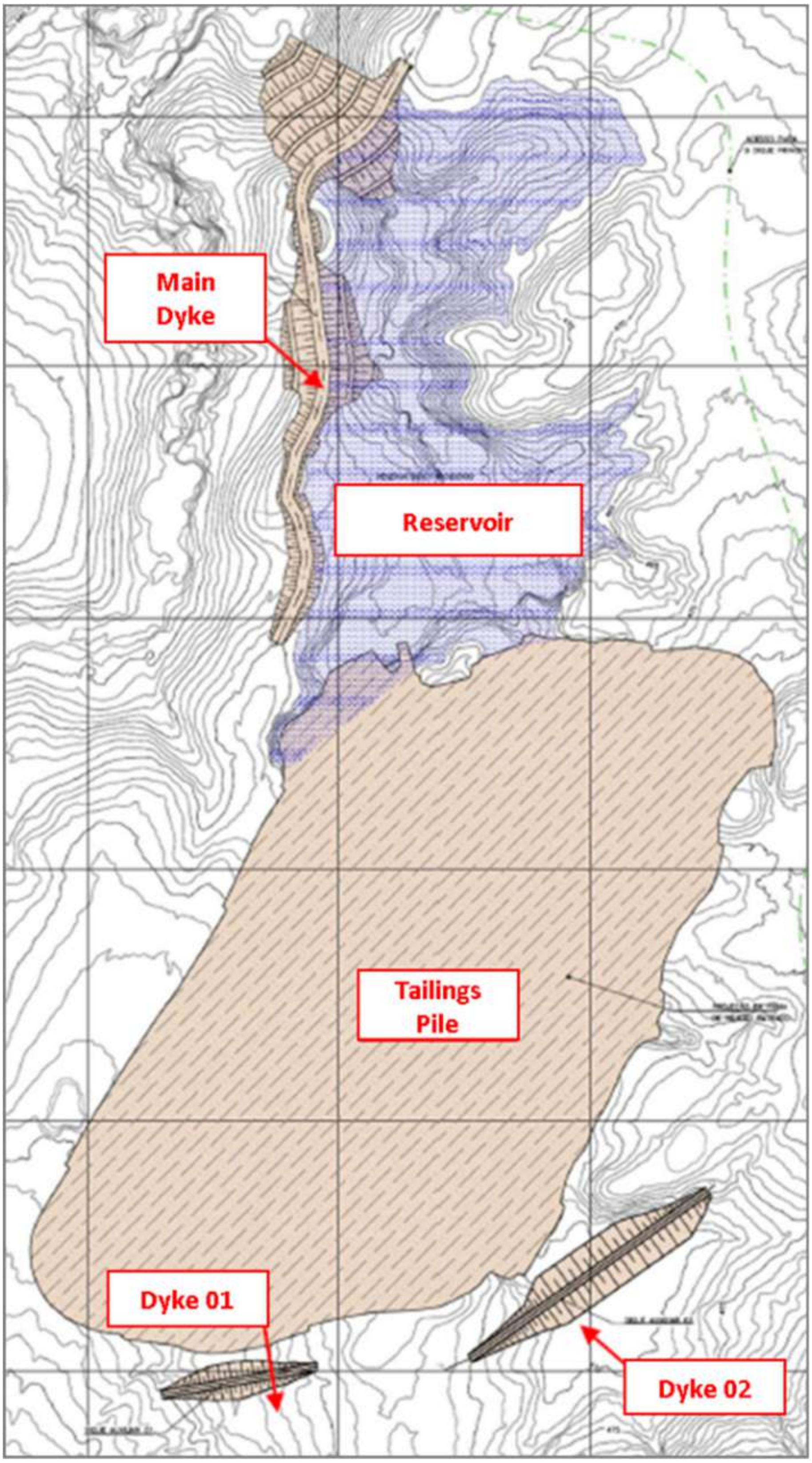


Figure 18-7: Filtered Tailings Pile Final Area

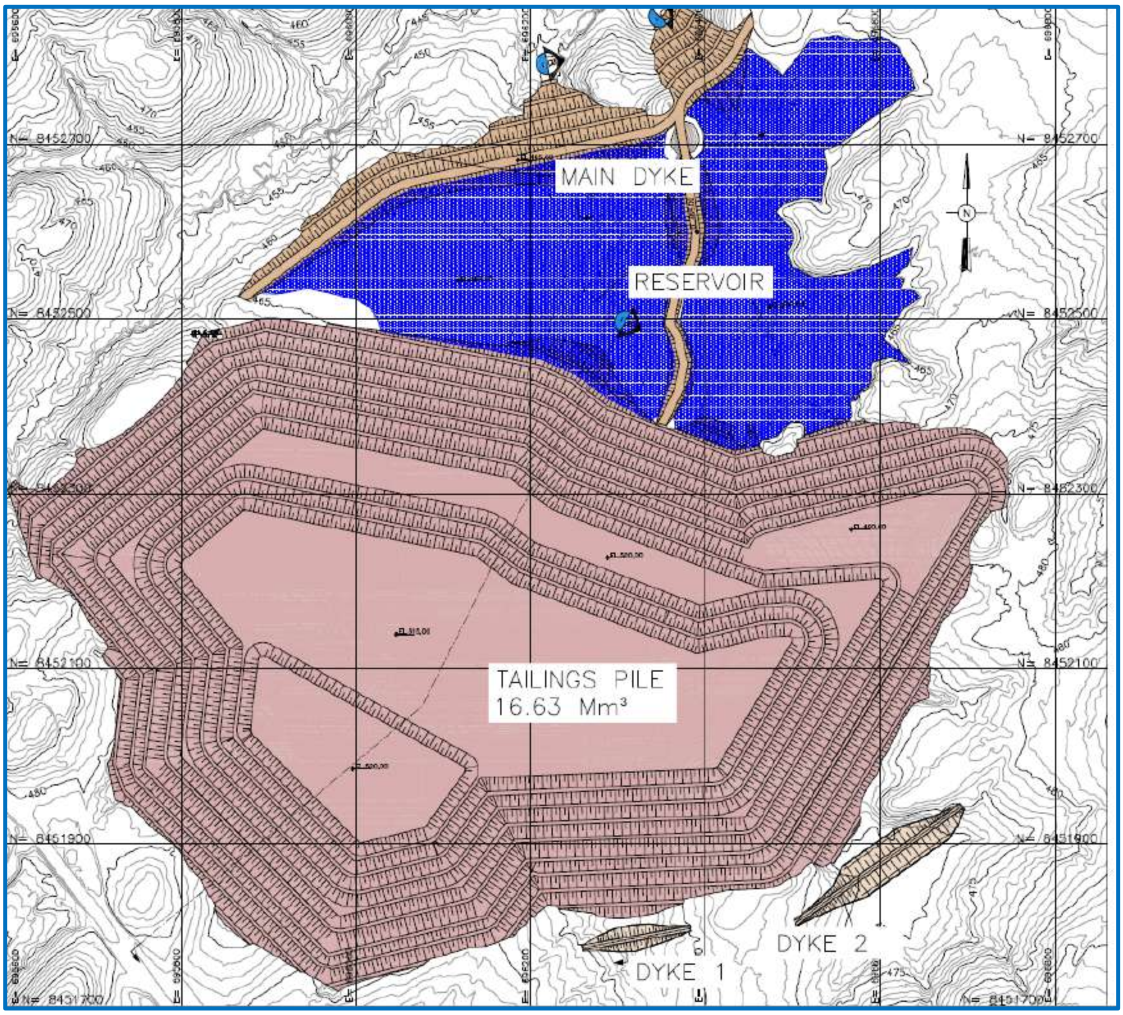


Figure 18-8: Vertical Cross Section of the Filtered Tailings Pile – Reservoir Slope

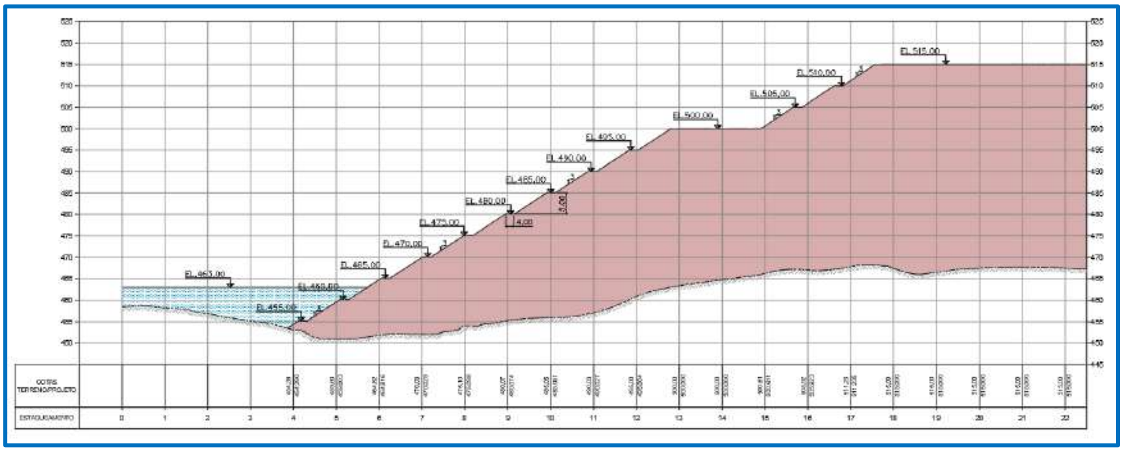
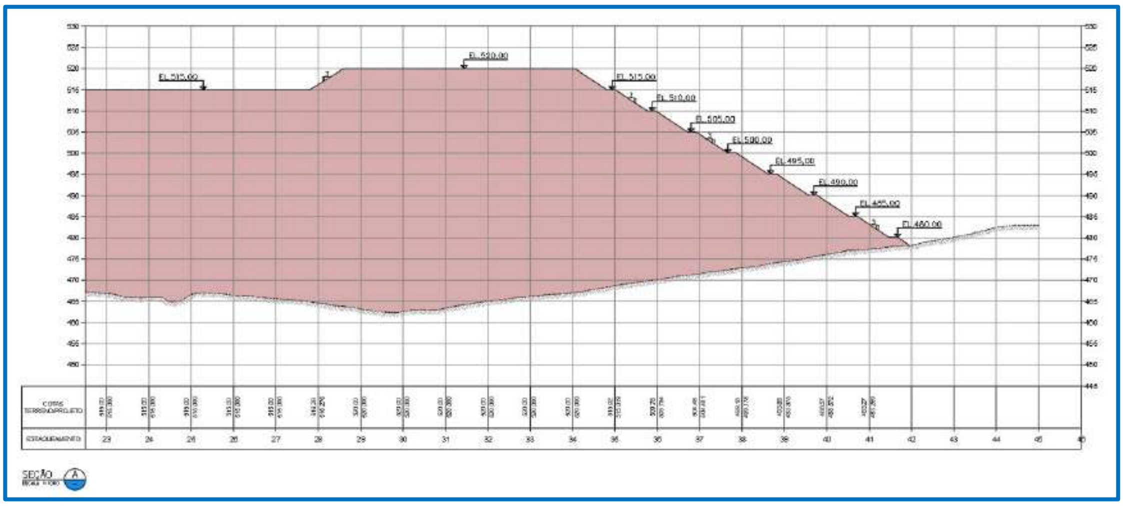


Figure 18-9: Vertical Cross Section of the Filtered Tailings Pile – Dikes 1



18.11.1 Tailings Pile Design Parameters

The geometry of the tailings pile and dikes were determined according to Brazilian Standard #13029 ("**ABNT**") - Deposition of Waste Pile, and the stability safety criteria established by ICOLD.

The geometry of the tailings pile for the LOM and the first two years are shown in Table 18-3 and Table 18-4, respectively.

Table 18-3: LoM tailings pile

Parameter	Unit	Pile	Dyke 1	Dyke 2	Dyke 3	Main Dyke
Total Height	m	58	3	3	3	17
Maximum Elevation	m	510	475	475	465	459
Bench Height	m	5	-	-	-	5
Berm Width	m	3	-	-	-	3
Slope Angle	V:H	1V:3H	1V:3H	1V:3H	1V:3H	1V:3H

Table 18-4: Tailings pile – stage 1 and stage 2

Parameter	Unit	Pile	Pile	Dyke 1	Dyke 2	Main Dyke
		1st phase	2nd phase			
Maximum elevation	m	480	490	475 m	475 m	459 m
Bench height	m	5	5	-	-	5 m
Berm Width	m	3	3	-	-	4 m
Slope Angle	V:H	1V:3H	1V:3H	1V:3H	1V:3H	1V:3H

18.11.2 Hydrological Criteria – Hydraulics

To characterize the basin, the area was delimited, and key aspects of the cover were determined. With these data, a number of physiographic indexes of interest in the basin were calculated, such as the slope and concentration time, among others. The length of each thalweg was obtained from the topography presented. In sub-basin 01, the thalweg is 680m length and with a 45m height difference. In sub-basin 02, the thalweg is 520m length and with a 43m height difference.

During the construction of the pile, there will be an additional thalweg to these two. This will represent the maximum contribution area between the pile's second and fifth operation year. At this stage, the thalweg will be 357m in length with a 14m height difference.

From the data extracted from monthly a number of evaluations were performed out of climatological precipitation based on the nearest meteorological station, located in Pirenópolis 200km south of Mara Rosa (Table 18-5), that meets the minimum requirements to be used in this study (30 years of readings). The average of each of the conditions was considered. The volume of water drained each year in the evolution of the filtered tailings was subsequently determined.

Table 18-5: Rainfalls data in Pirenópolis, Goiás

Period Month	Inferior Tertile mm	Median mm	Upper Tertile mm
January	236	260	318
February	196	266	258
March	200	219	277
April	122	165	180
May	16	23	31
June	0	0	2
July	0	0	0
August	0	0	11
September	9	37	55
October	80	146	170
November	234	265	286
December	246	293	325

The surface run-off coefficient is defined as the ratio between the volume of water drained superficially and the volume of precipitated water, (rain). This coefficient can vary substantially (rain falling on very dry ground will behave differently compared with rain falling on ground saturated with previous rainfalls).

For the drainage area in question, there are two types of coverage. For natural thalweg, a run-off coefficient (C) of 0.35 was adopted, since the cover corresponds to a natural field with low vegetation. For the drainage of the filtered tailings pile, the run-off coefficient adopted was 0.40, which corresponds to the waste dumps.

Using this data and values for the annual precipitation the volume drained in each stage of development of the filtered tailings pile was subsequently determined for the first two stages, as shown in Table 18-6 and Table 18-7.

The volume to be stored in the second stage is lower than the first stage due to the fact that in the second stage the area exposed of geomembrane, that is, the area without the pile cover is larger. Therefore, as the geomembrane has a zero-water absorption capacity, the volume is higher in the first stage.

Assuming that at the beginning of the rainy season (October) the reservoir will be empty, it is estimated that it will be completely full in approximately 12 months in case of a dry year or 8 months in case of a rainy year. Control of this volume will be by redirection of water back to the plant.

Table 18-6: Run-off volume – stage 1

Surface	Superficial Run-off Coefficient (C)	Area of Influence	Rainfall Volume (m^3)		
#	#	m²	Dry Year	Average Year	Rainy Year
Pile	0.55	162,810	119,803	146,264	171,328
Geomembrane	1	185,901	248,717	303,650	355,684
Natural Terrain	0.4	88,849	47,548	58,050	67,998
Total			**416,068**	**507,965**	**595,010**

Table 18-7: Run-off volume – stage 2

Surface	Superficial Run-off Coefficient (C)	Area of Influence	Rainfall Volume (m^3)		
#	#	m²	Dry Year	Average Year	Rainy Year
Pile	0.55	205,545	151,249	184,655	216,298
Geomembrane	1	143,166	191,542	233,848	273,920
Natural Terrain	0.4	88,849	47,548	58,050	67,998
Total			**390,340**	**476,553**	**558,216**

18.11.3 Drainage

The purpose of the diversion of the water falling on areas upstream of the filtered tailings pile is to transport this water to the Corrego de Araras stream (downstream from the filtered tailings pile facility) preventing the water from being contaminated by the tailings. This will be undertaken by conducting the water through a circular gallery composed of ARMCO type tubes. These galleries will start upstream of the auxiliary dikes and will run under the tailings pile, provisional reservoir and main dike.

Regarding the installation of ARMCO tubes, due to the irregular uneven topography, the tubes were positioned forming higher angles than the maximum deflection angle allowed in the installation. Thus, in some points it will be necessary to use reinforced concrete passage boxes to allow the necessary deflections.

The project to size surface drainage consists of developing hydrological and hydraulic studies of the research area to design drainage devices to obtain a system capable of collecting and conducting run-off from the areas of precipitation to the appropriate destinations.

The drainage devices include excavated channels, prefabricated sections, concrete collection boxes, concrete stepped drainage ladders and also the use of PVC tubes to final channel water to the temporary reservoir.

18.11.4 Geotechnical Studies and Stability Analysis

21 percussion drill holes were performed within the future foundation boundaries of the pile and dikes. The tests were performed by GEONORTE Geotecnia e Fundações company, in October 2019.

Disturbed and undisturbed samples were collected for laboratory tests, such as:

- Soil and tailing characterization tests;
- Compression tests; and
- Triaxial stress test UUsat and CIUsat.

In general, the results show spatial variability of the material. The local soil is filite clay of silty residue, unsaturated, with predominantly medium to hard consistency. The percussion impenetrable limit is reached between 4 to 15 meters in depth. In a few drill holes, a soft consistency was observed but limited to shallow depths.

In some drill holes it was possible to observe layers of silt clay with rock fragments. The thickness was around 1.5m of hard to very hard consistency. In the west side of the pile some drill holes showed fine sands with rock fragments with 5m of thickness and, predominantly, medium to hard consistencies.

The soil that will be used in the construction of the dikes is essentially the same material found in the surface layers of the pile foundation of clay-silt soil.

No water was found in the drill holes.

The geotechnical parameters adopted to size the structures design were based on results of field and laboratory tests along with benchmarks. For laboratory tests, moisture content was considered, defined by h = Mw/(Ms+Mw) = 19%. The moisture content of the tailings should be equal to or less than 19%. The parameters adopted are presented in Table 18-8.

Table 18-8: Run-off volume – stage 2

Parameter	Foundation	Dikes	Tailings	Rockfill	Geomembrane
Specific Gravity(kN/m³)	19.0	19.0	14.4	19	10.0
Undrained Strength	-	-	160	-	-
Cohesion	45	10	0	0	0
Friction Angle (°)	28	28	28	40	16

Slope stability analyses were performed in order to determine the slopes of the tailings pile and dikes, ensuring appropriate safety factors (“**SF**”) for the short-term (final construction) and long-term (operation) at its final total capacity, as indicated in Table 18-9.

Table 18-9: Run-off volume – stage 2

Item	End of Construction	Operation
Tailings Pile Reservoir Slope	2.11	1.51
Dike 1 Upstream	1.72	-
Dike 1 Downstream	1.63	2.57
Dike 2 Upstream	1.58	-
Dike 2 Downstream	1.61	2.63
Main Dike Upstream	1.56	-
Main Dike Downstream	1.38	2.54

The safety factors of pile and dike slopes are based on factors recommended by ICOLD –

International Commission on Large Dams and the final construction condition: SF ≥ 1.3 and operation condition: SF ≥ 1.5, all safety factors obtained are complies with these standards.

18.11.5 Geotechnical Instrumentation

The instrumentation and auscultation systems to be deployed in the pile area should be incorporated in two steps in order to ensure monitoring for approximately 2 years of operation. The system consists of Stage 1 and Stage 2 comprising:

- Stage 1:
  - 8 (eight) surface marks for monitoring settling and displacements distributed in the tailings pile. These land marks must be installed at specific locations indicated in the executive design.
  - 3 (three) reference marks for monitoring settling and displacements to be installed outside the pile. These marks must be installed at un-displaced locations or have the reading corrected from an un-displaced reference mark installed in the works area. These reference marks will be used as support for the reading of the surface marks.
  - 8 (eight) electric piezometers will be distributed at specific points in the pile area. The depths established for each piezometer are differentiated, especially where there is expected to be higher concentration of pore pressures; and
- Stage 2 where further instrumentation is added in the second stage to complete the following total of instrumentation:
  - 15 (fifteen) surface marks for monitoring settling and displacements distributed in the tailings pile. These marks should be installed at specific locations which are usually on the berms of the slopes.
  - 3 (three) reference marks for monitoring settling and displacements to be installed outside the pile. These marks must be installed in un-displaced locations or have the reading corrected from an un-displaced mark installed in the works area. These marks will be used as support for the reading of the surface marks.
  - 15 (fifteen) electric piezometers to be distributed at specific points in the pile area. The depths established for each piezometer are differentiated and especially where greater concentration of pore pressures is expected.

18.12 Water Dam

In order to provide water for the processing plant, a water dam will be constructed with a crest limited to 466m elevation. It designed for a storage capacity of 701,700m^3 at 464m elevation considering a freeboard of 2m. The dam will be built on compacted landfill using construction material from borrow areas where geotechnical investigations are being conducted. The upstream slope of the upper section should receive a rip-rap layer in order to avoid erosions arising from reservoir waves.

18.12.1 Design Parameters

The main characteristics of the dam are shown in Table 18-10.

Table 18-10: Water Dam Design Parameters

Parameter	Unit	Value
Crest elevation	m	451
Crest width	m	10
Maximum height	m	14.22
Upstream slope	V:H	1:2
Downstream slope	V:H	1:2
Embankment volume	m³	42,474
Reservoir: total area occupied	m²	72,031
Crest length	m	~280
Excavation volume	m³	96,219
Reservoir capacity	m³	701,700

18.12.2 Hydrological Criteria – Hydraulics

Hydrological and hydraulic studies were developed to size the hydraulic structures in a safely manner.

The dam drains a total area of 0.5km^2. This value was obtained from a topographic survey. The length of the main thalweg was obtained from computer-assisted planimetry using regional topography. The value found was 0.9km.

The overflow system will consist of a trapezoidal cross-section open channel associated with hydraulic descent on 0.5m steps with sizing of the section to be defined. It will be implemented on the right Water Dam shoulder with sill at 464.0m elevation. The project is based on a 10,000-year event.

The surface drainage system will consist of half pipe channels to receive water flow from the berms and perimeter channels along the embankment. The berms will have a cross slope of 5% and longitudinal inclination of 0.5%, in addition to concrete half pipe channels. The dimensioning of the berm drainage system is based on a 100-year event whilst the perimeter channels are sized for a 500-year event.

18.12.3 Stability Analysis

Based on analyses, it is possible to analyse the slopes stability by using the limit equilibrium theory in the SLOPE/W module. The method chosen to calculate the safety factor of stability analysis was the Spencer method, that considers circular rupture surfaces which were defined by mesh centres and tangent lines, passing through the dike and foundation of the dam.

The geotechnical parameters used in stability analyses were based on GHT's professional experience with similar projects.

The safety factor values obtained (SF obtained) in the analyses and required values (SF REQ) are by ABNT guidelines and are shown in Table 18-11 according to the safety criteria adopted and considered conditions.

Table 18-11: Stability Analysis Results of B1 Dam Slope

Condition	Obtained Safety Factor	Required Safety Factor
Final upstream construction	1.4	1.3
Final downstream construction	1.36	1.3
Downstream long term	1.67	1.5
Berms long term	1.81	1.1
Maximum NA	1.37	1.3
Fast Lowering	1.45	1.1
Pseudo-static (Seismic)	1.28	1.1

Based on the results of the safety factors obtained in the stability analyses in all cases evaluated show values higher than those required by Brazilian NORM # 13.028/2017 (ABNT- Brazilian Association of Technical Norms).

18.12.4 Internal Drainage

Internal drainage systems have been designed to control percolations through the embankment and foundation. These systems consist of a vertical sand filter connected to a horizontal draining mat. The water collected by the mat is sent to the lowest point of the embankment in upstanding drain located downstream. Considering a more critical operating condition the unit percolation flow that crosses the dam was calculated to be 6.4 * 10 -7 (m^3/s) /m.

The internal drainage system was dimensioned from the flow analysis using Darcy`s Law. The dimensioning resulted in a 0.70m thickness of the vertical filter and a thickness of 0.70 of the horizontal mats in the main section. Results of the sections located on the dam’s shoulder and their respective flows showed mat thickness of 0.35m, with a transition length of 5.0m between the different thicknesses.

The vertical chimney filter installed is a homogeneous drain, made of coarse sand. The chimney drain discharges at a blanket drain, also homogeneous and made of coarse sand. This system discharges at a toe drain made of gravel and manual stone, meeting the filter criteria proposed by Bertram and Terzaghi.

18.12.5 Geotechnical Instrumentation

The geotechnical monitoring program provides for the installation of instruments that will allow monitoring of the development of neutral pressures in the foundation and the embankment, the monitoring of the water level in the dam, the flow of the internal drainage system of the dam and definition of topographic reference points for displacement and deformation control.

There is also provision for the installation of instruments in 3 critical sections of the dam. Table 18-12 lists the number of instruments in each section.

Table 18-12: B1 Dam Instrumentation

Instrument	Dam B1 SECTION A-A	Dam B1 SECTION B-B	Dam B1 SECTION C-C	Total
Piezometer Stand Pipe	2	2	2	6
Water level metres	2	3	2	7
Surface Displacement Mark	2	2	2	6

18.13 Waste Dumps and Low-Grade Stockpile

The mine will generate amounts of waste which need to be stored safely. Also, a low-grade stockpile will be required. These waste and low-grade piles are designed in order to assist the technical constraints established by the owner of the operation, obeying any relevant laws and following good engineering practice.

The waste deposits are divided into three disposal piles and one pile for low grade ore. Internal drainage systems include a series of excavations filled with granular material, the function of which will be to capture the percolated flows through the stored materials and transfer to sumps which surround each deposit.

Based on the information provided by geotechnical drilling, the material in the area where the pile will be installed is mainly composed of residual soils including clayey silts of soft and medium consistencies with fragments of laterite and micas.

According to the information obtained from the probing, excavations were projected at 2 metres average depth until the right material is found to support the loading of the ore to be deposited.

18.13.1 Geometrical Parameters

The main geometrical characteristics of deposits are presented in Table 18-13.

Table 18-13: Waste Dumps and LG Stockpile Geometrical Parameters

Parameter	Unit	Waste Dump 1	Waste Dump 2	Waste Dump 3	Waste Dump 4	Waste Dump 5	Waste Dump 6	LG Stockpile
Bench Height	m	10	10	10	10	10	10	10
Berm Width	m	10	10	10	10	10	10	10
Batter Angle	H:V	2:1	2:1	2:1	2:1	2:1	2:1	2:1
Maximum Height	m	72	65	60	50	100	96	34
Maximum Elevation	m	460	480	480	490	510	540	475
Storage Volume	Mm^3	4.0	6.4	3.7	4.2	16.9	15.1	1.5

18.13.2 Drainage

The surface drainage pile will be composed of pre-shaped half pipe concrete trenches and water decline on steps.

The Wilkins Methodology was adopted for the dimensioning of the internal drainage system. This method applies to turbulent flow conditions observed in waste piles, where the bottom drains are formed by angular rock blocks or open particle size gravels.

18.13.3 Stability Analysis

The analyses are based on a series of activities that enabled the identification of the physical stability conditioning factors of the deposits. The minimum safety conditions of slopes of the material storage deposits are established by the Brazilian Association of Technical Standards (“**ABNT**”) who defines minimum safety factors (“**SF**”) for short-term conditions, and occurrence of seismic events. The minimum SF corresponding to the conditions evaluated are as follows:

- Construction process (short term): SF ≥ 1.3;
- Maximum load (long term): SF ≥ 1.5;
- Seismic event occurrence (pseudo-static analysis): SF ≥ 1.1.

The elaboration of the geological-geotechnical profile used for the stability analyses was based on the drilling percussion probes, as well as GHT’s professional experience. It is necessary to supplement this information with more detailed geotechnical investigations and complement them with laboratory tests.

Cross-sectional structures were chosen to evaluate the physical stability of critical points of the pile. The safety factors obtained are summarized in Table 18-14:

Results show that the safety factors obtained in the stability analyses, in all cases, values are above those required.

Table 18-14: Water dam safety factors

Item	Condition	Slope	Estimated Safety Factor	Required Safety Factor
Waste Dump 1	Static - Long term	Right	1.72	1.5
	Pseudo-Static	Right	1.37	1.1
	Static - Long term	Left	1.50	1.5
	Pseudo-Static	Left	1.11	1.1
Waste Dump 2	Static - Long term	Right	1.67	1.5
	Pseudo-Static	Right	1.30	1.1
	Static - Long term	Left	1.88	1.5
	Pseudo-Static	Left	1.42	1.1
Waste Dump 3	Static - Long term	Right	1.51	1.5
	Pseudo-Static	Right	1.13	1.1
	Static - Long term	Left	1.60	1.5
	Pseudo-Static	Left	1.25	1.1
LG Stockpile	Static - Long term	Right	1.5	1.5
	Pseudo-Static	Right	1.1	1.1
	Static - Long term	Left	1.6	1.5
	Pseudo-Static	Left	1.2	1.1

18.13.4 Geotechnical Instrumentation

The geotechnical monitoring program provides installation of instrument to accompany the development of neutral pressures in the foundation and the displacements over time. Table 18-15 presents the instrumentation summary to be installed in the piles.

Table 18-15: Instrumentation

Item	Electric Piezometer	Displacement Mark
Waste Dump 1	2	7
Waste Dump 2	2	9
Waste Dump 3	2	7
LG Stockpile	2	6

19 MARKET STUDIES AND CONTRACTS

No formal market studies have been undertaken. Gold in bullion is the principal commodity at the Posse Gold Project and is freely traded, at prices that are widely known, so that prospects for sale of any production are virtually assured. The Posse Gold Project will produce gold bars which will be refined to produce 99.9% purity gold and the refined gold will be sold to banks or other financial institutions either in Brazil or offshore on a spot price basis to capture the highest price.

A gold price of US$1,450/oz was used for the Mineral Reserve estimate. SRK considers that this price is reasonable and notes that gold has been trading above this price over the last years. The Base Case for the financial model was US$1,600/oz.

There are no material contracts in place as of the effective date of this Technical Report.

Refining contracts are typically put in place with well recognized international refineries and sales are made on spot gold prices. The ability to get a contract in place for the sale of doré prior to start of production is not seen as a risk to the Project.

It is anticipated that the following major contracts will be established to support operations:

- Processing plant engineering;
- Mining;
- Tailings dam construction contract;
- Power supply;
- Fuel supply;
- Dore transport and refining; and
- Site security.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

DBO Engenharia Ltda (“**DBO**”) completed a consolidated Environmental Impact Study (“**EIS**”) on behalf of Amarillo in compliance with the existing Brazilian legal requirements to obtain an environmental permit to construct and operate the Posse Project. This section summarizes the information compiled and analysed in the EIS. In addition, a description of the socio-environmental diagnosis and the proposed control measures will be described.

20.2 Governing Policies and Commitments

Amarillo has established a commitment to conduct several environmental monitoring and control programs in the Project area and the nearby community of Mara Rosa. To demonstrate socio-environmental responsibility, actions such as continuous engagement with local government institutions and organizations, implementing plans to maintain and improve the quality of the workforce, as well as actions to communicate the progress achieved and the development of the undertaking are foreseen.

20.3 Regulatory Framework and Permitting Status

Amarillo’s Project will be installed at the site of the former Mina de Posse located in the area comprised by the Mining Leases 1,783/86, 1,784/86, and 1,785/86, referenced to the National Department of Mineral Production (“**ANM**”) case #’s 862,000/84, 861,241/80 and 860,952/80, respectively, all located in the municipality of Mara Rosa, State of Goiás.

An environmental license is legally required to construct any project that may potentially impact the environment. The obligation to obtain the license is a requirement of both the State Environmental Agencies and of the Brazilian Environmental Institute (“**IBAMA**”) as they are members of the National Environmental System (“**SISNAMA**”). The main norms for environmental licensing are expressed in the National Environmental Policy (Law 6,938/81) and the National Environmental Council (“**CONAMA**”) Resolutions 001/86 and 237/97.

In the case of mining projects, there are other important resolutions such as the CONAMA Resolution 009/90 which establishes the procedure for obtaining the environmental licensing of mineral extraction activities. The ANM resolution #68 establishes that mineral exploration activities must submit a Closure Plan for approval; Law # 6567, dated 09/24/1978 and changed by Law #13,975 dated 01/07/2020. This Law provides for a specific regime of exploration regarding the use of mineral substances, pointed out in Art. 2: *“Licensed mineral exploitation is* *granted exclusively to the soil owner or whoever has explicit authorization unless the mineral deposit is located on properties belonging to a legal entity under Public Rights Law as in the hypothesis provided in paragraph 1 of Art.10"*. Additionally, mining activities must be regular, or they will suffer penalties and detention as ruled by the Environmental Crimes Law ("**Law #9,605/1998**").

The State of Goiás Department for the Environment and Sustainable Development is the agency responsible for the project's environmental licensing. In May 2016, Amarillo received the LP. The LI application was filed on December 13, 2019. During 2021 and the first quarter of 2022, Amarillo received the LI of several components. All the licenses are summarized in Table 20-1 below:

Table 20-1: Licenses and Water Grants for the Posse Gold Project

Type	Number	Issuance	Expiration	Comment
LP	792/2016	05/05/16	-	Posse Mine
LI	06/2021	02/02/21	02/02/22	Substation Expansion in Porangatu City
	45/2021	02/03/21	01/29/27	Construction Site, Access Roads, and Topsoil Deposition
	226/2021	05/18/21	05/18/22	Waste Rock Pile 1
	245/2021	05/28/21	01/29/27	Waste Rock Pile 2
	309/2021	06/30/21	01/29/27	Waste Rock Pile 4
	418/2021	10/15/21	10/15/27	Waste Rock Pile 3
	421/2021	10/19/21	10/19/31	138 kV Power Line
	474/2021	12/14/21	12/14/27	Low-Grade Ore Pile
	34/2022	02/02/22	02/02/28	Mine Pit
Environmental Registry	-	10/18/21	-	69 kV Power Line
Water license	1412/2020	07/15/20	04/08/30	Rio do Ouro - River
Authorization	17/2021	05/03/21	05/03/23	IPHAN – Archeology ordinance
Archeology Registry	-	10/05/21	-	Term of commitment – 69kv
Authorization	2649/2020	08/12/20	-	Rescue and Conservation of Terrestrial Fauna

Amarillo is implementing the following social and environmental programs which are benefiting the community of Mara Rosa:

- Negotiation program with landowners;
- Social Communication Program;
- Environmental Education Program;
- Surface and Groundwater Quality Monitoring Program;
- Terrestrial and Fauna Monitoring Program;
- Qualification, Training and Selection Program for Local Labour with the installation of the CTS (Selection and Training Centre); and
- Municipal Strengthening Program - Sub-Program for the Development of Local Suppliers ("**PDF**") and Certification of Companies.

20.4 Environmental and Socio-Economic Studies (DBO, 2015)

Once Amarillo commences extraction activities, few natural areas will be affected as the area has been previously impacted by other uses, such as cattle ranches. However, several environmental studies were completed to update the socio-environmental diagnosis of the region and forecast the impacts resulting from the planned mining activities.

Fieldwork was conducted to collect primary data in the entire area of direct influence, as well as in some points of the indirect area of influence. Results of the findings are summarized below:

20.4.1 Physical Environment

According to the Environmental Impact Study (DBO, 2015), the direct area of influence defined for the undertaking corresponds to approximately 940ha. The Project area is characterised by a hot semi-humid climate, with 4 to 5 dry months.

The annual average temperature in the region is 22.9ºC, with an average temperature range of around 7.5ºC. The average annual total evaporation is 1,842.5mm (1977-2010), with the highest rates observed in May to October. The annual average precipitation is 1725.9mm.

Hydrogeologic studies were conducted by HIDROVIA Hidrogeologia e Meio Ambiente Ltda. The project is located mainly in gneisses and meta volcano-sedimentary rocks domain of the Mara Rosa Sequence, which belongs to the Arco Magmático de Goiás (Magmatic Arch of Goiás State). Mina de Posse (Posse Mine) is in the eastern segment known as Faixa Leste (the Eastern Range), in an area of transgressive faults towards the northeast and where biotite-gneiss comes into contact with amphibolite. This contact is characterized by the presence of muscovite-quartz-biotite schist, the main ore zone. The fresh rock occurs after about 35m of depth where very closed fracture zones occur. The few open zones found are discrete and localized. Accordingly, the Project area has low underground water availability. Out of 80 water points analysed, 43 are intermittent, 16 are ephemeral, and only 25 are perennial, demonstrating the low rock water storage capacity in the area.

The main use of ground and surface water identified is for human consumption, animal consumption, and irrigation on a very small scale. The rural properties that are found in the area are small farms or small rural communities that cultivate the land and own livestock for their subsistence.

The Mara Rosa region is located near an area known as the SW-NE seismic strip - State of Goiás and Tocantins. It is a low-intensity seismic zone, where the likelihood of a major earthquake occurring is very low. Seismic considerations have been taken into consideration during the elaboration of engineering designs.

A speleological study was carried out as part of the Environmental Impact Study. The exercise confirmed the nonexistence of natural caves, burrows, and or shelters in the project footprint.

20.4.2 Biological Environment

The vegetation survey conducted in the direct areas of influence of the Project, which was carried out as part of the time that the Environmental Impact Study, indicates an advanced extent of forest degradation.

Based on direct field observation and phytosociological research, 151 plant species were identified and distributed into 61 families totalling 2,286 units catalogued. The five tree species that were registered with the highest importance in value (frequency + density + dominance) were the Physocallimma scaberrimum, Astronium (aroeira), Parapiptadenia rigida (angico-vermelho), Astronium fraxinifolium (gonçalo-alves), and Tapirira guianensis (pau-pombo).

As for the number of species per habitat, 36 species were cataloged based on their physiognomy typical of the high open Cerrado (tropical savanna), 85 species for the Cerrado stricto sensu, 72 species for the seasonal semideciduous forest, 108 species for the gallery forest and 45 species for the wooded pasture. Out of the total number of species sampled, about 100 species had their frequency in more than one physiognomy. The Cerrado stricto sensu sampling unit presented the largest number of cataloged species.

Only one species cataloged is in the Official List of Endangered Species of Brazilian Flora, the Myracrodruon urundeuva (aroeira).

In general, impacts on the flora caused by the implementation of the Posse Gold Project will occur directly in the areas used for extracting and processing ore due to the clearing of individual forests or arboretum trees. Germplasm of native trees found in these areas will be collected to create seedlings for later use in the mine closure program.

In the survey of the herpetofauna, a total of 34 species of amphibians were identified. As for reptiles, 38 species and a total of 17 families were registered. About avifauna, a total of 4,072 orders of 218 species were catalogued, which are distributed in 25 orders, 61 families.

None of the registered species are on the Ministry of the Environment's list of endangered species. A total of 39 species of mammals were registered, based on direct observations by sightings and indirect evidence, such as footprints. Lastly, regarding the ichthyofauna, a total of 312 specimens were collected in the upper basin of Rio do Ouro in Mara Rosa. They were identified and distributed into 3 orders, 11 families, 4 subfamilies, and 19 genera.

Cerrado biome has species of mammals, birds, terrestrial invertebrates, and fishes included in the List of endangered Brazilian fauna (IN003-03MMA).

As a way to offset potential impacts from the project, the land was acquired in the conservation unit of the Terra Ronca State Park (see Section 3.10).

20.4.3 Socio-Economic Environment

Mina de Posse is located approximately 7.0km north of Mara Rosa city and close to Araras stream, in the municipality of Mara Rosa, located on the northern part of the State of Goiás. The city of Mara Rosa is approximately 350km to the NW of Brasilia and the area can be accessed via Brasilia and/or Goiânia.

Mara Rosa’s economy is based on agriculture and livestock. Most properties in the municipality are destined for subsistence agriculture described as family farming.

About the transportation infrastructure, the municipality of Mara Rosa is strategically located off interstate BR-153, the main road coming and going from North to South of Brazil. The North-South Railroad borders the town and the project’s location, which is currently under construction.

According to the last official Brazilian Institute of Geography and Statistics census from 2010, the total population is 10,649 inhabitants, 74.76% of the population lives around the town centre. The Municipal Human Development Index of Mara Rosa was 0.691 in 2010, which places the municipality’s Average Human Development between 0.6 and 0.699. The sector that grew the most was Education (0.229), between the years 2000 and 2010, followed by Income (0.096) and Longevity (0.066), the same is true for the rest of Brazil and the state of Goiás.

According to information from the Unified Health System (“**DATASUS**”), medical infrastructure in the municipality consists of 02 hospitals, one public and one private. There are a total of 31 beds, 27 of which belong to the Unified Health System.

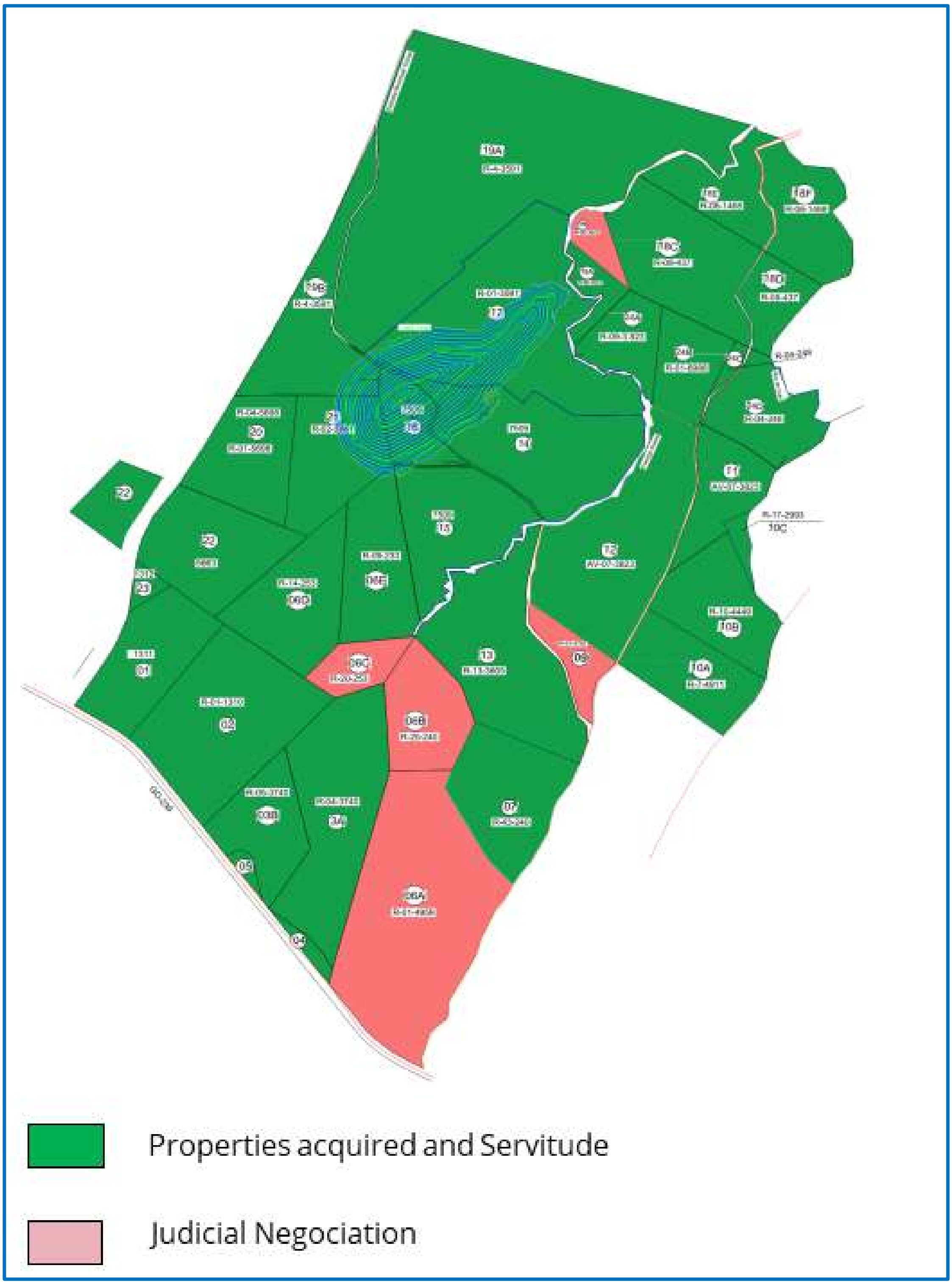
During the construction phase of Posse Gold Project, 700 direct jobs and 2,000 indirect jobs will be generated with a total investment of approximately R$980m.

The archaeological survey conducted confirmed the presence of anthropic activities in the area of the Project. According to the information collected and especially based on the presence of an important site in the region, the “**Petróglifo de Mara Rosa**” (the “*Petroglyph of Mara Rosa*”), it can be concluded that the area was once a place where there was the presence of humans. All the artifacts recovered were sent to the Historical Museum of Jataí, after IPHAN’s approval. Traces of pre-colonial ceramic were found, probably dating back to the mid-17th century.

20.5 Land Tenure

The Project has acquired 926.2ha of the 1,070.2ha land package required. The remaining 144 hectares are under judicial negotiation and are expected to be finalised by the beginning of construction. The land where the pit is located has been acquired.

Figure 20-1: Land Tenure Map



20.6 Environmental Control Plan

An Environmental Control Plan has been developed for the Project. It consists of 13 environmental programs and 5 management plans. Each program will be conducted according to a specific schedule. The programs and plans that compose the Environmental Control Plan are:

- Environmental Control Program for Construction Activities;
- Air Quality, Noise, and Vibration Emissions Control Program;

- Surface and Groundwater Quality Monitoring Program;
- Water and Groundwater Monitoring Program;
- Water Quality Control Program;
- Erosion Control Program;
- Solid Waste Management Program – PGRS;
- Program for the Recovery of Degraded Areas – PRAD;
- Vegetation Suppression Program;
- Vegetation Rescue Program and Monitoring of Modifications in Vegetation Cover, Composition and Diversity Rural Area Support Program – PAAR;
- Road Control and Conservation Program - PCCV;
- Environmental Education Program – PEA;
- Archeological Cultural-Historical Heritage Program;
- Biodiversity management plan;
- Stakeholder Engagement Plan;
- Communication plan;
- Emergency preparedness and response plans (coordinate with health, security safety, and communities); and
- Mine closure plan.

20.7 Additional environmental considerations

20.7.1 Industrial Effluents

The Project replaced the wet tailings dam that was initially planned for a dry stack. Amarillo will install 4 filter presses, with a total capacity of 459m^3/h (661tonnes/h). The dry stack will occupy a smaller area than the original TSF footprint approved. This change in the Project is already contemplated in the LI.

The improvement has reduced the environmental footprint of the Project, including a significant decrease in the water intake from Rio do Ouro.

The water recovered in the filter presses will be reused at the processing plant. The contact water from the dry stack will be collected and detoxified before being stored in water reservoir B1.

20.7.2 Domestic Effluents

The wastewater from the administration buildings will be collected and treated in a sewage treatment plant.

20.7.3 Waste Management

Non-hazardous waste will be disposed of in the Mara Rosa municipal landfill. Hazardous wastes will be disposed of in approved landfills.

Waste management will be carried out by specialized registered companies

20.8 Atmospheric Emissions

The atmospheric emissions will be generated in the processing plant mostly from the natural gas boiler and furnace of the smelter and gases from the regeneration of carbon, electrolytic cells, and acid washing.

The Air Quality, Noise, and Vibration Emissions Control Program is part of the Environmental

Control Plan. The program aims to collect data on air quality noise and vibrations. Collected data will be continuously evaluated to make sure controls are in place and working properly, keeping levels within the standards established by legislation.

20.9 Geochemistry

Amarillo developed two geochemistry studies to determine whether Posse ore and waste rock could have Acid Rock Drainage ("**ARD**") potential. The first by Coffey and the second by Golder, both in 2015.

Coffey collected three composite ore samples which were subjected to humidity cell testing over 18 weeks. They concluded the materials tested are unlikely to pose a threat to the receiving environment.

Golder did a field inspection of the historic Posse Mine and the historic Baribras TSF, focusing on looking for evidence of ARD occurrence. There was no evidence of ARD except at a single sample in the Baribras TSF. This TSF will be reclaimed and sent to the dry stack (lined).

Although ARD potential is low, the Project decided to convert the waste storage facility #3 to receive potential Potentially Acid Generating ("**PAG**") material. This storage facility will be lined and contact water generated from it will be recirculated to the processing plant. During operation, ABA testing will be continuously conducted to be able to identify any PAG material.

20.9.1 Vegetation Clearing

The clearing of vegetation will occur in areas to be occupied by the Project's infrastructure (buildings, access roads, pits, waste and tailings piles, water dam, etc.). According to information obtained in the Rural Environmental Registry System ("**SICAR**"), which was updated on 08/24/2019, fragments of the legal reserve were identified in the area to be occupied by the Project that will be offset by the conservation unit mentioned above.

To compensate for this clearing an offset has been established in the Parque Estadual de terra Ronca. This offset is underway, 95% of the land needed has been acquired, the remainder 5% is under judicial negotiation.

20.9.2 Water Supply and Water Quality

To provide water for the processing plant, a water reservoir (B1) will be constructed to provide water. The Reservoir B1 has been designed for a storage capacity of approximately 510,000m^3. According to the water balance study and simulations made by GBM (2021), this volume will be sufficient to supply the Project with the needed freshwater demand.

The environmental agency granted the water intake of 91,392m^3/month in January and February and of 97,92000m^3/month in March to May and in December from Rio do Ouro.

The water reservoir will also receive water from the pit.

A potable water treatment plant will be installed.

The Surface and Groundwater Quality Monitoring Program is being executed since 2011 and all the reports are presented to the Regulatory Agency ("**SEMAD**") and the Public Ministry. The monitoring plan has fifteen (15) surface water points and twelve (12) groundwater points, in a total of twenty-two (22) points.

During the campaign made in February 2020, the following parameters did not comply with the legislation for surface water (CONAMA 357/2005): dissolved aluminium, coliform thermotolerant, true colour, dissolved iron, total phosphorus, total manganese, dissolved oxygen, and pH. For groundwater, the parameters presented concentrations above the CONAMA 396/2008 were total iron total manganese, and coliform thermotolerant. These non-compliances could reflect land use and occupation in the region, like agriculture. Some exceedances could also be related to the local geology, for example, aluminium, iron, and manganese.

It is important to note that the water quality characteristics reflect the baseline collected in the area before Amarillo commencing any activities on site.

20.10 Mine Closure

A preliminary mine closure plan was developed by Ramboll in 2020. The plan includes specific plans to closure each mine component.

The anthropic terrains, i.e., the pit, the piles of sterile waste material, and the dray stack, will be closed and revegetated with herbaceous and shrubby species and equipped with instruments for geotechnical monitoring.

The water reservoir and the water uptake system, the transmission line, and the substation will be donated to the municipality or the state. The remaining industrial, operational, and administrative support facilities will be decommissioned. Any recyclable equipment and materials will be sold. The remaining materials will be sent to landfills.

The closure schedule is of eight years: 1 year of closure activities during the last year of operations, 2 years of final closure at the end of life of the mine, and 5 years of post-closure monitoring activities.

The closure cost was updated by Amarillo in 2021 to reflect more accurate unit costs. Table 20-2 presents the closure costs for the Posse Gold Project.

The closure cost estimate does not include the revenue eventually generated from the sale of equipment in industrial facilities or recyclable materials.

Table 20-2: Estimated general costs for the closure of Posse Gold Project

Item \#	Cost (US$k)
Assessments, Projects, Maintenance, and Monitoring Activities	1,501
Mining Area: Top Bench (not submerged)	352
Waste Storage Facility 1	305
Waste Storage Facility 2	513
Waste Storage Facility 3	317
Waste Storage Facility 4	387
Waste Storage Facility 5	585
Waste Storage Facility 6	761
Tailing Pile and Dike	1,123
Industrial, Operational and Administrative Support Facilities	9,614
Remaining Areas	2,723
Total Cost	**18,182**
Total Cost with contingency	**20,000**

SRK recommends periodic updates to the mine closure plan to adjust the socio-environmental conditions of the region, seeking to ensure post-closure sustainability in the generation of income and conservation of the environment and to comply with ANM 68/2021 which requires an update every five years.

21 CAPITAL AND OPERATING COSTS

21.1.1 Capital Cost Estimate

The capital cost of the Project has been estimated based on the scope of work defined in the section below. The parties below have contributed to the capital cost estimate in specific areas, as listed.

- **Ausenco:** Crushing; Process; Ancillary buildings; Utilities; On-site infrastructure; Service roads; Indirect cost; Contingency; and Water treatment;
- **Amarillo**: Mining; Water dam; Filtered TSF; Waste dumps; Power line; Hardware and Equipment; Software and communication systems; Temporary creek diversion; and
- **Ramboll**: Mine closure

21.1.2 Capital Cost Assumptions

The cost estimate for the Posse Gold Project consists of mining, crushing, processing plant and associated infrastructure.

All costs are presented in United States dollars ("**US$**") unless otherwise indicated. The estimate has been prepared based on an exchange rate of R$5.05/US$.

The base date of all estimates is the second semester of calendar year 2021 (H2 2021). No allowance has been included in the estimates for escalation beyond this date.

The estimates have an overall accuracy range of -10% to +15% for their scope.

Indirect costs have been factored from the direct cost, using percentages established from experience from similar operations in the region.

The capital cost estimate of the processing plant and infrastructure presented in the study is a total cost estimates and include growth factors for supply, installation and civil works.

The estimates do not include or allow escalation and foreign exchange fluctuations.

Mining is based on a contractor operation. It is thus anticipated that no acquisition of primary mining equipment will be required.

The pre-stripping and all costs incurred during the pre-production period were considered as capital cost.

The following sections describe the basis of the cost estimate.

Taxes

Taxes included in the cost estimate are:

- ISS (Municipal service tax);
- ICMS (Tax on the circulation of goods and transportation and communication services);
- DIFAL (ICMS tax difference, applied to interstate operations);
- PIS (Employees' profit participation program);
- COFINS (Social contribution for social security financing);
- IPI (Tax on industrialized goods);
- II (Importation tax);
- AFRMM (Merchant marine renewal tax); and
- IOF (Financial operations tax);

No fiscal incentives were considered.

Contingency

Contingency covers unknown costs that are unexpected to be incurred within the defined scope of the project but cannot be defined and identified at this stage of the project. The contingency allowance specifically excludes cost arising from scope changes, project risk factors and other items that are excluded from the capital cost estimate. The project contingency is meant to cover the normal inadequacies that are inherent in any project estimate due to the dynamic nature of project engineering and construction.

Contingency was estimated as 8% to 10% of capital costs.

Sustaining Capital

An estimate of the replacement of equipment, optimization of facilities and expansion of filtered tailings pile and additional waste dumps have been included.

Sustaining capital associated to the filtered tailings pile and water dam are estimated based on specific engineering requirements.

21.1.3 Capital Cost Summary

The summary of capital costs for the LoM is shown in Table 21 1. The next sections describe every component of the initial and sustaining capital cost criteria and results.

Table 21-1: Capital cost estimate summary

Item	Initial Capex (US$k)	Sustaining (US$k)	Total (US$k)
Processing plant and infrastructure(1)	112,882	0	112,882
Power line	13,805	0	13,805
Mining (pre-stripping)	9,299	0	9,299
Waste dumps and low-grade stockpile	19,503	24,703	44,206
Araras creek diversion	-	212	212
Water dam	2,000	0	2,000
Filtered tailings pile	0	9,951	9,951
Owner costs	13,369	5,000	18,369
Subtotal	**170,857**	**39,866**	**210,723**
Contingency	14,284	3,487	17,770
Subtotal	**185,141**	**43,352**	**228,493**
Working Capital	8,876	0	8,876
Total capital cost	**194,017**	**43,352**	**237,369**
Mine closure w/ 10% contingency	-	-	20,000

(1) With exception of owner cost, electric equipment and working capital.

The mine closure cost estimate is shown in Table 21-2.

Table 21-2: Mine closure cost estimate summary

Item	Total (US$k)
Pre-closure Cost	1,131
Closure Cost	15,915
Post-closure Cost	954
Subtotal	**18,000**
Contingency	2,000
Total	**20,000**

21.1.4 Process Plant and Infrastructure Capital Cost Estimate

The estimate conforms to AACE Class 3 guidelines for a Feasibility Study Level Estimate with a -10% to +15% accuracy. Table 21-3 provides a summary of the estimate of the Process plant and infrastructure on site.

Table 21-3: Mine closure cost estimate summary

Description	Total (US$k)
Direct Cost	
General Project Cost	4,499
Temporary Facilities	200
Access and Service Roads	700
Mine Preproduction and Miscellaneous	11,645
Process Plant	72,552
Utilities and Services	3,175
Power Line	13,805
Electric Systems	3,850
On-Site Infrastructure	7,876
Subtotal Direct Cost	**118,302**
Indirect Cost	
EPCM and Consulting Services	8,384
Owner Costs	13,369
Subtotal Indirect Cost	**21,753**
Subtotal Direct + Indirect	**140,055**
Working Capital	8,876
Project Contingency	11,820
Total Initial Capital Cost	**160,751**

21.1.5 Filtered Tailings Pile, Water Dam, Waste Dumps and Araras Creek Diversion Capital Cost Estimate

The capital and sustaining costs estimated for the construction of the filtered tailings pile, water dam, waste dumps, low grade stockpile, and the Araras creek diversion are presented in Table 21-4.

A 10% contingency factor was applied to the initial capital cost. For the sustaining capital, a contingency of 10% was assumed for the costs incurred in the Araras creek diversion and the construction of waste dumps 4, 5 and 6, while a 21% factor was applied to the filtered tailings pile expansions.

Table 21-4: Tailings Pile, Water Dam, Waste Dumps, LG Stockpile and Creek Diversion Capital Cost Summary

Item	Initial Capex (US$k)	Sustaining (US$k)	Total (US$k)
Water dam	2,000	0	2,000
Filtered tailings dump	0	9,951	9,951
Low grade stockpile	2,271	0	2,271
Waste dumps 1, 2 and 3	17,232	0	17,232
Waste dumps 4 and 5	0	17,227	17,227
Waste dump 6	0	7,476	7,476
Creek diversion	0	212	212
Subtotal	**21,503**	**34,866**	**56,368**
Contingency	1,720	3,487	5,207
Total	**23,223**	**38,352**	**61,575**

21.1.6 Power transmission line

The power electric line construction is estimated as of US$13.8m.

21.1.7 Mining Capital Cost Estimate

All production primary mining operations will be contracted, including drilling, blasting, loading, haulage and material disposal. Thus, initial capital and sustaining costs will be restricted to:

- Pre-stripping;
- Owner costs during the pre-production period;
- Auxiliary components such as the core shed, hardware, light vehicles, software and others;

A summary of the mining capital and sustaining costs is shown in Table 21-5.

Table 21-5: Mining Capital Cost Summary

Item	Initial Capex (US$k)
Pre-stripping	4,894
Owner costs	3,730
Hardware and Equipment	118
Software	114
Core Shed	108
Other Expenses	34
Explosives Magazine	300
Subtotal	**9,299**
Contingency	744
Total	**10,043**

21.1.8 Mine Closure Cost Estimate

The mine closure is divided in three periods: pre-closure, closure, and post-closure period. A summary of the mine closure estimate is shown in Table 21-16.

Table 21-6: Mine Closure Cost Summary

Item	Cost (US$k)
Assessments, Projects, Maintenance, and Monitoring Activities	1,501
Mining Area: Top Bench (not submerged)	352
Waste Storage Facility 1	305
Waste Storage Facility 2	513
Waste Storage Facility 3	317
Waste Storage Facility 4	387
Waste Storage Facility 5	585
Waste Storage Facility 6	761
Tailing Pile and Dike	1,123
Industrial, Operational and Administrative Support Facilities	9,614
Remaining Areas	2,723
Total Cost	**18,182**
Total Cost with contingency	**20,000**

21.2 Operating Cost Estimate

The operating cost estimate is broken down by area including mining, processing, G&A and tailings logistics. The processing, G&A, mining and tailings logistics operating costs were estimated by Amarillo based on updated quotes. The operating costs are reported in US$.

21.2.1 Operating Cost Summary

Table 21-7 shows the operating cost summary, which amounts to US$23.06/t processed over the LoM.

Table 21-7: Operating Cost Estimate Summary

Item	Unit	Operating Cost
Mining	US$/t processed	9.97
Processing w/ 5% allowance	US$/t processed	10.89
G&A w/ 5% allowance	US$/t processed	1.20
Tailings Haulage and Disposal	US$/t processed	1.00
Total	**US$/t processed**	**23.06**

Table 21-8 shows the estimated cash cost over the LoM for a total gold production of 811koz.

Table 21-8: LoM Cash Cost Estimate

LOM Cash Cost Estimate	Total Cost (US$k)	Unit Cost (US$/oz)
Operating Cost Estimate		
Mining	237,431	292.8
Processing w/ 5% allowance	259,234	319.6
G&A w/ 5% allowance	28,566	35.2
Tailings Haulage and Disposal	23,805	29.4
Operating Cost	**549,036**	**677.0**
Adjusted Operating Cost Estimate		
Refining, Transportation, Insurance	9,732	12.0
Royalties	79,553	98.09
Adjusted Operating Cost	**638,321**	**787.1**
All-in Sustaining Cost ("AISC") Estimate		
Sustaining Capital	43,352	53.5
AISC	**681,673**	**840.6**

21.2.2 Process Plant Operating Cost

The following criteria were used to estimate the process plant operating cost:

- no escalation was considered;
- average production considered is 2.5Mtpa;
- off-site gold refining, insurance, and transportation costs are excluded (these costs are included in the financial model);
- power cost of US$0.0565/kWh, with no tax considered due to local tax benefits as per Amarillo information;
- labour costs are based on 8h shifts;
- grinding media consumption was estimated by a grinding media vendor;
- reagent consumption rates were based on test work and process plant design criteria and mass balance; and
- tailings filtration area costs were from the conceptual level tailings filtration engineering study. As this study was at a conceptual level an allowance of 5% on total plant costs was added (contingency).

The process plant operating cost is divided in the following categories:

- **fixed costs:** labour; general and administration (G&A); and
- **variable costs:** power; reagents and consumables; maintenance.

To estimate these costs, data from the process criteria and plant design were used for many of the quantities. For reagents and consumables, the source of the data is the test work and mass balance. As for power consumption, loads from the mechanical equipment list were used to prepare an electrical load list to reach actual consumption values.

The annual process operating costs is shown in Table 21-9, in the categories previously mentioned.

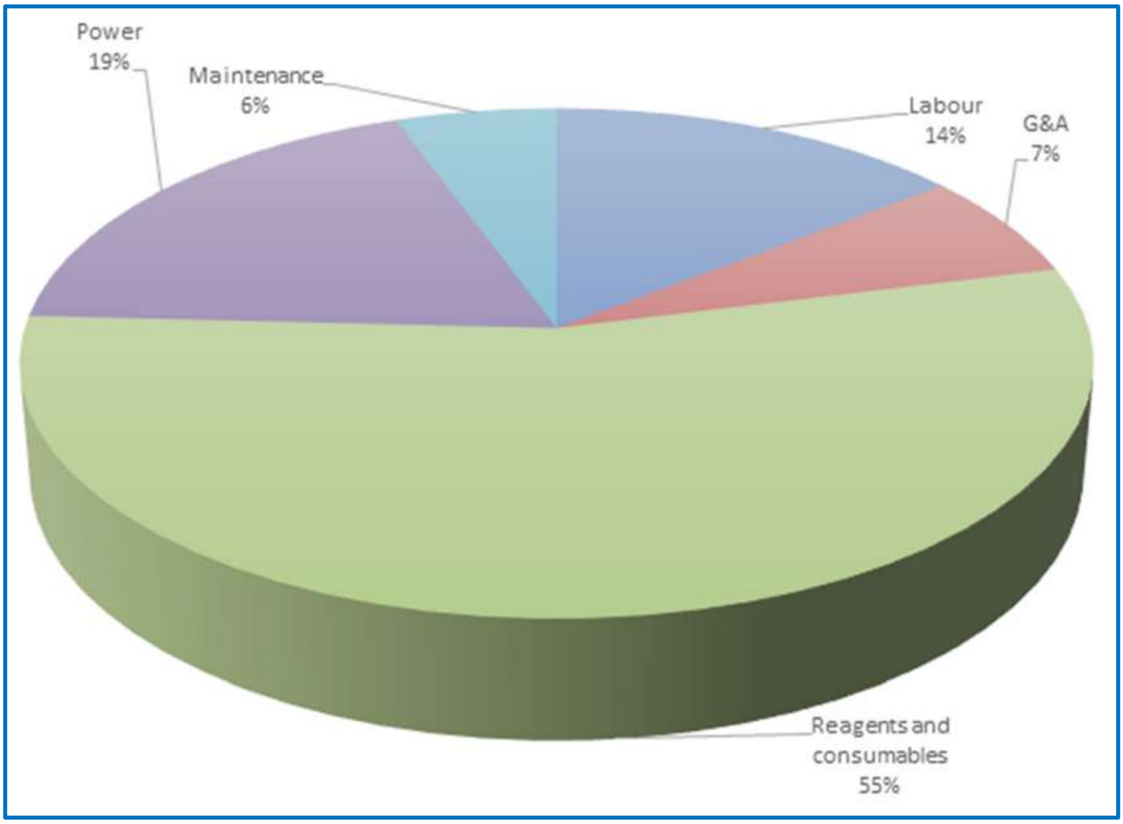
Figure 21-1 presents the distribution of each category of the operating cost (fixed plus variable).

Table 21-9: Process Plant Operating Cost Estimate Summary

Item	US$m/year	US$/tonne
Fixed costs		
Labor	4.15	1.66

Item	US$m/year	US$/tonne
G&A	1.92	0.77
Subtotal (fixed cost)	**6.07**	**2.43**
Variable costs		
Reagents and consumables	15.74	6.30
Power	5.36	2.15
Maintenance	1.61	0.64
Subtotal (variable cost)	**22.72**	**9.09**
Subtotal (without allowances)	**28.79**	**11.52**
Allowance	1.44	0.58
Total	**30.23**	**12.09**
Tailings haulage	2.50	1.00
Total with Filtration Logistics	**32.73**	**13.09**

Figure 21-1: Operating Cost Distribution, with values in US$/t



Labour

The labour rosters were assembled by Amarillo. The roster for G&A is shown in Table 21-10 and for process and maintenance, in Table 21-11.

The roster is based on eight-hour shifts. For continuous operations, four people are required.

The salary costs used are appropriate for the plant location and consider taxes and benefits as per Brazilian law requirements, such as transport, PPE, health plan and others.

Labour is equivalent to approximately 14% of the total process plant operating cost at US$1.66/$t_{milled}$, with a total of 156 people.

Table 21-10: G&A Roster

Labour / Contractor Summary	# / Shift	# Shifts	Quantity
Gatehouse			
Security personnel	1	1	1
Scale operator	1	1	1
Receptionist	1	1	1
Total			**3**
Main office / Mine			
General manager	1	1	1
Administration coordinator	1	1	1
Administration supervisor	1	1	1
Reception / clerk	1	1	1
Accountant	1	1	1
Senior accounting assistant	1	1	1
Junior accounting assistant	1	2	2

Labour / Contractor Summary	# / Shift	# Shifts	Quantity
Senior tax analyst	1	1	1
Junior tax analyst	1	2	2
Lawyer	1	1	1
Receptionist	1	1	1
Procurement coordinator	1	1	1
Senior procurement personnel	1	1	1
Junior procurement personnel	1	1	1
Safety and environmental coordinator	1	1	1
Junior safety engineer	1	1	1
Safety technician	1	2	2
Senior environmental engineer	1	1	1
Environmental technician	1	1	1
HR coordinator	1	1	1
Senior HR analyst	1	1	1
Junior HR analyst	1	2	2
Technician (community)	1	1	1
IT (senior)	1	1	1
IT (junior)	1	1	1
Subtotal			**29**
Medical and fire suppression			
Nurse	1	5	5
Driver	3	1	3
Doctor (partial time)	1	1	1
Subtotal			**9**
Total			**41**

Table 21-11: Operation and Maintenance Roster

Labour / Contractor Summary	# / Shift	# Shifts	Quantity
Process			
Process coordinator	1	1	1
Senior metallurgist	1	1	1
Metallurgist	1	1	1
Junior metallurgist	1	1	1
Shift supervisor	1	5	5
Subtotal			**9**
Warehouse			
Warehouseman	1	4	4
Warehouse assistant	2	1	2
Subtotal			**6**
Workshop			
Maintenance coordinator	1	1	1
Senior maintenance planner	1	1	1
Maintenance planning technician	1	1	1
Electrical engineer	1	1	1
Mechanical engineer	1	1	1
Electrical supervisor	1	1	1
Mechanical supervisor	1	1	1
Instrumentation supervisor	1	1	1
Instrumentation technician	1	2	2
Mechanic	2	4	8
Electrician	2	4	8
Electrician assistant	2	4	8
Mechanical assistant	2	4	8
Truck operator	1	1	1
Lubrication technician	2	1	2
Subtotal			**45**
Laboratory			
Chemist	1	1	1
Technician	2	1	2
Sample preparation personnel	2	4	8
Subtotal			**11**
Crushing control room			
Operator	1	4	4
Operation assistant	1	4	4
Subtotal			**8**
Grinding control room			
Senior operator	1	4	4
Operation assistant	1	4	4
Subtotal			**8**
Operation			
Operator	3	4	12
Operation assistant	4	4	16
Subtotal			**28**
Total			**115**

G&A

The G&A costs were given to Ausenco by Amarillo. They include the following items:

- general expenses, such as personnel travel and consultants;
- health, safety and environmental programs and community activities;
- human resources costs, including recruiting costs;
- administration, such as communications, software, insurance and banking fees;
- outsourced services, such as security, cleaning and catering; and

- other items, such as general consumption and freight.

G&A represents approximately 7% of the total process plant operating cost at US\$0.77/t_{milled}.

Power

Power costs were calculated using the installed motor power for the equipment in the plant and their expected power loads. The total installed load is 14.6MW and the operating load is 11.9MW. The total power consumption considered annually is approximately 90,500MWh/a. or 36kWh/t_{milled}. It includes the process plant facilities and auxiliary facilities, such as the laboratory and the administrative buildings.

Power is equivalent to approximately 19% of the total process plant operating cost at US\$2.15/t_{milled}.

Maintenance

Costs with maintenance and fuels/lubricants were calculated based on the capital cost of mechanical equipment installed per area. The factors used were 3.5% for maintenance and 1.5% for fuel and lubricants. This item represents approximately 6% of the total process plant operating cost at US\$0.64/t_{milled}.

Consumables

Consumption rates for the reagents and consumables were defined based on metallurgical test work results, Ausenco's in-house database and experience on similar projects, typical industry practice and vendor advice. Reagent and consumable costs were sourced by Amarillo.

Reagents and consumables are equivalent to approximately 55% of the total process plant operating cost at US\$6.30/t_{milled}.

Table 21-12: Consumables

Description	Consumption		Unit cost	Total
	kg/t	t/a	US\$/t	US\$/year
Reagents				
Hydrated lime	6.5	16,182	168	2,714,156
Sodium hydroxide	0.21	520.3	991	515,542
Sodium cyanide	0.47	1,173.9	2,956	3,469,951
Flocculant	0.02	50.3	4,339	218,109
Hydrochloric acid	0.07	167.7	860	144,161
SMBS	1.64	4,099.7	375	1,537,814
Copper sulphate	0.09	236.5	3,723	880,505
Total Reagents (US\$)				**9,480,238**
Consumables				
Grinding Media–(total)	1.21	3,025	1,225	3,704,822
Primary Mill Liners (sets/y)	1.50	1.50	218,317	327,475
Secondary Mill Liners (sets/y)	1.50	1.50	218,317	327,475
Jaw Crusher Liners (sets/y)	4.00	4.00	10,991	43,964
Cone Crushers Liners (sets/y)	12.00	36.00	16,586	597,085
Activated Carbon	50.00	125.00	3,592	448,977
Tailings Filter Media (sets/y)	4.00	4.00	148,897	595,590
Laboratory Consumables	-	-	-	214,286
Total Consumables (US\$)				**6,259,674**
Total (US\$)				**15,739,912**

21.2.3 Mining Operating Costs

The primary mining activities are assumed to be undertaken by a contractor, including drilling, blasting, loading, haulage, stockpile re-handling and material disposal. To estimate these costs, Amarillo requested specific proposals from different contractors operating in Brazil which based their estimate on the Posse DFS mine schedule. Amarillo evaluated and equalized the proposals and selected those contractors with the best technical specifications and economic terms.

The mining owner costs estimate includes a variety of activities which will be completed by Amarillo, including: Amarillo's mining team, hardware and software, dewatering, survey services, light vehicles, communication systems, material re-handling and others.

All operating costs incurred during the pre-production period of time were incorporated into the capital cost structure. A summary of the average mining operating costs is presented in Table 21-13. These costs include diesel.

Table 21-13: Mining Operating Cost Summary(1)

Item	Mining Operating Cost	
	By Individual Activity (US$/dmt)	Global Contribution (US$/dmt)
Owner Costs	0.16	0.16
Ore Grade Control	0.25	0.05
Ore Drilling	0.47	0.09
Ore Blasting	0.33	0.06
Ore Load, Haul and Dump	1.01	0.19
Waste Drilling	0.21	0.17
Waste Blasting	0.33	0.27
Waste Load, Haul and Dump	1.12	0.91
Total (US$/dmt)		**1.89**
Re-handling Cost - Stockpile	0.87	-

(1) Notes: Costs and tonnes of pre-stripping period not included. Diesel cost included.

The following sections provide the basis and inputs of the operating cost estimate.

Owner Costs

The owner costs consist of the following items:

- Amarillo's mining workforce;
- Dewatering (power and services);
- Material re-handling, not related to production;
- Communication systems;
- Light vehicles;
- Survey services;
- Others.

These costs were estimated on an annual basis based on quotations as presented in Table 21-14. The composition of the Amarillo's workforce is shown in Table 16-6.

From 2030 onwards the primary crusher is only fed by the re-handling of the low-grade stockpile. Thus, it was assumed that no mining owner costs are incurred from this year as the ore re-handling activity can be easily managed by the processing plant management.

Table 21-14: Mining Operating Cost Summary

Item	Annual Cost (US$k)
Owner Workforce	1,163
Hardware and Software Licenses	11
Dewatering Power	215
Dewatering Services	86
Materials Re-handling	294
Communication	3
Pick-ups	61
Survey Services	43
Grade Control	541
Other Expenses	206
Sub Total	**2,624**
Contingencies	32
Total	**2,656**

Diesel

Amarillo is responsible for supplying diesel to the mining contractors. The diesel cost is US$0.92/L and includes the entire refuelling infrastructure provided by a specialized contractor. The diesel cost is net of 9.25% Brazilian taxes (PIS and COFINS), as it is subject to tax recovery by Amarillo.

The diesel consumption by operating activity was estimated by mining contractors based on the DFS mine plan. The total diesel consumption of 46.3ML was estimated for the LoM with an annual peak of 7.05ML.

Drilling and Blasting

The unit operating costs related to drilling and blasting activities are shown in Table 21-15, including the diesel cost. The blasting services encompass emulsion pumping, explosive accessories and charging services. One hundred per cent of the fresh rock and 30% of saprolite will be drilled and blasted.

Table 21-15: Mining Operating Cost Summary

Item	Ore Rock		Waste Rock	
#	Cost US$/dmt	Diesel L/dmt	Cost US$/dmt	Diesel L/dmt
Drilling	0.48	0.07	0.22	0.03
Blasting	0.33	0.00	0.34	0.00

Load, Haul, Dump and Re-handling

The unit cost of load, haul and dump are summarized in Table 21-16, including the diesel cost. The mobilisation, demobilisation, grade control drill rig and dispatch management system are part of the mining contractor package. A US$0.87/dmt re-handling cost was estimated, including the diesel cost.

Table 21-16: Load, Haul and Dump Unit Operating Costs

Distance (m)		Soil		Backfill		Saprolite		Fresh Rock	
From	To	Cost US$/dmt	Diesel L/dmt	Cost US$/dmt	Diesel L/dmt	Cost US$/dmt	Diesel L/dmt	Cost US$/dmt	Diesel L/dmt
0	500	-	-	1.09	0.30	0.99	0.27	0.83	0.23
5,00	1,000	1.21	0.33	1.17	0.32	1.06	0.29	0.89	0.24
1,000	1,500	1.27	0.36	1.23	0.34	1.12	0.31	0.94	0.26
1,500	2,000	1.34	0.38	1.29	0.36	1.17	0.33	0.99	0.27
2,000	2,500	1.40	0.40	1.35	0.38	1.23	0.34	1.03	0.29
2,500	3,000	1.46	0.42	1.41	0.40	1.28	0.36	1.08	0.30
3,000	3,500	1.54	0.44	1.48	0.42	1.34	0.38	1.13	0.32
3,500	4,000	-	-	-	-	-	-	1.19	0.34
4,000	4,500	-	-	-	-	-	-	1.24	0.35
4,500	5,000	-	-	-	-	-	-	1.30	0.37
5,000	5,500	-	-	-	-	-	-	1.35	0.38
5,500	6,000	-	-	-	-	-	-	1.41	0.40

22 ECONOMIC ANALYSIS

22.1 Introduction

The overall economics of the Project have been evaluated using conventional Discounted Cash Flow (“**DCF**”) methods. The production schedules, capital expenditures and operating costs have previously been discussed in this Report. The following key parameters were integral to the construction of the cashflow model and the economic results:

- A Base Case gold price of US$1,600/oz was used;
- The cost estimate was based on an exchange rate of R$5.05/US$;
- The economic analysis was based on 100% equity financing with no debt component; and
- All revenues and costs are reported in ‘real’ constant U.S$ terms without escalation.

The economic analysis presented in this section contains forward-looking information with regards to the Mineral Reserve estimates, commodity prices, exchange rates, proposed mine production plan, projected recovery rates and processing costs, infrastructure construction costs and schedule. Conservative assumptions regarding withholding tax on private royalties have been included, which could be subject to reductions (from 4.25% to 3.75%). A tax structure has been included within the economic analysis, which follows current Brazilian Laws and includes conservative assumptions regarding the usage of tax-credits. The results of the economic analysis are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here.

In addition to the financial evaluation performed on the Base Case (US$1,600/oz gold price), a scenario using a gold price of US$1,450/oz was developed for the purpose of confirming the economic viability of the Mineral Reserve.

22.2 Cashflow Assumptions

22.2.1 Mine Production Sequence

Table 16-2 and Table 16-3 shown previously in this Report outline the proposed Mining and Processing Schedules.

22.2.2 Capital and Operating Costs

The capital and operating costs used in the financial model are shown and detailed in Section 21.

22.2.3 Metallurgical Recovery

The following metallurgical recoveries were used in the modelling:

- Recovery (%): [(Au - (0.0854 x Au0.8718 + 0.023)) / Au] x 100%; and
- Average LoM Recovery: 89.9% Au recovery

22.2.4 Metal Prices and Net Revenue

The long-term gold price incorporated into the cashflow model was US$1,600/oz. Table 22-1 shows the calculation of net revenue in US$/oz, after adjustment for refining charges and royalties.

Table 22-1: Net Revenue Calculation

Revenue and Selling Costs	DFS Case	Basis
Gold price	US$1,600/oz	Amarillo
Refining, transportation, insurance	US$12.0/oz	Amarillo
Gold price net of refining, transport, and insurance	US$1,588/oz	Calculation
Average LoM Royalties – Landowner (0.75% of gold price)	US$6.60/oz	Amarillo
Royalties – CFEM Federal Tax (1.5% of gold price)	US$24.00/oz	Amarillo
Royalties – Royal Gold and Franco-Nevada (4.25% of gold price net of refining, transport, and insurance)	US$67.49/oz	Amarillo
Average LoM net revenue	US$1,490/oz	Calculation

22.2.5 Salvage Value

No salvage value was used in the financial and cost modelling.

22.2.6 Taxes

A Brazilian income tax rate of 25% was used. Provision for the 9% Brazilian Social Contribution Tax (“**CSLL**”) was also used, bringing the effective tax rate to 34%.

22.2.7 Depreciation

The capital and sustaining expenditures, including the development costs, have been fully depreciated or amortized on a unit production basis over the LoM. Expenditures for land acquisition have not been depreciated.

22.2.8 Working Capital

The LoM working capital was calculated using the following assumptions defined by Amarillo:

- Inventory: 15 days;
- Receivable: 3 days; and
- Payable: 60 days.

The LoM working capital has a positive effect in DCF (@ 5% discount rate) of US$0.04m.

In addition, Amarillo has estimated US$8.89m for working capital to be included in the initial capital expenditure. This approximately accounts for the two-month ramp-up period with 50% and 75% production. This amount has not been included in the discounted cash flow calculation as it is already in there as operating costs.

22.2.9 Mine Closure

The mine closure cost of US$20.0m estimated by Amarillo was used in the financial and cost modelling.

22.3 Discounted Cashflow Result

Table 22-2 is a summary of the LoM physicals inputs.

Table 22-2: Net Revenue Calculation

LoM physicals	Units	Value
Material movement LoM		
Total material movement (TMM)	Mt dry	129.5
Total waste movement	Mt dry	105.7
Total ore mined	Mt dry	23.8
Average ore grade	g/t	1.18
Processing LoM		
Ore processed	Mt dry	23.8
Ore grade processed	g/t	1.18
Average recovery	%	89.9%
Gold produced	koz	811

The LoM financials from the financial model are shown in Table 22-4 and Table 22-5. The accumulated undiscounted cash flow is US$262.8m. The Net Present Value ("**NPV**") @ 5% annual discount rate is US$154.6m and the resulting internal rate of return (IRR) is 19%. The payback period based on the undiscounted cash flow is 3 years from the start-up date.

In addition to the financial evaluation performed on the Base Case (US$1,600/oz gold price), a scenario with a gold price of US$1,450/oz was developed in line with the economic cut-off calculations of the Mineral Reserve (Section 15.7). A positive US$186.8m undiscounted free cash flow over the LoM was determined confirming the economic viability of the mineable inventories.

Table 22-3: LoM Financials (Undiscounted)

LoM Financials (Undiscounted)	US$1,600/oz USk	US$1,450oz US$k
Gross Revenue	1,297,635	1,175,982
Operating Costs	-549,036	-549,036
Refining, Transportation, Insurance	-9,732	-9,732
Royalties	-79,553	-72,056
Mine Closure Costs	-20,000	-20,000
EBITDA	**639,314**	**525,158**
Depreciation	-239,560	-239,560
Less Tax	-179,510	-141,337
Net Profit	**220,244**	**144,260**
Depreciation	239,560	239,560
Plus Tax Benefit	40,347	40,347
Salvage Value	0	0
Less Initial Capital Costs	-194,017	-194,017
Less Sustaining Costs	-43,352	-43,352
Less Δ Working Capital	0	0
Free Cash Flow	**262,782**	**186,798**

The results of the DCF analysis for the Base Case (US$1,600/oz gold price) are shown in Table 22-4.

Table 22-4: DCF Results for the Base Case

Results	Annual discount rate			
	5%	**8%**	**10%**	**15%**
Pre-tax NPV (US$m)	**269.6**	204.9	169.1	99.6
Pre-tax IRR (%)	**28%**	28%	28%	28%
After-tax NPV (US$m)	**154.6**	107.5	81.6	31.4
After-tax IRR (%)	**19%**	19%	19%	19%
Tax rate (%)	**34%**	34%	34%	34%

Table 22-5: LoM Financials Year by Year (Base Case)

LOM Financials US$m	Total A(1)	Total B(1)	Y00 2021	Y01 2022	Y02 2023	Y03 2024	Y04 2025	Y05 2026	Y06 2027	Y07 2028	Y08 2029	Y09 2030	Y10 2031	Y11 2032	Y12 2033	Y13-Y17 2034-2038
Gross Revenue	965.5	1,297.6	0.0	0.0	0.0	147.8	173.5	162.9	169.9	126.8	141.2	137.9	161.0	51.3	25.4	0.0
Operating Costs	(408.3)	(549.0)	0.0	0.0	0.0	(64.7)	(68.3)	(68.9)	(69.1)	(70.5)	(62.3)	(49.5)	(42.9)	(33.9)	(19.1)	0.0
Refining, transportation, insurance	(7.2)	(9.7)	0.0	0.0	0.0	(1.1)	(1.3)	(1.2)	(1.3)	(1.0)	(1.1)	(1.0)	(1.2)	(0.4)	(0.2)	0.0
Royalties (Gross Revenue) - Land owner	(4.0)	(5.4)	0.0	0.0	0.0	(0.6)	(0.7)	(0.7)	(0.7)	(0.5)	(0.6)	(0.6)	(0.7)	(0.2)	(0.1)	0.0
Royalties (Gross Revenue) - CFEM Federal Tax	(14.5)	(19.5)	0.0	0.0	0.0	(2.2)	(2.6)	(2.4)	(2.5)	(1.9)	(2.1)	(2.1)	(2.4)	(0.8)	(0.4)	0.0

LOM Financials	Total	Total	Y00	Y01	Y02	Y03	Y04	Y05	Y06	Y07	Y08	Y09	Y10	Y11	Y12	Y13-Y17
US$m	A(1)	B(1)	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034-2038
Royalties (Gross Revenue - Refining) - Royal Gold and Franco-Nevada	(40.7)	(54.7)	0.0	0.0	0.0	(6.2)	(7.3)	(6.9)	(7.2)	(5.3)	(6.0)	(5.8)	(6.8)	(2.2)	(1.1)	0.0
Mine Closure Costs	(10.9)	(20.0)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	(20.0)
EBITDA	**479.9**	**639.3**	**0.0**	**0.0**	**0.0**	**72.9**	**93.2**	**82.8**	**89.1**	**47.6**	**69.2**	**78.9**	**107.0**	**13.9**	**4.5**	**(20.0)**
Depreciation	(171.8)	(239.6)	0.0	0.0	0.0	(22.8)	(23.4)	(24.0)	(24.7)	(24.6)	(24.6)	(24.5)	(24.3)	(23.4)	(23.4)	0.0
Less Tax	(133.6)	(179.5)	0.0	0.0	0.0	(15.0)	(27.2)	(24.7)	(26.7)	(12.1)	(19.4)	(22.3)	(32.1)	0.0	0.0	0.0
Net Profit	**174.5**	**220.2**	**0.0**	**0.0**	**0.0**	**35.1**	**42.6**	**34.1**	**37.7**	**10.9**	**25.2**	**32.1**	**50.6**	**(9.4)**	**(18.8)**	**(20.0)**
Depreciation	171.8	239.6	0.0	0.0	0.0	22.8	23.4	24.0	24.7	24.6	24.6	24.5	24.3	23.4	23.4	0.0
Plus Tax Benefit	30.4	40.3	0.0	0.2	0.1	5.1	5.2	5.2	5.2	5.3	4.8	3.9	3.4	1.3	0.7	0.0
Salvage Value	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Less Initial Capital Costs	(185.8)	(194.0)	0.0	(134.7)	(59.4)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Less Sustaining Costs	(36.3)	(43.4)	0.0	(2.0)	(2.0)	(13.0)	(8.0)	(7.7)	(9.2)	(0.8)	(0.4)	(0.3)	0.0	0.0	0.0	0.0
Less Δ Working Capital	0.0	0.0	0.0	0.0	0.0	0.2	(0.2)	0.1	(0.0)	0.4	(0.3)	(0.2)	(0.3)	0.7	0.2	(0.5)
Free Cash Flow	**154.6**	**262.8**	**0.0**	**(136.5)**	**(61.3)**	**50.3**	**63.0**	**55.7**	**58.4**	**40.4**	**53.9**	**59.9**	**78.1**	**15.9**	**5.4**	**(20.5)**

(1) A: Discounted cashflows at 5% real; B: Undiscounted cashflows

22.4 Sensitivity Analysis

SRK undertook a sensitivity analysis on the DCF model to check the impact on the NPV by varying the operating costs, capital costs (including sustaining) and gross revenue (gold price). The results are shown in Table 22-6 and Figure 22-1.

The Project is most sensitive to revenue, and least sensitive to capital expenditure.

Table 22-6: Sensitivity Analysis Result (Opex, Capex and Revenue)

Variance	Operating Costs	Capital Costs	Revenue
	NPV @ 5% annual discount rate (US$m)		
25%	88.8	110.7	303.5
20%	102.0	119.4	274.0
15%	115.1	128.2	244.5
10%	128.3	137.0	214.6
5%	141.5	145.8	184.6
0%	**154.6**	**154.6**	**154.6**
-5%	167.8	163.4	124.7
-10%	180.9	172.2	94.7
-15%	194.1	181.0	64.7
-20%	207.3	189.8	34.6
-25%	220.2	198.6	4.4

Figure 22-1: Sensitivity Spider Chart (Opex, Capex and Revenue)

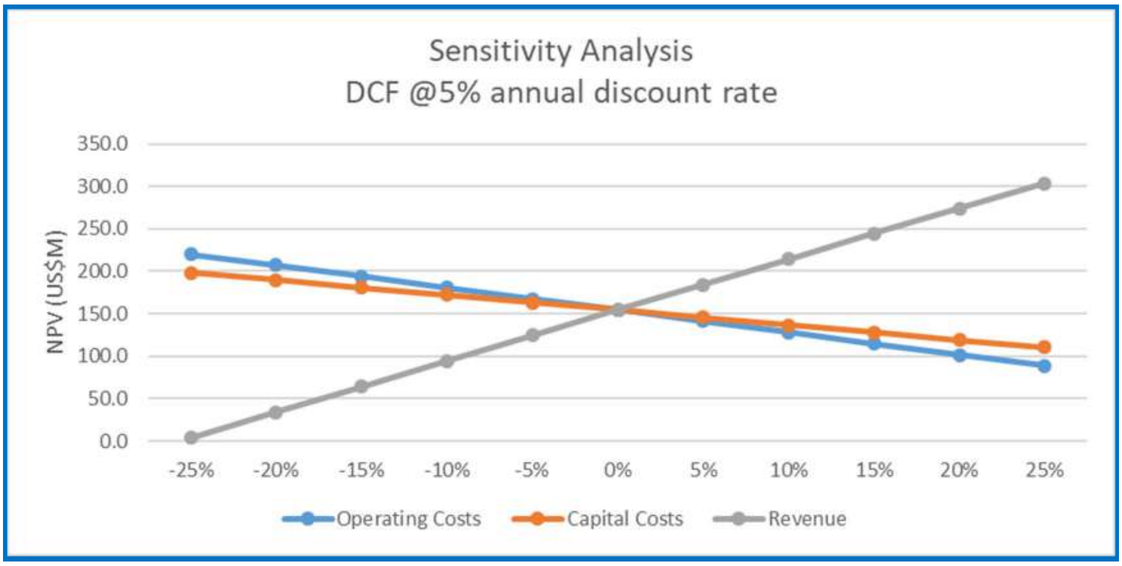


Table 22-7 and Table 22-8 show the sensitivity of various financial parameters to gold price and exchange rate variations.

Table 22-7: Sensitivities to Revenue

Gold price per ounce	1,300	1,400	1,450	1,500	**1,600**	1,700	1,800	1,900	2,000
R$ to US$	5.05	5.05	5.05	5.05	**5.05**	5.05	5.05	5.05	5.05
After-tax NPV 5% (US$M)	42.2	79.7	98.4	117.2	**154.6**	192.1	229.5	266.6	303.5
After-tax IRR	9%	13%	15%	16%	**19%**	22%	25%	28%	30%
After-tax payback (years)	5	4	4	3	**3**	3	2	2	2

Table 22-8: Sensitivities to Exchange Rate

Gold price per ounce	1,600	1,600	1,600	1,600	1,600	**1,600**	1,600	1,600	1,600
R$ to US$	3.05	3.40	3.80	4.20	4.60	**5.05**	5.40	5.80	6.20
After-tax NPV 5% (US$M)	-51.4	2.3	51.6	91.3	124.0	**154.6**	174.9	195.1	212.7
After-tax IRR	1%	5%	9%	13%	16%	**19%**	22%	24%	26%
After-tax payback (years)	7	6	5	4	3	**3**	3	2	2

The capital and operating cost estimates are based primarily on quotes by vendors (materials, supplies, equipment, and installation) and mining contractors. Most of these quotes were provided in local currency (R$), although some items are highly influenced by exchange rate variations (R$ to US$). Therefore, to simulate the sensitivities to exchange rate, it was necessary to assume the currency portion in the quotations of each area as shown in Table 22-9.

Table 22-9: Currency Portion

Area	R$ portion fixed in US$	R$ remaining portion fixed in R$
Processing	42%	58%
Mining	10%	90%
Tailings pile, waste dumps and water dam	42%	58%
Power line	23%	77%
Owner's cost	0%	100%

22.5 Comparison to Previous Studies

Table 22-10 compares the Updated DFS (2021) study with the previous reports.

Table 22-10: Comparison to Previous Studies

Category	Units	2017 SRKBR PFS	2018 SRKAU PFS	2020 SRKBR DFS	2021 SRKBR DFS Update
Exchange rate	US$ / R$	3.2	3.6	4.2	5.05
Initial capital (including initial working capital)	US$M	132.3	122.9	145.2	194.0
Sustaining capital	US$M	16.5	17.4	20.5	43.4
Total LOM capital	US$M	148.8	140.3	165.7	237.4
After-tax NPV @ 5%	US$M	178.3	244.3	183.1	154.6
After-tax IRR	%	35.2	50.8	25.1	19.4
Cash operating cost (excluding royalty & refining)	US$/oz	545	545	615	677
Cash operating cost (including royalty & refining)	US$/oz	603	633	706	787
AISC (including sustaining capital & closure)	US$/oz	627	655	738	841
Tonnes of ore processed	Mt dry	19.0	23.8	23.8	23.8
Grade of ore processed	g/t	1.63	1.42	1.18	1.18
LOM strip ratio (waste: ore)	t:t	4.5: 1	4.84: 1	4.44: 1	4.44: 1
Resources Measured & Indicated	contained koz	1,260	1,300	1,200	1,200
Resources cut-off grade	g/t	0.35	0.20	0.35	0.35
Resources average grade (M&I)	g/t	1.50	1.30	1.10	1.10
Reserves Proven & Probable	contained koz	998	1,087	902	902
Reserves cut-off grade	g/t	0.38	Variable	0.37	0.37
Reserves average grade	g/t	1.63	1.42	1.18	1.18
Gold price	US$/oz	US$1,200/oz 0.80 Revenue Factor Pit Shell US$1,200/oz Financials	US$1,300/oz 0.85 Revenue Factor Pit Shell US$1,300/oz Financials	US$1,400/oz 0.92 Revenue Factor Pit Shell US$1,400/oz Financials	US$1,450/oz 0.81 Revenue Factor Pit Shell US$1,600/oz Financials
Mining dilution & loss	%	3% dilution & 3% loss factors	3% dilution & 3% loss factors	Regularized mining model (4% dilution & 4% loss)	Regularized mining model (4% dilution & 4% loss)
Metallurgical recovery	%	92%	Variable 90.6% (LOM Average)	Variable 89.9% (LOM Average)	Variable 89.9% (LOM Average)

23 ADJACENT PROPERTIES

Numerous mineral deposits occur in the Mara Rosa region including the Posse gold deposit, the Zacarias gold-silver-barite deposit and the Chapada copper-gold deposit, in addition to numerous historic prospects and garimpos.

A detailed discussion of these various properties was provided in Hoogvliet Contract Services & Australian Exploration Field Services Pty Ltd (2011) filed at Sedar.com and to which the reader is referred.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Execution Plan

24.1.1 Introduction

Ausenco developed an implementation plan on behalf of Amarillo that addresses the Posse Gold Project schedule, engineering and construction management, procurement, logistics, construction, construction contracting, temporary facilities, project planning/execution and reporting, pre-commissioning and commissioning, and start-up/turnover.

24.1.2 Responsibilities

The Amarillo engineering team, with the support of subcontractors, will be responsible for the following activities:

- Procurement: qualification and contracting of vendors, ensuring compliance with contracting, mobilization and equipment delivery deadlines;
- Follow up on the planned implementation;
- Tracking schedule, cost and overall performance indicators;
- Report on the progress of activities through periodic reports;
- Management of construction-related activities, such as earthworks, construction sites, civil works, electromechanical assembly, health and safety aspects and impacts on the community; and
- Detailed engineering: follow up and approval of technical documents.

All contractors will be required to provide an updated organization chart, consistent with the project communication matrix.

24.1.3 Procurement plan

The scope will be divided into packages where the interfaces shall be managed by Amarillo for greater control over the development of the project and optimization of tax expenses. Technical analysis of proposals will be developed by the engineering contractor.

The procurement strategy has the following main considerations:

- Amarillo will directly manage all supply items;
- Hiring of the same company responsible for the development of detailed engineering for construction support engineering at the site;
- Hiring a medium / large construction company to carry out civil works;
- Hiring a specialized contractor for electromechanical assembly;
- The equipment will be quoted in the domestic market, and in some cases may also be quoted abroad, and potential suppliers must present complete documentation in compliance with the established requirements; and
- Commissioning and ramp-up by Amarillo together with the electromechanical assembly and engineering company.

Expediting and inspection

Expediting and inspection refers to the process of follow up on the progress of procurement packages by suppliers and the compliance with all requirements set forth in the Technical Requisitions provided by Amarillo.

The expediting plan contains the process and responsibility for expediting the supply of equipment and materials, level of quality control for each supply package, as established in the TR's and including the health, safety and environmental requirements, in compliance with Amarillo's requirements.

The inspection and testing plan will define in general terms all manufacturing and certification processes to be closely tracked by Amarillo as required. This may also include the inspection of suppliers own facilities and fabrication equipment critical to the procured items.

Criteria for Inspection of Received Equipment and Materials

The criteria for inspections of received equipment and materials at site must comply with warehouse requirements. A specific process for receiving all equipment and components for the project shall be implemented, in accordance with specific quality control requirements from Amarillo.

Logistics

A robust logistics strategy shall be developed to ensure effective delivery of materials and equipment to site, considering transport infrastructure (road, port, airport), customs clearance legislation and requirements, need of special transport (oversize/overweight) and temporary storage, among others.

Acceptance and unloading of materials and equipment at site, either under responsibility of suppliers or the erection contractor, shall follow specific rules to ensure that the operation is safe and that components are properly controlled and received free of defects and in accordance with the technical requisitions.

Contracting process

Technical Requisitions are intended to support the acquisition of special equipment and materials as well as the contracting of services. This document, to be issued by Amarillo, must constitute the Request for Proposal together with other information such as contractual drafts, confidentiality and guarantee terms, criteria for proposal submission, security requirements, environment etc., to be prepared by the areas of supplies, legal and HSE.

The main contractual models to be adopted are:

- Services: EPC and EPCM (lump sum models), Turnkey, Unit price (DBB) and fixed price;
- Equipment: Acquisition, rental or leasing; and
- Materials: Acquisition or supplied by contractor.

When contracting service providers, the procurement team must comply with Specific Procedures for Supplier Contracting. Specific rules apply for payment procedures, compliance with legal and corporate requirements as well as subcontracting.

Procurement planning

The procurement map is the main document containing all information along the procurement process, from vendor list definition until commissioning.

A specific vendor list encompassing equipment, materials and services for this project has been developed and approved by Amarillo.

An approvals framework shall be put in place to regulate authority levels for approvals, containing related positions, accountabilities and breadth, and also deadlines and processes.

Contractor and supplier access to the project premises shall be regulated under specific procedures.

The Contract Management process shall watch closely on the follow-up and monitoring of contractual obligations such as time, cost and quality, thus providing timely indications that preventive or corrective action are needed for contractors to meet performance requirements.

Each contract shall be initiated with a “Kick-off meeting”, which purpose is to clarify each party’s obligations and expectations to ensure a mutual understanding of contractual scope. It should address the main requirements contained in the technical requisition and an agreement on initial activities to ensure optimized ramp up of the contract.

The services progress and invoicing procedures shall be included in the bidding documentation to all suppliers and compliance to them confirmed in the received technical and commercial proposals. At the contractual kick-off meeting the legal representatives from Amarillo and contractor, as well as others involved in the invoicing process, shall be formally introduced and registered.

Contractual close-out

A contractual closeout procedure shall be in place to ensure that all required contractual milestones are completed before a Preliminary Handover Term is released. At its sole discretion, Amarillo may release the term even if milestones and requirements are not fully met, but then a punch list where outstanding issues are clearly listed and expected completion dates registered shall be agreed. Once all issues are finally resolved by the contractor and all project deliverables are completed Amarillo shall release the Final Handover Term. According to each contract, some of its terms may still remain in effect afterwards, particularly those related to warranties.

Demobilization

Service contractors shall present a demobilization plan to be approved by Amarillo's Supervision and Management, which must include, among other things, the demobilization schedule, waste management procedures, plans land recovery and reuse of temporary facilities.

Contract close-out term

Once all contractual obligations including warranties are fulfilled, and where defined by contract, a close out term may be signed by both parties exempting each other of any issues thereafter.

24.1.4 Quality plan

A Quality Plan will be implemented aiming to ensure compliance with quality requirements. The plan will set out policies and procedures needed to ensure that the project meets its objectives, as well as compliance with deadlines, budget and adequate levels of facilities safety, availability, operability and maintainability.

Quality Management System

The Quality Management System is structured to ensure that project coordination and other stakeholders have controlled, and reliable information related to the project.

Documentation requirements

Specific procedures will be implemented to ensure proper control of quality records as well as retention times. All contractors will be required to implement their own Quality Plans in line with the guidelines set out by the project and shall describe their specific quality management structure and programs specific for the project, specifically identifying roles and responsibilities.

Quality Management System (QMS) documentation applicable to the project

The project will develop and implement all documents and requirements for compliance with

NBR ISO 9001 related to its own processes. This will include Quality Policy and Goals, a specific Quality Plan and all supporting procedures to ensure processes related to the implementation are properly managed and controlled.

In addition, contractors will be required to implement specific Execution Plans in line with the best practices in project management, such as:

- Project Execution Plan;
- Planning and Controls Execution Plan;
- Procurement Execution Plan;
- Health, Safety and Environment Execution Plan; and
- Conditioning and Preservation Execution Plan.

Contractors will also be required to manage all records necessary to meet the requirements of the NBR ISO 9001 Standard in its latest version, legal requirements and requirements established by Amarillo.

Subcontracting of services must be previously approved by Amarillo and subcontractors will also be required to present their own quality plans and to comply with the project quality requirements as applicable to their activities.

Document control

A specific structure will be in charge of the Document control processes and requirements, such as ensuring consistency with Amarillo standards, registering all received/released documents, controlling and signalling of any outdated documentation, among other activities.

With respect to technical documents in particular, a clear process will ensure that before being distributed to the construction teams they are properly verified and released by qualified professionals and any construction or assembly is executed based only on those with status "APPROVED FOR CONSTRUCTION".

Any change to designs developed at site will be documented and registered by the construction support engineering team. All of such changes will be duly controlled so that related documents are properly updated and released in "AS BUILT" status before project completion, for later reference by the maintenance and operations teams.

Quality control records will be duly filed for at least 5 years to provide evidence of compliance with the project quality procedures and requirements. As a contractual requirement for close out of any contract, all quality control records related to delivery of services or equipment shall be incorporated to the relevant Data Books.

24.1.5 Risk management

A risk management process will be implemented in the Posse Gold project with the main purpose of reduce the level of uncertainty related to the project as well as minimizing possible impacts of negative events that may occur and maximizing the outcomes of positive risks (opportunities).

Risk management is responsibility of all project areas, which shall provide proper input and participation on the identification of risks, performance of qualitative and quantitative assessments and development and implementation of risk response plans.

A risk assessment committee made up of senior professionals will regularly assess project status against any previously identified or emerging risk factors and take required actions to preserve the project objectives and best interests of Amarillo.

Risk Breakdown Structure (RBS)

A risk breakdown structure has been developed so that all risks are categorized and classified in relation to the main aspects of the project that may be impacted. Table 24 1 presents the RBS at current status.

Table 24-1: Project risk breakdown structure

RBS LEVEL 1	RBS LEVEL 2	RBS LEVEL 3
0. All sources of risks related to the project	1. Technical Risk	1.1 Scope
		1.2 Requirements
		1.3 Technology
		1.4 Quality
		1.5 Designs
	2. Management Risk	2.1 Project Management
		2.2 Organization
		2.3 Communication
		2.4 Resources
	3. Commercial Risk	3.1 Contractual
		3.2 Service contractors
		3.3 Client stability
		3.4 Partnerships
		3.5 Internal procurement
	4. External Risk	4.1 Legal
		4.2 Exchange rates
		4.3 Licensing
		4.4 Market

Description of the RBS risk categories:

- **Technical Risk:** all risks related to civil, electrical, mechanical and hydraulic design, equipment affecting capacity, term and cost;
- **Management Risk:** referring to the foundations on which the management of the project is structured: finance, people, logistics, project management, communication, internal organization;
- **Commercial Risk:** risks related to the contract, suppliers and service providers, partnerships; and
- **External Risk:** all issues that not dependent directly on owner and contractors, such as legislation, approval and regulation, exchange rates, etc.

24.1.6 Communication plan

A communication plan will be implemented to ensure all relevant stakeholders have access to adequate information related to the project.

Communication processes will be structured around internal and external stakeholders, with specific strategies developed for each one. Special care will be taken with local communities surrounding the site to ensure the project progress remains unhindered by any dissatisfaction or undesirable event in such areas and the project's "*social license*" is strengthened.

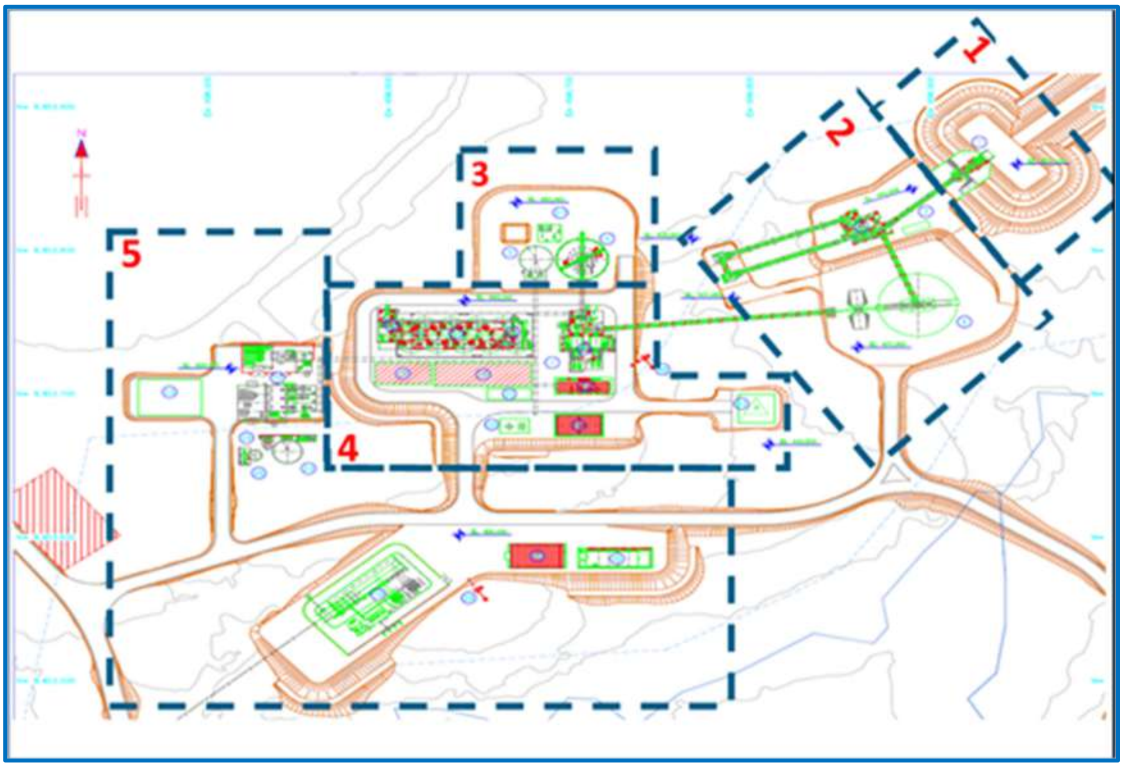
24.1.7 Communication plan

A preliminary construction plan has been developed, to be further detailed and validated together with the construction and erection contractors prior to the commencement of the works.

Construction strategy

The construction strategy has been conceived to optimize the productivity and logistics of the work fronts and, after initial activities of land clearing and earthmoving, the sequence of works is according to the numbered areas above, starting with the primary crushing and RoM stockpile (Figure 24-1).

Figure 24-1: Construction strategy



Implementation methodology

The implementation methodology has been developed in order to optimize construction performance, ensuring that all required materials, teams and equipment are available at the required work fronts in accordance with the construction plan.

All construction disciplines (civil, structural, mechanical, electrical, automation) will follow execution and technological control requirements so that quality is consistent with Amarillo's minimum standards.

24.1.8 Safety plan

In line with Amarillo's safety at work standards and policies, a Safety Plan was established in order to set general guidelines for the mobilization and activities of contractors on the construction site, as well as the requirements for the health, safety and environment management system ("**HSEC**").

This plan sets comprehensive standards based not only on Amarillo's but also on legal requirements for contractor mobilization, erection and maintenance of temporary facilities and housing for employees. It also defines mandated procedures for mobilization, training, performance of activities at site – with particular focus on those involving increased personal risk, demobilization and overall HSEC management. The environmental aspects are addressed as well, through stringent requirements for waste and effluent management, both industrial and domestic.

A specific set of KPIs and targets have been defined for HSEC related aspects of the project, as indicated in Table 24-2.

Table 24-2: Safety KPIs

KPI	Target	Estimated target	Global target
TRIFR: Total Recordable Injury Frequency Rate	<2.0	<0.8	0
LTIFR: Lost Time Injury Frequency Rate	<1.0	<0.5	0
SPIFR: Severe Potential Injury Frequency Rate	<2.0	<1.0	0
Mean duration rate of lost work-days due to LTI	<5 days	<2 days	0
Significant environmental incidents	0	0	0

KPI	Target	Estimated target	Global target
Number of Environmental Non-conformities	0	0	0
Incident reports	24 hrs	12 hrs	Immediate
Behavior based safety observation	2 ps/pw	2ps/pw	>2ps/pw
HSEC inspections	16	16	> 20
HSEC Management inspections	2	4	> 5
Workshops – Zero Harm	1	2	4
Contractor audits	1 pm	1 pm	>2
HSEC assurance reports	Q1,2,3,4	Q1,2,3,4	Q1,2,3,4
HSEC training	24 hrs pp	24 hrs pp	> 24 hrs pp

24.1.9 Planning and Control

The execution planning is guided by the scope of the work, as well as the action plan and control and monitoring plan. Such action plan is designed from the definition of work packages, estimation of execution times and corresponding resources, activity networks finally consolidated in the project schedule.

Each large work package is divided into actions, activities and tasks, detailing at managerial level the scope of the work package. For other levels and better follow up of the work, the details of each proposed activity should be considered.

The activity times were estimated based on the similarity and expertise of the team involved and reconciled with information from suppliers regarding the time of manufacture and assembly of the main equipment of the work, all defined in 8 hours per day working days

For the resources estimation Amarillo considered information from a database with historical productivity rates for services and equivalent supplies, as well as on productivity information from bidders.

Based on this information a detailed schedule may be constructed as management basis for the enterprise.

A Monitoring and Control Plan was developed containing all activities required for regular and consistent execution performance tracking and assessment of results. Such results will be verified based on:

- Matrix of expected results;
- Monitoring and Evaluation Spreadsheet (table of KPIs);
- Risk analysis (risk identification).

24.1.10 Implementation schedule

Figure 24-3 shows a summary level implementation schedule. The project plan has been developed for the duration of 2 years.

The activities related to grinding mills define the implementation critical path, therefore they will have special focus from the management and execution teams to ensure project targets are achieved (Figure 24-4).

Project "S" Curve

The "S" curve (Figure 24-2) presents the planned progress distributed along the project timeframe and the cumulative progress in the period, being also the main tool for tracking and controlling actual progress against plan.

Figure 24-2: Implementation "S" Curve

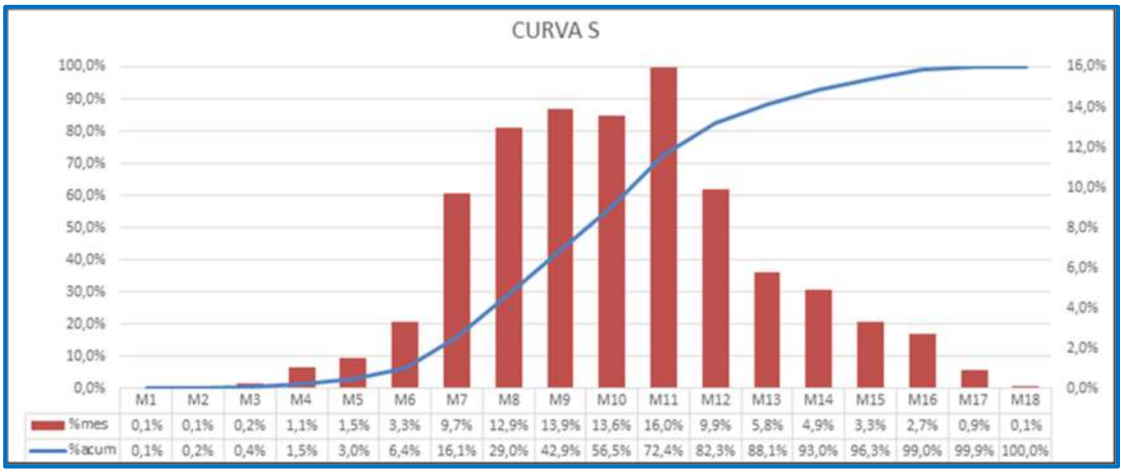
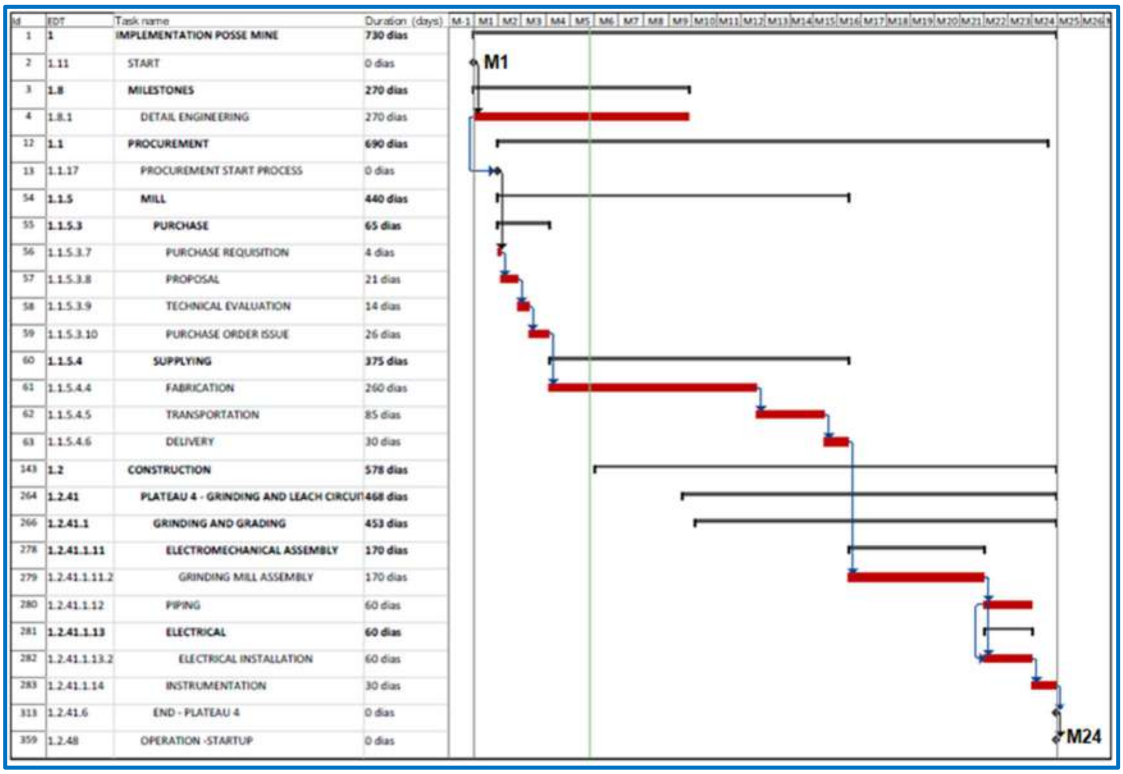


Figure 24-3: Implementation Summary Schedule

Figure 24-4: Implementation Critical Path



24.1.11 Commissioning plan and ramp-up

As the project approaches construction completion management focus will gradually shift towards commissioning and ramp-up activities. To ensure a safe and smooth start of the plant systems and a short ramp-up time until nominal capacity, thus maximizing project value, a detailed plan for these activities will be developed during detailed design. A specific team for commissioning will be formed as appropriate to ensure required levels of expertise are allocated to these activities.

Commissioning activities detailed plan will follow the sequence of six steps below:

- **Verification of Plant and Equipment (C0):** performed by the erection contractor and verified by Amarillo, to ensure the field assembly of mechanical and electrical equipment comply with the designs and construction standards. No equipment is powered up at this time;
- **Dry commissioning (C1):** the completion of all electromechanical erection activities indicates the completion of this phase;
- **Cold commissioning (C2):** equipment are powered up and run with inert fluids, such as water or air. Once all systems are stabilized this phase may be completed and the responsibilities of the electromechanical erection contractor are finalised;
- **Hot commissioning (with load) (C3):** this phase is conducted by Amarillo's commissioning team, where the plant is fed with raw material;
- **Performance tests (C4):** – where applicable performance tests will be executed under supervision of representatives of the relevant suppliers, and any performance deviations from contractual values identified at this time shall be addressed with their support;
- **Final acceptance and start of operations (C5):** at this phase Amarillo provides final acceptance of the plant and all installed equipment to contractors and suppliers, project

team responsibilities are completed, and the plant is handed over to the operations team; and

- **Ramp-up (C6):** operations bring plant output to nominal capacity according to a specific ramp-up schedule.

24.2 Implementation Risk Analysis

24.2.1 Risk Evaluation Summary

This section presents the main findings of the risk workshop held at Belo Horizonte in 14th January 2020 as part of the DSF review of Posse Gold Project. A group from Amarillo (including consultants) and Ausenco gathered within the purpose of assess and identify risks and define proper response plans for these unwanted events. The workshop was focused on the project implementation phase. As a result, a total of 41 risks formed the project risk profile.

The risk exposure or remaining threats that could not be treated or addressed by both preventative or mitigation actions and controls can be considered moderate, with a total of 3 significant risks. In contrast, 3 opportunities were raised and must be explored to leverage their potential results and benefits to project plan.

Worries regarding the possible delay to obtain the installation license (that includes all structures, except the transmission line, that is being treated apart) and also the delay to approve funds for project implementation were classified as highest risks, followed by the mills long lead time and possible accidents involving fatalities during construction. Other important risks with substantial impact on cost and schedule are also in place and of considerable importance.

Control and mitigation actions were described in terms of what is possible or is already in place that could act as a barrier for risk occurrence. Appropriate follow-up is imperative to guarantee the response plan will be effective and generate the risk profile severity reduction.

In addition to the January's risk analysis Amarillo organized a Hazard and Operability ("**HAZOP**") workshop to identify and evaluate issues that may represent risks to personnel and equipment of the Project.

24.2.2 Introduction

Risk assessments are a formal instrument to enhance business performance. The outcome of this exercise is critical for the project and the attention given to risk assessments shall be accordingly, to pro-actively identify, manage, monitor and review all project risks throughout project lifecycle.

This document describes the risk management processes, activities and highlights. It also describes how the associated risks are identified, evaluated and managed. Risk management activities include risk evaluations as well as response interventions that will be worked on throughout the project implementation.

Risk management defines the risk picture as a basis for decisions and reduces number and consequence of undesirable incidents by minimization of controllable risks as far as profitable and the identification and reduction of consequences of uncontrollable risks. The term risk is commonly used in a negative context. However, it covers both threats and opportunities, i.e., upsides and downsides.

Risk Management

Provides early identification of factors that adds value (opportunities), allowing for making plans for utilization of these;

- Provides early identification of factors that may decrease value, allowing early attention towards potential undesired development and avoidance of problems before they arise;
- Encourage management to look ahead;
- Enhance prioritizing and resource utilization;
- Increase budget and financial control;
- Facilitate communication with stakeholders; and
- Provides alignment and ownership of threats and opportunities.

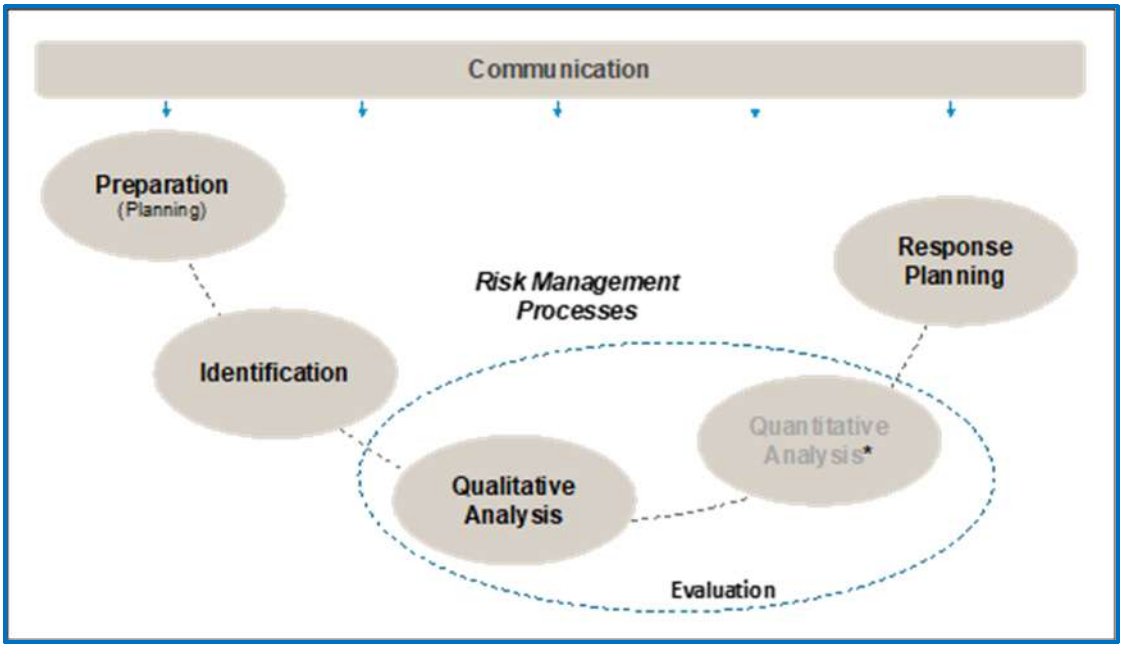
24.2.3 Risk Workshop

The risk workshop was carried out in in Belo Horizonte, on 14th January 2020, join professionals with different background and areas of expertise from Amarillo, independent consultants and Ausenco. The meeting was held and conducted by a risk management expert.

24.2.4 Risk Management Process

Risk management process was suggested by the risk management expert and based on a consolidated methodology that complains 5 different processes: preparation, identification, evaluation, response planning and communication (Figure 24-5). The Monte Carlo simulation was not used for contingency calculation; however, a "*semi-qualitative / quantitative*" risk matrix was used, giving numerical references of severity of time and cost impacts.

Figure 24-5: Risk management processes



Preparation

This process is the first and potentially the most important step in the risk management process. It defines the specific objectives of project risk management were the project articulates its objectives and defines the external and internal parameters to be considered when managing risks within a defined scope.

Identification

It aims at identifying all risks that may influence the project negatively or positively. A risk is normally only relevant during a certain time. The time horizon assessed complains a period where all events can cause an impact on the decisions to execute the project ore one of its activities or plans.

During identification, before adding a potential project risk item to the risk register, an evaluation has to be made, to ensure it is a real "risk issue" as opposed to a normal day-to-day challenge. A proposed list of origins and sources of risk is given to ensure a comprehensive scope is addressed.

Evaluation

Evaluation is a subjective analysis of the risk severity and allows the classification of risks. Through a 5 by 5 risk matrix, a risk can be described and measured by the probability and impact of its occurrence. The probable consequence of the risk (Expected Value) is the probability multiplied by the impact and both can be classified in 5 different levels (5x5 Matrix). Once this evaluation is complete, each risk shall be classified with a risk level: low, medium, significant or high as shown in Table 24-3.

Opportunities are classified using the same scale of threats, but with a positive impact. Impacts are refereed by a minimum 1-Insignificant to 5-Catastrophic and are rated in 7 distinct impact types: cost, schedule, health and safety, environment, social, legal and reputation. If there is more than one area of impact, the consequence level is defined by the highest one.

Table 24-3: 5x5 Risk Matrix (Risk Levels)

PROJECT RISK MATRIX					
IMPACT	1 - Insignificant	2 – Minor	3 – Moderate	4 – Major	5 - Catastrophic
PROBABILITY	Risk Level				
5 – Almost Certain	11 (Medium)	16 (Significant)	20 (Significant)	23 (High)	25 (High)
4 – Likely	7 (Medium)	12 (Medium)	17 (Significant)	21 (High)	24 (High)
3 – Possible	4 (Low)	8 (Medium)	13 (Significant)	18 (Significant)	22 (High)
2 – Unlikely	2 (Low)	5 (Low)	9 (Medium)	14 (Significant)	19 (Significant)
1 – Rare	1 (Low)	3 (Low)	6 (Medium)	10 (Medium)	15 (Significant)

Legend for probability levels can be found and assessed in Table 24-4

Table 24-4: Probability Scale

Probability Scale	Guidelines for Risk Matrix
5 – Almost Certain (90%)	Almost certain chance of occurrence
4 – Likely (75%)	Greater chance to occur
3 – Possible (50%)	Same chance of occurring or not
2 – Unlikely (25%)	Greater chance to not occur
1 – Rare (5%)	Remote chance of occurrence

Response Planning

Risk response planning is the determination of intervention actions, treatment and mitigation of the identified risks, in order to maximize the positive impacts, minimize potential negative impacts, and ensure the achievement of the expected result. In this process, the responsibility of risk owners is established for the definition of the answers and the next specific actions; quantifying resource needs, predicting the duration and reporting progress against the planned action.

Risk response planning shall also establish the basis for cost provision when mitigation actions entail expenses, in which case the active risks shall be linked to the budget as contingency values.

Guidance on how to act and define the response plans is given in Table 24-5.

Table 24-5: Risk Response Strategies

Risk Rating	Risk Level	Guidelines for Risk Response Planning
21 to 25	High	A high risk exists that management's objectives may not be achieved. Appropriate mitigation strategy to be devised immediately.
13 to 20	Significant	A significant risk exists that management's objectives may not be achieved. Appropriate mitigation strategy to be devised as soon as possible.

Risk Rating	Risk Level	Guidelines for Risk Response Planning
6 to 12	Medium	A moderate risk exists that management's objectives may not be achieved. Appropriate mitigation strategy to be devised as part of the normal management process.
1 to 5	Low	A low risk exists that management's objectives may not be achieved. Monitor risk, no further mitigation required.

It is important to guarantee that all the risks where a proposed mitigation action was put in place must be re-evaluated in terms of probability and impact. This result reflects the post-mitigation scenario of the risk severity.

Communication

Communicate the risks and actions to the project team, management and other stakeholders throughout the project. Top risks shall be included in project reports. Key risks shall be followed-up on a regular basis.

24.2.5 Risk Profile

A total of 41 risks were identified and assessed. From this number, 38 have negative impacts on project objectives (threats) and 3 opportunities were raised. The areas that concentrate the great number of risk events are showed in Table 24-6.

Table 24-6: Risk by Area

Area	Risk Events		
	Threats	Opportunities	Total
Engineering and Construction	7	1	8
Financial	3	-	3
Human Resources	1	-	1
Legal e Taxes	4	1	5
Licensing and Permitting	3	-	3
Mine	6	-	6
Procurement	3	1	4
Safety, Health and Environment	7	-	7
Social and Communities	4	-	4

The number of risks plotted on Risk Matrix according to the severity can be found in Figure 24-6. This distribution shows a moderate risk exposure, with the majority of the events out of the critical positions.

Figure 24-6: Risk Levels

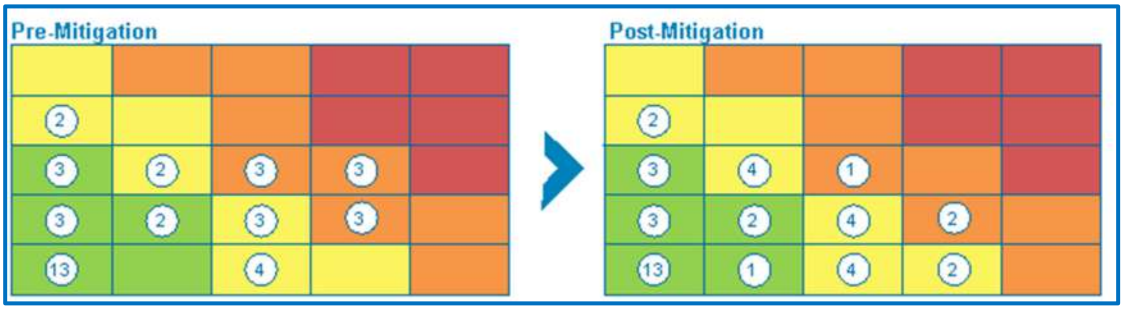


Table 24-7 presents the Top 10 Risks of the implementation phase besides the appropriate mitigation required to reduce their impact.

Table 24-7: Risk by Area

No	Description	Post-Mitigation Level	Main Actions	Responsible
1	Delay to obtain the Installation License	14 (Significant)	- Hire specialized company to conduct the licensing process; - Sign-off Letter of Intent with governor and secretaries directly involved on licensing process; - Detailed project presentation for the Environmental Agency; - Prompt answers to eventual Environmental Agency requests.	Amarillo: Arão Portugal
35	Sinergy between contractors / packages *(Opportunity)*	13 (Significant	- Hire only "Class A" companies; - Management / diligence; - Define and execute the better contract type for each scope.	Amarillo: Arão Portugal
28	Accidents during construction	10 (Medium)	- Define and elaborate specific procedures for each risk situation; - Hire only "Class A" companies; - Define programs for employees' awareness.	Amarillo: Arão Portugal
4	Accesses problems to transmission line right of way	9 (Medium)	- Risk being followed / no mitigation action defined for instance(1).	Amarillo: Arão Portugal
16	Reduce lead time of critical equipment *(Opportunity)*	9 (Medium)	- Risk being followed / no mitigation action defined for instance(1).	Amarillo: Arão Portugal

No	Description	Post-Mitigation Level	Main Actions	Responsible
27	High traffic on federal, state, and local roads	9 (Medium)	- Hire dedicated companies for transportation; - Define programs for employees' awareness; - Define and elaborate specific procedures.	Amarillo: Arão Portugal
37	Delay in commercial production statement	9 (Medium)	- Risk being followed / no mitigation action defined for instance[(1)].	Amarillo: Frank Baker
20	Delay to acquire lands necessary	8 (Medium)	- Pay greater values to guarantee the accesses (without legal actions) - contingency must contain an expected amount for it.	Amarillo: Arão Portugal

[(1)] According to the Risk Methodology, the mitigation for a medium risk can be devised as part of the normal management process

24.2.6 Conclusion

The identification and evaluation of the risks of Posse Gold Project implementation shows that only one significant risk may threat it. The greater impacts are bringing timing consequences that also trigger potential cost overruns. Uncertainties regarding the outstanding installation license approval, although estimated in approximately three months in the risk evaluation, involves complex scenarios, potentially out of Amarillo's actions coverage, and depends on a list of primary concessions or permits like vegetation suppression authorization. A close contact and follow-up with the environmental agency are crucial to guarantee that this process will not affect the Project schedule more than the suggested risk scene.

A second group of events, for now classified as medium risks, are of considerable importance, e.g., the risk of not procure all the lands necessary for construction and operation. Further efforts must be put in place to manage those risks with the severity level they present for instance.

Opportunities raised must be pursued to leverage both time and cost potential impacts. The synergy between contractors during construction was considered significant and appropriate negotiations, decisions about contract types and packaging strategies were listed during response planning.

The completion of the risk management process is mandatory for Project success. Additional assessments must be conducted to follow-up the implementation and effectiveness of controls and mitigation actions suggested. Complementary and specific risk assessments should identify new risks, as per project gains maturity and more detailed information is available.

25 INTERPRETATION AND CONCLUSIONS

25.1 Mineral Processing and Metallurgical Testing

The test work described herein has provided support for the proposed flowsheet to be applied at Mara Rosa and is considered adequate to take into process design. The flowsheet being to crush, grind, leach at 53µm for 36 hours at a pH of 12.0 at anticipated temperatures of +35°C generated as a consequence of grinding effort. The work has shown the carbon characteristics remain in the range typical of the industry, even though elevated pH is present. The work has also shown that SO_2/air cyanide detoxification is applicable using reagent doses and residence times again typical of the gold industry.

To reduce capital cost, the decision to take the tailings thickener out of the flowsheet has been made. Filtration testing at a nominal pulp density of 40% and 50% solids has shown filtered solids can be generated at moisture contents that will allow handling and placement. Press type filter technologies appearing the most appropriate.

The samples used in the test work have been sourced from a large number of drill holes and from varying depths along strike. The basic work (both earlier work by Coffey and latter work managed by Amarillo directly) to define the flowsheet has been conducted on a number of composites suggesting "average" or "typical" performance will provide high leach extractions in the 90% range. As the test work programs have progressed, and as test work control has improved, the Locality Composites tested have provided very consistent results in both extraction outcomes and reagent demands. This lack of variability suggests the Mara Rosa material can be expected to provide consistent leach extractions in the 90% range and also supports adequate coverage of the deposit by the samples selected. That is sensitivity to sample location is minor and is not a key driver with regard to the metallurgical responses.

There do not appear to be any deleterious elements or compounds present. An exception may be considered to be the presence of auriferous tellurides themselves. However, as the flowsheet has provided high leach extractions, these tellurides are no longer considered deleterious. The extractions achieved are high even by typical free milling ores in this head grade range.

25.2 Geology and Mineral Resource Estimate

The Posse Deposit is hosted by a mylonitic shear hosted zone in a high greenschist to low amphibolite metamorphic terrain. The ore body strikes NE-SW and dips about 50o to the NW. On average, the ore body is about 30m wide. Alteration is dominated by silicification, sericitization, K- feldspar flooding and pyritisation. Gold is positively correlated with the intensity of silicification and total sulphide content and occurs as 10-100 micron sized particles along the margins of silicates and in association with pyrite (FeS_2) and frohbergite ($FeTe_2$).

The Mineral Resource, first reported as part of the 2020 DFS, reported in this document, has established a Mineral Resource of around 32Mt containing around 1.2MozAu, at a grade of 1.10g/tAu, above a cut-off grade of 0.35 g/t in the Measured and Indicated Mineral Resource categories. A further 100kt containing 1.7kozAu, at a grade of 0.52g/tAu, above a cut-off of 0.35g/tAu has been classified as Inferred Mineral Resource.

The Mineral Resource includes Measured, Indicated and Inferred Mineral; Resource categories. Drilling completed in 2019 and reported as part of this report has significantly increased the confidence in the current mineral resource estimate compared to that reported in 2018. The resource has been extensively tested as part of a risk review, Section 12.7, this work suggests it is still appropriate to use the resource reported as part of the 2020 DFS as the current mineral resource for the Posse deposit. Appropriate recommendations to improve the geological understanding of the deposit and future resource estimates are made in Sections 2.16.2 and 26.2.

Table 25-1: Changes in Mineral Resource 2018 against 2020[1]

Category	Changes in resource 2018 to 2020 2018 % of resource	2020 % of resource	Change in Ounces (kozAu)
Measured	35	43	-50
Indicated	44	53	-70
Inferred	20	4	-328

(1) Due to rounding numbers may not sum.

Note that the decrease in contained troy ounces is related to a change in cut-off grade used from 0.20g/tAu in 2018 to 0.35g/tAu in this CPR. Additionally, the nominal pit shape has imposed a reasonably tight constraint on the material classified as part of the Mineral Resource.

Comparing the 2018 Mineral Resource with the current estimate at a 0.20g/t cut-off without the pit constraint shows a 30kozAu increase in Measured Mineral Resource ounces, and a 125kozAu increase in Indicated Mineral Resource ounces. The Inferred Mineral Resource ounces show a 300kozAu decrease.

The opinion of AEFS is that the character of the Mara Rosa Property, the Posse Deposit and the Mineral Resource Estimate reported herein is appropriate to support the continued development of the Posse Gold Project and valuations which may be derived from the current knowledge of the project.

25.3 Mining and Mineral Reserve Estimate

It is SRK's opinion that the Mineral Reserve estimation is compliant with the NI 43-101 standards.

To convert Mineral Resources to Mineral Reserves, consideration was given to forecasts and estimates of gold price, metallurgical recovery, mining dilution and ore loss factors, royalties and costs associated to mining, processing, overhead, refining and logistics. Since the completion of the DFS 2020, some of these parameters were updated to reflect more accurately the current economic conditions of the Project, including:

- Long term gold price;
- Processing operating costs;
- Mining operating costs;
- G&A costs; and
- Project implementation schedule.

SRK verified the effect of these changes on the economic cut-off grades and pit design. No material impact was noted. Therefore, the Mineral Reserve estimated in the DFS 2020 remains unchanged. Specifically, the Mineral Reserve estimated in 2020 reached 23.8Mt (dry) at an average grade of 1.18g/tAu. The detailed breakdown of the Mineral Reserve is presented in Table 15-1. This Mineral Reserve is estimated on the basis of currently available information. The Reserve classification reflects the level of accuracy of the updated DFS.

A sensitivity analysis was completed to verify the impact on the mining inventories. The results indicate that the Project is resilient to negative variations of 10% in the mining and process costs, and 5 percentage points in the metallurgical recovery.

The pit design is based on the pit slope angles recommended by a geotechnical study completed by Coffey in 2013. This study is based on sparse geotechnical data with ATV (acoustic televiewer) logging limited to six boreholes, of which only two are drilled perpendicular to the mineralized schist and continue into the footwall. Since 2013, no additional geotechnical data collection, testing and studies have been undertaken. Further investigation is necessary to confirm the pit slope angles or suggest new angles.

The mine schedule achieved a production target of 2.5Mtpa with a maximum annual rock movement (ore and waste) of 20.0Mtpa. A variable cut-off grade strategy was implemented by which the high grades were mined in the early periods while leaving the low grades for the end of the mining sequence. The LoM sequence encompasses a 15-month pre-stripping phase between October 2022 and December 2023 followed by 8 years of primary ore mining and, finally, 2 years of re-handling low grade ore.

The mining equipment selected involves small backhoe excavators (74-t op. weight) and on-road mining trucks (45-t capacity). The ore will be drilled by top-hammer drill rigs in 5-m high benches, while the waste will be drilled by DTH rigs in 10-m high benches. It is SRK's opinion that the method is appropriate to the orebody geometry, mineralization style, production rate, and is benchmarked with similar mining operations.

25.4 Recovery Methods

The Project process plant will have a capacity of 2.5Mtpa. The process plant includes crushing, milling, pre-leach thickening, pre-oxidation and CIL adsorption, desorption, regeneration and gold room. The process plant also includes tailings detoxification and filtration. The filtered tailings are transported and stored in a tailings pile.

The process flow sheet proposed for the Posse Gold Project involve mostly well proven technologies in the gold/silver processing industry and thus, no significant risks are anticipated and there are no deleterious elements in the feed. Novel recirculation and high shear technologies are used in the pre-aeration and CIL circuits to inject oxygen.

25.5 Project Infrastructure

The Project infrastructure consists mainly of the process plant, buildings, power line, water dam, filtered tailings pile, waste dumps and low-grade stockpile.

The Project access and most of service roads are existing roads, minimizing earthworks and clearing vegetation.

The Project requires the construction of 67km of a 138kV transmission line to link Porangatu and the mine site.

An executive project was developed for the filtered tailings pile to meet the tailings requirements until Year 2. Thereafter, a design at a PFS level was created to accommodate the tailings produced until the end of the Posse Gold Project. The total capacity of the filtered tailings pile will reach 16.53Mm^3.

The waste will be disposed in six waste dumps. Waste dumps 1, 2, 3 and 4, which account for 36% of total waste produced over the LoM, have sufficient capacity to meet a period of 3.5 years of operation, including the pre-stripping stage, and have already been granted the Installation License ("**LI**"). Further designs were developed for waste dumps 5 and 6.

25.6 Environmental Studies and Permitting

Permits are well underway with no foreseen delays.

According to the water balance study and simulations made by GBM (2021), the volume of the water reservoir (B1) is sufficient to supply the Project with the necessary water.

Regarding social and community, there is no risk identified. Although there are some land purchases still pending, it is not expected that this will have an impact on the Project schedule.

A preliminary mine closure plan was developed that includes closure activities for each phase of the Project. Amarillo updated the closure cost in 2021.

25.7 Capital and Operating Costs

Capital and operating costs have been estimated at a level appropriate for a Feasibility Study. Overall accuracy is estimated at ±15% for both capital and operating costs.

All key capital and operating cost items were supported by vendor quotes.

Mining costs were derived from quotes provided by contractors based on the DFS mine plan.

25.8 Economic Analysis

The economic model on the Base Case of the Project demonstrates that under the current set of economic assumptions the Posse Gold Project provides an accumulated undiscounted cash flow of US$262.8m and a robust positive post-tax Net Present Value ("**NPV**") of US$154.6m @ 5% annual discount rate over the LoM. The Project showed a post-tax IRR of 19% and a post-tax payback period, based on the undiscounted cash flow, of 3 years from the start-up date.

In addition to the financial evaluation performed on the Base Case (US$1,600/oz gold price), a scenario with a gold price of US$1,450/oz was developed in line with the economic cut-off calculations of the Mineral Reserve (Section 15.7). A positive US$186.8m free cash flow total over the LoM was determined confirming the economic viability of the mineable inventories.

26 RECOMMENDATIONS

26.1 Mineral Processing and Metallurgical Testing

The metallurgical performance is a function of gold head grade, the deportment of which is understood per the reserve model. Gold head grade providing a means to estimate recovery per the algorithm presented herein. The test work also suggests a telluride association as would be expected given the mineralogy of the ore and this may improve the accuracy of the recovery estimates. There is no tellurium model available for the reserve at this time.

To understand the metallurgy in the operating stage of the Project, it is recommended:

- Grade controls samples be subjected to a standardised leach test and include tellurium head assay so as to establish a data set of gold and tellurium head grades and extraction behaviour;
- That some grade control samples be subjected to the same leach test but at two alternative pH levels. This will allow the operations to associated gold and tellurium grade with benefit of higher and lower pH considering reagent demands and extraction; and
- Grade control sample viscosity also be determined. This being the only physical characteristic of the samples tested noted to be potentially problematic, albeit sporadic. A simple viscosity funnel test could be employed to simplify the data collection, combined with periodic cross-checks with a proprietary viscometer capable for presenting variable shear rates.

In the pre-operational stage and during operations, it is recommended:

- Future drilling of the resource/reserve includes sulphide sulphur and tellurium assays with a view to build a tellurium and possibly sulphide model in the future; and
- Some metallurgical test work be conducted to establish gold-tellurium-sulphide influences on extraction and improve the prediction thereof.

26.2 Geology and Mineral Resources

The work undertaken to calculate the current Mineral Resource has indicated the need for further work including the following:

- Ensure that future diamond drilling is conducted in such a way that geological information is maximised and recorded in an appropriately structured database so that it can be used for future mineral resource development;
- Ensure that accurate rock density data is collected as a regular part of diamond drilling;
- Carry out further drilling to test the areas under the old Posse north pit to upgrade Indicated resource to Measured;
- Test drill the historic waste dumps to test the degree of mineralization in waste dumps;
- Ensure the check drilling of the backfill in the historic Posse pits is conducted early in the mine development to determine if the material is mineralised and represents unrecognised mineralisation and to confirm volumes;
- Updating of lithological and mineralisation wireframes;
- All re-assay results should be incorporated into the drillhole database as preferred assays and used for future modelling work together with the results of the 2021 drilling;
- The volume of underground workings, while small, should be recognised and removed from future models;
- The newly acquired SG data should be modelled as part of any future resource; and

- There is now assay data for a range of elements other than Au, those that have potential to interfere with metallurgy or which may indicate potential for AMD should be modelled as part of any future model.

26.3 Mining and Mineral Reserves

- The geotechnical study is based on a limited number of geotechnical boreholes. It is recommended that additional geotechnical boreholes be drilled to collect additional data to update of the geotechnical characterization;
- Major structures need to be mapped in the old pit once access is re-established and used to develop a working structural geological model to assist pit design;
- Standard ground control/slope management procedures need to be adopted so that the design assumptions are validated during mining and the design is further optimized. Mapping of the footwall structures will be very important to maintain the optimal pit production as well as checking for the potential for adverse footwall structures that could be unstable;
- The intermediate cutbacks were designed using slope angles recommended for the walls of the ultimate pit. Good quality blasting of final walls and major intermediate cutbacks will be critical to good performance, so pre-splitting (or similar blasting techniques) should be adopted;
- A mine-to-mill approach should be considered to optimize the overall costs of mining and processing operations; and
- Develop detailed grade control procedures to improve the mining model accuracy and grade estimates.

26.4 Recovery Methods

- Adding quick lime directly to the ball mils may consume more quick lime than by adding a lime slurry to the cyclone feed pump box. Studying the capacity of the plant lime slurry system is recommended to assess the benefit of increased capacity to add lime to the grinding pump boxes;
- The total cost of ownership of the mixing system should also be investigated further to verify the technology; and
- Cyanide detoxification may be able to be optimized and incorporated as part of the water treatment system. It's recommended that the water treatment system be evaluated to determine its suitability for cyanide detoxification.

26.5 Infrastructure

- The tailings generated from the ore processing plant will be accommodated in a pile after filtering in a dedicated facility. The pile design is based on tests of tailings samples to determine resistance characteristics. The following additional studies are recommended for tailings characterization:
  - Improve the knowledge of the physical indexes and geotechnical parameters of the tailings to better estimate the safety and economic factors. The solid size distribution and the mineralogy of the fine fraction of tailings are essential to optimize the performance of filtering.
  - Specify the tailings compaction conditions, such as moisture content, as they are intrinsically associated to the pile configuration.
  - Undertake detailed studies of densification / compressibility, due to the impact of these

parameters on the undrained behaviour of the material and on the predicted pore-pressure conditions.

- Further investigate the liquefaction effect. Static liquefaction is activated in saturated tailings when these are subject to shear stress, mainly, within dykes at certain levels of disposal rates.
- Design experimental fills to create the conditions for testing the Normal and Modified Proctor;

- Executive projects for the filtered tailings pile and waste dumps were developed for the two first years of production. Additional studies and designs at a PFS level were then completed to accommodate the remaining materials until the end of the life of the mine. Future engineering iterations should increase the level of accuracy of these studies as required by the mine;
- Twenty-one drill holes were completed within the limits of the planned tailings pile to assess the geotechnical conditions. However, no samples were collected for laboratory analysis. The geotechnical parameters of the foundation need to be better understood through shear stress tests under dry and wet conditions covering all concerned lithologies;
- The stability analysis showed safety factors above the minimum limits established by the legal regulations under the specified premises. Hence, it is suggested that additional evaluations under pseudo-static conditions both the pile and the dyke;
- Thirty-seven drill holes were undertaken to assess the foundation of the planned waste dumps WD1, WD2 and WD3 and define the required excavation. It is recommended that laboratory tests be performed to estimate the geotechnical parameters; and
- The waste dumps 1, 2, 3 and 4 account for 36% of the total waste dumping capacity of the Project, which is sufficient to meet 3.5 years of operation, including the pre-stripping phase. These waste dumps have already been granted the Installation License ("**LI**"). Additional areas will be required for future waste dumping (waste dumps 5 and 6). SRK recommends planning early to obtain the required environmental licenses so the waste dumps can be built in a timely manner.

26.6 Environment

- Develop a plan to obtain the operation license ("**LO**") according to schedule; and
- SRK recommends periodic updates of the mine closure plan to consider any changes in the socio-environmental conditions of the region, seeking to ensure post-closure sustainability in the generation of income and conservation of the environment and to comply with ANM 68/2021 which requires an updated every five years.

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Glossary

Glossary – Mineral Resources and Ore Reserves

Mineral Resource	A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.
Indicated Mineral Resource	An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation. An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve
Inferred Mineral Resource	An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.
Measured Mineral Resource	A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation. A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.
Mineral Reserve	A Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at Pre-Feasibility or Feasibility level as appropriate that include application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified. The reference point at which Mineral Reserves are defined, usually the point where the ore is delivered to the processing plant, must be stated. It is important that, in all situations where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported.

	The public disclosure of a Mineral Reserve must be demonstrated by a Pre-Feasibility Study or Feasibility Study.
Probable Mineral Reserve	A Probable Mineral Reserve is the economically mineable part of an Indicated, and in some circumstances, a Measured Mineral Resource. The confidence in the Modifying Factors applying to a Probable Mineral Reserve is lower than that applying to a Proven Mineral Reserve.
Proven Mineral Reserve	A Proven Mineral Reserve is the economically mineable part of a Measured Mineral Resource. A Proven Mineral Reserve implies a high degree of confidence in the Modifying Factors.

Glossary – Development Stages

Producing Property	Mineral assets for which current Ore Reserves are declared and mining and processing operations have been commissioned and are in production.
Development Property	Mineral assets for which Ore Reserves have been declared and are essentially supported by a minimum of a pre-feasibility study which on a multi-disciplinary basis demonstrates that the consideration is technically feasible and economically viable.
Pre-Development Property	Mineral assets for which Mineral Resources have been defined but where a decision to proceed with development has not been made.
Advanced Exploration Property	Mineral assets for which only Mineral Resources have been declared.
Exploration Property	Mineral assets for which no Mineral Resources have been declared.

Glossary – Terms, Abbreviations and Units

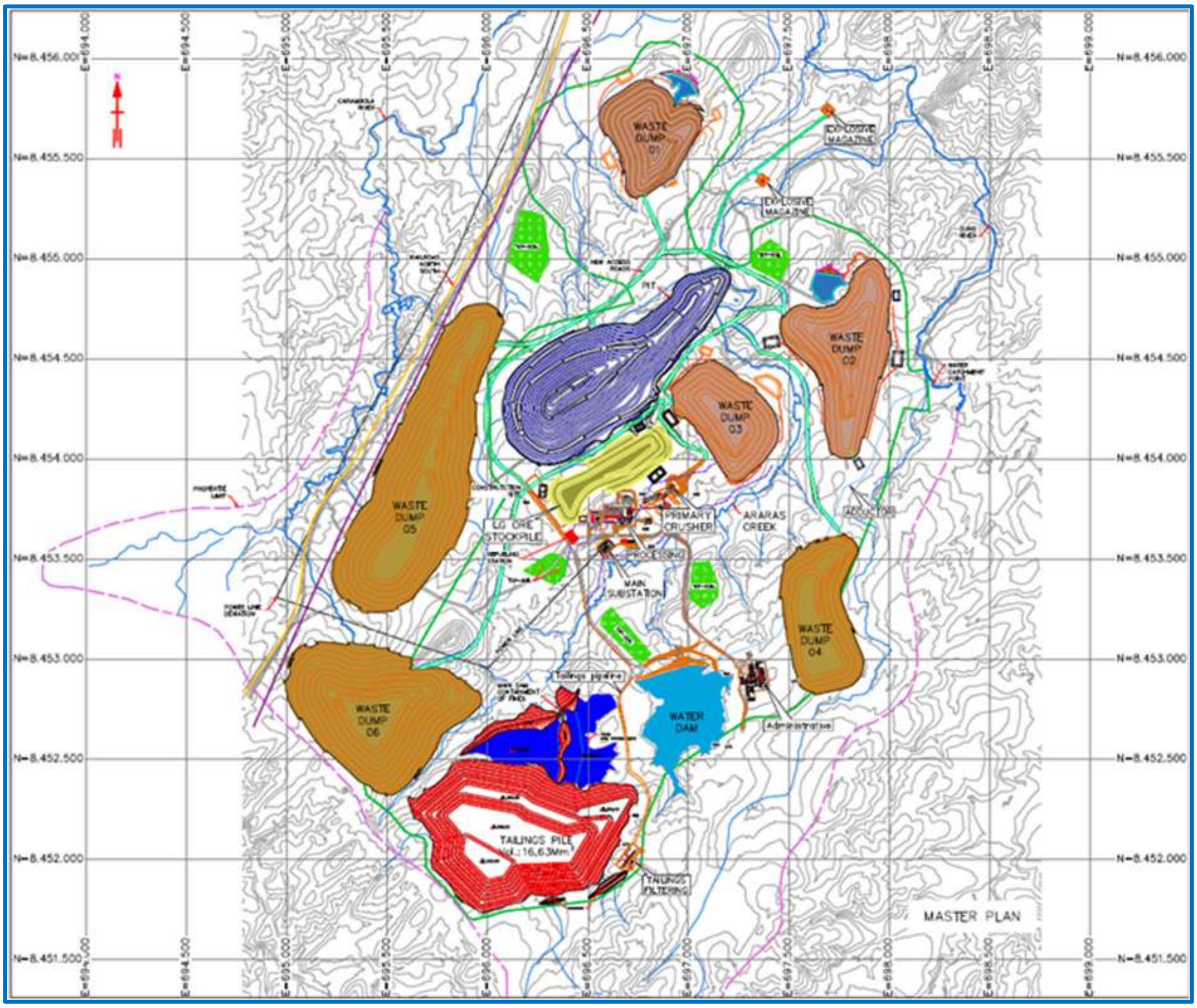
Abbreviation	Meaning
$	dollar sign
%	percentage sign
°C	Celsius sign
µm	micrometre
3D	three dimensional
AAS	atomic absorption spectroscopy
ACME	Acme Analytical Labs Ltd
AEFS	Australian Exploration Field Services Pty Ltd
Aurifex	Aurifex Pty Ltd
Amarillo	Amarillo Gold Corporation
ANFO	Ammonium Nitrate and Fuel Oil (FO)
ANM	Agência Nacional de Mineração
ASL	above sea level
ATV	acoustic televiewer
Au	gold symbol
Ausenco	Ausenco do Brasil Engenharia Ltda.
B/H	bench height
Barrick	Barrick do Brasil
bcm	bank cubic metre
BD	bulk density
BFA	bench face angle
BH	bench height
BHP	BHP Limited
BoD	basis of design
BSA	bench stack angle
BSH	bench stack height
BVP	BVP Engenharia
C	cohesion
CAPEX	capital expenditure

Abbreviation	Meaning
CC	correlation Coefficient
CCIC	Caracle Creek International Consulting Inc
CELG	Brazil-based company engaged in the electric power industry
CIL	carbon-in-leach
CIM	Canadian Institute of Mining
CONAMA	National Environmental Council, Brazil
CTD	conventional tailings disposal
DBO	DBO Engenharia Ltda.
DFS	definitive feasibility study
dmt	dry metric tonne
DTM	digital terrain model
DXF	drawing exchange format
DYKE	A subvertical intrusion of volcanic rock
E	east
E–W	east-west
FEL	front end loader
FoS	factor of safety
FS	feasibility study
FSL	full supply level
FW	footwall
g	gram
g/cc	gram per cubic centimetre, a density measure, see Kg/m3
g/t	gram per tonne
garimpos	An artisanal miner, at Mara Rosa the term *garimpos* is also used to describe workings dug by artisanal miners
GB	geotechnical berm
GBI	geotechnical blockiness index
GDM	geotechnical domain model
GHT	GeoHydroTech Engenharia
GIS	geographic information system
GMA	Goiás Magmatic Arc
GO	Goiás State, Brazil
GPS	global positioning system
GSBW	geotechnical safety berm width
GTR	grind –throughput recovery
ha	hectares
HARD	half absolute relative difference, used in conjunction with the statistical analysis of duplicate sampling. HARD plots measure precision of duplicate sample pairs using the absolute difference between pairs divided by the mean of the pair.
HCS	Hoogvliet Contract Services
HME	heavy mining equipment
Hr	hydraulic radius
HW	hanging wall
ICP	inductively coupled plasma
IDS	a modelling algorithm where points are weighted by the square of the distance
IOSA	indicative overall slope angles
IPC	in-pit crushing
IRA	inter ramp angle
JORC Code	Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves prepared by the Joint Ore Reserves Committee of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Minerals Council of Australia (JORC), 2012.
k	thousand
kg	kilogram
kg/m^3	kilograms per cubic metre, the SI unit for measuring density, a density of 1000 kg/m^3 is equivalent to 1 g/cc or 1 t/m^3 or 1t/bcm
kj	kilojoule
kL	kilolitre
KNA	Kriging neighbourhood analysis
kt	kilotonne
kV	kilovolt
kW	kilowatt
L	litre

Abbreviation	Meaning
L/s	litres per second
lcm	loose cubic metre
LG	Lerchs-Grossman
LOM	life of mine
LOSA	limiting overall slope angle
m	metre
M	million
Ma	Million years ago
m RL	metres reduced level
m^3	cubic metre
Maxibor	A type of downhole survey instrument
MBL	Metallica Brasil Ltda
MCAF	mining cost adjustment factor
mE	metres east
MIK	multiple indicator kriging
Minere	Minere Mineração Ltda.
MIP	maximum intensity projection
MJSA	Mineração Jenipapo S.A.
mL	millilitre
ML	megalitre
mN	metres north
MR files	Mara Rosa files
MRMR	mining rock mass rating
mS	metres south
MSSO	MineSight Scheduling Optimiser
Mt	million tonnes
Mtpa	million tonnes per annum
MVA	mega volt amperes
MW	megawatt
N	north
NE	northeast
NPV	net present value
NSR	net smelter return
NW	northwest
OB	overburden
OPEX	operating expenditure
OS	oversize
oz	troy ounces
Pnn	percentage of material passing a size measure. For example, P80 indicates that 80% of the material is smaller than a specified size.
PCAF	processing cost adjustment factor
PFS	pre-feasibility study
PMF	probable maximum flood
Q	Barton Q value
Q'	modified Q value
QA/QC	Quality assurance / quality control
QP	Qualified Person
QQ	quantile / quantile plot
RC	reverse circulation
RF	revenue factor
RG	Royal Gold Inc.
RL	reduced level
RMR	rock mass rating
ROM	run of mine
RQD	rock quality designation
S	south
SANEAGO	State water company, Goiás, Brazil
SBW	spill berm width
SE	southeast
SG	specific gravity also known as density
SRK	SRK Consulting / SRK Consultores do Brasil Ltda
SW	southwest
t	tonne

Abbreviation	Meaning
t/m^3	tonnes per cubic metre, a density measure, see Kg/m^3
t/bcm	tonnes per bcm, a density measure, see Kg/m^3
tpa	tonnes per annum
TSF	tailings storage facility
TTD	thickened tailings disposal
US	United States
UTM	Universal Transverse Mercator projection, all data in this report has been referenced the WGS 84 datum projected to UTM Zone 22 South
W	west
WFS	waste storage facility
WGS84	World geodetic system 1984
WMC	Western Mining Corporation
wmt	wet metric tonne
WSF	water storage facility

APPENDIX
A Integrated Project Layout and End-of-Period Layouts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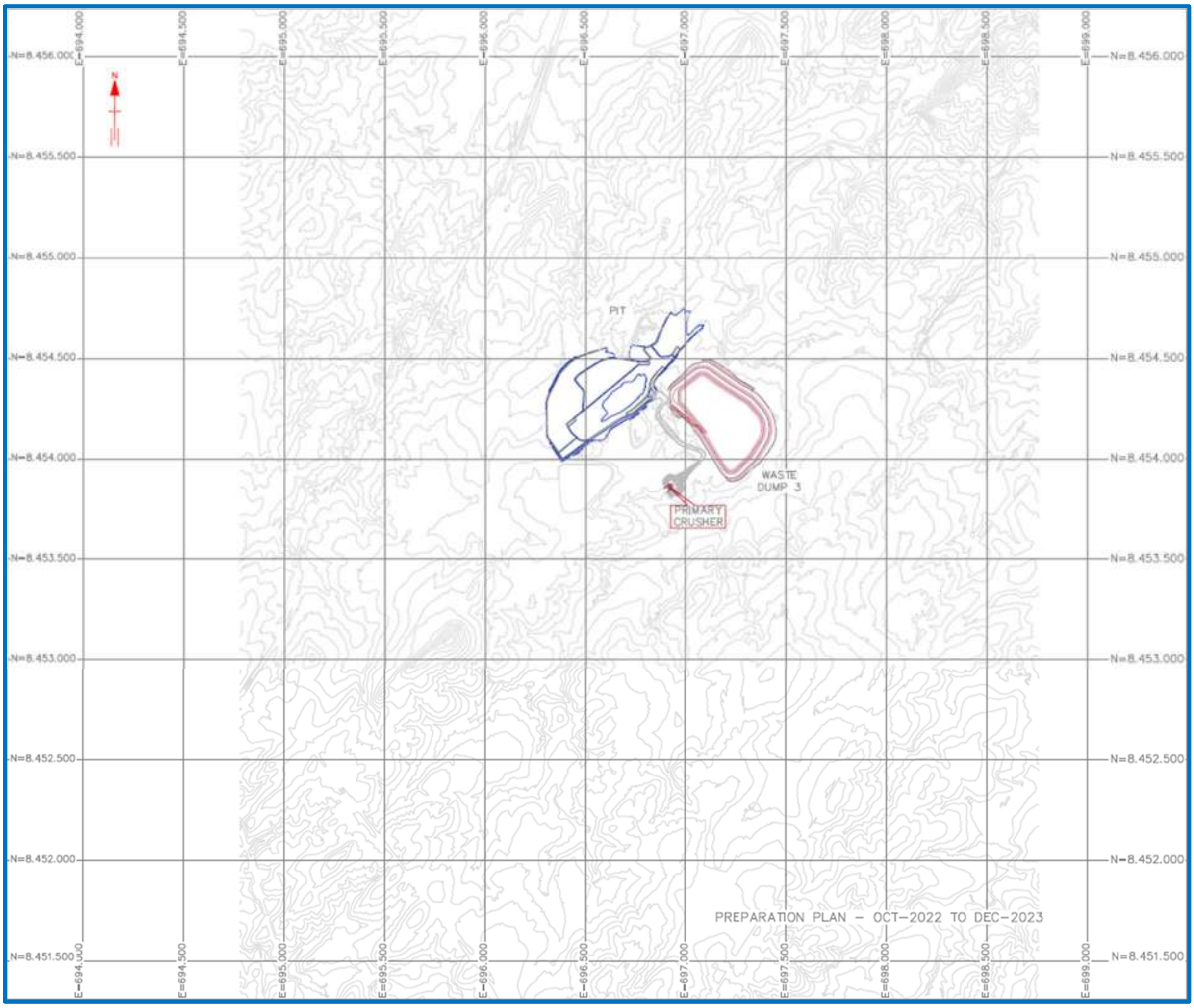
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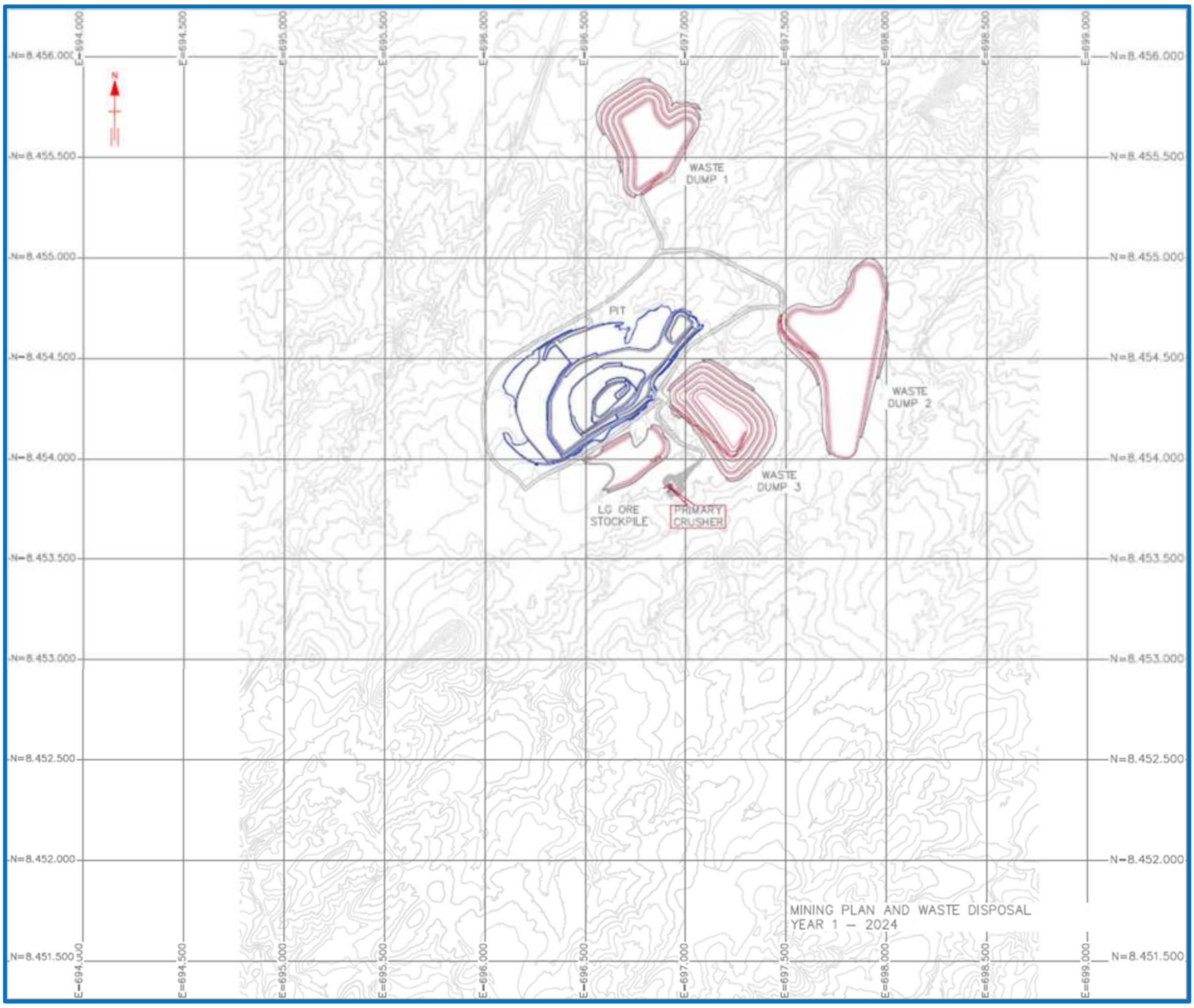
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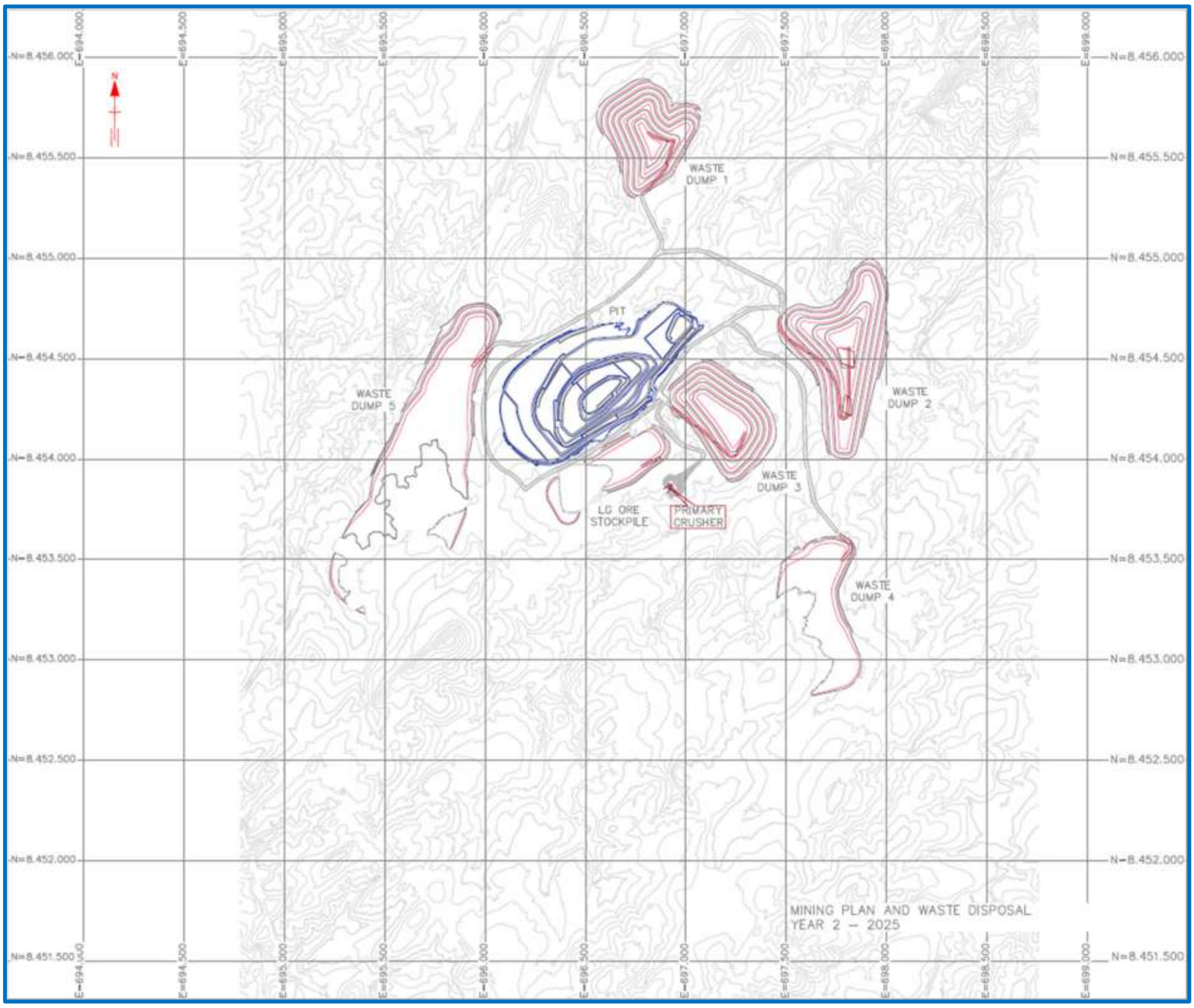
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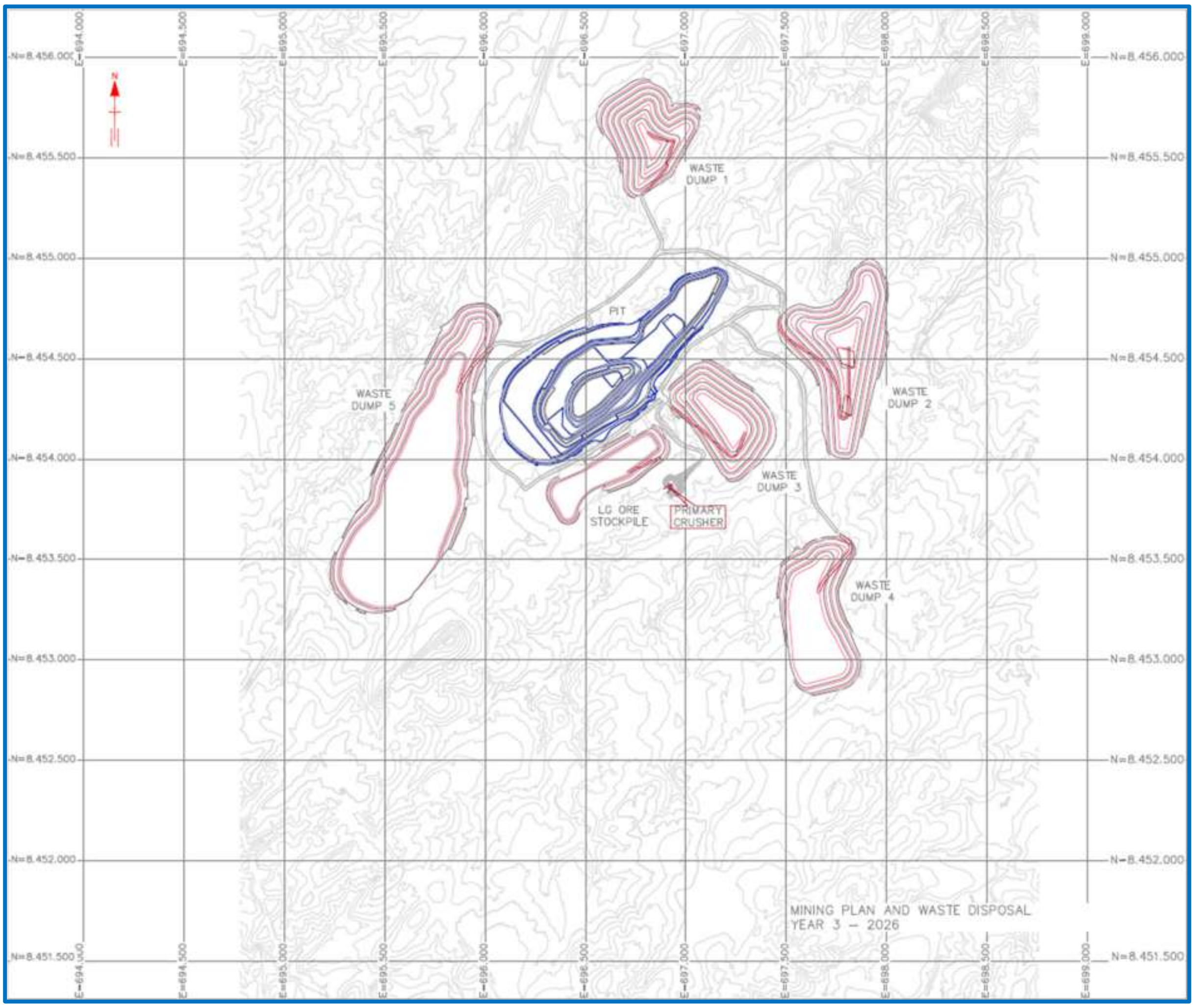
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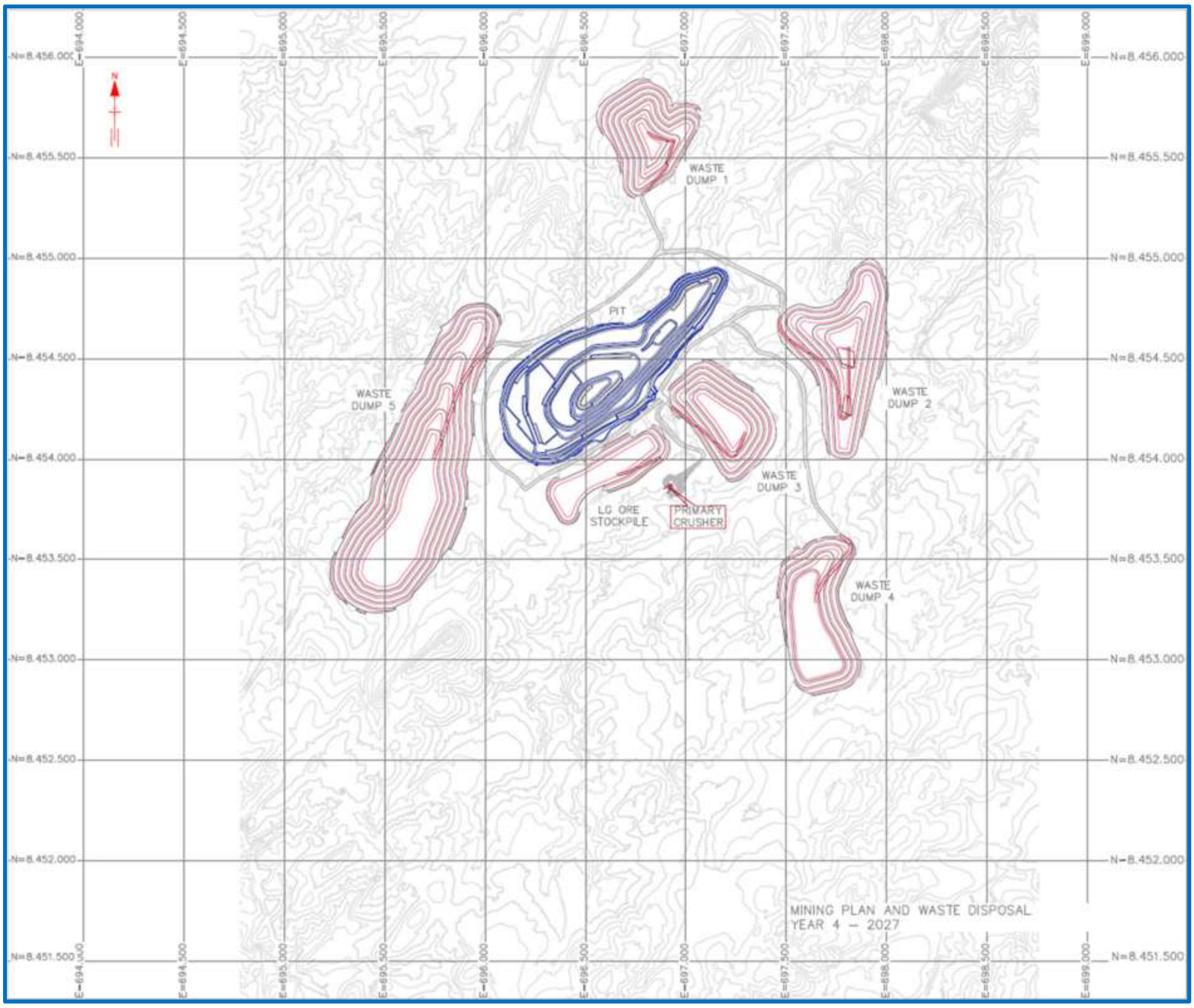
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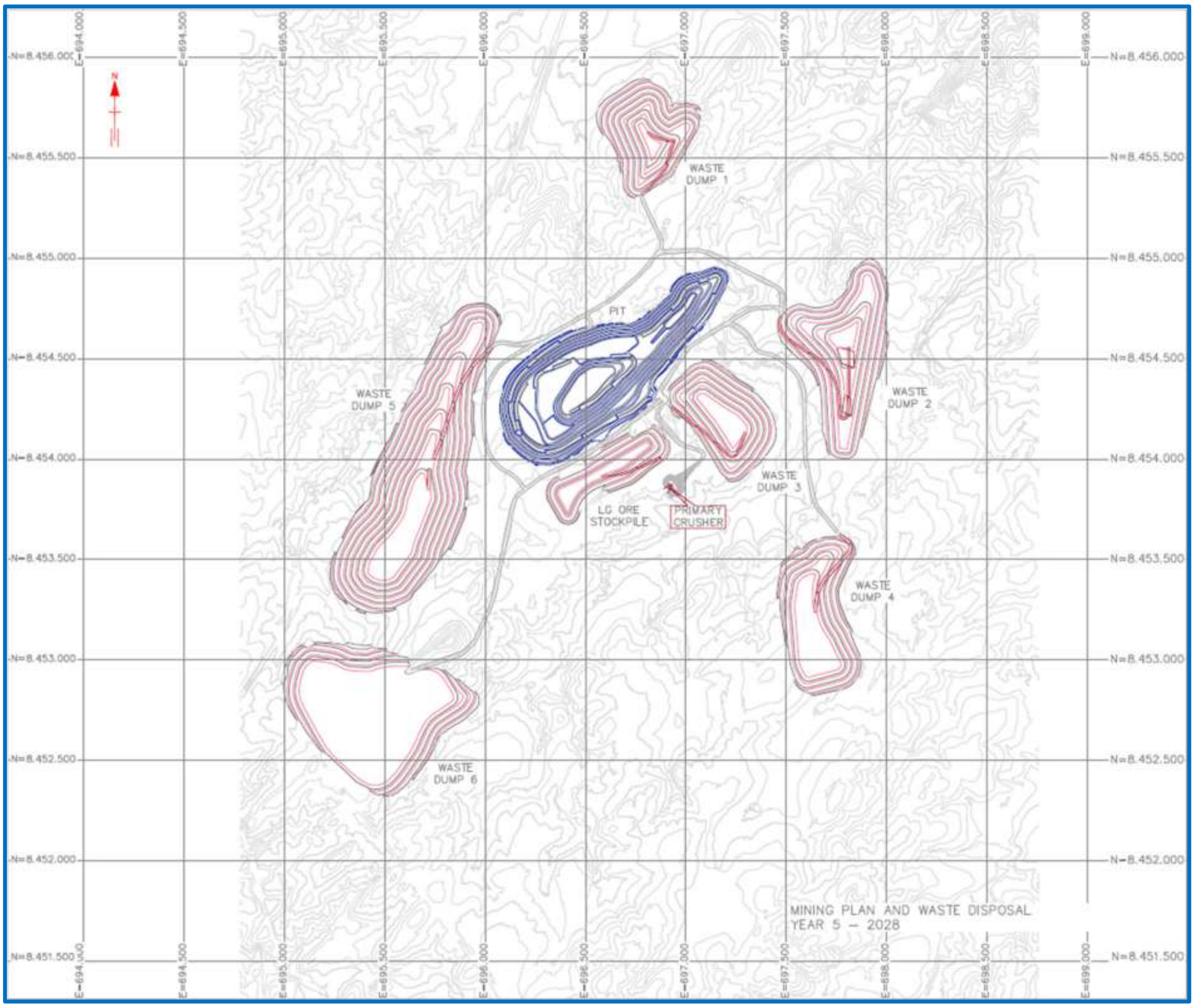
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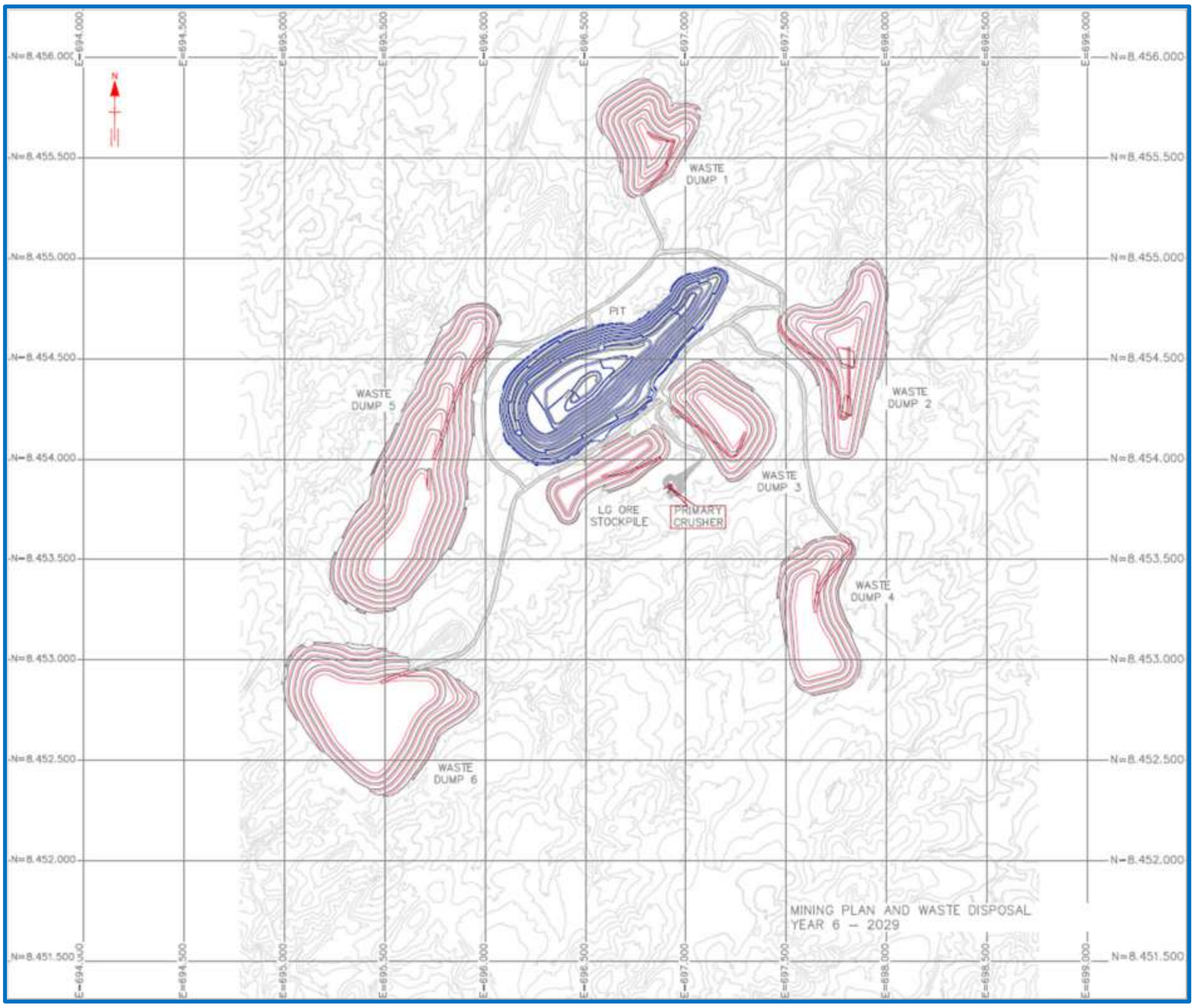
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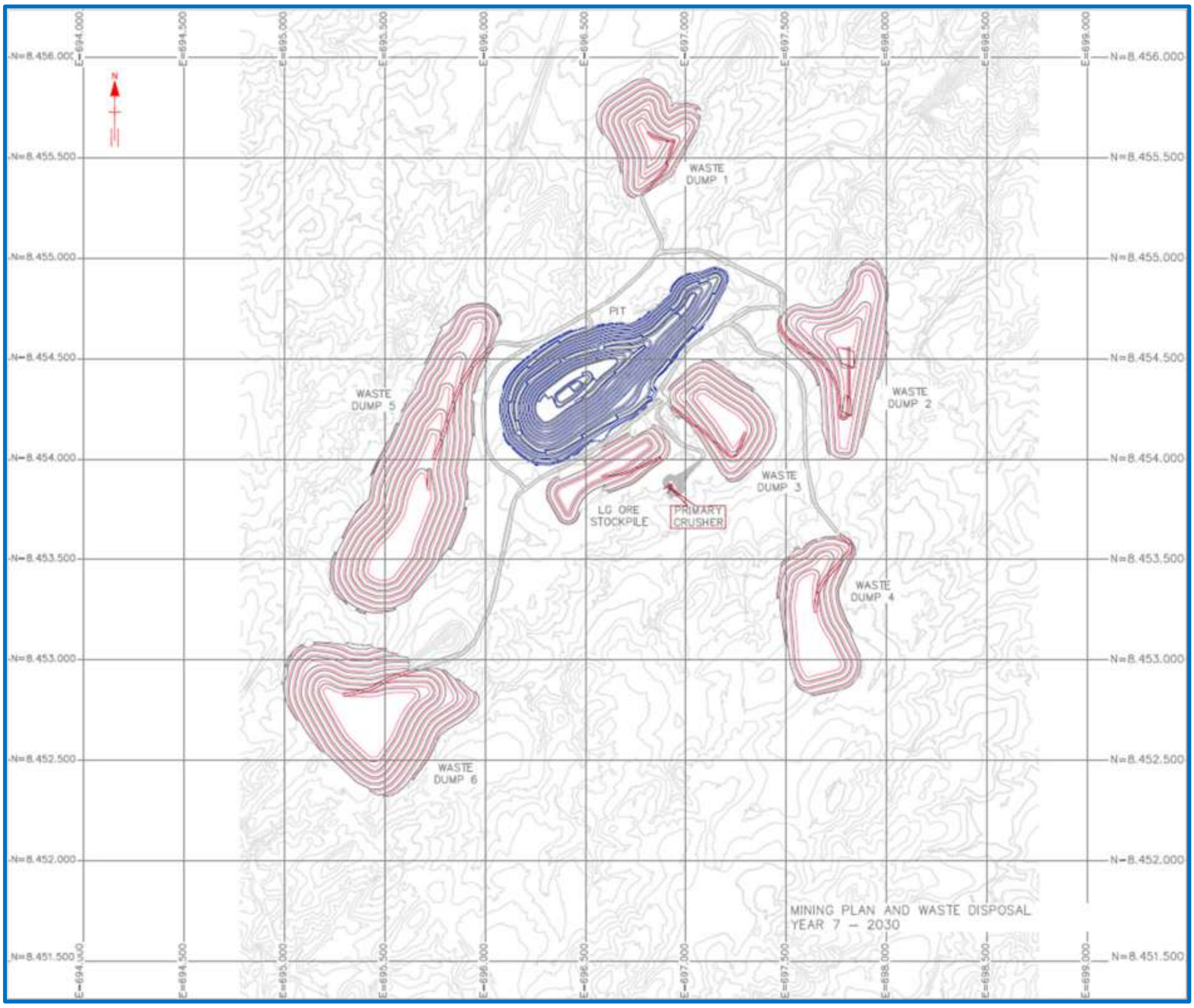
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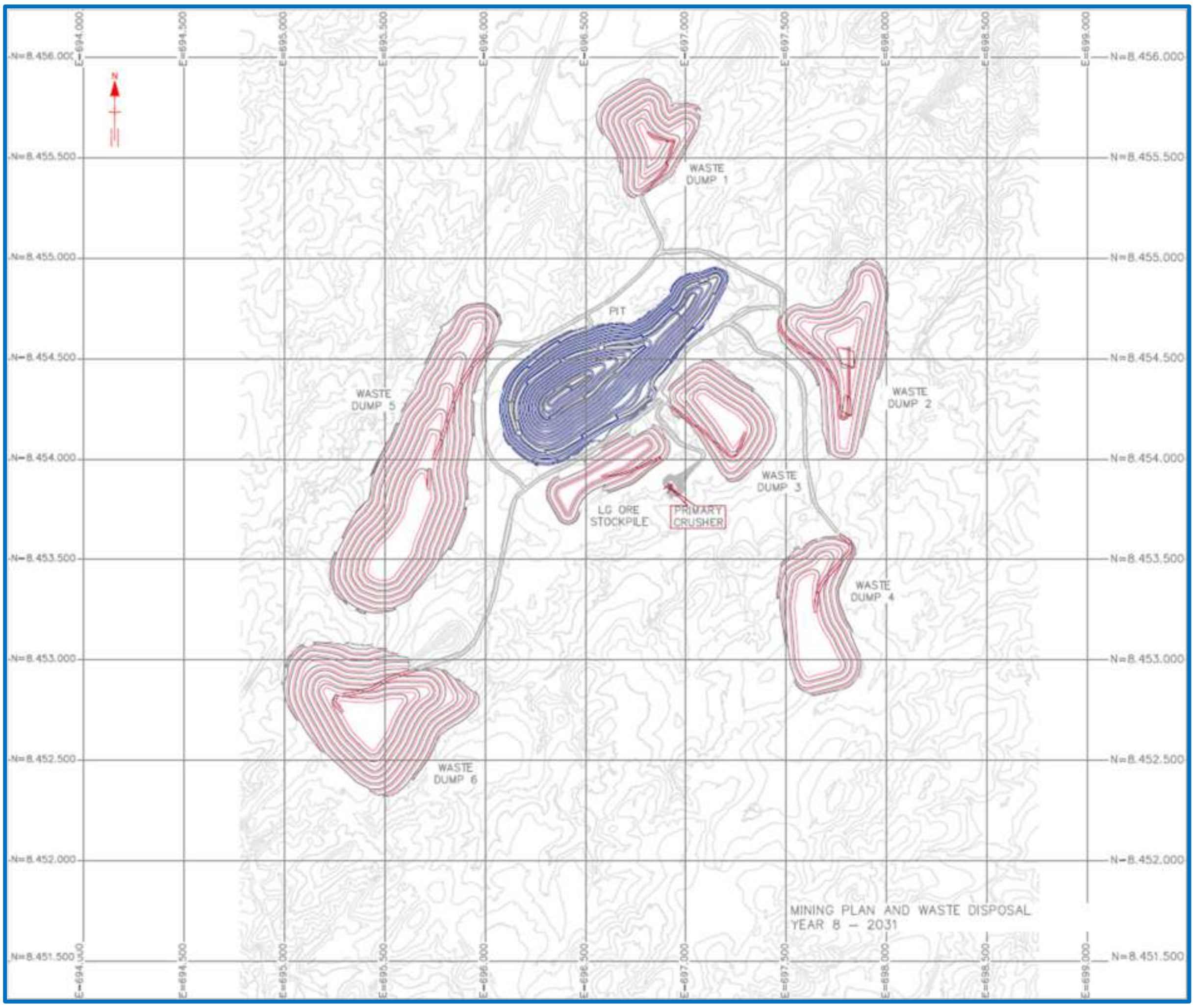
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