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Geometallurgical Study of a Gravity Recoverable Gold Orebody

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Abstract: Sheeted vein gold deposits are often characterised by multiple sub-parallel veins and free-milling coarse gold. Inherent mineralisation heterogeneity results in grade and process parameter variability, which increases project risk if not quantified. Measured grade variability is often exacerbated by poorly designed sampling and testwork protocols. Protocols that are optimised within the framework of the Theory of Sampling (TOS) to suit the ore type, together with quality assurance/quality control systems, will reduce variability and provide fit-for-purpose results. Geometallurgy can be broadly split into two key approaches: strategic and tactical (or operational). The strategic approach focuses on the whole orebody and long-term life-of-mine view, whereas tactical geometallurgy relates to a more short- to medium-term view during mining. The geometallurgical approach requires spatially distributed samples within a deposit to support variability modelling. Diverse attributes from core logging, mineralogical/textural determination and small-scale tests are used to measure variability. This contribution presents a case study that emphasises an early-stage strategic geometallurgical programme applied to a gravity recoverable gold (GRG) dominated deposit. It exemplifies how data can be acquired from a well-designed and planned programme to support resource estimation, a pre-feasibility study, trial mining and fast-track to production. A tactical geometallurgical programme is embedded into the mine operation.

Keywords: sheeted vein mineralisation; geometallurgy; gravity recoverable gold; sampling protocol optimisation; bulk sampling; trial mining

1. Introduction

1.1. Geometallurgy

The prime objective of geometallurgy is to improve the profitability of mines using spatial models of rock properties that have a significant impact on value [1–3]. It aims to correlate geology and mineralogy with data from metallurgical testwork and develop a model to predict variability. A key output is 3D mapping, where diverse attributes from core logging, mineralogical/textural determination and small-scale tests are used to resolve grade, process parameter and rock mass variability. The geometallurgical approach emphasises early stage intervention and progression during the project to optimise the mine plan. Geometallurgy can be broadly split into two key approaches: strategic and tactical [1,2]. The strategic approach focuses on the whole orebody and long-term life-of-mine view, whereas tactical geometallurgy relates to the short- to medium-term view during mining.

1.2. Defining Gravity Recoverable Gold

GRG is a gold recovery parameter based on liberated gold or high-grade composite gold particles that can be recovered by gravity methods such as batch centrifugal concentrators or shaking tables [4]. Method specific testwork includes the one- and three-stage GRG tests using a Knelson concentrator [4–6]. Recovery depends upon mineralisation type and comminution and extraction method used. For example, centrifugal concentrators may recover gold down to 20 µm in size, whereas jigs are more likely to be restricted to >100 µm. Any ore type bearing >25% of gold greater than 50 µm to 100 µm in size is likely to have a high GRG component.

1.3. Importance of Good Sampling

The sampling and analysis of waste and mineralised material is a key part of any geometallurgical programme [7,8]. Testwork and associated measurements throughout the mine value chain must be supported by representative samples [7,8]. There is a need to consider both in-situ and testwork sub-sample representativity. Sampling and testwork protocols must be designed to suit the style of mineralisation. In support, the a priori need for evaluation is characterisation of the mineralisation and domain definition. The entire sampling to assay process yields data that form the base for Mineral Resource and Ore Reserve estimates and subsequent economic studies (e.g., to be reported in accordance with The JORC Code 2012) [9].

Traditional metallurgical sampling and testwork are critical for plant design and are an inherent part of geometallurgy [1,8]. In a geometallurgical study, multiple spatially distributed small-scale tests are used as proxies for grade, mineralogy, process parameter, etc. These will generally be validated against traditional testwork results. In the context of geometallurgical programmes, metallurgical sampling and testwork are a critical input. Traditional testwork programmes might have a few hundred results at the feasibility level, where a strategic geometallurgical programme will result in thousands of spatially distributed data points [10].

1.4. Focus of This Contribution

The mining industry frequently fails to focus on the technical variability that impacts on the bottom line of a project. The geometallurgical approach is often applied to large-scale >1 Mt per annum operations (e.g., Olympic Dam, Australia) [3], but is rarely applied to small-scale <250,000 t per annum operations.

This contribution presents a case study, which shows for the first time how fit-for-purpose data can be gained from a well-planned and implemented geometallurgical programme in a small, high-nugget effect GRG-dominated deposit. It presents a drill core-based integrated grade and metallurgical recovery sampling and testwork programme, which was designed for the specific mineralisation. Additional information on comminution, mineralogy and geochemistry were also derived from the core. The grade and metallurgical recovery results were subsequently validated by bulk sampling, trial mining and production. All sampling activities were optimised within the framework of the TOS and quality assurance/quality control (QAQC) procedures applied. The paper provides an overview of the programme, particularly the development of the sampling and testwork protocol. It shows how the strategic geometallurgical approach contributed to a Mineral Resource estimate and associated pre-feasibility study, initial mine plan, change in mining strategy and fast-track to production. A tactical geometallurgical approach has been embedded into on-going ore control and resource development.

2. Theory of Sampling and Representative Sampling

2.1. Theory of Sampling

Sampling errors are defined in the TOS as promulgated by the works of Dr Pierre Gy [11,12]. The TOS errors relate to actions across the sample value chain that may lead to uncertainty and create an overall measurement error [13]. TOS attempts to break down this error into a series of contributions along the sampling value chain.

Uncontrolled sampling errors lead to an elevated nugget effect [14,15]. The fundamental sampling (FSE) and grouping and segregation (GSE) errors are irreducible random errors related to the inherent heterogeneity and characteristics of the material being sampled. They lead to poor precision and can only be minimized through good protocols. The other errors, delimitation (DE), extraction (EE), preparation (PE) and analytical (AE) errors, arise as a consequence of the physical interaction between the material being sampled and the technology employed to extract the sample [11,12]. They result in bias, which can be reduced by the correct application of sampling methods and procedures [11,12]. A sample can be described as being representative when it results in acceptable levels of bias and precision [12].

The FSE equation is used to optimise sampling protocols, where it addresses key questions related to the sampling of broken rock including mass and particle size reduction requirements [11,12,16]. The FSE is expressed as a relative error (precision) at a given confidence limit (reliability), usually 68% or 90%. All FSE calculations in this contribution are reported at the 68% reliability.

The FSE equation requires definition of the liberation diameter (d_L) [16–18]. The presence of discrete coarse gold particles even after comminution reduces the probability of any sub-sample containing a representative number of gold particles [19]. Consequently. for gold mineralisation, d_{95Au} is defined to represent the coarsest gold most influential particles, effectively the screen size that retains 5% of gold given a theoretical lot of liberated gold [14–16,20]. Application of the FSE equation represents an idealised expectation that may or may not be attained in practice, but provides a starting point from which protocols can be evaluated and optimised. Results of QAQC programmes provide evidence for representivity, particularly through the application of duplicate sample analysis [21].

2.2. Representative Sampling

A sample can be described as being representative when it results in acceptable levels of bias (accuracy) and precision. Whilst precision (reproducibility) can be determined, bias is less easy to quantify without generally impractical and costly experimental efforts. Pitard [14] stated that the total variance for resource and grade control sampling should not be more than ±32%, with the FSE component not more than ±16%.

All sampling variances are cumulative and contribute to the total, which in turn contributes to the sampling nugget effect. In reality, the FSE and GSE may contribute up to 90%, with the DE, EE and PE up to 25% of the total [12]. Analytical errors generally account for between 1% and 15% relative error [21]. It is generally recognised that the total sampling to assay error is dominated by the error associated with the initial sampling stage (e.g., sample collection process), which could potentially be in the range of 15% to 60% [8,21].

The 'Danish Hoizontal Standard' (DS3077) [22] recommends that the total sampling variability be quantified by the relative sampling variance (RSV): the percentage coefficient of variance for repeat sample values. The RSV comprises all stages of the sampling protocol, including all errors incurred by mass reduction as well as analytical error. RSV measures the total empirical sampling variance influenced by the heterogeneity of the lot being sampled under the current sampling procedure [22]. The accepted value of RSV is up to the practitioner and is based upon the nature of the mineralisation in question, the data quality objectives and what is cost-effective and practical.

In coarse-gold dominated mineralisation, it may be impractical to collect representative samples [6,15,23]. A theoretical field sample mass of many tonnes may be required to achieve an acceptable RSV. A reasonable strategy is to collect multiple samples across a given domain. Each sample may be locally unrepresentative, but appropriately spaced samples informing an optimised kriged block model will provide a robust estimate.

A sampling and testwork programme must produce data that are fit-for-purpose for their proposed usage [24]. In this context, fit-for-purpose refers to the production of data that enables the practitioner to make technically correct decisions. In most cases, results must be fit to contribute to a Mineral Resource (and/or Ore Reserve) that can be reported in accordance with the 2012 JORC Code (or other codes). Sampling and sub-sampling should result in representative samples. A critical input is that of QAQC to maintain data quality through documented procedures, sample security, and monitoring of precision, accuracy and contamination. If a batch of samples is deemed to be representative and assaying (measurement) complies with QA documentation and QC metrics, then the data are fit-for-purpose.

2.3. Bulk Sampling and Large-Scale Testwork

Grade and metallurgical validation and metallurgical scale-up work generally involves the use of pilot plant testing of bulk samples or full processing of trial mining lots [18,25]. Such activities are often a critical part of the pre-feasibility or feasibility study stages. They require careful planning and management to ensure that they provide validation of a given grade and/or metallurgical model [8,18,25].

The collection of large samples from surface and/or underground locations characterises pilot and plant testwork. Bulk samples tend to be smaller (from a few tonnes to 250 t), more numerous samples that may be from single locations or grouped together from underground development or surface trenches. In some cases, a bulk composite may be formed from drill core material. Trial mining activities tend to represent underground stope lots or open pit benches. Such lots may yield thousands of tonnes of mineralisation for processing. A well-planned programme will account for mineralisation variability, which may require several sample collection areas in specific geometallurgical domains. In addition, any programme needs to account for grade variability—honouring the grade distribution—and not only focus on high-grade or run of mine (ROM) mineralisation.

3. Case Study Overview

The case study presents a geometallurgical approach instigated at the San Antonio project, located in South America. The deposit is characterised by sheeted vein mineralisation, which comprises multiple parallel to sub-parallel veins. The planned programme aimed to investigate and quantify grade and metallurgical recovery variability within the ore zone and to inform a pre-feasibility study (PFS). A staged diamond drilling programme during 2006–2008 focused on the sheeted vein zone and resulted in the definition of an Inferred Mineral Resource of 1 Mt at 5.3 g/t Au (for 5.3 t or 170,000 oounces Au). An Exploration Target of 3 Mt to 6 Mt with a grade range of 4–8 g/t Au was defined along the 2.5 km zone. In 2013, a four-stage programme was instigated, including:

(1) Underground access and ore characterisation study.
(2) Geometallurgical drilling programme.
(3) Bulk sampling programme.
(4) Resource estimate update and pre-feasibility study (PFS).

The ore characterisation study aimed to access the mineralisation and undertake orientation testwork to support design of the geometallurgical programme. The overall programme aimed to upgrade part of the Inferred Mineral Resource to the Indicated category to support the first few years of production and a PFS. The programme was designed to maximise data outputs and to produce both grade and metallurgical recovery block models. The geometallurgical drilling programme was completed in late 2015 together with an updated resource estimate. The bulk sampling programme and additional drilling was completed in 2016, the PFS and updated resource estimate was completed in May 2017, and trial mining undertaken during June–July 2017. Continuous production commenced in September 2017.

4. Geology and Mineralisation

The deposit comprises a sheeted vein zone (SVZ), which can be traced along strike for 2.5 km and is hosted in a composite granodiorite intrusion. It strikes 075–085° and dips at 80–90° E. The deepest diamond drill hole confirms the zone to 475 m below surface. The resource zone represents 550 m of strike to a depth of 350 m. The SVZ retains a relatively consistent width of 15 m (ranging 12 to 16.5 m) along the drilled zone. All veins are dominated by quartz, with locally up to 20% sulphides. Sulphides are typically pyrite with traces of galena and sphalerite. The pyrite contains free-gold <100 μm in size. Individual veins range from a few mm to 35 cm in width.

A central 4–5 m wide core zone (CZ) displays the highest density of veins, strongest alteration and highest grades (Figure 1). Within the CZ, some veins reach 35 cm in width. The hangingwall (HW) and footwall (FW) zones are characterised by a lower density of veins, weaker silicification and lower grades. Outside of the SVZ, the host granodiorite displays weak sericitic alteration. There is localised strong silicification, which results in quartz-rich zones comprising quartz veins and silicified wall rocks.

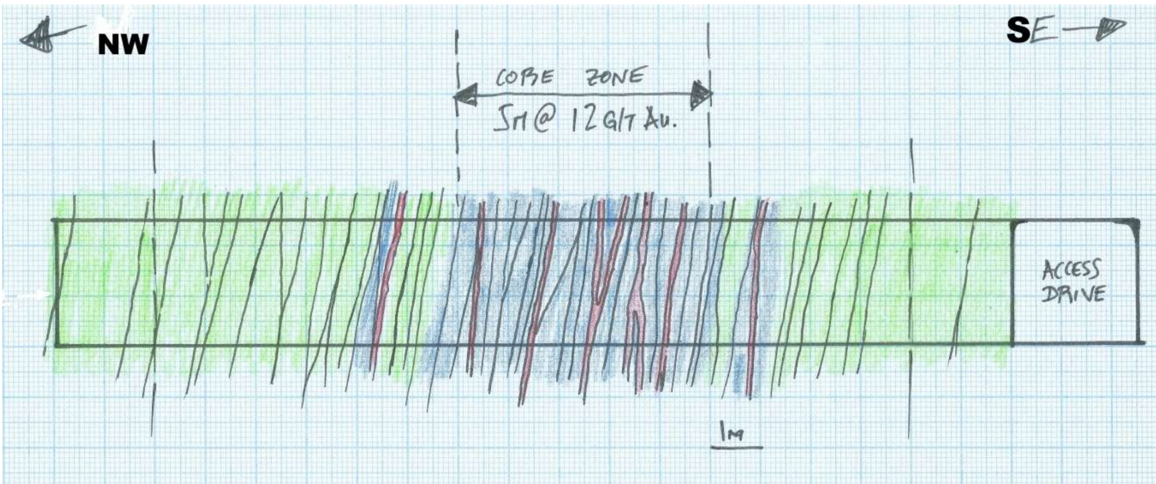

Figure 1. Original geological field sketch of cross-cut #2 through the SVZ (XC.2; see Figure 2) NE wall. Green: weak to moderate silicification; and Blue: strong silicification. Grades from wall channel samples.

5. Resource Development

A staged 14,000 m diamond drilling programme was undertaken between 2005 and 2008, on a variable 40–100 m by 40–100 m pattern producing NQ drill core. NQ is a standard wireline diamond drill type, where the core diameter is 47.6 mm. An Inferred Mineral Resource of 1 Mt at 5.2 g/t Au at a cut-off of 4 g/t Au was reported in accordance with the 2004 JORC Code. The resource was based on a bulk-mine scenario across the SVZ.

The geometallurgical drilling programme was undertaken during 2015, which reduced the drill spacing to 20 m by 20 m. A resource estimate based on the 2005–2008 and 2015 drilling yielded 1.5 Mt at 6.4 g/t Au. This comprised 630,000 t at 6.6 g/t Au in the Indicated category. The resources were reported in accordance with The JORC Code 2012 [9]. The resource was based on a bulk-mine scenario across the SVZ and reported at a 4 g/t Au cut-off.

The ore characterisation study accessed a 65 m strike section of the SVC via a hangingwall drive and cross-cuts (Figures 1 and 2). Comparison of drilling results from this area and elsewhere in the resource demonstrated that the proposed area displayed no principal differences with respect to geology, mineralogy and grade distribution.

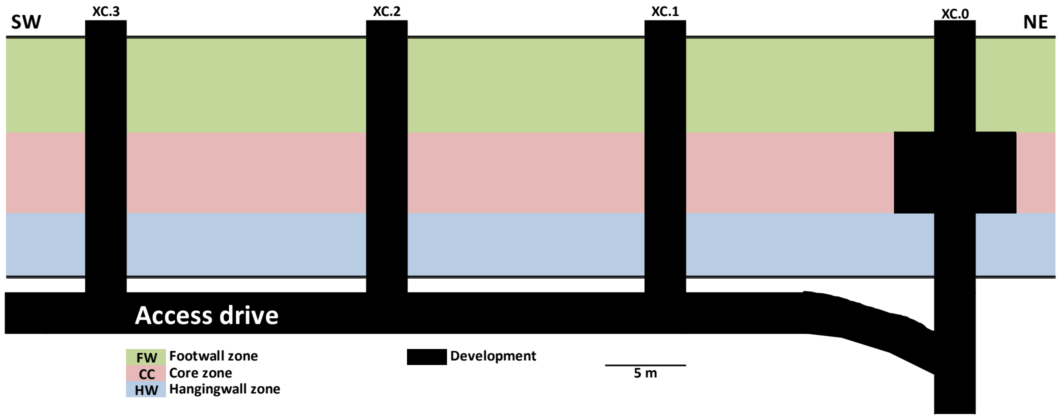

Figure 2. Plan of underground exploratory development within the SVZ.

All development was undertaken by hand-held mining. Three 20 m (2.5 m by 2.5 m) cross-cuts (XC.1–3) were placed 15 m apart across the mineralised SVZ (Figure 2). Subsequently, a fourth cross-cut (XC.0) was developed in early 2016 as part of the PFS (Figure 2). During this programme, the side walls of all cross-cuts were both saw-cut channel and chip-channel sampled every metre. Across the two stages of development, a total of 1330 t of mineralised SVZ material was mined and stockpiled for processing.

An additional 2000 m of drilling was undertaken as part of the PFS, which focused on part of the Indicated resource around the underground development. This resulted in a drill spacing of approximately 10 m by 10 m. In this case, samples were collected as 1 m whole core composites and processed via the geometallurgical drilling programme protocol.

6. Ore Characterisation Study

6.1. Rationale

The ore characterisation study was a pre-cursor planning stage to the geometallurgical drilling programme. It aimed to define gold particle size distribution, gold deportment, and metallurgical characteristics. The study included review of previous drill core and assay results. New testwork was based on micro-bulk samples collected from underground.

6.2. Sampling

Stage 1 of the study involved cutting channels in each wall of the three cross-cuts and collecting a sample every 1 m. Channels were cut using a diamond saw to achieve a high quality and a consistent support of 3 kg/m. Diamond saw cut channels generally show lower DE, EE and WE [8]. Each sample was submitted to a laboratory for a total screen fire assay (SFA). All cross-cut walls were mapped, particularly noting vein texture, mineralogy and presence of visible gold. The area was domained into four grade categories: very low grade (<0.5 g/t Au), low-grade (0.5–3 g/t Au), run of mine grade (3–10 g/t Au) and high-grade (>10 g/t Au).

Stage 2 involved the collection of a series of 30 kg samples of which eight were re-combined after crushing to form three 240 kg master composite samples from each grade domain (low to high grade). Multiple channels were cut using a diamond saw to achieve a consistent 8 kg/m sample. Each field sample was sealed into plastic pails as four approximately 30 kg lots for shipping to the laboratory.

6.3. Testwork

At the laboratory after drying, each 30 kg sub-sample was crushed to P_{80} −5 mm. All sub-samples were recombined via a rotary sample divider to form a 240 kg master composite. Each master composite represents a grade domain and was used as the test sample. Each master composite was treated via an established protocol [26]. This protocol is designed to investigate coarse-gold ores, principally from a perspective of gold particle size determination. It is not intended to replace the GRG test, though provides a preliminary evaluation of GRG. The protocol is based on three stages comprising: (1) preliminary geological and mineralogical characterisation; followed by two processes addressing; (2) metallurgy; and (3) gold particle size determination after liberation.

In addition to the gold testwork, a series of samples were collected for Bond ball mill work index tests. Ten initial 15 kg samples were collected from the cross-cut walls, based on the proportion of quartz (vein and/or silicification) versus host granodiorite.

6.4. Study Outcomes

6.4.1. Data Overview

The study confirmed the nature of the mineralisation; the key results are presented in Tables 1 and 2 and Figure 3. The overall conclusion was that the deposit is coarse gold-dominated (>40% of a 6.5 g/t Au grade), which indicates both gravity processing amenability and sampling challenges. Gold fineness ranges between 880 and 910, generally around 900.

Table 1. Summary of ore characterisation testwork results in the SVZ.

Master Sample	Head Grade (g/t Au)	GRG (%)	Leach Recovery (%)	Total GRG and Leach Recovery (%)	Gold >100 µm	d_{95Au} (µm)
MC.1	2.3	14	67	81	15%	150
MC.3	5.8	56	33	89	40%	650
MC.2	10.9	67	27	94	49%	1100
All	6.3	57	34	91	42%	680

Table 2. Optimal sample mass required to achieve theoretical "field" precision of ±20% at the 90% confidence limits for different grade-liberation diameter scenarios within the SVZ.

Grade Type	Grade (g/t Au)	d_{95Au} (µm)	Optimum Sample Mass (kg)	Sampling Constant—K (g/cm$^{1.5}$)
High [1]	11	1100	185	5700
Run of mine (ROM) [2]	6	650	70	5300
Low [3]	2	150	3	1800

[1] Table 1 sample MC.2; [2] Table 1 sample MC.3; [3] Table 1 sample MC.1.

Sampling d_{95Au} values were up to 1100 µm (Table 2). The lower grade sample was dominated by finer gold, whereas the higher grade samples contain more coarse gold (Figure 3). This testifies the presence of free, often visible gold which is confirmed from field observation and mineralogical testwork.

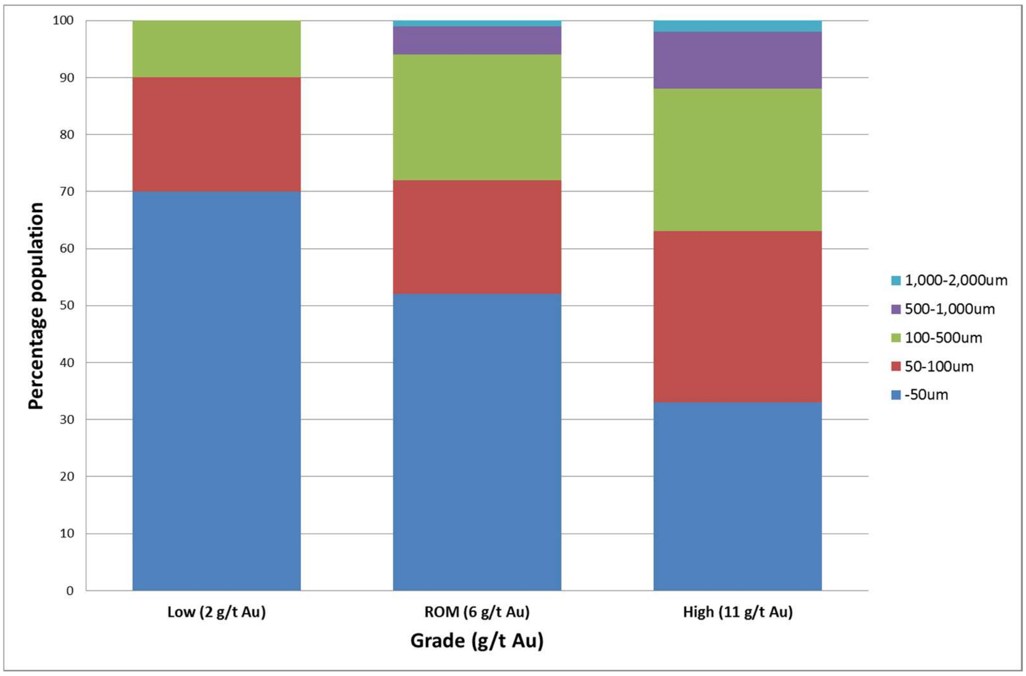

Figure 3. Percentage gold particle distribution by mass for SVZ grade domains.

ROM grade (nominally 6 g/t Au) yields a d_{95Au} of 650 µm, which gives a sampling constant (K) of 5300 $g/m^{1.5}$ (Table 2). K values between 1000 $g/m^{1.5}$ and 5000 $g/m^{1.5}$ indicate major sampling challenges that are likely to require specialised protocols. Values >5000 $g/m^{1.5}$ indicate the need for specialised protocols and potentially bulk sampling. The K value confirms the potential challenges of evaluating San Antonio mineralisation.

6.4.2. Optimum Mass Requirements

Review of core logs indicates that, whilst rare gold particle clustering exists it is not likely to materially impact sampling. Based on the ROM grade and d_{95Au} of 650 µm, the optimum field sample mass to achieve 15% precision is 70 kg based on Poisson statistics [26]. It should be noted that the d_{95Au} value is a minimum value, given that the test protocol method will result in some gold particle size reduction.

Analysis of previous drill data and the characterisation data shows that the background SVZ grade is around 2–3 g/t Au. This appears to corroborate the low grade master composite sample (MC.1) of 2.3 g/t Au (Table 1). This yielded less coarse gold compared to the other master composites and a lower d_{95Au} of 150 µm. Gold deportment in this sample relates to more sulphide-hosted gold. Low-grade background mineralisation indicates an optimum field sample mass of 3 kg.

6.4.3. Screen Fire Assay–GRG

A study was undertaken to investigate the use of screen fire assay as a proxy for GRG. Thirty 20 kg channel samples were collected from the underground development across the different grade domains. Each individual sample was saw-cut, sealed in a plastic pail and shipped to the laboratory. After drying, each sample was crushed to 80% passing (P_{80}) −1.5 mm and 5 kg split off by a rotary sample divider. The remaining 15 kg was submitted for GRG testing and the 5 kg retained for screen fire assay. The splitting of 5 kg from 20 kg at a P_{80} −1.5 mm yields an FSE of ±21%, which is moderately high though has to be tolerated as 15 kg is required for the GRG test. The 15 kg sub-sample was submitted for a single-stage GRG test [5] and the 5 kg sub-sample for a screen fire assay. Both sample sets were pulverised to P_{80} −100 µm. A disposable nylon screen was used for the screen fire assay. The results are shown in Figure 4 and indicate a reasonable correlation.

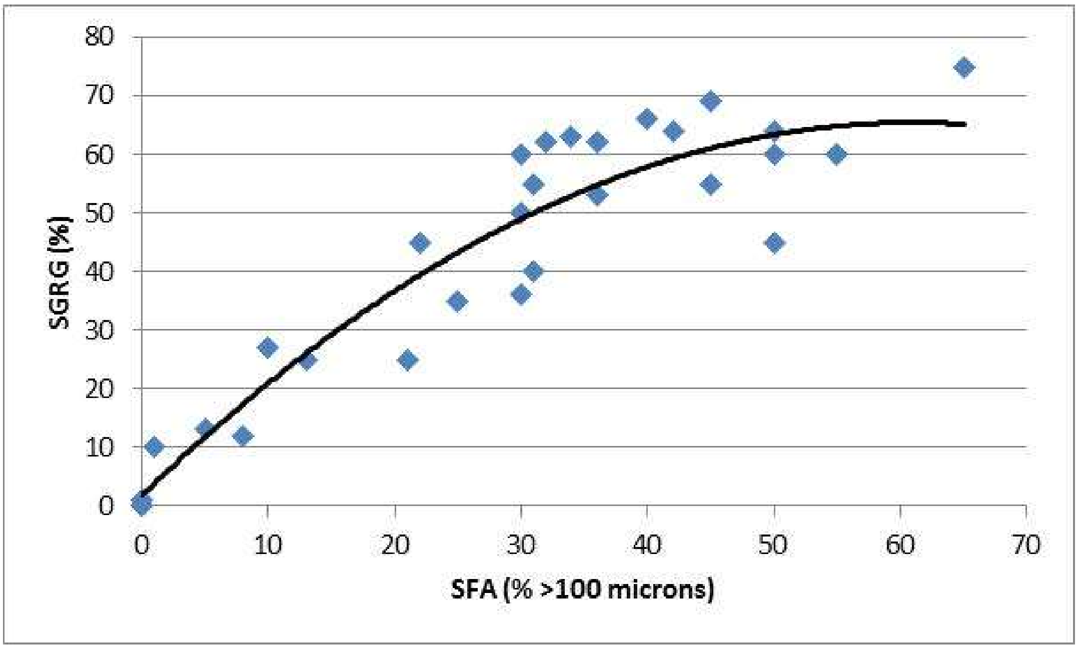

Figure 4. Relationship between single-stage GRG (SGRG) and screen fire assay (SFA) coarse gold values in the SVZ.

6.4.4. Previous Sampling Protocol

The characterisation study showed that mineralisation is coarse gold-dominated and that additional sampling should reflect this, particularly as it would be used to support a PFS. Previous drilling utilised 1 m half core samples, which were crushed and pulverised in their entirety and subsequently 30 g taken for fire assay. This approach is considered inappropriate for coarse gold mineralisation as it is prone to high FSE (±35% in this case) and risks high GSE during splitting of the pulp [23,24,27].

7. Geometallurgical Drilling Programme

7.1. Introduction

The geometallurgical drilling programme marked the commencement of working towards a PFS and increase in resource. The project owners were focused on defining resources for the first 2–3 years of a mining operation and getting into production quickly. As a result, they saw value in the geometallurgical approach to understand the SVZ. The key driver was to resolve variability within the deposit and feed this into the PFS and reduce technical risk. The success of the project was envisaged through ensuring good communication between disciplines, strong project management and work protocols, the right facilities and tools, and an open and collaborative team environment.

7.2. Drilling and Sampling Strategy

The programme included 45 HQ holes, where HQ is a standard wireline diamond drill type which yields a core diameter of 63.5 mm. A spacing of 20 m by 20 m was applied, based on variographic analysis of previous drill data to achieve an Indicated resource and support an optimal estimation block size. The programme was required to provide maximum information from the drill core, to include geology/lithology, geochemistry, rock mass properties, metallurgical recovery and grade.

During the design stage, discussions were held around the use of 1 m versus 2 m composites and half core versus whole core samples. The width of the mineralisation and likely bulk mine approach allowed the use of 2 m composites. Several issues arise when cutting a drill core in half, which relate to gold loss (e.g., extraction error) through: (a) actual cutting (10–15% of core lost to cuttings); and (b) plucking of gold and/or gold-bearing sulphides from the cut surface [14,15,23]. Given these issues and the coarse gold nature of the mineralisation, a large sample mass was preferred and the choice was made to use whole core samples.

On the project site, detailed core logging and high resolution photography was undertaken prior to laboratory submission. Prior to sample collection, a rigorous internal and third party peer review system was employed to verify the existence of the core, and to check log and photograph quality. Each sample was placed into a labelled self-locking plastic pail for transportation to the laboratory.

7.3. Laboratory Protocol Development

A specific geometallurgical drill core protocol was developed for the determination of key gold recovery and grade parameters (Figure 5). As drilling was based on 2 m HQ core, each whole-core sample had a mass of approximately 17 kg. After laboratory submission and drying, each sample was crushed to P_{80} −1.5 mm. A 0.5 kg sub-sample was split off via a rotary sample divider and retained for future reference, including geochemical and mineralogical analysis. The splitting of 0.5 kg from 17 kg at a P_{80} −1.5 mm yields an FSE of ±77%. Whilst high, this figure was accepted as it was important not to compromise the rest of the sample for testwork. Importantly, the 16.5 kg split from 17 kg at a P_{80} −1.5 mm yields an FSE of ±2.5%, which is acceptable.

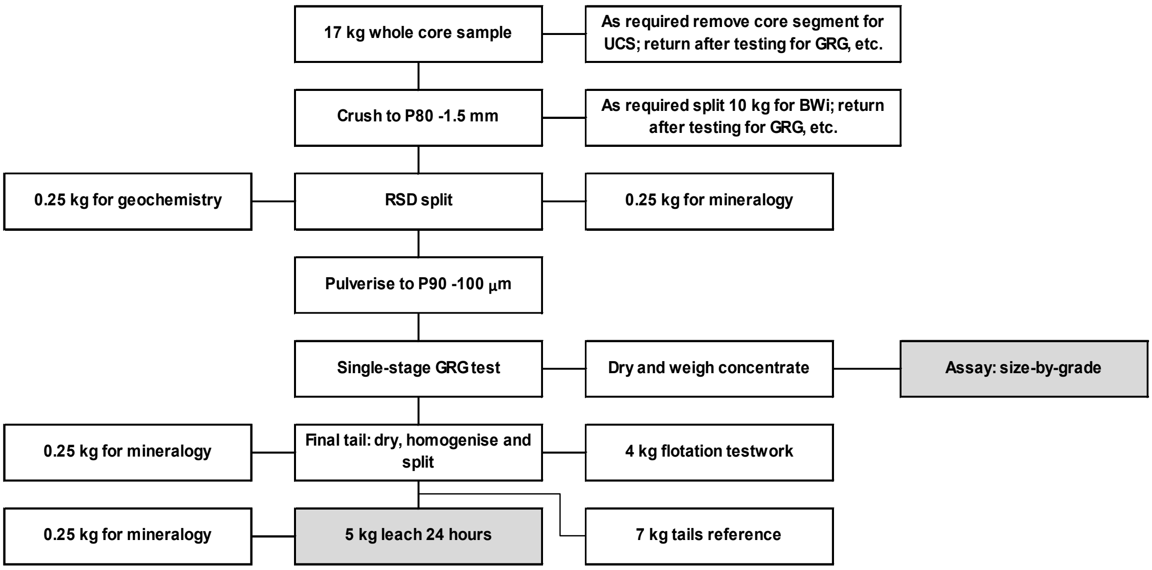

Figure 5. Geometallurgical testwork flow-sheet for drill core and underground samples. The GRG test concentrate grade and tails leach grade (grey shaded boxes) are recombined to provide a head grade for the sample. UCS: uniaxial compressive strength; BWi: Bond ball mill work index; RSD: rotary sample divider.

The 16.5 kg sample was then milled to P_{80} −100 µm prior to a single-stage GRG test using a Knelson concentrator [5]. The gravity concentrate is subjected to sizing analysis and assay, and the tails retained for leach, flotation and mineralogical analysis. A 5 kg sub-sample was split off for leaching. At this stage the coarse gold has been removed and the nominal P_{80} is −100 µm, thus the split FSE is ±1.5%. The tails leach result is recombined with the gravity grade to give a sample head grade.

7.4. Quality Assurance/Quality Control

QAQC is a major consideration in any programme that will support a Mineral Resource estimate. There are well-established approaches for grade samples, but less so for metallurgical testwork [8,28]. Table 3 summarises the QAQC undertaken throughout the geometallurgical drilling campaign (Figure 5). A key part of the study was to ensure that QC data were reviewed on a batch-by-batch basis and not left until the end of the programme.

Table 3. Summary of QAQC applied during the geometallurgical drilling programme.

Action/Activity	Responsibility	Action	Performance Expectation
Core logging and sampling Core verification	Project	Internal peer review of all photographs and logs against the core Third party peer review and verification of all core, photographs, logs and samples	Compliance to core logging protocol. Nominal expectation of 95% compliance
Sample security	Project	Core trays in secure storage All samples placed and transported in self-locking plastic pails Completion of chain of custody paperwork Third party supervision of collection and dispatch process	Must occur for all core/each and every sample
Field duplicates	-	None taken due to whole core sample/assay	-
Coarse duplicates	-	None taken due to whole core sample/assay	-
Pulp duplicates	Laboratory	LeachWELL tails duplicates (5 kg) at rate of 1 in 20	Duplicate sample grade within 90% ±10% HARD
Certified reference material (CRM)	Project and laboratory	Four CRMs applied ranging low to high grade. Used for LeachWELL determination on GRG tails. CRMs (0.2 kg) leached with fire assay on tail. Rate of 1 in 15	2δ–3δ ("warning") re-assay 25% of batch >3δ ("action") re-assay 100% of batch
Standards (In-house)	Laboratory	Used to monitor GRG test/Knelson concentrator efficiency. Fifteen kilograms of blank fine-crushed rock dosed with a known quantity of coarse gold. Inserted at 1 in 20	90% of samples to be within ±20% of expected grade (e.g., >80% recovery)
Blanks	Project and laboratory	Blanks granitic material, 50 assays gave grade <0.01 g/t Au 15 kg blank material processed through entire protocol at rate of 1 in 20 15 kg blank placed after sample with visible gold and processed through entire protocol 2 kg blank material leached at a rate of 1 in 20	Blank assay <0.1 g/t Au
Barren flushes of Knelson unit	Laboratory	2 kg of barren sand flushed the Knelson concentrator after each sample Knelson concentrator cleaning sand assayed by LeachWELL at 1 in 10. If visible gold reported in sample, cleaning sand automatically LeachWELL	Less than 1% gold loss in blank assay compared to primary sample head grade
Umpire	Laboratory	Duplicate tails samples submitted to an external laboratory for LeachWELL at a rate of 1 in 25	Duplicate sample grade within 90% ±10% HARD
Laboratory audit	Project	Weekly audits during the programme by project staff. Monthly audits by an independent third party	Full compliance of all procedures

HARD: Half Absolute Relative Difference.

7.5. Programme Implementation

Key implementation points were at the on-site core shed and off-site laboratory. The programme produced 5500 m of core and 340 primary samples. A core shed flowsheet was instigated and written protocols produced for each activity (e.g., geological and geotechnical logging; EQUOtip usage; specific gravity determination; sample collection; and security). Staff training was undertaken prior to commencement. During the programme, on-going staff mentoring and review was undertaken by consultants, and the project manager, geologist and metallurgist. The independent laboratory was included in all discussions regarding the protocol. Laboratory protocols and staff training were undertaken prior to the programme. Most of the activities in the laboratory were fairly standard, though it was critical to ensure that the protocol was followed to maintain quality. The QAQC programme was instigated to ensure this aim (Table 3).

7.6. Geometallurgical Drilling Programme Results

7.6.1. Data Summary

The geometallurgical programme included 5500 m of core drilling, which provided 340 samples. The core was logged and sampled to provide data for several disciplines (Table 4). All data are stored in a relational database.

Table 4. Summary of data collected during the geometallurgical drilling programme.

Key Orebody Knowledge	Data Collection	Data Statistics/Number of Tests/Samples
Rock type	Geological and structure logs Core photography	All core logged and photographed wet and dry
Geochemistry and mineralogy	Handheld X-ray fluorescence Total sulphur Base metals Optical and automated mineralogy	Approximately 1000 m core scanned 340 340 12 elements 340
Rock mass properties	Fracture logging and core recovery EQUOtip Bulk density Uniaxial compressive strength	All core geotechnically logged Reading points every 20 cm 205 ore/50 waste 48 ore/15 waste
Comminution	Bond ball mill work index	48 ore/8 waste
Metallurgical recovery	GRG (primary sample) Leach (tailings sub-sample) Flotation (tailings sub-sample)	340 340 105
Grade	Gold grade	340 back-calculated head grades

7.6.2. Grade

The original drilling, based on 1 m half core samples, yields a mean grade of 2.7 g/t Au, RSV of 360% and a nugget effect of 78%. The geometallurgical drilling, based on 2 m whole-core samples, yields a mean grade of 3.3 g/t Au, RSV of 120% and a nugget effect of 55% (Table 5). Even for the larger support samples, the RSV was still high, which reflects the heterogeneous nature of the mineralisation. Compared to the original drilling, the new drilling displays an overall grade upside of 20%.

Table 5. Summary of data from the geometallurgical drilling programme across the SVZ.

Data Type	Metric	Summary of Results	Comment
Mineralogy	-	Sheeted veins: gold dominantly free in quartz veins, with some pyrite-hosted gold	
Geochemistry	-	Very low (<<0.1 ppm) Hg, As, Bi and Sb Key base metals include Fe, Pb and Zn which correlate with pyrite, galena and sphalerite	
Gold recovery	GRG	Data range: 65% Mean: 51% RSV: 71%	Overall shows dominance of GRG; generally 80% > 40% GRG. When GRG drops, then finer gold is locked in sulphide which is recovered by leaching
	Leach	Data range: 61% Mean: 35% RSV: 64%	
Gold grade	Grade	Data range: 158.7 g/t Au Mean: 3.3 g/t Au RSV: 120%	Variability of gold grade across the deposit. RSV for population lower than previous half core sampling by fire assay (360%)

7.6.3. Gold Recovery

Gold recovery was determined through GRG and leach tests (Table 5). Recovery reflects the relative proportions of free gold in quartz versus sulphides. The presence of sulphide-rich vein sets reflects locally less GRG and increased fine gold. These features are reflected by two gold recovery domains within the Indicated resource model. The AUREC.1 domain reflects dominantly free-gold with high GRG potential (e.g., CZ), whereas the domain AUREC.2 reflects the sulphide-hosted gold amenable to leaching (e.g., HW and FW zones).

7.6.4. Comminution

Comminution relationships were determined through 48 paired uniaxial compressive strength and Bond ball mill work index samples (Table 5), which were subsequently correlated with EQUOtip values. Comminution variability ranged from hard-very hard (18–27 kWh/t) in quartz-dominated silicified zones to soft (8–10 kWh/t) for kaolinised/sheared fault zones. The majority of the mineralisation relates to the hard classification. Three comminution domains are identified within the Indicated resource model; defined as COMM.1—very hard (80% of model); COMM.2—hard (15% of model); and COMM.3—soft-medium (15% of model). Localised soft zones (COMM.3) reflect cross-faulting which results in sheared granodiorite with some kaolinite alteration. The CZ is dominated by COMM.1. The RSV for the hard and very-hard (COMM.1–2) Bond work index data (n = 39) is 15%, indicating its low variability.

7.7. QAQC Results

Throughout the geometallurgical drilling programme a thorough QAQC programme was applied, which qualified the integrity of the testwork and assay results (Tables 3 and 6).

Table 6. Summary of geometallurgical drilling programme QAQC programme results.

Action/Activity	Responsibility	Performance Expectation	Actual Performance
Core logging and sampling Core verification	Project	Compliance to core logging protocol. Nominal expectation of 95% compliance	Minor logging issues noted during peer review, which were corrected prior to sampling
Sample security	Project	Must occur for all core/each and every sample	No security breaches. No seals broken or samples lost
Pulp duplicates	Laboratory	Duplicate sample grade within 90% ± 5% HARD	91% ± 10% HARD
Certified reference material (CRM)	Project	2δ–3δ (“warning”) >3δ (“action”)	92% within 2δ; 5% within 2δ–3δ; 3% above 3δ Relative bias within ±9%
Standards (In-house)	Laboratory	90% of samples to be within ±20% of expected grade (e.g., >80% recovery)	93% within ±20%
Blanks	Project—sample blanks Laboratory—Leach blanks	Blank assay <0.1 g/t Au	98% <0.1 g/t Au 2% 0.1-0.3 g/t Au
Barren flushes of Knelson unit	Laboratory	Less than 1% gold loss in blank assay compared to sample head grade	100% <1% gold loss
Umpire	Laboratory	Duplicate sample grade within 90% ± 10% HARD	95% ± 10% HARD
Laboratory audit	Project	Full compliance across procedures	A number of minor issues were noted. None deemed material. All matters corrected

7.8. Geometallurgical Drilling Programme Risk

It is critical that sample representivity, testwork quality and overall fit-for-purpose application of results are communicated to stakeholders, particularly when they are publicly released [8]. Table 7 provides an overview of representivity and fit-for-purpose nature of testwork results for the case study. The geometallurgical drilling programme was designed to suit the mineralisation style and reduce project risk—the programme indicates a low risk throughout.

Table 7. Risk review of geometallurgical drilling programme sampling, testwork and assay, after [8].

Key Parameter		Comment	Dominant TOS Error	Risk Rating
1	Spatial distribution and number of samples	20 m by 20 m (locally to 10 m by 10 m) drilling grid yielding 436 variability composites	-	Low
2	Sample mass	Multiple 17 kg 2-m whole-core samples Composite mass across the SVZ ranges between 85 kg to 120 kg dependent upon SVZ width and core intersection angle	-	Low
3	Degree of domaining	Domains sampled across FW, CZ and HW	-	Low
4	Collection and handling	All samples collected according to protocols written to comply with TOS All samples sealed prior to transportation	-	Low
5	Transport and security	Chain of custody procedures in place All samples secured into locked container for transportation	-	Low
6	Preparation	Full sample crushed and pulverised All equipment cleaned between samples	-	Low
7	Testwork (incl. QAQC)	Full sample through Knelson concentrator Compliant QAQC (Table 6), with documentation across sample collection, preparation and testwork Rigorous cleaning of laboratory equipment Accredited laboratory	-	Low
8	Assay (incl. QAQC)	Compliant QAQC (Table 6), with full documentation across sample collection, preparation and assaying Rigorous cleaning of laboratory equipment Accredited laboratory	-	Low
9	Validation	Geometallurgical drilling programme resource estimate validated by bulk sampling programme, including 5 t composite and five 5 t pilot samples, followed by 1330 t trial process lot	-	Low
			Sample representivity (1)–(5)	Low
			Testwork quality (6)–(8)	Low
			Fit-for-purpose rating (1)–(9)	Low

8. Bulk Sampling Programme

8.1. Introduction

Two stages of underground development were undertaken as part of the ore characterisation study and PFS programmes. In both cases, material across the FW, CZ and HW domains were kept separate for later processing. The bulk sampling programme aimed to verify local grade and metallurgical recoveries, as well as allowing detailed geological and geotechnical mapping. Three stages of pilot testing were undertaken, ranging from 100 kg sub-samples to full plant processing (Table 8).

Table 8. Summary of bulk sampling programme stages and tonnes treated from the SVZ.

Bulk Sampling Stage	Activity	Total Mass Treated
I	Fifty 100 kg sub-samples split from lot of 855 t for laboratory testing	5 t
II	Five 5 t sub-samples split from lot of 850 t for laboratory pilot testing	25 t
III	Thirteen batches for trial plant processing	1262 t

The bulk sampling programme area was in-fill drilled to a 10 m by 10 m spacing from 20 m by 20 m spacing (Figure 2). This was undertaken to permit comparison between models based on both drill patterns and avoid some of the issues of comparing bulk sample grades with wide-spaced drill estimates [25].

8.2. Bulk Sampling Programme Phase I

The stockpiled mineralisation extracted from the three cross-cuts was crushed in its entirety on site to P_{80} −1.5 cm. For Phase I, 5 t was split off via a linear splitter as fifty 100 kg sub-samples. These were shipped to a commercial laboratory for processing. At the laboratory, each 100 kg sub-sample was run through a small circuit comprising comminution (P_{80} −100 µm), gravity (Knelson concentrator) and leaching. Splitting of each 100 kg sub-sample yielded a fundamental sampling error (FSE) of ±31%. However, the combined 5 t total sub-sample from 855 t of crushed material yields an FSE of ±4%.

8.3. Bulk Sampling Programme Phase II

In the second pilot test, 25 t was split from the remaining crushed material by linear splitter. Five 5 t sub-samples were split to form the 25 t lot, where each 5 t split yielded an FSE of ±5%. At the laboratory, each 5 t sub-sample was batched through a pilot circuit with gravity (Knelson concentrator) and the tails leached for 24 h. The remaining crushed material was stored on site until such a time that it could be processed in its entirety or used for additional testwork.

8.4. Bulk Sampling Programme Phase III

Phase III of the pilot testing was undertaken at a process plant located 7 km from the mine site. All mineralisation was trucked to the plant as mineralisation domain batches by cross-cut (Table 9). The plant comprised primary and secondary crushers; ball mill (P_{80} −125 µm); gravity circuit (12″ Knelson and Wilfley table) and a carbon in leach circuit. Gold traps were located pre- and post the ball mill. The pre-mill trap caught minimal gold, whereas the post-mill trap caught gold particles >500 µm. Thirteen batches were fed through the plant at a rate of 5 t per hour (Table 9). A mill balance was undertaken at the end of each batch. A mechanical sampler was used on the tails stream.

Table 9. Development tonnes (t) mined across cross-cut and SVZ mineralisation domain.

Domain	XC.0	XC.1	XC.2	XC.3	Total
FW	114	110	117	115	456
CZ	95	97	93	94	379
HW	72	76	75	72	295
Total	281	283	285	281	1330

Rounded to the nearest tonne. Cross-cuts (XC) shown in Figure 2.

Between each batch, the plant was cleaned and 25 t of barren granodiorite was run to flush the system. For reasons of practicality, the ball mill was only cleaned out before and at the end of the programme. On final cleaning, 187 g of gold was recovered, which was apportioned to each lot based on fraction of total GRG produced. Table 10 shows the reconciled head grades across the cross-cuts and mineralisation domains.

Table 10. Plant head grades (g/t Au) for SVZ mineralisation domains across development.

Domain	XC.0	XC.1	XC.2	XC.3	Mean
FW	5.0	3.5	5.6	3.1	4.3
CZ	26.1	16.5	19.3	13.7	18.9
HW	4.2	6.5	5.6	5.3	5.4
FW + HW	Tonnage weighted grade				4.7
All	Tonnage weighted grade				9.5

Cross-cuts (XC) shown in Figure 2.

In the CZ domain of XC.0, two 5 m wide 2.5 m deep cuts were developed into the side walls (Figure 2). The two combined yielded a tonnage of 190 t, which were batched through the plant to give a head grade of 17 g/t Au. The mean grade of the entire 1330 t lot from the four cross-cuts was 9.5 g/t Au. The total grade of the FW and HW zones is 4.1 g/t Au, with the CZ (5 m wide) at 16.6 g/t Au. Table 11 shows the GRG and leach recoveries within the mineralisation domains. The CZ domain contains more GRG than the HW and FW domains. The CZ total gold recovery (GRG-leach) was 94%, with the overall for all domains at 92%.

Table 11. Gold recovery by SVZ mineralisation domain.

Domain	Head Grade (g/t Au)	GRG (%)	Leach (%)	Total Recovery (%)
FW	4.3	27	63	90
CZ	18.9	66	28	94
HW	5.4	31	61	92
All	9.5	45	47	92

8.5. Bulk Sampling Programme QAQC

Table 12 summarises the QAQC undertaken during the bulk sampling programme, which included barren flushes and blanks for all testwork, and blanks, CRMs and pulp duplicates for all assays.

Table 12. Summary of bulk sampling programme QAQC.

Bulk Sampling Stage	Testwork Scale	QA Activities	Insertion Rate	QC Metrics
I	Bench: 50 × 100 kg	Equipment cleaning via barren flushes (20 kg) Blank sample processing (25 kg) All subsequent laboratory assay apply standard QA including blanks, CRMs and pulp duplicates (where possible)	Between all 1 in 5 Generally 1 in 10	- Blanks <0.1 g/t Au Standard expectation (see Table 3)
II	Pilot: 5 × 5 t	Equipment cleaning via barren flushes (500 kg) Blank sample processing (1.5 t) All subsequent laboratory assay apply standard QA including blanks, CRMs and pulp duplicates (where possible)	Between all 3 processed Generally 1 in 10	- Blanks <0.1 g/t Au Standard expectation (see Table 3)
III	Plant: 13 × 72–117 t batches	Equipment cleaning via barren flushes (500 kg) Blank sample processing (5 t) All subsequent laboratory assay apply standard QA including blanks, CRMs and pulp duplicates (where possible)	Between all 10 processed Generally 1 in 10	- Blanks <0.1 g/t Au Standard expectation (see Table 3)

8.6. Analysis of Gold from Processing

All GRG concentrates underwent size-by-assay, where they were screened and smelted by fraction (Table 13 and Figure 6). Automated mineralogical analysis of leach feed and tailings confirmed that gold was generally below 100 µm, and mostly below 50 µm in size. The quantity of leach and tails gold was nominally assigned to the −50 µm fraction.

Table 13. Percentage gold particle distribution (by mass) for SVZ mineralisation domains.

Domain	−50 µm	50–100 µm	100–500 µm	500–1000 µm	1000–2000 µm	2000–3000 µm
FW	56	20	15	8	1	0
CZ	32	26	24	11	6	1
HW	53	22	13	7	5	0

The CZ contains 68% of gold >50 µm, which was recovered by GRG. The HW and FW zones contain more fine gold and less coarse gold. The d_{95Au} values across the domains were higher than the ore characterisation study results, resulting in higher K values (Table 14 and Figure 6). The differences were not substantial enough to require a change in sampling strategy.

Table 14. Sampling characteristics by SVZ mineralisation domain based on bulk sample processing.

Domain	Grade (g/t Au)	Percent > 100 µm	d_{95Au} (µm)	K (g/cm$^{1.5}$)	Max. Particle Size (µm)	Opt. Sample Mass (kg)
FW	4.3	24	800	12,700	1000	185
HW	5.4	25	950	13,000	1300	245
CZ	18.9	42	1300	5900	2000	180
All	9.5	32	1000	8000	2000	160

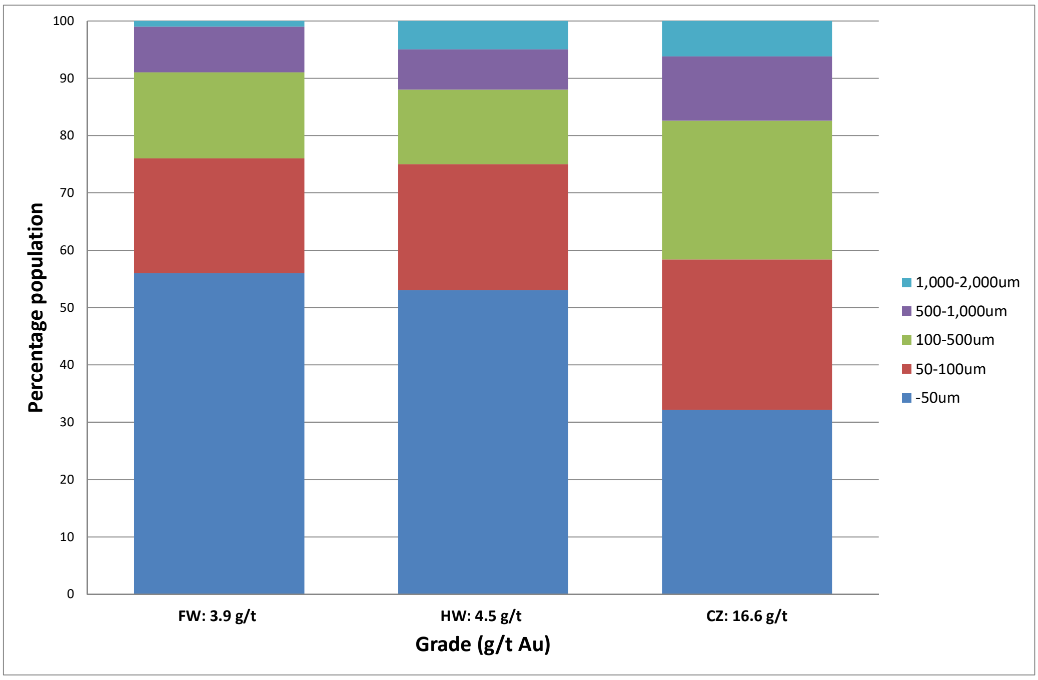

Figure 6. Percentage gold particle distribution (by mass) for SVZ mineralisation domains based on bulk sample processing.

The higher d_{95Au} values yield larger optimal sample masses in comparison to the ore characterisation study (Table 1). As the highest optimal sample mass values, the CZ and HW domains indicate that between eleven and thirteen 17 kg (2 m) whole core samples are required to provide a local grade estimate.

9. Reconciliation with Predictions

9.1. Block Model

The bulk sampling and processing results were compared with several gold grade and recovery estimates for the SVZ (Table 15). Estimates were based on a bulk mine approach to stoping the SVZ to around a 10 m width. Grade estimation was undertaken on a 6750 t test block, which contained the development (Figure 2). Grades and tonnages are reported at a zero cut-off grade, as the bulk sampling programme includes samples from the entire mineralised zone with no selectivity.

Conditional simulation was used to estimate the test area. The nugget effect was determined from the down-hole Gaussian variogram, with back transformation into non-Gaussian space giving a nugget value of 53%. Sequential Gaussian Simulation was applied, which requires that data be transformed into Gaussian space and that variograms of these values be defined. A 1 m by 1 m by 1 m grid was selected for the simulation, which would provide a good reflection of grade variability after regrouping into selective mining units (SMU). Each grid node within the orebody wireframe was simulated 100 times. The simulated points (nodes) were regrouped into 5 m (strike) by 5 m (dip) by 2 m (width) SMU blocks. The average of 100 simulations was taken as the mean grade, which corresponds to the kriged mean grade. Table 15 compares estimated volumes with the actual plant results.

Table 15. Estimated tonnages and grade versus achieved values for bulk sampling programme across the SVZ. Bulk sample location and details given in Figure 2.

Estimate	Estimation Type	Estimation Block Size [4]	Input Data	Tonnes (t)	Grade (g/t Au)	% Difference
Development horizon [1]	CS	4 × 4 × 2 m	All drilling (to 10 m by 10 m)	6750	8.8	−7
Mined development (XC.0–3) [2]	CS	4 × 4 × 2 m	All drilling (to 10 m by 10 m)	1330 [5]	10.6	+11
Development horizon [1]	OK	12 × 12 × 2 m	Geometallurgical programme drilling (20 m by 20 m)	6750	7.1	−25
Development horizon [1]	OK	15 × 15 × 2 m	Pre-geometallurgical drilling programme (30–40 m by 30–40 m)	6750	6.4	−33
Actual plant [3]	All of bulk sample		1300 t [6]		9.5	-

OK: ordinary kriging; CS: conditional simulation; [1] 65 m long by 15 m wide by 2.5 m high block around development; [2] mined development; four 2.5 m by 2.5 m by 15 m drives; [3] actual tonnes extracted from development; [4] all estimation block sizes optimised by kriging neighbourhood analysis, supported by composite variogram model; [5] includes allowance for dilution; [6] actual processed, excludes 30 t of pilot testing.

The development horizon was best estimated by the close-spaced drilling and simulation to achieve ±12% of the plant grade. The wider-spaced kriged estimates were understated plant grade by 25–33%.

Gold recovery was estimated into each SMU block via simulation (Table 16). Analysis of the geometallurgical drilling programme whole core data was used to determine the grade-recovery relationship. Prediction of the estimated total recovery was reasonably accurate, though GRG prediction was overstated, whereas leach recovery was understated.

Table 16. Estimated gold recovery and achieved values in the SVZ.

Estimate	Tonnes (t)	Grade (g/t Au)	Est. GRG Recovery (%)	Est. Leach Recovery (%)	Est. Total Recovery (%)
Development horizon	6750	8.8	49	41	90
Mined development (XC.0–3)	1330	10.6	52	42	94
Actual plant	1300	9.5	45	47	92

The Pilot Phase II results compare most favourably with the full plant Phase III results (Table 17). The Phase I results understate grade and GRG, indicating that the small samples were not representative of the coarse gold population.

Table 17. Grade and gold recovery values for bulk sampling programme Phases I to II compared to development estimate.

Pilot Stage	Tonnage	Grade (g/t Au)	Grade RSV (%)	GRG (%)	Leach (%)	Total Recovery (%)
I	50 × 100 kg (5 t)	6.2	85	37	42	79
II	5 × 5 t (25 t)	8.2	49	48	42	90
III (plant)	1300 t	9.5	-	45	47	92
CS estimate	1330 t	10.6	-	52	42	94

9.2. Variability

The project included a number of different sample types and supports, ranging from half and whole core through to 5 t pilot lots and full plant processing (Table 18).

Table 18. Summary of sample type and mass, and FSE and RSV for different stages of SVZ evaluation.

Project Stage	Sample	Individual Sample Mass	Total Field Sample Mass	Number of Assays or Tests	Assay/Test Mass	Total Assay/Test Mass	FSE (%)	RSV (%)
Orig. drilling (pre-geomet)	1 m half HQ	3.7 kg	1.55 t	412	30 g	12.4 kg	±54	360
OCS	Saw-cut channels	240 kg	720 kg	3	120 kg	360 kg	±14	105 [4]
PFS	Saw-cut 2 m channels	17 kg	1.02 t	60	16 kg	960 kg	0 [3]	145
PFS	Chip 2 m channels	5 kg	300 kg	60	5 kg	300 kg	0 [3]	296
Geomet drilling	2 m full HQ	17 kg	5.8 t	436	16.5 kg	7.2 t	±3	120
PFS: BS-I	Bulk lot	100 kg	855 t	50	100 kg	5 t	±31 [1] ±4 [2]	85
PFS: BS-II	Bulk lot	5 t	850 t	5	5 t	25 t	±5 [1] ±2 [2]	49 [4]
PFS: BS-III	Bulk: FW	115 t	450 t	4	115 t	450 t	0 [3]	31 [4]
PFS: BS-III	Bulk: CZ	142 t	555 t	6	142 t	555 t	0 [3]	27 [4]
PFS: BS-III	Bulk: mHW	76 t	295 t	4	76 t	295 t	0 [3]	25 [4]
PFS: Plant	Bulk: Dev XC.0–3	285 t	285 t	4	285 t	1330 t	0 [3]	17 [4]

[1] FSE for individual test mass; [2] FSE for composite test mass; [3] where samples show zero FSE, entire field sample was tested/assayed; [4] RSV values are based on very small datasets. OCS: ore characterisation study; PFS: pre-feasibility study; BS: bulk sampling programme.

The highest variability was seen in the 1 m half-core fire-assay results where the FSE and RSV values were ±54% and 360%, respectively. A substantive reduction in both RSV (120%) and FSE (0%) was obtained through the introduction of whole-core sampling and assay. The saw-cut channels were designed to provide a similar support to HQ core, yielding an RSV of 145% compared to 120% for the core. FSE values were effectively zero, as the entire field sample mass was assayed or tested. As expected, the nugget effect reduces as sample support increases (Table 19). The chip samples display the highest nugget effect, which reflects both their relatively small support (e.g., 2.5 kg/m) and low quality based on high DE and EE.

Table 19. Summary of sample type and support with FSE, RSV and nugget effect. All data from the SVZ.

Project Stage	Sample Type	Assay Protocol	Support (kg/m)	FSE (%)	RSV (%)	Nugget Effect (%)
Geomet drilling	2 m full HQ	Geomet protocol [3]	8.5	±6 [1]	120	53
PFS	Saw-cut 2 m channels	Geomet protocol [3]	8.0	0 [2]	145	59
Orig. drilling (pre-geomet)	1 m half HQ (Comp. to 2 m)	Fire assay	3.7 (7.4)	±54 (±45)	360 (277)	87 (71)

[1] Whole core/sample tested/assayed, FSE value reflects reference sample extracted after crushing; [2] where samples show zero FSE, entire field sample was tested/assayed; [3] geometallurgical drilling programme protocol Figure 5. ND: not determined; PFS: pre-feasibility study.

The lowest nugget effect is displayed by the geometallurgical drilling programme whole-core samples. These have the greatest support (e.g., 8.5 kg/m), high quality (e.g., low DE and EE; high core recovery) and low sampling errors post-drilling. As a result, the 53% nugget effect is likely to be dominated by the true in-situ nugget effect. The PFS saw-cut channel samples have a very similar support to the drill core and display a slightly higher nugget effect at 59%.

10. PFS Conclusions and Trial Mining

10.1. PFS Outcome

10.1.1. Mining Strategy

During the PFS, consideration was given to mining strategy with underground scenarios across: (1) a 300,000 t per annum bulk-mine operation based on longhole stoping; and (2) a high-grade selective small-scale operation based on shrinkage stoping. The conclusion was that a selective operation was the best option allowing for good project economics, a small footprint, fast production ramp-up (to 50,000 t per annum within 6 months), less capital expenditure and aligned better with the expectations of local stakeholders. In addition, the mine site and infrastructure would be used as a base for regional exploration and for the evaluation of a number of other small historic mines within 2 km of San Antonio. Overall, the selective option was considered a more economically, environmentally and socially sustainable option.

The selective mining scenario is based on a narrow high-grade zone (HGZ) within the centre of the CZ (Figures 7 and 8). It is based on a consistent 2–3 m wide zone comprising a number of 10–35 cm wide veins that run in a parallel to anastomosing fashion. The HGZ shows strong silicic alteration. Individual intersection grades across the HGZ vary from 0.01 g/t Au to 105 g/t Au, with most from 10 g/t Au to 50 g/t Au. The HGZ contains around 60–65% of the grade across the 10 m wide mineralised SVZ.

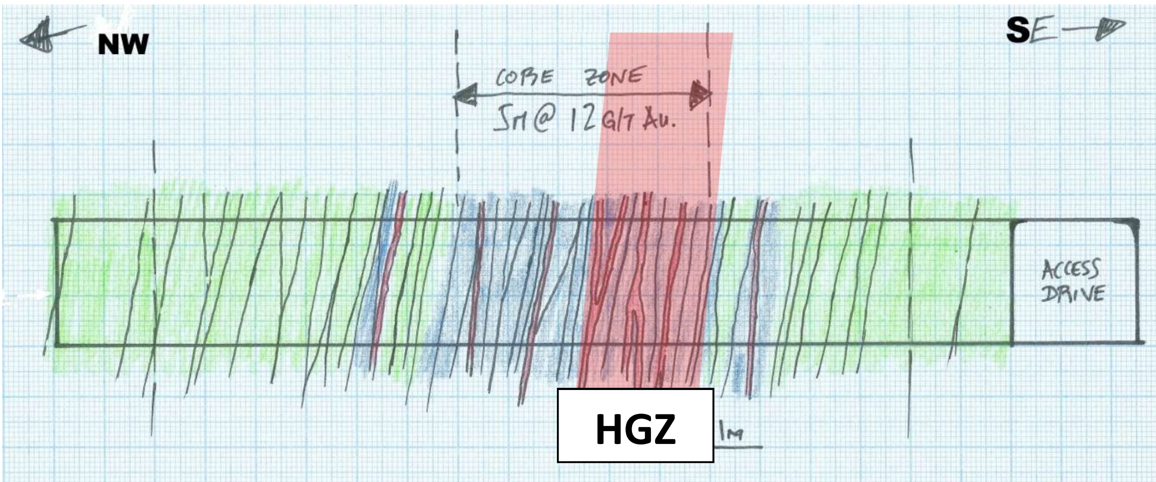

Figure 7. HGZ (2.6 m width at 19.1 g/t Au) identified within the CZ. Geological field sketch of cross-cut #2 (XC.2; Figure 2) NE wall. Green: weak to moderate silicification; and Blue: strong silicification.

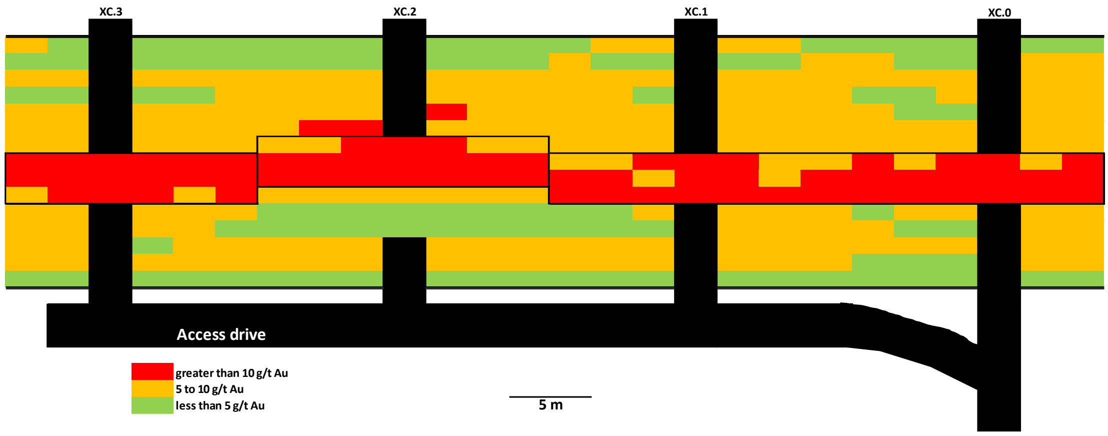

Figure 8. High-resolution grade-control model by CS showing HGZ amenable to narrow vein mining. Trial stope focussed on areas defined by black outlines.

10.1.2. Mineral Resources and Ore Reserves

A resource estimate for the HGZ yielded an Indicated Mineral Resource of 155,000 t at 21.9 g/t Au (±17%) reported at a 7 g/t Au cut-off grade to provide a base for the first 2.5 years of production. The Indicated resource yields a Proven Ore Reserve of 130,000 t at 20.3 g/t Au. An Inferred Mineral Resource of 255,000 t at 17.8 g/t Au (±44%) provides the base for Years 2.5 to 5 of production. Precision values were estimated via Conditional Simulation and reported at 80% reliability.

10.1.3. Project Financial Metrics

The financial metrics of the selective option were positive, where a five-year mine life yields a post-tax NPV_{20} of US $32M and an IRR of 140%. It is noted that 2.5 years of the five-year production period are based on Inferred resources. The operator has a high confidence in conversion of Inferred to Indicated resources based on drilling and mapping of the mineralisation. As a private entity, the operator is not required to publicly report its results and the project is fully funded by the company. The expected mine life is beyond five years, where the HGZ Exploration Target (0.5–1.0 Mt at a grade range of 10–20 g/t Au) requires drilling to define resources.

From commencement of the ore characterisation programme to completion of the PFS took three years and cost US $6.5M. The operator then committed to the trial mining programme, which required minor site works and upgrade of the processing plant costing a total of US $1M.

10.2. HGZ Trial Stoping Campaign

The company instigated a trial mining programme based on a shrinkage stope placed above the existing exploratory development (Figures 7 and 8). After preparation, the stope was extracted on a daily production basis of 25–50 t over a two-month period. The trial stope aimed to extract 2200 t at 23 g/t Au with a modelled GRG recovery of 61%. If the trial stope was successful, production would commence incrementally, whilst further development was undertaken to access additional stope blocks.

10.3. Processing

The plant used for the bulk sampling Phase III programme was used for the trial mining. Several modifications were undertaken, principally related to the decommissioning of the carbon in leach circuit due to age. A second gravity circuit was added to reprocess the primary tails. Removal of the leach circuit was considered not to be an issue, given that the HGZ had a greater dominance of GRG compared to sulphide-hosted fine gold. A moisture sample point and weightometer were added to the feed belt.

The overall performance of the campaign was excellent, yielding a reconciled head grade of 20.5 g/t Au against an estimate of 22.6 g/t Au (Table 20). The lower tonnes (−17%) represent a tight stope width with reduced planned dilution from the wall rocks.

Table 20. Comparison between estimates and actual reconciled trial stope head grade and tonnage.

Estimate Type	Input	No. Samples in Panel	Tonnes (t) (Diluted)	Grade (g/t Au) (Diluted)	% Difference to Actual
Ordinary kriging	20 m by 20 m geomet drilling	6	2200	17.6	−14
Ordinary kriging	10 m by 10 m geomet infill drilling	11	2200	22.6	+10
Mean	2 m by 2 m stope channel samples	85 [1]	2140	31.6 (uncut) 22.9 (cut)	+54 +12
Mean	Plant head samples	250	500 kg	27.8 (uncut) 24.2 (cut)	+36 +18
Actual trial stope	Process plant	-	1815	20.5	-

[1] Geometallurgical core protocol as per Figure 5; collected as 1 m composites at approximately 4–5 kg/m.

In-stope hand-cut chip-channel samples were taken on an approximate 2 m grid from the stope backs. These overstated grade by 54% (uncut) and 12% (cut), representing the relatively imprecise nature of the samples. The raw data shows an RSV of 205%.

The modified plant recovered 75% of the gold via gravity alone, with the higher performance related to the dual gravity circuit. The concentrates from the plant were acid cleaned and subjected to size-by-assay (Table 21). The maximum particle size recovered from the gravity circuit was 2500 µm, with an estimated d_{95Au} of 1700 µm. Testwork on the tails has shown that with the addition of a flotation circuit, gold recovery can be improved to 91–95%.

Table 21. Percentage gold particle distribution (by mass) for 20.5 g/t Au HGZ mineralisation.

−50 µm (Est.)	50–100 µm	100–500 µm	500–1000 µm	1000–2000 µm	2000–3000 µm	d_{95Au} (µm)	Max. Particle Size (µm)	Ms Opt. (kg)
29%	18%	29%	13%	9%	2%	1700	2500	200

Ms opt., optimum sample mass.

10.4. Duplicate Field Sampling Programme

During trial mining, a duplicate channel sampling programme was undertaken to quantify sampling, preparation and analytical error (Table 22). Sixty saw-cut channel samples were collected within the target HGZ from the cross-cuts and along the stope sub-level. All samples were collected as 1 m lengths at 8 kg/m to mirror HQ drill core and processed via the geometallurgical drill core protocol (Figure 5).

Table 22. Analysis of duplicate sampling data.

Type	Description	Number	Error Type	Split FSE at ROM Grade	Component Relative Error
Field	Channel samples 2 × 8 kg collected next to each other	20	Sampling	-	±51%
Coarse	2 × 4 kg taken after crushing to 1.5 mm	20	Preparation	±22%	±18%
GRG concentrate (pulp)	2 × split of concentrate for 30 g fire assay	30	Analytical	<±5%	±4%
Tailings (pulp)	2 × 4 kg taken after pulverising to −100 µm for LeachWELL	30	Analytical	±1%	±7%
	Total				±55%

The highest error is the sampling component, which reflects both the in-situ nugget effect and physical sample collection (e.g., DE and EE). The channels were collected to minimise both DE and EE, thus the 51% error is dominated by the in-situ nugget effect. The preparation and analytical error components are low and within expectation for gold mineralisation [21]. The FSE for the coarse and pulp splits is low and reflected in the preparation and analytical error components. During sample collection and splitting, the DE, EE and PE were minimised. Given the use of a rotary splitter for all sub-sampling actions, any GSE was minor, although could relate to liberated gold segregating during crushing and grinding and biasing a number of the splits. This is ameliorated during normal processing, given that the entire sample is put through the geometallurgical drill core protocol (Figure 5). In this case, the preparation error is very close to zero, thus the total sampling error is likely to be around 52%.

10.5. Selective Mining Production Results

During the half-year period July to December 2017, stoping yielded 33,150 t at 23 g/t Au against an estimated grade of 23.7 g/t Au (Table 23). The estimate was dominated by 20 m by 20 m spaced drilling augmented with a small amount of 10 m by 10 m drilling and development sampling. The estimate overstated the mined grade by 3% over the period.

Table 23. Summary of half year production reconciliation of selective mining the HGZ.

Estimate/Actual Mined	Reserve Physicals				Monthly Variability Range Across Period		
	Tonnes (t)	Grade (g/t Au)	Contained Ounces (Au)	Recovered Ounces (Au)	Tonnes (t)	Grade (g/t Au)	Ounces (Au)
Estimate (plan)	32,500	23.7	24,770	16,595 (Rec: 67%)	-	-	-
Actual	33,150	23.0	24,520	18,145 (Rec: 74%)	±5%	−20% to +18%	−18% to +24%
Actual/estimate (%)	102%	97%	99%	109%	±5%	±20%	±25%

Like many operations displaying a relatively high-nugget effect, on a month-by-month basis, reconciliation is variable. In this case, the critical parameter is gold grade, which varied between −22% and +18% monthly (Table 23). This overall variability is within ±22%, which is reasonable for an Indicated resource category in high-nugget mineralisation [29]. The low tonnage variability reflects good ground conditions within the stopes and good mining practice by the crews. In-stope sampling allows the mining crews to focus each bench onto the highest grade areas within a minimum to maximum stope width of 1.5 m to 3.0 m. The plant recovered 74% (17 g/t Au) of the gold by gravity alone, against an estimated recovery of 63%.

11. Tactical Geometallurgy Application

11.1. Overview

With commencement of production, the tactical geometallurgical approach has been incorporated into the operation. This is primarily focused on defining feed variability for forecasting and blending purposes. Initial production is 145 t per day, although an increase to 290 t per day during Year 3 of operation is under consideration.

11.2. Ore Control

Ore control is a critical part of tactical geometallurgy, where the focus is on defining and controlling plant feed variability (e.g., grade, recovery, throughput, etc.) for forecasting and blending purposes.

It is effectively the traditional grade control approach, but extended to include parameters beyond just grade.

Development faces are saw-cut channel sampled at a length of 1 m ± 0.2 m to provide a target mass of 5 kg/m. Face samples will be used for development control, tracking grade distribution and contribution to a local ore-control model for planning. In-stope samples are hand chip-channelled from the backs to achieve lengths of 1 m ± 0.2 m and a target mass of 5 kg/m. Some 6–12 samples are collected per stope per day, depending on mineralisation width (e.g., 2–3 m) and bench length. Sample data are used for stope control and tracking grade distribution. Core sampling remains essentially the same, where 1 m ± 0.2 m samples are collected to yield approximately 8.5 kg/m (±20%) to resolve the narrow mineralisation target.

All core and ore control samples pass through the geometallurgical drilling protocol, including assay for sulphur, lead and zinc. The on-site laboratory runs two separate process circuits, one for ore control (Figure 9) and the other for the drill core (Figure 5). Samples are currently batched through a 2.5 kg capacity ring pulveriser. The application of a large capacity pulveriser (up to 8 kg) is being investigated to speed up the process. The laboratory currently has a small LeachWELL bottle roll facility capable of processing around ten 2 kg bottles per 6 h period. This is being expanded to around 30 bottles per 6 h period to allow for on-site processing of both ore control and plant tailings samples (Table 24). QAQC is applied to all sample processing and assaying (Table 3).

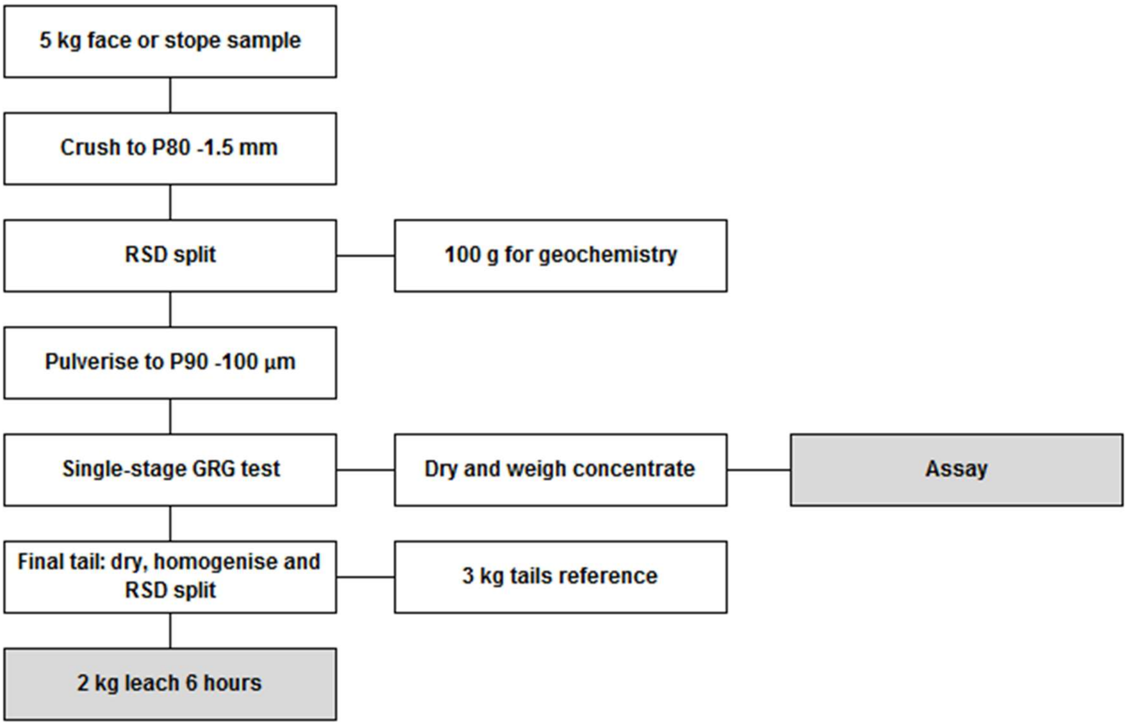

Figure 9. Flow sheet for ore control samples. The GRG test concentrate grade and tails leach grade (grey shaded boxes) are recombined to provide a head grade for the sample.

Table 24. Summary of sample processing locations, activity and turn-around times.

Sample Type	On-Site Activity	Off-Site Activity	Timing
Geomet core (Figure 5)	Dry, crush, pulverise, Knelson GRG, concentrate cleaning and fire assay	Comminution tests, all geochemistry and mineralogy, flotation testing and LeachWELL tails	Head grade within 5 days All results within 21 days
Ore control (Figure 9)	Dry, crush, pulverise, Knelson GRG, concentrate cleaning and fire assay, and LeachWELL tails	Sulphur, lead and zinc analysis	Preliminary grade from GRG concentrate weight within 2–3 h Second preliminary grade from GRG concentrate fire assay within 3–5 h Head grade within 9 h Sulphur and base metal assays within 3 days
Plant tailings	Hourly composites dry and pulverise	Sulphur, lead and zinc analysis LeachWELL assay	All assays within 24 h

A further study is underway to investigate the application of screen fire assay to the ore control sampling programme. If a 5 kg sample can be pulverised in one pass, screened and fire assayed, the on-site turnaround time can potentially be reduced. It is key however to retain the GRG information, therefore the screen fire assay–GRG correlation is being further scrutinised (Figure 4).

Critical to mine-to-mill reconciliation, the plant tailings sampler is set to cut six 0.5 kg samples per hour equating to around 3 kg for 12 t of ore feed (Table 24). The split is designed to achieve an FSE of less than $\pm3\%$ given the fine state of the tails and removal of coarse gold. Each 3 kg sample is pulverised and subjected to total sample LeachWELL. A sub-split of 50 g is taken for geochemical analysis.

11.3. Tactical Geometallurgical Modelling

The medium-term (>6 months) resource model will to be based on diamond core drilling on a 20 m by 20 m spacing to achieve an Indicated Mineral Resource. The core samples have been reverted to 1 m to better resolve the narrow HGZ. The 1 m whole core samples will be processed via the geometallurgical drilling protocol to produce grade, GRG, sulphur and base metal data. Core logging continues to record mineralogical, textural, structural, geotechnical and hardness data.

The short-term ore control model (<6 months) is based on core drilling, augmented by development samples as they become available. Key outcomes across both the short- and medium-term models are: (1) grade; (2) GRG recovery; (3) sulphide content; (4) ore density; and (5) ore hardness models (Table 25).

Table 25. Geometallurgical programme block model inputs and outputs.

Data Input (Units)	Primary Model	Secondary Model
Grade (g/t Au)	Grade	GRG recovery
GRG recovery (%)	GRG recovered grade	
Sulphur (%)	Sulphur	Sulphide Density
Iron (%)	Iron	
Lead (%)	Lead	
Zinc (%)	Zinc	
Density (t/m^3)	Density	-
EQUOtip	Hardness	Throughput
Rock quality designation	Rock quality designation	

GRG recovery cannot be modelled directly, so estimation applies the head grade and GRG recovery data to calculate a GRG recovered grade (e.g., GRG grade = head grade $\times$ GRG recovery fraction). Each block then has an estimated GRG recovered grade, which is reverted to GRG recovery by the application of the block grade. Flotation recovery is estimated from the pyrite content, which is correlated to gold grade.

Ore zone hardness is modelled through the EQUOtip. Overall hardness variability is generally low, except in close proximity to low-displacement cross-cutting faults where kaolinisation can be strong and result in weaker ground conditions and relatively clay-rich ore. Sulphur, lead and zinc assays are used for direct estimation and then are recombined to produce a sulphide distribution model (e.g., pyrite, galena and sphalerite) and density model. Throughout the HGZ, density is controlled by sulphide content and may vary from 2.7 t/m^3 to 3.4 t/m^3.

11.4. Short-Term Stope Panel Modelling

The initial stope panel size is approximately 60 m by 20 m, including lower development to yield 8500 t assuming a stope width of 2.5 m and making no allowance for mining recovery. Estimation is based on 10 m (strike) by 10 m (dip) by 1 m (width) blocks, with sub-cells of 2.5 m by 2.5 m by 0.25 m. Generally, 24–36 estimation blocks represent a stope panel. Estimation is undertaken using ordinary kriging, though additional studies using simulation are in progress. For each stope panel, the block model is compiled to yield key variables (Table 25).

In the stope panel example presented in Table 26, the reconciled head grade and tonnage are 108% and 104%, respectively, of the prediction. The GRG predicted recovery was 70% versus 76% actual recovery.

Table 26. Example of stope panel data at the ore control model stage and post-mining reconciliation.

Parameter	Panel Estimate	Range	Reconciled Actual
Tonnage	7360 t	-	7655 t
Width	2.25 m	2.00–2.45 g/t Au	2.30 m
Grade	22.7 g/t Au	19.9–26.3 g/t Au	24.6 g/t Au
GRG recovery	70%	95–75%	76%
Sulphide	4%	2–6%	5%
Density	2.97 t/m^3	2.85–3.20 t/m^3	3.10 t/m^3
Hardness	Hard	Hard-very hard	Hard
RQD	Excellent	Good-excellent	Good-excellent

The block model is used to support mine design, planning and ore blending. Each estimation block informing the panel is reviewed and applied to final stope design (e.g., smaller stopes will be designed as required and/or low grade areas planned as pillars where possible). The stope panel grade, mineralogy, recovery, hardness and RQD are used to design a final ore control reserve model incorporating pillars and dilution. The final outcome is a dollar cost and value for each stope.

12. Discussion

12.1. Overview

This study aimed to link ore characteristics with key operating parameters, principally grade and GRG recovery, but also including geochemistry/mineralogy, rock mass properties and plant throughput. The outcome was the production of 3D block models to describe variability (Table 25). San Antonio is a relatively small project, but shows that the geometallurgical approach is as valid as in large deposits. The PFS provided two mining scenarios for bulk and selective narrow-vein stoping. Both scenarios related to mineable ore zones with different characteristics (Table 27). Strategic considerations included selective campaign mining and contribution to a cluster of small operations feeding a central plant. The PFS led to a mining scenario to support an early start-up low-CAPEX selective narrow-vein operation. The key driver for the narrow vein option was that high-grades in the HGZ displayed greater than expected continuity.

Table 27. Comparison between bulk-mine ore zone and selective narrow-vein ore zone.

Mining Scenario/Ore Type	Minable Width (m)	Grade (g/t Au)	Core RSV (Nugget Effect)	% >100 μm	d_{95Au} (μm)	K (g/cm$^{1.5}$)	Ms Opt. (kg)	Ms Opt. Per m (kg/m)	GRG (%)	Gold Deportment
Bulk stoping FW, CZ and HW	10–15	6–10	120% (53%)	30	650	5300	70	5–7	30–40	Free gold, finer gold within pyrite Coarse gold, with more fine gold
Narrow vein stoping HGZ	2–3	20–24	138% (66%)	55	1700	6600	200	67–100	60–75	Free gold Coarse gold, with less fine gold

Ms opt.; optimum sample mass.

12.2. Sample Mass Requirements

The HGZ requires a larger optimum sample mass of 200 kg compared to the broader SVZ, which relates to higher d_{95Au} values (Table 27). Over the expected 2–3 m target mineralisation, this equates to 67 kg/m to 100 kg/m of sample required. The total mass across the mineralisation based on whole HQ core is between 26 kg and 36 kg dependent upon intersection angle. No change in sampling strategy is anticipated, given that a stope panel will be intersected by a minimum of three core holes. This equates to between 78 kg and 108 kg as a minimum total core sample mass. In addition, approximately 65 development faces will also be sampled around the stope panel, giving between 130 and 195 individual samples yielding between 650 kg and 975 kg of total sample. Across the drill core and development samples, a minimum total mass between 0.7 t to 1.1 t informs a given stope panel. If the total sample mass is 0.7 t, then the theoretical precision is ±10% based on ROM HGZ ore (Table 27). The total number of samples in each case relates to HGZ width. Stope mapping has indicated that gold particle clustering may be important within the HGZ, although the overall large composite mass ameliorates this.

12.3. Sampling, Testwork and Assaying

The study reiterates the link between sampling errors and magnitude of the nugget effect. In this case, a rigorously executed whole-sample assaying programme led to a reduction in nugget effect. The nugget effect measured from the geometallurgical programme is dominated by the in-situ nugget effect, given that many sampling errors were effectively zero. The duplicate sampling programme based on channel samples (of a similar support to the HQ drill core) confirmed dominance of the sampling error over preparation and analytical errors (Table 22).

The original drilling programme applied the traditional coarse-gold sampling paradigm of half core crushed, pulverised and a 30 g fire assay is flawed [13,23,24,27]. The approach is flawed and prone to high FSE, together with high GSE, DE and EE particularly when the assay charge is scooped from the pulp. The change from half-core fire-assay samples to the geometallurgical programme whole-core full-assay protocol showed a 20% increase in global grade in the SVZ Indicated resource zone. The project will continue to define HGZ Indicated resources using 20 m by 20 m drilling, although 10 m by 10 m will be applied as required.

12.4. Orebody Variability and Prediction

Gold grade is notably variable within the SVZ, reflected by a 53% nugget effect based on the geometallurgical drilling programme samples. Reconciliation has been possible across bulk sampling, trial mining and production. At the bulk sampling stage, grade reconciled to within ±11% and ±25% for the 10 m and 20 m drill grids respectively (Table 15). At the trial stope and production stages, mining had re-focused to the HGZ. The trial stoping grade reconciled to within ±10% and ±15% for the 10 m and 20 m drill grids respectively (Table 20). The six-month production period reconciled to within ±3% for the period, but to within ±22% on a month-by-month basis (Table 23). The nugget effect for 1 m whole-core and development saw-cut channel samples within the HGZ is 66%. Overall, the grade reconciliations are acceptable given the nugget nature of the mineralisation and Indicated resource classification, validating the sampling protocols applied.

Prediction of gold recovery was reasonably accurate, though has become less critical within the current HGZ mining area. Review of historical records indicates that, even within the HGZ, local zoning may occur, with some segments of the mineralisation containing more fine gold and sulphides (locally up to 30%), and thus a reduction in GRG. Within the HGZ, more sulphides equate to denser ore, less GRG recovery and more leach or flotation recovery. Further work is being undertaken to refine recovery prediction.

Variability in comminution ranges from very hard in quartz-dominated silicified zones to soft in kaolinised/sheared fault zones. The majority (95%) of the mineralisation relates to the hard-very hard classification, including the HGZ. Plant throughout is currently optimised to 50,000 t per annum, at around 6 t per hour based on hard/very hard feed. The small amount of soft material (5%) will be blended with hard material.

12.5. Vein Paragenesis and Links to Metallurgical Recovery

Geological mapping, mineralogy and metallurgical testwork has permitted resolution of vein paragenesis and associated metallurgy (Table 28) [30]. The altered wall rocks hosting the veins bears fine-gold hosted in pyrite, which shows grades to 5 g/t Au in highly silicified zones within the HGZ. Elsewhere in the SVZ, wallrock grades range from <0.5 g/t Au 2 g/t Au. The wider composite veins of the HGZ represent a later veining stage that cuts the two sheeted-vein phases (SV-I and -II: Table 28). The HGZ composite veins can locally attain grades of up to 100 g/t Au. The LV are not observed in the initial mining area, but become more significant in Year 3 onwards.

Table 28. Vein paragenetic sequence within the SVZ and mineralogical and metallurgical characteristics.

Type/Paragenetic Sequence	Occurrence in SVZ Domain	Structure Type	Width	Mineralogy	Gold Type/Host	Gold Particle Size	Metallurgy
Wall rocks	FW, CZ and HW	Wallrock alteration within SVZ	8–11 m (excluding veins)	Disseminated pyrite (0–5% py)	Free gold Pyrite	Fine gold, <150 μm	Leach
SV-I (early)	FW, CZ and HW	Steep-dipping sheeted veins	<1–5 cm	Pyrite, trace galena, sphalerite and chalcopyrite (0–5% py)	Free gold Pyrite	Some coarse, up to 0.25 mm	GRG (up to 30%) +leach
SV-II	Principally CZ	Steep-dipping sheeted veins. Different strike (±5–10°) to SV-I; crosscut SV-I	<1–10 cm	Pyrite, trace galena, sphalerite and chalcopyrite (0–5% py)	Free gold Pyrite	Some coarse, up to 0.5 mm	GRG (up to 30%) + leach
CV	HGZ	Steep-dipping composite veins, crosscut SV-I and -II	10–35 cm	Pyrite, trace galena and sphalerite (0–10% py)	Free gold Quartz Pyrite	Coarse up to 2.5 mm	GRG (up to 80%)
LV (late)	Principally cuts CZ	Flat-dipping veins. Cut all SV	0.25–1 m	Sulphide-rich; pyrite (25–100% py)	Free gold Pyrite	Minor coarse <0.2 mm	Leach

SV, sheeted veins; CV, composite veins; LV, late veins; py, pyrite.

13. Conclusions

(1) San Antonio represents a moderate-grade bulk-mine sheeted-vein (e.g., HW, CZ and FW) gold deposit, with a high-grade selective-mine zone (e.g., HGZ). Both deposit types pose challenges for grade and metallurgical sampling by virtue of their heterogeneous gold particle and grade frequency distributions. The project is relatively small, but shows that the geometallurgical approach is appropriate, where a programme must be tailored to the deposit in question.

(2) The PFS concluded that the selective operation was a better option allowing for a smaller footprint, fast production ramp-up, less capital expenditure and aligned better with the expectations of stakeholders. In addition, the mine site and infrastructure could be used as a base for regional exploration and for the evaluation of several other historic mines within 2 km of San Antonio. Overall, it is a more sustainable option.

(3) Metallurgical (composite and variability) samples should be undertaken early in the mine value chain to assess ore characteristics. Gold characterisation is required to optimise subsequent resource grade and metallurgical sampling protocols. If characterisation concludes that large samples are required, then strategies to deal with this can be put in place using multiple smaller samples followed by bulk sampling.

(4) The protocol presented was designed to respond to the coarse-gold nature of the mineralisation and to provide maximum data from drill core to support a resource estimate and PFS. The change from 1 m-half core samples to 2 m whole-core whole-sample assay in the Indicated resource area showed a 20% increase in grade. In addition, the protocol change resulted in a reduction of the nugget effect and RSV. The core sampling strategy reverted to 1 m whole-core samples in response to the change to selective narrow-vein mining. The geometallurgical core protocol continues to be applied.

(5) As whole core sampling is undertaken, no mineralised reference core is retained. Whilst contentious, this is done to ameliorate the coarse-gold nature of the mineralisation and to avoid core cutting losses. In addition, it permits maximum information to be gained from the core. An independent third party reviews and verifies the drilling, logging, sampling and sample dispatch process.

(6) A fit-for-purpose sampling, testwork and assaying programme was designed to support a Mineral Resource and Ore Reserve reported in accordance with the 2012 JORC Code. To ensure fit-for-purpose data, a rigorous QAQC programme was developed to support sampling and testwork. The data quality and drill spacing achieved an Indicated Mineral Resource across the first 2–3 years of production.

(7) Within the broader SVZ, metallurgical results indicate two grade recovery domains and three comminution domains. Within the selective-mine HGZ, gravity-only recovery is achieving 70–75% GRG at grades >12 g/t Au. The FW and HW zones require either leaching or flotation to provide recoveries above 30%.

(8) A bulk sample programme was undertaken across the SVZ, which was processed through a pilot plant as part of the PFS. Application of pilot plant testwork is a critical stage in validating previous metallurgical testwork. The results were also used to validate the resource model.

(9) A trial mining programme to test the selective HGZ scenario was successful in proving the mining method, validation of estimated grade and process flow-sheet design. Based on this success, the project passed from trial mining seamlessly into production within a month.

(10) There is a greater need towards the quantification of sampling and analytical errors to better communicate uncertainty and risk [8,31]. A first step is the application of the protocols and RSV metric presented in DS3077 [22]. Resolution of component relative errors across sampling, preparation and analysis can be gained from duplicate sample pairs [21].

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