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Integrating the Theory of Sampling into Underground Mine Grade Control Strategies: Case Studies from Gold Operations

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Abstract: Grade control aims to deliver adequately defined tonnes of ore to the process plant. The foundation of any grade control programme is collecting high-quality samples within a geological context. The requirement for quality samples has long been recognised, in that these should be representative and fit-for-purpose. Correct application of the Theory of Sampling reduces sampling errors across the grade control process, in which errors can propagate from sample collection through sample preparation to assay results. This contribution presents three case studies which are based on coarse gold-dominated orebodies. These illustrate the challenges and potential solutions to achieve representative sampling and build on the content of a previous publication. Solutions ranging from bulk samples processed through a plant to whole-core sampling and assaying using bulk leaching, are discussed. These approaches account for the nature of the mineralisation, where extreme gold particle-clustering effects render the analysis of small-scale samples highly unrepresentative. Furthermore, the analysis of chip samples, which generally yield a positive bias due to over-sampling of quartz vein material, is discussed.

Keywords: gold mineralisation; grade control; Theory of Sampling; sampling errors; representative sampling; fit-for-purpose sampling

1. Introduction

1.1. Rationale for this Contribution

Underground mine grade control aims to deliver quality tonnes to the process plant via the definition of ore and waste. The traditional role of grade control samples relates to the following: mine development ore/waste decisions; investigation of ore limits; identification of grade trends and/or continuity along development or stopes; and local "grade control" estimation. Where samples feed into ore/waste decisions, the risks of potential misclassification and its economic impacts must be considered [1–3].

Many small- to medium-sized (production <250,000 t per annum) and some large mine operations rely on face samples for grade control. These may be used to update a resource/reserve model that is publicly reported. In some larger operations, the need for face sampling is removed by the application of grade control/stope definition drilling.

This contribution places the review of Dominy et al. into a practical framework [2]. Three case studies are evaluated in terms of ore characterisation, Theory of Sampling and component error analysis. The objective is to assess sampling risk and the fit-for-purpose nature of each sampling programme. Where appropriate, the effects of poor practice are discussed and solutions provided. Additionally, this and the previous paper [2], provide a benchmark to guide the design of grade control sampling protocols for new or developing underground operations.

1.2. Importance of Sampling

Sampling remains a critical component throughout the mine value chain. Without being able to analyse all material in advance, sampling of both in-situ and broken material serves to inform geological (resource and grade control), geoenvironmental and geometallurgical based mine planning and decision-making [1,3–5]. Sampling errors can generate both monetary and intangible losses [3,5–10].

Representative samples are required to effectively evaluate the style of mineralisation in question [2,3,5]. This can be particularly challenging in deposits with coarse gold (>100 µm particles dominate) [11,12], where large field samples and special preparation-assay protocols may be required [3,9,13,14]. Unrepresentative samples will not describe the true in-situ gold grade distribution and the overall result generally leads to a lower (undervalued) deposit mean grade. This is attributed to small samples having a high probability of missing influential coarse gold particles and reporting at the lower end of the grade distribution. As a result, there will be overestimation of block grades below the economic cut-off value, that is, blocks which report as waste. Any fine-gold background population is likely to be represented relatively well by small samples [11]. At the other extreme, samples may report as "false" high grades when they occasionally contain coarse gold particles. For example, 1 m of half HQ core (c. 4 kg based on 63.5 mm diameter core) will yield a grade of 40 g/t Au and 295 g/t Au if it contains a single 2.5 mm or 5 mm gold particle respectively. Note that the presence of rare coarse gold particles in small samples may positively bias the deposit mean grade.

Quality assurance/quality control (QAQC) is critical to maintain data integrity through documented procedures, sample security and monitoring of precision, accuracy and contamination [2,15–17]. The ultimate test of any grade control programme comes through reconciliation of actual mine performance versus that predicted by grade control samples [18].

1.3. Theory of Sampling

1.3.1. Overview of Theory of Sampling

The Theory of Sampling (TOS) aims to provide answers to two questions: *how should a sample be selected* and *how much material should be taken*? It defines a series of sampling errors which, if not minimised, lead to error and uncertainty in the final assay value [3,7]. TOS attempts to break down this error into a series of contributions from sample collection through to assaying (e.g., the sampling value chain; Table 1).

The heterogeneity of a given variable (e.g., grade) can be quantified through the nugget effect and has a direct link to TOS [3,9,12,19]. The nugget effect is a quantitative geostatistical term describing the inherent variability between samples at very small separation distances. In reality, the nugget effect has a wider remit than just differences between contiguous samples and its magnitude relates to the small-scale geological variation and sample measurement error [3,9,12,19].

Table 1. Sampling value chain from programme planning to assaying.

Stage	Planning	Collection	Transport	Laboratory Preparation	Assaying
	1	**2**	**3**	**4**	**5**
Activity	Scope Develop Execute	Observe Collect Bag and tag QAQC Integrity/security Chain of custody	Integrity/security Chain of custody	Equipment operation Equipment clean QAQC Integrity/security	Equipment operation Equipment clean QAQC Integrity/security
Sampling errors		GNE, FSE, GSE DE, EE, WE	PE	FSE, GSE DE, EE, WE, PE	PE AE
Dominant effect on results		Precision Bias	Bias	Precision (if splitting) Bias	Bias

GNE: geological nugget effect; FSE: fundamental sampling error; GSE: grouping and segregation error; DE: delimitation error; EE: extraction error; PE: preparation error; WE: weighting error; AE: analytical error.

The geological component of the nugget effect expresses short-range data variability, which is particularly significant when samples are small and protocols not optimised. The sampling component of the nugget effect expresses errors induced by inadequate sample mass, poor sample collection and preparation methods and poor analytical procedures. Throughout the mine value chain, sampling protocols should be optimised to reduce the sampling nugget effect which, in turn, reduces the total nugget effect, data skewness and the number of extreme data values [3,9,12,13].

1.3.2. FSE Equation and its Application

An important TOS error is the Fundamental Sampling Error (FSE), which characterises the compositional (e.g., grade) heterogeneity of the material in question [3,7]. The FSE is controlled via optimisation of the sample mass and size reduction process. For any process where the FSE is large, there is an associated loss due to uncertainty which, in terms of grade control, relates to ore/waste misclassification [3,8]. An enhanced FSE increases the total nugget effect [3,9,19].

FSE can be modelled using the so-called "FSE equation" provided that certain characteristics are known or reasonably inferred [3,7,20]. The equation is applicable to grade control samples once they are collected (e.g., a face sample is broken rock in a sample bag or core have been crushed). The FSE equation ("François-Bongarçon modified" version) is given as [20]:

$$\mathrm{FSE}_{(\text{relative variance})} = f\, g\, c\, (d_{95\mathrm{Au}})^{b}\, d_{\mathrm{n}}^{\alpha}\, (1/M_{\mathrm{S}} - 1/M_{\mathrm{L}}) \tag{1}$$

where f = shape factor; g = granulometric factor; d_{n} = nominal material size; c = mineralogical factor; $d_{95\mathrm{Au}}$ = liberation diameter; $b = (3 - \alpha)$, where α is determined experimentally from duplicate series analysis tests or a default value of $\alpha = 1.5$ is applied [20,21]; M_{S} = sample mass (in grams); and M_{L} = lot mass (in grams). The sampling constant (K) is a product of: $f\, g\, c\, (d_{95\mathrm{Au}})^{b}$, where its value reflects the samplability of an ore type. A value >1000 g/cm$^{1.5}$ indicates a problematic ore type, that will require careful sampling programme design. Using this formula, it is possible to: (a) calculate the FSE for a given sample size split from the original or (b) calculate what sub-sample size should be used to obtain a specified FSE at a given reliability [3,7,22].

The key input that requires determination is the liberation diameter of the gold, which is a particle size parameter [3,20,22]. For gold mineralisation, it can be defined as the screen size that allows 95% of gold given a theoretical liberated lot to pass ($d_{95\mathrm{Au}}$) [20]. If gold particles are clustered, then $d_{95\mathrm{Au}}$ should be defined as the cluster diameter—d_{Auclus} [11]. Approaches to $d_{95\mathrm{Au}}$ determination range from field observation, optical and automated mineralogical studies, dedicated metallurgical test work and/or heterogeneity tests [3,14,21,23].

In this contribution, all FSE values are estimated using the François-Bongarçon modified equation (Equation (1)) and reported at the 68% reliability (e.g., one standard deviation). Calculations are made at the breakeven cut-off grade (BCOG), given that this is the critical economic ore/waste decision point. A total FSE value is given for each protocol, representing all comminution and subsampling steps. FSE levels are defined as: >±30% high; ±20–30% moderate; and <±20% low [3].

1.4. Determining Sampling Variability-Duplicate Pair Analysis

Errors representing the repeatability of assay results can be estimated by pairwise analysis of field, coarse and pulp duplicates [15–17]. Sampling protocols include several stages of comminution and subsampling, where duplicates can be taken at every stage to allow estimation of the total sampling precision error and the relative contributions at the different stages of the sampling protocol (e.g., sampling, preparation and analysis error; Table 2).

Table 2. Examples of duplicate pair analysis from contrasting styles of gold mineralisation.

Sample Type/Error (Preparation Route)	Sampling (%)	Preparation (%)	Analytical (%)	Total (%)
Fine-gold disseminated	Half core	Split at −2 mm	Split at −75 µm (FA30)	-
NQ core	17	11	5	21
Number of pairs	245	175	150	-
Coarse-gold vein	Half core	Split at −1.5 mm	Split at −75 µm (FA30)	-
NQ core	70	38	32	86
Number of pairs	125	125	155	-
Expected error range for gold mineralisation	20–90	5–40	1–25	20–100

Pairs cover range of mineralisation grades from low to high. NQ core (diameter 48 mm).

Stanley and Lawrie [15] have shown that the coefficient of variation (COV) estimated from paired data produces stable and robust estimates of sampling precision and is independent of statistical distributions of the studied variables and presence of outliers.

The Danish Horizontal Standard [24] uses the relative sampling variability (RSV) metric to measure sampling variability–effectively the COV. As the COV and RSV are the same, they can be calculated as a metric of a set of samples (e.g., channel samples) or for pairwise duplicates (e.g., duplicate channel samples). RSV measures the total empirical sampling variance influenced by the heterogeneity of the lot sampled under the current sampling procedure.

Component errors reflect the ore type, sample type and collection and proceeding preparation and analysis. Total sampling error (as COV) is likely to be in the range 25–100% for gold ores, with components of 20–90% (sampling), 5–40% (preparation) and 1–25% (analytical) respectively. Table 2 shows errors values for two ore types, where the disseminated mineralisation displays minimal error due to fine gold particles. The coarse-gold mineralisation shows large errors, particularly at the analytical stage where 30 g fire assays (FA) are drawn from the pulp. This indicates the presence of coarse gold particles in the pulp.

Throughout this contribution, sampling relative precision from duplicate pairs is calculated via the COV at the 68% reliability as described in Stanley and Lawrie [15]. The RSV metric is applied to total populations of samples.

1.5. Sampling Programme Risk Review

The public reporting of Mineral Resources based on grade control sampling, require the Competent Person(s) to provide commentary on the sampling and assaying process via the JORC Code Table 1 Section 1 (“Sampling Techniques and Data”: [25]).

The authors recommend creating a simple overview table, which summarises key aspects of the sampling programme (Table 3). Individual TOS errors cannot be quantitatively evaluated but their impact can be inferred from duplicate sample analysis and resolution of the sampling, preparation and analytical error components (Table 2). Individual TOS errors can be evaluated based on knowledge of ore characteristics and the protocol(s) (field and laboratory) applied. These components can be grouped to represent key parameters and applied as a guide to the fit for purpose nature of the overall programme (Table 3).

Table 3. Risk review for a low-coarse gold mineralisation grade control sampling programme used for resource definition.

<table>
<tr><th colspan="2">Key Parameter</th><th>Comment</th><th>[1] Component Error</th><th>TOS Error</th><th>[2] Error Rating</th></tr>
<tr><td>1</td><td>Spatial distribution and number of samples</td><td>Samples collected at approx. 1.8 m intervals along drives and raises
Each stope block informed by around 25–35 samples</td><td rowspan="4">32%</td><td rowspan="2">GNE</td><td>Low-mod.</td></tr>
<tr><td>2</td><td>Sample mass (representativity)</td><td>Each face composite sample approx. 2–4 kg; total sample mass collected approx. 75–100 kg; indicated optimum mass around 120 kg to achieve 90% ±20%</td><td>Low-mod.</td></tr>
<tr><td>3</td><td>Collection and handling</td><td>Samples collected by chip-channel
Reasonably consistent sample mass collected
All samples placed into calico bags and tied</td><td>EE</td><td>Low-mod.</td></tr>
<tr><td>4</td><td>Transport and security</td><td>Chain of custody recorded between mine and off-site laboratory</td><td>-</td><td>Low</td></tr>
<tr><td>5</td><td>Preparation</td><td>Entire sample lot crushed and pulverised and 1 kg riffle split for screen fire assay</td><td>12%</td><td>-</td><td>Low</td></tr>
<tr><td>6</td><td>Assay</td><td>Fire assay process undertaken correctly</td><td>11%</td><td>-</td><td>Low</td></tr>
<tr><td>7</td><td>QAQC</td><td>Four CRMs used from low to high grade
CRMs and blanks inserted at 1 to 25 rate; performance within expectation
Full written protocols for the sampling-assaying process</td><td>-</td><td>-</td><td>Low</td></tr>
<tr><td>8</td><td>Validation/variability indicators</td><td>Sample population RSV: 155%
Total nugget effect: 40%
Grade reconciles to ±15% quarterly based on Indicated Mineral Resources</td><td>Total 36%</td><td></td><td>Mod.</td></tr>
<tr><td colspan="6">Summary</td></tr>
<tr><td colspan="5">Representativity (1)–(3)</td><td>Low-mod.</td></tr>
<tr><td colspan="5">Preparation and assay (4)–(7)</td><td>Low</td></tr>
<tr><td colspan="5">Fit-for-purpose</td><td>Yes</td></tr>
</table>

[1] Component errors from duplicate pair analysis; [2] Indicative total error rating; red: high (>±50%); orange: moderate (±25–50%); low-moderate (±20–35%); green: low (<±25%).

An important outcome for any sampling programme is that it must produce data that is fit-for-purpose [2,4,10]. In this context, fit-for-purpose refers to grade data that is of an appropriate quality to contribute to a resource/reserve estimate that will be reported in accordance with the 2012 JORC Code. Development of sampling and assaying protocols in the context of TOS must be based on the specific mineralisation. If a batch of samples is deemed to be representative and associated assaying complies with QA documentation and QC metrics, then fit-for-purpose is indicated. In addition, the fit-for-purpose test must consider the spatial distribution and number of samples.

An error rating is provided for sample representativity and preparation and assaying based on the key parameters (Table 3; e.g., key parameters 1–7). The total error rating definitions are based on the recommendations of Pitard [3,26]. The fit-for-purpose rating is based on variability indicators (e.g., nugget effect and RSV), performance of QC and reconciliation over a given period within the context of resource classification [27]. The ratings currently are guided by quantitative assessment (e.g., duplicate pair analysis) and Competent Person judgement. Work is in progress to define a spatially quantitative sampling error assessment.

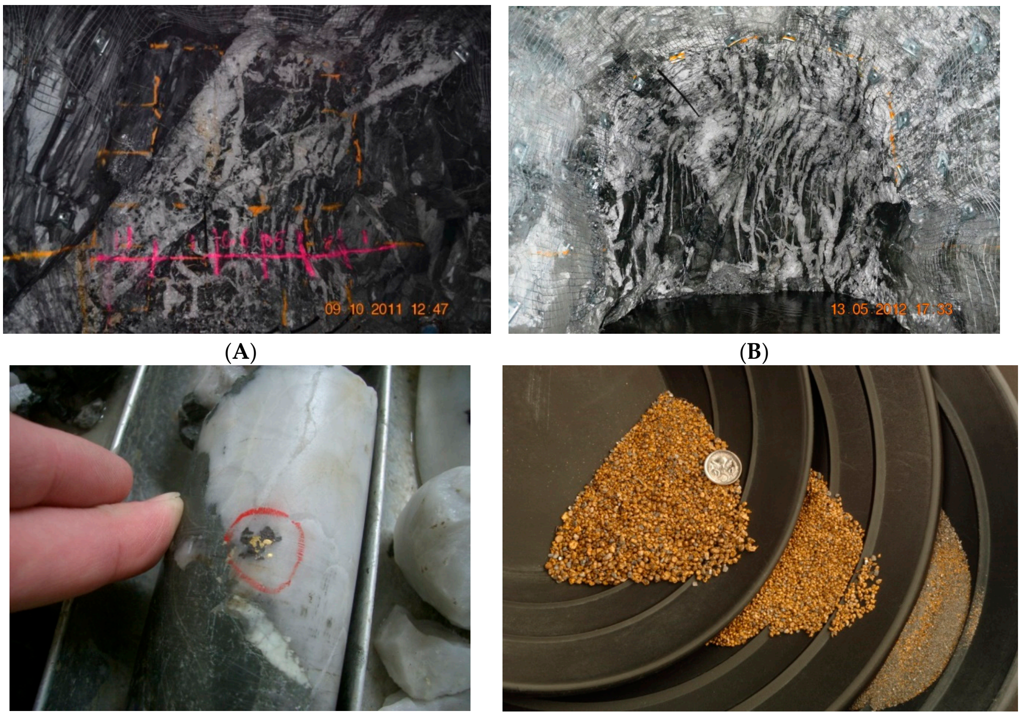
2. Case Study 1: Ballarat Mine, Australia

2.1. Introduction

The underground Ballarat mine is operated by Castlemaine Goldfields Pty Ltd. (CGT), a subsidiary of LionGold Corporation Ltd. The mine is located beneath the city of Ballarat, in the State of Victoria, Australia. The goldfield extends over a strike length of 3 km and has produced over 1.2 Moz Au from underground sources since 1858. The current mine plan is based on a combination of ore generated from lode development and longhole bench stoping. CGT recommenced production at Ballarat in 2011, since when they have mined 1.4 Mt at 5.3 g/t Au and recovered 242,100 oz Au [28]. The previous operator extracted 350,000 t at 3.1 g/t Au for 25,400 oz Au (recovered) between 2006 and 2010. The 2018–2019 CGT budget aims to extract 270,000 t at 5.9 g/t Au from an Inferred Mineral Resource of 414,000 t at 10.2 g/t Au [28].

2.2. Geology and Mineralisation

Gold mineralisation at Ballarat is hosted in lode structures that comprise discrete quartz veins or stockworks (Figure 1) [29]. All vein sets are quartz-dominated and related to low-displacement W-dipping faults (≤45°) that transect the core and/or eastern limb of tight, asymmetric N–S-trending anticlines. The W- and E-dipping veins range from 1 m to 5 m in width. The stockwork zones comprise a number of narrow <5 cm thick veins spaced between 5 cm and 20 cm apart. The lodes typically have dip extents from 5 m to 65 m, widths of ≤20 m and strike lengths of a few hundred metres. Their strike continuity is disrupted by oblique, low-displacement cross-course faults.



(**A**) (**B**) (**C**) (**D**)

Figure 1. (**A**) West-dipping Sovereign Mako lode (width of face approx. 5 m); (**B**) Llanberris Mako Tiger lode (width of face approx. 5 m); (**C**) Gold particles in core; and (**D**) liberated gold from processing.

The lodes are characterised by distinct phases of sulphide paragenesis with minor gold-arsenopyrite-pyrite defining the early sulphide stage. Late-stage coarse-gold was precipitated with galena-sphalerite ± pyrrhotite ± chalcopyrite. The estimated percentage of sulphide minerals in the veins is 2–5%. Sulphides occur in a broad spatial relationship with gold. Mineralisation is characterised by notable quantities of coarse gold hosted in quartz veins (Figure 1).

2.3. Gold Particle Size and Liberation Diameter

2.3.1. Process Plant Batch Analysis

Gold particle size distribution was determined from batch processing through the plant, where products were sized prior to smelting (Table 4 and Figures 1D and 2). The data shows a strong coarse gold component across low, BCOG, run of mine (ROM) and high grades.

Table 4. Summary of gold particle size distribution by mass across different grades based on 2000–5000 t trial process plant lots.

Grade (g/t Au)	<100 µm	100–2000 µm	+2000 µm	>100 µm	d_{95Au} (µm)
2	64%	28%	8%	36%	1800
4 (BCOG)	44%	36%	20%	56%	2500
6 (ROM)	40%	34%	26%	60%	3500
10	37%	36%	27%	63%	4000

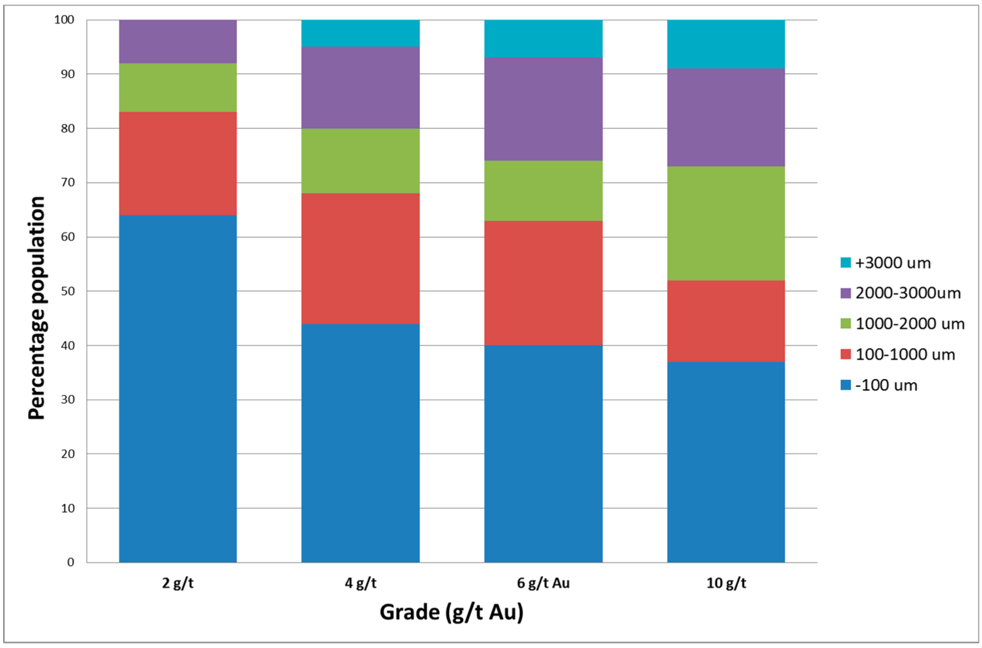


Figure 2. Gold particle size distribution by mass across different grades (after Table 4).

A number of different d_{95Au} values are indicated depending upon the determination method used (Tables 5 and 6). The laboratory-based and heterogeneity test methods generally understate the expected d_{95Au}, a common challenge with the application of such methods [23]. The BCOG (4 g/t Au) d_{95Au} determined from batch processing was around 2500 µm (Table 4); however this value could change so a range of values were applied to the FSE calculations (Table 6). Clustering is observed in drill core and mine faces, though appears to be local and is not considered to be material. The higher d_{95Au} values in Table 6 account for some clustering.

Table 5. Summary of d_{95Au} values determined from various methods.

Test Type	Mass of Lot	Grade (g/t Au)	d_{95Au} (µm)	K	Comment
Laboratory	204 kg	6.3	700	5700	Face sample composite put through a gravity process
Heterogeneity test	500 kg	9.9	900	5700	330 × 1.5 kg rock fragments from mill feed belt [23]
Heterogeneity test	1 t	13.0	150	355	500 × 2 kg drill core pieces [23]

Table 6. Range of potential d_{95Au} values for BCOG.

d_{95Au} (µm)	Comment
500	Individual particles
1000	Individual particles
2500	Individual or clustered particles
3000	Individual or clustered particles

2.3.2. Pulp Heterogeneity Testwork

The fire assaying of a sample to extinction is a useful method to investigate pulp heterogeneity. A test was undertaken on 1 m of whole NQ2 core (50.6 mm diameter) containing six visible gold particles ranging from 250 µm to 850 µm in size. The core weighing 4.88 kg was crushed and pulverised and 163 replicate FA30 (e.g., 30 g fire assay) were performed until total sample extinction (Table 7).

The spread of FA30 assay values is marked, ranging from 0.01 g/t Au to 364 g/t Au, indicating the large number of potential outcomes. Of note, is that 151 out of 163 assays (93%) are below the BCOG. Sixteen lots of randomly chosen groups of 10 assays were compiled, where each group was averaged (Table 7). An average coarse gold particle size of 400 µm was calculated via the method proposed in Pitard and Lyman [30], which accords with core observations. This single test indicates the potentially high pulp heterogeneity at Ballarat.

Table 7. Fire assay extinction test results for NQ2 core pulp.

Sample	Number of Assays	Range (g/t Au)	Mean (g/t Au)	RSV
FA30	163	0.01–364	5.13	602%
Composites	16	0.76-38.1	5.13	169%

2.4. Theoretical Sample Mass

The theoretical required field sample mass can be estimated using Poisson statistics to achieve a ±15 per cent precision at 90% reliability [31]. The range of mass values for BCOG mineralisation is given in Table 8.

Table 8. Range of theoretical sample mass values for BCOG mineralisation.

d_{95Au} (µm)	Sampling Constant (K)	Optimum Mass
500	5400	50 kg
1000	15,200	350 kg
2500	60,300	5 t
3000	79,300	9 t

An individual sample mass of 50 kg to 9 t is not practical to collect as single samples but provides an indication of the nature of the mineralisation. The conclusion is that a given ore zone probably requires between 2000 and 4000 spatially distributed NQ (47.6 mm diameter) 0.3–0.7 m (nom. mean 2.2 kg) core samples. These numbers are realistic, for example the Normanby Mako lode (resource of 100,000 t) contains around 3000 samples giving >6 t of NQ core [28].

2.5. Sampling Protocol Development

2.5.1. Project History

The Ballarat project has been operated by a number of companies since 1983. For the purpose of this discussion it will be split into the periods 1983–2006 (Ballarat Goldfields NL and Ballarat Goldfields Ltd.: BGF), 2007–2010 (Lihir Gold Ltd.: LGL) and 2011 to present (Castlemaine Goldfields Ltd.: CGT).

2.5.2. BGF (1983–2006)

During 1985 to 1988, half-core 1 m composites were used. Intervals with no visible gold, where pulverised and split into a 50 g FA charge. If visible gold was observed, then screen fire assay (SFA) was applied to the entire pulp.

Between 2003 and 2006, three methods were applied: FA, SFA and cyanide leaching (e.g., LeachWELL 2 kg charges: LW2000). The protocols are summarised in Table 9, together with an analysis of the FSE for each. The FA route reports high FSE values, recording that a 25–50 g sub-sample is extracted from the original 2.5 kg lot. Given that the entire sample is pulverised and gold will be liberated, the GSE potentially becomes material during pulp splitting (e.g., taking a 30 g FA charge from 2.5 kg of pulp).

Table 9. FSE for NQ/NQ2 core protocols at the breakeven cut-off grade (BCOG) (4 g/t Au) applying the $d_{95\text{Au}}$ scenarios defined in Table 6. Fundamental sampling error (FSE) assumes the presence of coarse gold in the pulp.

Stage	Type	Protocol	[1] FSE	Comment
Exploration/resource development (1983–2006)	Half core (2.5 kg)	Jaw crush all to P_{90} −6 mm Pulverise in LM5 to P_{90} −75 Riffle split FA50	±30% ±50% ±100% ±115%	FA for non-visible gold bearing core
Exploration/resource development (2003–2006)	Half core (2.5 kg)	Jaw crush all to P_{90} −6 mm Pulverise in LM5 to P_{90} −75 SFA2500 or riffle split for LW2000	±5% ±5% ±5% ±5%	SFA or LW for visible gold bearing core

[1] FSE error definition: red: high (>±30%); orange: moderate (±20–30%); green: low (<±20%).

When the FA50 protocol was used, if the resulting grade was >0.5 g/t Au, then the pulp was re-assayed using either SFA or LW. Given the high FSE values, it is possible that a FA50 sub-sample will contain no coarse gold and hence understate the true sample grade. The probability of not drawing a 500 μm coarse gold particle in a 50 g sub-sample is around 80% assuming a 4 g/t Au lot. The BGF period typifies many gold projects where different sample supports (e.g., half versus whole core), assay methods (e.g., aqua regia, FA and SFA) and triggers for different methods (e.g., visible gold and/or grade >0.5 g/t Au) are used. These introduce bias into the assay data, which increases the nugget effect [9,30,31].

2.5.3. LGL (2007–2010)

The evaluation approach used by Lihir ranged from wide-spaced exploration drilling [L1] to close-spaced stope design drilling [L4] and production sampling [L5] (Table 10).

Table 10. Lihir gold limited (LGL) evaluation stages.

Stage	Type	Aim	Spacing
L1: Exploration	Diamond core (NQ or HQ)	Exploration targeting	500 m by 100 m
L2: Exploration	Diamond core (NQ or HQ)	Resource growth	100 m by 20 m 50 m by 15 m
L3: Resource	Diamond core (NQ, NQ2 or HQ)	Resource shape definition	33 m by 10 m
L4: Resource	Diamond core (NQ, NQ2 or HQ) RC and sludge	Stope design	20 m by 5 m
L5: Production	Face and grab samples	Development control	Development face or blast lot

Between September 2007 and March 2008, samples were crushed on site and sent to an external laboratory for assay. From April 2008 onward, samples were prepared and assayed at the on-site laboratory. Table 11 documents the protocols applied and an analysis of FSE for each.

Table 11. Protocols and FSE for L1-L5 (Table 10) at BCOG across d_{95Au} scenarios (Table 6) for the LGL operating period. FSE calculations assume coarse gold in the pulp.

Stage	Type	[2] Protocol	[1] FSE	Comment
L1/L2	Half NQ/NQ2 core (2.5 kg)	Jaw crush all (P_{80} −10 mm) Pulverise all in LM5 to P_{90} −75 µm Scoop 2 kg for LW2000	±2% ±3% ±6% ±7%	Noted that assay pulp (2 kg from 2.5 kg) scooped from LM5; problematic from a GSE, DE and EE perspective
L1/L2	Half HQ core (4 kg)	Jaw crush all (P_{80} −1 mm) Riffle split 2 × 2 kg, retain one Pulverise all in LM5 to P_{90} −75 µm Scoop 2 kg for LW2000	±20% ±35% ±70% ±80%	Substantial FSE at post-crusher split. Noted that assay pulp scooped from LM5; problematic from a GSE, DE and EE perspective
L3/L4	Whole HQ core (4 kg)	Jaw crush all (P_{80} −10 mm) Crush all to P_{90} −3 mm RSD split 2 kg Pulverise 1 × 2 kg sub-sample in LM5 to P_{90} −75 µm Scoop 2 kg for LW2000	±40% ±70% ±140% ±160%	Substantial FSE at Boyd crusher split. Noted that assay pulp scooped from LM5; problematic from a GSE, DE and EE perspective
L3/L4	Sludge (6 kg)	Sludge chippings (3–6 kg) Crush to P_{90} −3 mm RSD split 2 kg Pulverise in LM5 to P_{90} −75 µm LW2000	±55% ±90% ±180% ±210%	Substantial FSE at Boyd crusher split
L5	Face (2–4 kg)	Face chippings (2–4 kg) Crush to P_{90} −3 mm RSD split 2 kg Pulverise in LM5 to P_{90} −75 µm LW2000	±38% ±65% ±128% ±147%	Substantial FSE at Boyd crusher split
L5	Grab (10 kg)	Grab 2 × 10 kg at sub-10 cm size Crush all in LM5 to P_{90} −3 mm RSD split into 2 kg lots Pulverise to P_{90} −75 µm LW2000	±170% ±290% ±580% ±665%	Sub-10 cm material collected from 120 to 150 t rock pile. Substantial FSE at Boyd crusher split
L5	HQ core, sludge or grab (5–10 kg)	Crush all to P_{80} −1 mm Pass through gravity concentrator Gravity concentrate assay by 2–3 × PAL500 RSD split 1.5 kg tails for 3 × PAL500	±20–40% ±20–40% ±20–40% ±20–40%	FSE calculations assume most coarse gold above 500 µm is recovered but some 200–500 µm remains

[1] FSE error definition: red: high (>±30%); orange: moderate (±20–30%); green: low (<±20%). 2 Boyd crusher and LM5 pulveriser used; All LW2000 24 h leach time; Analysis by AAS; FA50 of tails at rate of 1 in 20; RSD: rotary sample divider; PAL: pulverise and leach.

The L1/L2 exploration protocols show a low FSE, given that most of the sample was assayed. However, the assay pulp was scooped from the pulveriser bowl, which gives potentially high GSE, DE and EE values [3,9,32]. The pulp contains liberated gold which is likely to have settled to the bottom of the bowl, thus any sample scooped from the top will not delineate or extract a sub-sample properly. Gold could have been lost at the base of the bowl as it was not sampled correctly. The L3/L4 resource protocols show high to extreme FSE values related to sample splitting after crushing. Where any oversize pulp is reduced by scooping, the GSE issues raised above are applicable.

During 2006 to 2007 LGL trialled the use of sample processing via gravity concentrators (Figure 3). Large samples (e.g., HQ core, sludge and grab samples) were crushed and passed through a gravity spinner. The concentrate and tails samples were submitted for pulverise and leach (PAL) assays. It was recognised that fine gold was lost to the tails as it was not liberated. Actual gold recovery ranged from 40% to 80%. The trials were stopped due to backlogs in sample processing. Using a whole sample processing methodology is a valid approach to deal with coarse gold-bearing samples, though in practical terms requires substantial facilities and staffing in a large operation [14].

Figure 3. LGL gravity concentrator facility.

During the LGL period, minimal face sampling was undertaken with a preference for grab sampling of development rock for grade control. Grab samples were used for decisions such as ending a drive, orientation of mining relative to grade and for tracking grade trends. Samples were generally collected at underground or surface stockpiles, rarely at the mine face. Two 10 kg samples were collected of generally less than fist size material. Some trials of face samples were undertaken, where 3–5 kg samples were collected as nominal 1 m composites across the face (Table 11).

2.5.4. CGT (2010-present time)–Resource Drill Core Sampling

Diamond drilling is undertaken on approximately 25 m sections and from 15 m to 5 m down-dip [28]. Approximately 4.9 km of underground diamond drilling per month is planned for the 2018–2019 year. In 2011, CGT introduced a whole core sampling approach on nominal 0.4 m lengths for NQ2 core and 0.5 m lengths for LTK60 core (44 mm diameter). The nominal lengths were selected to generate approximately 2 kg of sample material for full LW analysis. The change to full core sampling was made to increase the volume of samples collected from diamond drill holes. This reduction in sample length also increased the resolution of grade within mineralised zones making it easier to discriminate narrow high grade structures.

From 2014, sampling intervals have a nominal length of 0.7 m with a minimum length of 0.3 m of NQ core (1.6 kg to 3.5 kg mass range). Where the sample size exceeds the maximum LW weight of 2.3 kg, it is split down to a 2 kg to 2.3 kg sub-sample by RSD and the remaining material retained (Table 12). All sample preparation and assaying is undertaken on site (Figure 4).

Table 12. FSE for nominal drill core protocol at BCOG across d_{95Au} scenarios (Table 6) for Castlemaine Goldfields Pty Ltd. (CGT) (post-2014). FSE calculations assume the presence of coarse gold in the pulp.

Stage	Type	Protocol	[1] FSE	Comment
Exploration/resource development	Whole core (1.4–3.4 kg)	Jaw crush all to P_{90} −6 mm Pulverise in LM5 P_{90} −75 RSD split to 2.3 kg for LW2300	±3% ±5% ±9% ±10%	FSE relates to splitting the pulp to achieve a target 2.3 kg for LW when pulp is >2.3 kg

[1] FSE error definition: green: low (<±20%).

Figure 4. Current on-site laboratory—Clockwise: received samples; Boyd crusher and rotary splitter; LM5 pulveriser; and LeachWELL bottle roll unit.

2.5.5. CGT (2010-present time)—Grade Control Sampling

During the 2018–2019 period, approximately 1.1 km of development will be mined resulting in 315 faces yielding around 1600 samples. Face samples are collected as 1 m composites of 2 kg, where the entire sample is crushed, pulverised to P_{90} −75 µm and assayed by LW. Where a face cannot be sampled, a grab sample may be taken from the rock pile underground or stockpile on surface. Multiple 2 kg grab samples are taken; these are assayed in total via LW.

2.6. Quality Assurance/Quality Control

QC includes the use of Certified Reference Materials ("CRM") and blanks. CRMs are inserted at a rate of 1 in 20 (e.g., 1 CRM per 20 samples). Blanks are inserted at a rate of 1 in 15 in mineralised zones and after visible gold occurrences. Up to three blanks may be placed after rich visible gold samples. Testwork has shown that LW generally recovers 96–99% of gold on a 24 h leach.

2.7. Duplicate Pair Analysis

Duplicate sets are available for Ballarat mineralised material from different stages of the project life (Table 13).

Table 13. NQ/NQ2 and HQ core duplicate pair analysis.

Sample Type/Error (Protocol)	Sampling (%)	Preparation (%)	Analytical (%)	Total (%)
GF half NQ/NQ2 core (Table 9)	Half drill core	-	Split at −75 µm (FA50)	-
Diamond drill core	74	-	49	89
Number of pairs	89	-	134	-
LGC/CGT (L1/L2) half HQ core (Table 11)	Half drill core	Split at −1 mm	Split at −75 µm (LW2000)	-
Diamond drill core	70	37	20	79
Number of pairs	167	125	125	-

Pairs cover range of mineralisation grades from low to high.

The sampling error is high on half core pairs (74% and 70%), testifying to the presence of coarse gold. The high preparation (37%) and analytical (20%) errors for the LGL HQ core also reflect the presence of coarse gold. Of particular interest is the analytical error for the BGF NQ/NQ2 core, where an FA50 from 2.5 kg of pulp yields a very high error (49%). This indicates coarse gold in the pulp and the challenge of using FA when sub-sampling from a pulp (see Section 2.3.2: Table 7). All values indicate that half core samples are not optimal, verifying the use of whole core samples and a large assay mass. The current approach uses whole core and assay, thus the sample preparation and assay error is minimal (Table 12).

2.8. Discussion

Ballarat mineralisation is dominated by coarse gold across all grades, which increases the challenge of achieving representative samples. Over time, protocols were developed which dissolve half or whole core with FA or aqua regia leach, through to SFA and LW and ultimately to whole-core LW2300 assay.

The current protocol involves the collection of 0.3 m to 0.7 m runs of whole NQ core. Preparation includes drying, crushing and pulverising prior to LW2300. The 2018 resource was informed by 38,730 samples (90.2 km of drilling across 570 holes) of which 97% are whole core samples [28]. The 0.7 m composites across different orebodies give a range of RSV values between 240% and 610% (grades 0.01 g/t Au to 1274 g/t Au). These values testify to the heterogeneous nature of the mineralisation. Individual samples are not locally representative, though >2000 samples (e.g., >4.4 t) are likely to achieve reasonable representativity across a given mineralised zone.

Whole core sampling followed by full to split sample LW2300 assay, yields a low FSE value. Where samples are >2.3 kg, they are split down at the pulp stage to 2.3 kg by RSD. In the worst case, this yields an FSE of ±10% (Table 11), though there is potential for GSE when splitting pulp containing coarse gold. The duplicate pulp analysis shown in Table 12 confirms residual variability in some pulps due to coarse gold. With good laboratory practice through the use of an RSD, the GSE can be minimised. Arguments against whole core sampling revolve around no reference core remaining, though with high-resolution digital photography, detailed logging and internal/external peer review this does not have to be an issue [14]. A risk review of the Ballarat core sampling programme used during production for resource estimation is given in Table 14.

Table 14. Risk review of the Ballarat core sampling programme (Table 12) used during production for resource definition.

	Key Parameter	Comment	[1] Component Error	TOS Error	[2] Error Rating
1	Spatial distribution and number of samples	Approx. 25–50 m by 5–15 m drill grid A mineralised zone may be informed by 50–200 holes yielding 3000–13,800 samples	[3] ND.	GNE	Mod.
2	Sample mass (representa-tivity)	Each NQ core sample approx. 0.3–0.7 m for 1.4–3.4 kg; total sample for a mineralised zone can theoretically range 5–30 t Indicated optimum mass around 50 kg to 9 t to achieve 90% ±15%			High
3	Collection and handling	Core recovery generally good, though some localised areas of poor total core recovery Core trays delivered directly from the mine to site core shed		EE	Low
4	Transport and security	All samples placed into plastic bags and sealed All samples delivered directly to the site laboratory		-	Low
5	Preparation	Entire sample of 1.4–3.4 kg crushed and pulverised to P_{85} −75 µm, with 1.4–2.3 kg RSD split for assay Equipment cleaned between samples Error from splitting, where pulp duplicate analysis indicates an analytical component which will dominantly comprise splitting error		GSE, FSE, DE, EE, PE	Low
6	Assay	LW2300 process undertaken correctly	[4] 20%	-	Low
7	QAQC	CRMs (1 in 20) and Blanks (1 in 20) within expectation Written protocols for the sampling-assaying process	-	-	Low
8	Validation/variability indicators	Duplicate analysis indicates a high total sampling error, supporting the need for whole core sampling Nugget effect of 65–85% and RSV 240–610% Monthly grade reconciliations variable, with annual reconciliations within expectation for Inferred Mineral Resources	[3] ND.	GNE, GSE, DE, EE, WE	Mod.
		Summary Sample representativity (1)–(3) Sample preparation and assay (4)–(7) Fit-for-purpose			 Mod. Low Yes

[1] Component errors from duplicate pair analysis; [2] Indicative total error rating; red: high (>±50%); orange: moderate (±25–50%); green: low (<±25%); [3] Not determined as current approach uses whole core; [4] Based on pulp duplicates; current practice only splits pulps when required (e.g., pulp is >2.3 kg).

Annual reconciliation at Ballarat typifies the high nugget environment, where global estimates are reasonable (e.g., 2017–2018 period tonnage +1% and grade −6%) while month-by-month variability is large (e.g., 2017–2018 period tonnage ±18% and grade −50% to +33%) [28]. Reconciliation is impacted by a number of inputs other than from sampling, including drill spacing, estimation method and top-cut applied. An inverse distance weighting interpolator is used, which is sub-optimal [28]. Year-on-year, over 90% of the reported Mineral Resources at Ballarat are classified into the Inferred category, which broadly accords with both the nature of the mineralisation and performance of the resource estimate [28].

The sampling regime at Ballarat is overall fit-for-purpose given the nuggety nature of the mineralisation and the use of whole core samples. Improvements to reconciliation on a month-on-month basis will most likely come from closer drill spacing and the implementation of a kriging-based grade interpolator.

3. Case Study 2: San Christina Mine, South America

3.1. Introduction

The San Christina mine is located in South America and for reasons of confidentiality its exact location is not disclosed. The project is privately owned and operated. During the period 2005–2012, the mine underwent a period of evaluation and subsequent mining. The programme yielded 75,840 t of ore at a reconciled head grade of 23.2 g/t Au for 42,290 oz Au recovered. It was known that the mineralisation was dominated by coarse gold and that a significant nugget effect existed. As part of the mine re-evaluation, commencing in February 2018, a series of characterisation tests were undertaken to investigate gold particle sizing and grade variability. An Inferred Mineral Resource of 55,000 t at 9.5 g/t Au and an Indicated Mineral Resource of 16,500 t at 25.7 g/t Au were declared in late 2018 in accordance with the JORC Code. Mining recommenced in February 2019.

3.2. Geology and Mineralisation

The sub-vertical vein system is hosted in a series of volcanic rocks. The veins comprise massive, brecciated to laminated quartz, with up to 10% galena and sphalerite in the oreshoots. Individual vein widths vary from 0.5 m to 1.5 m, with an average of 1.2 m. Outside of the oreshoots, the vein may reduce to a few cm or fault gouge. Locally the main Veta (vein) Christina splits, with splays emanating into short-lived to continuous structures. The split hinge zones sometimes host (>15 g/t Au) oreshoots. As well as the Veta Christina, four other vein systems have been identified where shallow historical workings confirm gold mineralisation. The additional reefs contain an Exploration Target of 250,000 t to 500,000 t with a grade range of 10 g/t Au to 25 g/t Au.

Economic grades are located within steeply plunging oreshoots that are traceable for 25 m to 60 m along strike and >300 m down-plunge. All vein structures contain low-grades up to 2 g/t Au, with the oreshoots historically containing recoverable grades of between 15 g/t Au and 30 g/t Au. The veins that comprise the oreshoots are generally continuous, through grades are variable and discontinuous low-grade zones can be present. Individual oreshoots generally represent between 30,000 t and 75,000 t of mineralisation. Criteria for the recognition of an oreshoot, other than gold grade relate to: (1) laminated vein with thickness greater than 0.5 m; (2) the presence of galena-sphalerite and locally visible gold; and (3) moderate to strong wallrock silicification.

3.3. Gold Particle Size and Liberation Diameter: Characterisation via Bulk Sampling

A characterisation programme was undertaken based on a series of bulk samples collected on the accessible 2 level on the Veta Christina reef. After detailed geological mapping and chip sampling of the drive backs, a series of locations were defined. At each location an approx. 1.5 m to 2 m long (strike) by 1.5 m deep (up-dip) by 1.2 m wide cut was mined into the level back to yield approx. 9 t to 12 t of mineralised vein sample. This material was stage processed through a modified plant to liberate gold over a series of crush-grind concentration steps. The gold liberated from each stage was sized and assayed to provide an indication of gold particle size distribution (Table 15 and Figure 5).

Table 15. Summary of gold particle size distribution by mass across different grades based on 18–24 t bulk sample processing.

Grade (g/t Au)	+100 μm	+500 μm	+1000 μm	d_{95Au} (μm)	d_{maxAu} (μm)
2	53	0	0	300	400
[1] 6F (BCOG)	54	7	0	500	700
[2] 6C (BCOG)	66	21	5	1000	1500
9	59	5	0	600	725
13	67	22	8	1100	1500
25 (ROM)	70	27	9	1500	2000

[1] Fine-gold dominated; [2] Coarse-gold dominated.

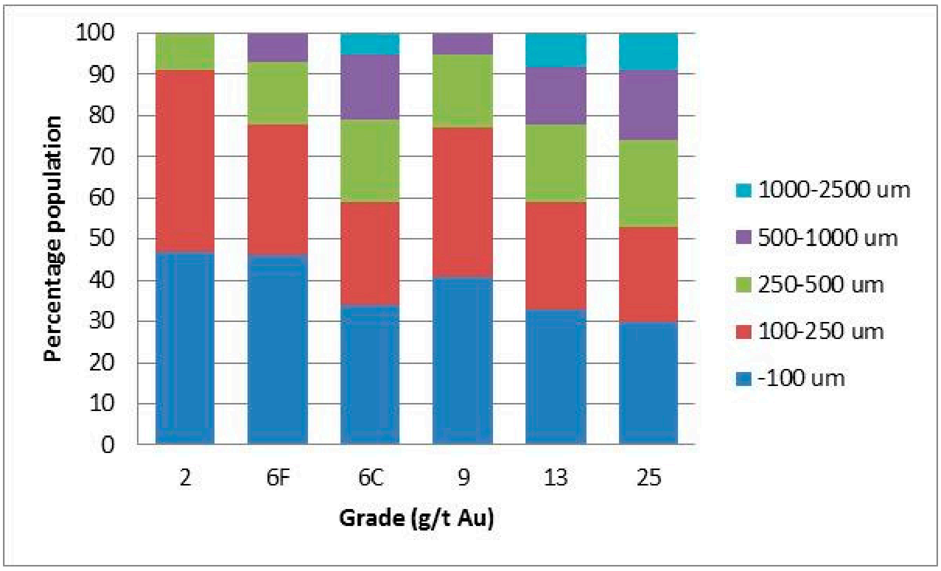


Figure 5. Gold particle size distribution by mass across different grades on the Veta Christina reef.

There is consistency of the sub-100 μm to 500 μm fractions with increasing grade and a relatively small increase in the >500 μm fractions. There is a distinct variation in population around 6–9 g/t Au, where in some cases a finer gold population dominates (Table 15 and Figure 5; refer 6F) but in others a coarse population exists (Table 15 and Figure 5; refer 6C). These indicate a potentially more disseminated finer gold background population that may be easier to sample.

Core logging and face mapping reveal that gold particle clustering becomes locally material. These grade hotspots relate to 1–3 cm^3 of clustered >250 μm gold, which provide gold-only composite clusters up to 1 cm^3 [11]. These can be missed by small channel samples (13.5 kg/m) and core samples (4.9 kg/m). If intersected in small volume samples, they will provide extreme value grades. In one case, an approx. 2 cm^3 cluster in half core gave a grade of 985 g/t Au over 0.5 m.

3.4. Theoretical Sample Mass

A theoretical field sample mass can be estimated using Poisson statistics to achieve a given precision (e.g., ±15% at 90% reliability) [31]. The range of mass values for BCOG and run of mine mineralisation is given in Table 16. The major driver for large sample mass at San Christina is the gold particle clustering effect.

Table 16. Range of theoretical sample mass values for BCOG and ROM mineralisation.

Grade (g/t Au)	d_{95Au} (μm)/ [d_{Auclus} (μm)]	Clustering Effect	Sampling Constant (K)	Optimum Mass
2	300	None	5000	10 kg
6 (BCOG)	500 1000	None	2700 10,000	30 kg 150 kg
9	600	Potentially	2400	35 kg
13	1100 [5000]	High	6200 52,500	120 kg 8 t
24 (ROM)	1500 [10,000]	Very high	28,500 80,500	120 kg 35 t

3.5. Sampling Protocol Development

3.5.1. Channel Sampling and Assay Methodology

Channel samples were cut using a diamond saw to produce a near uniform 10 cm wide by 5 cm deep channel to yield around 13.5 kg/m. Two 5 cm saw cuts, 10 cm apart were cut and a hammer and chisel used to break the intervening block of rock out of the channel. Samples were collected across the vein as 0.4–0.5 m lengths (5.4–6.7 kg/m) from the reef hanging- to foot-wall.

Samples were weighed and compared to their expected mass, which was 6.75 kg for the dominant 0.5 m samples. Around 75% (N = 206) were within ±15% of the target mass (e.g., ±1 kg: 5.75–7.75 kg) based on 275 samples. Overall this was a good result, given the inherent challenges of collecting channel samples [2]. Mass variability related to the interrelationship between DE and EE, where the saw-cut depth could vary depending upon face profile. The extracted material depended on effort to remove the delimited sample and loss through fines and fly-rock.

Due to the presence of coarse gold, SFA was considered the most applicable technique. All channel samples were bagged and secured on-site and transported to an independent laboratory. Samples were dried and crushed to P_{90} −3 mm, one-third RSD split off and pulverised to P_{95} −75 μm and then split for two to three SFA1000.

3.5.2. Diamond Drilling Programme

During 2002–2003, a 49-hole surface diamond-drilling programme was undertaken on the Veta Christina reef to assess the predicted oreshoot from surface to a depth of 225 m. The NQ (4.9 kg/m) holes were drilled on an approx. 10–20 m by 10–20 m grid. Samples were collected across the vein as 0.4–0.5 m lengths (1.9–2.4 kg/m) from the reef hanging- to foot-wall. A 0.4 m sample was taken into both the hanging- and foot-walls for assay. After logging and photography, cores were cut in half and one half sent to an external laboratory. The samples were dried and crushed to P_{90} −3 mm and pulverised to P_{95} −75 μm and riffle split into two halves and both screen fire assayed.

3.5.3. Duplicate Pair Analysis—Channel and Core Samples

Duplicate pair analyses were undertaken for channel and core samples (Tables 17 and 18). The channel samples (79%) show a smaller sampling error component compared to the core (93%), probably relating to a larger mass. Both values are relatively high and reflect a high geological nugget effect. The relatively high preparation error of the channel samples (32%) indicates the presence of coarse-gold at the split stage. The analytical error components for both channel and core samples are slightly higher than expected, reflecting the presence of residual coarse gold in the pulps. Overall the results show that channel samples are of better quality than core, though both error values are high.

Table 17. Face channel sample duplicate pair analysis.

Sample Type/Error (Preparation Route)	Sampling (%)	Preparation (%)	Analytical (%)	Total (%)
	Duplicate face	−3 mm split	−75 um split	-
Face channel	79	32	18	87
Number of pairs	75	150	150	

Pairs cover range of mineralisation grades from low to high.

Table 18. Core sample duplicate pair analysis.

Sample Type/Error (Preparation Route)	Sampling (%)	Preparation (%)	Analytical (%)	Total (%)
	Half drill core	No split	−75 um split	-
Diamond drill core	93	-	21	95
Number of pairs	100	-	200	-

Pairs cover range of mineralisation grades from low to high.

3.6. Bulk Sample Trials

3.6.1. Bulk Sample Strategy

The high variability of channel and drill samples lead the project team to consider the use of bulk sampling to overcome the nugget effect. During underground development, a programme of bulk sampling was undertaken. Bulk samples comprised three types: (1) drive round of approx. 25–30 t; (2) raise development of approx. 10–15 t; and (3) drive back cut of approx. 15–20 t. All development faces were channel sampled and mapped to guide the bulk sampling process. Individual rounds were transported to surface and kept isolated prior to processing.

3.6.2. Bulk Sample Processing

It was considered best practice that each bulk sample be processed in its entirety through an on-site plant. Each bulk sample lot was batch milled through a surface-based gravity plant that was able to process 8 t per hour (e.g., target 160 t per day). The plant was used for both bulk samples and production batches. Ore was passed through jaw and gyratory crushers prior to being fed into a 30 t capacity fine ore bin. This bin was attached to four strain gauges to provide a weight determination. The fine ore bin fed into a ball mill, yielding a P_{90} −150 µm. The ball mill was fitted with a large access panel to allow cleaning and a bunded wash area immediately below to allow access and containment for washings. Washings were collected, tabled and assayed.

A simple gold trap was located at the outflow of the ball mill, which typically collected 5–15% of the gold in a sample (usually particles >0.1 mm in size). This trap was cleaned out after every sample. The feed then passed through a 0.5 mm screen, with the undersize passing to a 250 mm Knelson concentrator. The oversize recirculated to the ball mill.

The mill circuit was flushed out after every sample with 2 t of waste rock and stripped/cleaned after every 4th sample (e.g., 120 t). Gold recovered during stripping was proportionally re-combined with the previous bulk samples based on their percentage gold yield. This was found to be the best way in which to deal with recovered gold in-circuit, as the higher the grade of ore processed the greater the problem. Between 5% and 10% of the batch gold yield was usually recovered from the ball mill. Minimal gold was recovered from elsewhere.

All concentrates were weighed, combined and tabled, prior to size by assay to extinction. Tails samples of 1 kg were collected every 15 min by an automatic Vezin splitter. Every hour, the composite tails samples were removed for drying and on-site pulverisation to P_{80} −100 µm and then RSD split down to 1 kg. The series of hourly 1 kg splits were submitted to an external laboratory for SFA1000. A 30 t primary bulk sample yields 22 kg of tails sample, with ultimately eight 1 kg samples assayed. Reconciliation of mill gold yield and tails assays showed a recovery of >75% for grades >6 g/t Au and up to 95% for grades >25 g/t Au.

3.6.3. Bulk Sampling of 2 Level Veta Christina South Reef

Bulk sampling was undertaken along the 2 level drive on the Veta Christina South reef. Sixty-five metres of 2.5 m by 2.5 m was driven along the reef, which varied in width from 1 m to 1.5 m. Based on bulk sample grade and geological features, the oreshoot zone was represented by 43 m of strike, comprising 24 bulk samples. Each round was mucked carefully to ensure collection of all broken material. All bulk samples were processed through the pilot plant. All faces were channel sampled and mapped. Table 19 shows a comparison of sample types along the Veta Christina South 2 level drive.

Table 19. Comparison of samples along the Veta Christina South 2 level drive.

Sample Type	Bulk	Grab	Channel
Sample mass/Total mass	18 t 432 t	5 × 5 kg (25 kg) 1 t	13.5 kg/m 472 kg
No. of samples	24	120	24
[1] Mean grade (g/t Au)	27.3	38.4	14.7
Min. grade (g/t Au)	4.1	1.54	0.01
Max. grade (g/t Au)	69.6	452.3	225.5
RSV	73%	297%	306%
Nugget effect	48%	ND	90%
Difference with respect to bulk sample grade	-	+28%	−46%

[1] Grades back-calculated to minimum mining width. ND: not determined.

The bulk and channel samples grades are back-calculated to a 1.5 m minimum mining width. The lowest variability is displayed by the bulk samples (18 t over the minimum mining width), with an RSV of 73% and nugget effect of 48%. In contrast, the smaller channel samples show a high RSV of 306% and an extreme nugget effect. The grab samples were diluted with material outside of the minimum mining width, showing a high RSV and mean grade, despite the dilution. The challenges of fines bias during grab sample collection (e.g., high DE and EE) are well-known [2].

If the bulk samples are taken to be drive-grade (e.g., 432 t at 27.3 g/t Au), neither the grab or channel samples predict grade well. The channel samples under-call the grade by 54%, whereas the grab samples overestimate the grade by 41%.

As bulk sampling progressed, it was possible to undertake duplicate pair analysis. The results display a total sampling error of 44%, comprising a sampling component of 40% and analytical component of 18% (Table 20). These values indicate the validity of the bulk sampling approach, where a sampling component of 44% can be considered acceptable given the clustered nature of the gold.

Table 20. Bulk sample duplicate pair analysis.

Sample Type/Error (Preparation Route)	Sampling (%)	Preparation (%)	Analytical (%)	Total (%)
	Rounds	**-**	**−500 µm split**	**-**
Bulk samples	40	-	18	44
Number of pairs	128	-	65	-

Pairs cover range of mineralisation grades from low to high.

3.6.4. Head Split Bulk Sample Grade Determination

Given the reliance on full bulk sample processing, an alternate way of determining bulk sample grade was sought. A sub-sampling option was devised which ran the 15–30 t bulk sample through the primary and secondary crushers to achieve a P_{90} −4 mm product. A linear splitter was installed after the secondary crusher to take a 2.25 kg sample every 2.5 min. Multiple increments were considered the best option to increase the probability of gold being encountered in the split. For a 30 t bulk sample, an approx. 200 kg sub-sample was collected. This was then processed via a laboratory-based process unit, where a preliminary recovered grade was declared in approx. 6 h from arrival at the plant. A 10 kg tails sample was collected manually from the Knelson concentrator underflow.

For the first 30 bulk samples, two head-splits were taken and the remainder of each bulk sample processed in its entirety. The head grade of the 30 bulk samples (approx. 900 t) was 20.6 g/t Au and the grades of the two sets of head-splits were 19.2 g/t Au and 22.3 g/t Au respectively. These lie within ±10% of the full bulk sample grade, which is an acceptable result.

Duplicate pair analysis of the head split bulk sample protocol yielded a total error of 57%, with relative components of 50%, 21% and 19% (Table 21). These values indicate that the protocol worked well, where clusters were broken down during crushing and multiple increments reduced periodic variability. The company opted not use this option routinely, as they wished to recover gold from the pilot plant for sale.

Table 21. Bulk sample head-split duplicate pair analysis.

Sample Type/Error (Preparation Route)	Sampling (%)	Preparation (%)	Analytical (%)	Total (%)
	Rounds	**−4 mm split**	**−500 µm split**	**-**
Bulk samples	50	21	19	57
Number of pairs	30	30	30	-

Pairs cover range of mineralisation grades from low to high.

3.7. *Quality Assurance/Quality Control*

QC for core and channel samples included the use of CRMs, blanks, pulp quality checks, umpire assays and duplicates. CRMs were inserted at a rate of 1 in 20, and blanks inserted at a rate of 1 in 20 and after visible gold occurrences. Duplicates were collected at a rate of 1 in 20, though this varied depending upon the testwork being undertaken. Pulps were checked for quality at a rate of 1 in 20. Umpire pulp splits were taken at a rate of 1 in 30 and submitted to a second external laboratory. All samples (e.g., core, channel and grab) collected underground were removed to the on-site logging and sample preparation facility.

QC for the bulk sample circuit included blanks (2 t) at 1 in 15 and barren flushes (2 t) between all samples, which were assayed at a rate of 1 in 5. All concentrate and tails assays related to bulk sampling have the same QC as other samples. Samples going to the external laboratory were secured into boxes and transported by road in locked containers. QA documentation of activities included sample collection, security and transport, through to preparation and assaying.

3.8. *FSE Analysis of Sample Protocols*

An analysis for FSE was undertaken for each protocol applied at San Christina (Table 22). The highest error related to grab samples and the collection of 25 kg from a 30 t pile. The method was discontinued. The channel samples also displayed a high FSE relating to the splitting of the sample post-crushing.

Table 22. Protocols and FSE for resource evaluation and mine development stages at San Christina. FSE calculations based on the BCOG and run of mine grade scenarios (Table 15). FSE calculations assume coarse gold in the pulp.

Stage	Type	Protocol	[1] FSE	Comment
Resource evaluation	Core (half core)	Half NQ core (1.9–2.4 kg) crush to P_{90} −3 mm Pulverise to P_{95} −75 µm and riffle split in half 2x SFA1200	±0% ±0%	No FSE as entire sample prepared and assayed
Resource evaluation	Core (whole core)	Whole NQ core (2.8–4.8 kg) crush to P_{90} −3 mm Pulverise to P_{95} −75 µm and riffle split in half 2x LW2500	±0% ±0%	No FSE as entire sample prepared and assayed
Mine development	Face channel	5.4–6.7 kg crush to P_{90} −3 mm, then RSD split off one third Pulverise to P_{95} −75 µm and RSD split into thirds 2–3 × SFA1000	±49% ±34%	Substantial FSE at post-crusher split
Mine development	Bulk (full sample pilot plant)	15–30 t crushed, pulverised and fed through gravity concentrator 22 kg of tails incrementally linear split at P_{90} −500 µm 8 kg RSD split and pulverised to P_{95} −75 µm 8 × SFA1000	±5% ±5%	Entire sample processed through plant FSE relates to tails sample splitting
Mine development	Bulk (head coarse split)	15–30 t primary and secondary crushed 200 kg incrementally linear split from the 30 t bulk sample at P_{85} −4 mm200 kg crushed, pulverised and fed through gravity concentrator 10 kg of tails incrementally collected at P_{90} −500 µm 10 kg RSD split and pulverised to P_{95} −75 µm, 2 × 1 kg sub-samples taken for SFA1000	±15% ±25%	Most FSE relates to the primary split Recoverable gold grade determined from gravity concentrate FSE relates to tails sample splitting
Mine development	Grab	5 kg crush to P_{90} −3 mm, then RSD split 2.5 kg Pulverise to P_{95} −75 µm and riffle split in half 2x SFA1250	±305% ±205%	Sample collected at 5 × 5 kg of sub-8 cm material Large FSE on collection of 25 kg from 30 t lot FSE given for entire process

[1] FSE error definition: red: high (>±30%); orange: moderate (±20–30%); green: low (<±20%).

3.9. Reconciliation between Grade Control Sampling and Production

As part of the original evaluation phase, a trial mine shrinkage stope was extracted on the Veta Christina south between levels 1 and 2. This yielded 2692 t at a head grade of 26.7 g/t Au compared to an estimated 23.6 g/t Au via bulk sampling (Table 23).

Table 23. Comparison between diamond drill, channel samples and bulk sample estimates for the Veta Christina south trial (1-2 level) stope panel with the plant head grade. All estimated grades account for intentional and unintentional dilution.

Sample Type	No. Samples	Total Assayed/Processed	Estimated Stope Grade (g/t Au)	Difference with Reconciled Grade
Diamond drill	5	39 kg	6.1	−77%
Face channel samples	44	891 kg	10.0	−63%
Bulk sample (full)	44	792 t	23.6	−12%
Reconciled head	-	2692 t	26.7	-

The bulk sample grade understated the plant head grade by 12%. Other estimates based on diamond core and face channel samples understated the plant head grade by >49% (Table 24). Over the production period (including development and trial mining), 75,840 t was produced at 23.2 g/t Au against a bulk sample predicted grade of 20.3 g/t Au (Table 24). The bulk samples clearly provided the best evaluation of mineable grade.

Table 24. Reconciliation between reconciled plant head grade and diamond drill core and bulk sample estimates for the 2005–2012 period. All estimated grades account for intentional and unintentional dilution.

Year	Tonnes Processed (t)	Reconciled Head Grade (g/t Au)	Predicted Grade (g/t Au)	
			Drilling	[1] Bulk
2005–2006	6,460	19.6	4.0	18.4
2007	9,510	24.5	6.8	20.1
2008	10,740	19.2	4.5	18.2
2009	12,630	26.7	14.3	21.9
2010	12,980	23.6	9.2	21.2
2011	11,950	28.1	-	25.7
2012	11,570	18.3	-	15.3
Total	75,840	23.2	8.4	20.3
Difference to reconciled grade	-	-	−64%	−13%

[1] From Q2 2011 all bulk sample grades as head split samples.

3.10. *Sample Application and Resource Estimation*

3.10.1. Sampling Strategy

The most effective sample type were the bulk samples, given that they yield a sampling component of 44–57% (Tables 20 and 21) and provide a reconciliation with mined grade to within ±20% (Tables 23 and 24). Allowing for development drive width (2.5 m), the effective mineralised bulk sample mass is 18 t based on a minimum mining width of 1.5 m, a drive height of 2.5 m and advance of 1.8 m. They are generally large enough to overcome the high geological nugget effect driven by the gold particle clustering. All small sample types (e.g., drill core and channel samples) have a low probability of intersecting clusters. Given that at a run of mine grade tonne of ore may only contain 1 to 4 gold clusters containing >70% of the grade, the probability of intersecting zero clusters ranges from 97% to 99% for core samples and 94% to 98% for channel samples.

All resources defined by development bulk samples are reported as Indicated Mineral Resources (6 month ±15–25%) and those solely by diamond core drilling as Inferred Mineral Resources (globally expected to be ±50%) [27].

3.10.2. Resource Estimation based on Diamond Drilling

Diamond core drilling provides a method to evaluate reef location, geometry and internal characteristics. The 2018 resource estimate was based on 49 NQ holes, drilled on a 10–20 m by 10–20 m grid and whole core sampled (sampled separately as both halves). The programme yielded 106 reef whole-core composites with a total mass of 400 kg, where 14 h contained visible gold and graded >15 g/t Au. An ordinary kriged block model yielded a global resource grade of 9.4 g/t Au. It is realised that global grade understates the mineable grade, which could be in the 22 g/t Au to 26 g/t Au range but provides confidence to commit to underground development.

The robust lower grade estimate likely reflects the presence of a finer more disseminated background gold population below 9 g/t Au (Tables 15 and 16 and Figure 5). This mineralisation requires a lower sample mass (e.g., 30–35 kg; Table 16), where the number of samples informing a given block estimate will be >30 kg in total mass.

3.10.3. Bulk Sampling of 4 Level Veta Christina South Reef

Grade evaluation during 2019 is utilising development bulk sampling and full processing. The advantage of this option is that gold is produced, for example a sample at 6 g/t Au (e.g., BCOG) will yield around 3 oz of saleable gold. An initial programme was undertaken on the Veta Christina South reef 4 Level in preparation for mining. A 70 m drive was developed along the reef, which varied in width from 1.25 m to 1.50 m. Based on bulk sample grade and geological features, the oreshoot zone was represented by 55 m of strike, comprising 30 bulk samples. All bulk samples were processed through the pilot plant. All faces were channel sampled and mapped. Table 25 shows a comparison of sample types along the Veta Christina South 4 level drive.

Table 25. Comparison between diamond drill, channel samples and bulk sample estimates for the Veta Christina South bulk samples with the plant head grade. All estimated grades account for intentional and unintentional dilution.

Sample Type	No. Faces	Total Assayed/Processed	Grade (g/t Au)	Difference with Bulk Sample Reconciled Grade
Face channel samples	31	633 kg	15.6	−36%
Bulk sample	30	982 t	24.5	-
Drill-only block model	-	-	9.4	−62%
Local simulated block model (core and channel samples)	-	-	14.9	−39%

As observed previously (Tables 23 and 24), the block models based on drilling or drilling and channel samples understate the bulk samples grades. The global bulk sample yielded 625 oz of gold bullion for sale. Between 2 and 4 levels (approx. 80 m vertically), bulk sampling has defined an Indicated Mineral Resource of 16,500 t at 25.7 g/t Au to provide a base for mining over 12 months.

3.11. Discussion

The San Christina reefs bears coarse gold-dominated mineralisation, where >65% of the gold is present in particles with a size greater than 100 μm for grades above 6 g/t Au. However, gold rarely occurs >2000 μm in size, the maximum gold particle size observed being 2500 μm. Traditional sampling methods such as face channel and diamond drill core samples understate the mean gold grade by 65% to 75%. This relates to gold particle clustering for grades nominally >9 g/t Au, where clusters of 0.5 cm to 2 cm of >500 μm gold increase the geological nugget effect. Given that at a run of mine grade tonne of ore may only contain 1 to 4 gold clusters that contain >70% of the grade, the probability of intersecting zero clusters is >94% for core and channel samples. These small samples fail to intersect the sparse clusters but will yield extreme value grades if they do. A number of sample types have been trialled at the project (Table 26).

Table 26. Summary of sample types used at San Christina.

Type	Stage	Period	Comment
Core (half)	Evaluation	2005–2011	Discontinued
Core (whole)	Evaluation	2011–2012 2019	Targeting and preliminary resource estimate Continued in 2019 for resource estimation
Channel	Evaluation	2005–2006 2019	Discontinued in 2006 Applied in 2019 for comparison, not used routinely during mining
Grab	Production	2005–2006	Discontinued
Bulk (full)	Evaluation	2005–2012 2019	Replaced by bulk (split) option in 2011 Continued in 2019
Bulk (head split)	Evaluation	2010–2012	Introduced to speed up development grade determination Maybe applied as required for fact grade determination

Based on a Poisson-based probabilistic method, a representative sample mass ranges from 30 kg to 35 t to achieve a precision of ±15% at 90% reliability [31]. The very large sample mass is driven 1–2 cm gold clusters within run of mine grade mineralisation. If these clusters did not exist, then a run of mine grade sample mass of <120 kg may be appropriate.

Given the challenges of small volume samples, a development drive bulk sampling programme was instigated where entire development blasts of approx. 30 t were taken and processed in their entirety through a surface plant. The 30 t development samples effectively contained a diluted 18 t sample of the mineralisation, based on a minimum mining width of 1.5 m (mean vein width 1.2 m). The composited bulk sample grades along the upper and lower drives of stopes provide a reliable estimate of stope grade. On a quarterly basis, reconciliation is generally within ±20% which accords with the Indicated Mineral Resource category applied. Reconciliation for the first month of production (March 2019) was based on a single stope panel of 1150 t at a bulk sample estimated grade of 22.5 g/t Au. Actual performance was 1275 t at 19 g/t Au, showing a grade under-call of 16%.

A risk review of the bulk sampling method used to support resource estimation is provided in Table 27.

Table 27. Risk review of the San Christina bulk sampling programme used during production for resource definition.

Key Parameter		Comment	[1] Component Error	TOS Error	[2] Error Rating
1	Spatial distribution and number of samples	Samples collected at approx. 1.8 m intervals along drives and some raises. Vertical drive separation approx. 15 m. Each stope block (~1600–3200 t) informed by between 40 and 80 bulk samples (upper and lower drives)	40%	GNE	Low-mod.
2	Sample mass (representativity)	Each sample 20–30 t; total sample mass collected around a stope block ranges between approx. 1200–2400 t Indicated optimum mass around 35 t to achieve 90% ±15% at ROM for clustered gold			Mod.
3	Collection and handling	Sample extracted by blasting Sample collection by small mechanised mucking unit. Floors cleaned as required by hand All samples transported to surface and kept separate prior to crushing and splitting		-	Low
4	Transport and security	Samples delivered directly from the mine to the on-site plant		-	Low
5	Preparation	Entire sample crushed and pulverised Plant cleaned and flushed between samples	18%	-	Low
6	Assay	Entire sample passed through gravity circuit Gold concentrates weighed and sent for fire assay to extinction Preparation and analytical error relate to tails sample split and assay		-	Low
7	QAQC	Duplicates and blanks within expectation CRMs within expectation Written protocols for the sampling-assaying process	-	-	Low

Table 27. *Cont.*

	Key Parameter	Comment	[1] Component Error	TOS Error	[2] Error Rating
8	Validation/variability indicators	Bulk sample RSV 65% Nugget effect 49% Quarterly grade reconciliation ±20% Stope grade reconciliation ±25% Inferred and Indicated Mineral Resources defined	Total 44%	-	Low
		Summary			
		Sample representativity (1)–(3)			Mod.
		Preparation and assay (4)–(7)			Low
		Fit-for-purpose			Yes

[1] Component errors from duplicate pair analysis; [2] Indicative total error rating; red: high (>±50%); orange: moderate (±25–50%); low-moderate (±20–35%); green: low (<±25%); ROM: run of mine grade.

The mine operator is a privately owned entity. Given that the company is not required to publicly report its resources, the extensive sampling regime may appear to be excessive. However, the group has a number of international investors and the owners may ultimately opt to publicly list the company. Importantly, they understand that quality data underpins quality decisions and require all its technical activities to be carried out to best practice.

4. Case Study 3: Nalunaq Mine, Greenland

4.1. Introduction

The Nalunaq mine is situated in southern Greenland, 86 km northwest of Kap Farvel. It was owned and operated by the former listed entity, Crew Gold Corporation between 1999 and 2009. Nalunaq was discovered in 1992, with extensive underground development and bulk sampling undertaken during 2000 to 2002 as part of a feasibility study [13]. At commencement of mining in 2002, the total Indicated Mineral Resource was 415,000 t at 26 g/t Au for 345,000 oz Au contained. The Indicated Mineral Resources were based on close-spaced channel-sampling of development drives with some drill intersections. The mine was designed to extract around 160,000 t per annum via longhole stoping. Between 2004 and 2009, the operation produced 655,000 t ore for 369,000 oz Au at a head grade of 17.5 g/t Au. Dominant high-grade production during this period came from the Target Block Main vein.

4.2. Geology and Mineralisation

Nalunaq is an orogenic narrow vein style gold deposit hosted by metavolcanic rocks [33]. The deposit lies in a metavolcanic thrust sheet resting on gently dipping meta-arkoses of the highest tectonic unit in the Psammite Zone. The rocks show amphibolite facies metamorphism and limited deformation. The metavolcanics are underlain and intruded by a late granitoid pluton, which is associated with a network of aplite sheets in the metavolcanics. The gross vein structure is a sheet with northeast-southwest strike and gross dip 35° southeast (range 20° to 55°). The total vein thickness ranges from less than 1 cm up to 3 m, with a mean of 0.7 m. The vein can be traced in outcrop for 1.4 km along the north side of Nalunaq Mountain and down the west side of Kirkspirdalen. The vein is emplaced in fine amphibolites and medium to coarse amphibolites interpreted as lavas and sills respectively. The vein dips at a slightly lower angle than the host succession and cuts across the lavas and sills.

The vein (Figure 6) commonly occurs as an array of quartz sheets (0.01 m to 1 m thick) that can be in direct contact with each other, separated by discontinuous, thin wall rock screens or separated by wide (0.1 m to 2.0 m) screens of wall rock. Locally, individual quartz veins diverge by more than 2 m from each other. Wall rocks show local alteration adjacent to the vein, usually within 1 m.

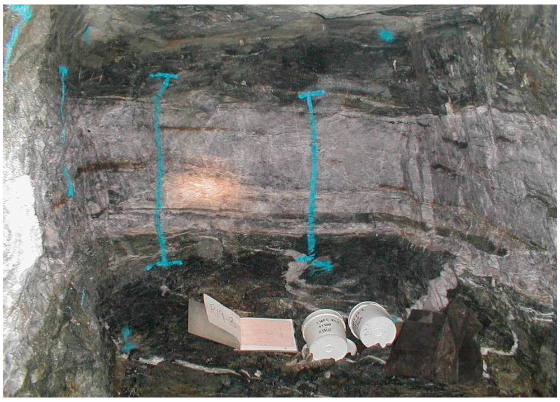


Figure 6. Target Block Main vein exposed in a raise, where the structure comprises a composite package of narrow veins separated by screens of wallrock. Vertical blue lines mark location of channel samples.

The vein mineralogy is simple, being mostly granular coarse quartz. Trace to minor amounts of lollingite, arsenopyrite, pyrite, pyrrhotite, chalcopyrite and bismuth sulphosalts occur in some veins. Gold occurs as fine (less than 100 µm) and coarse (greater than 100 µm) particles [13]. Significant amounts of visible gold are disseminated through the vein and locally cluster at the macro- to micro-scales to form high-grade zones (>25 g/t Au).

4.3. Gold Particle Distribution and Liberation Diameter

Gold particle size distribution at Nalunaq was determined from metallurgical testwork campaigns (Figure 7) [13]. Liberation diameter values for run of mine (20 g/t Au) and break even cut-off (8 g/t Au) are 850 µm and 400 µm respectively.

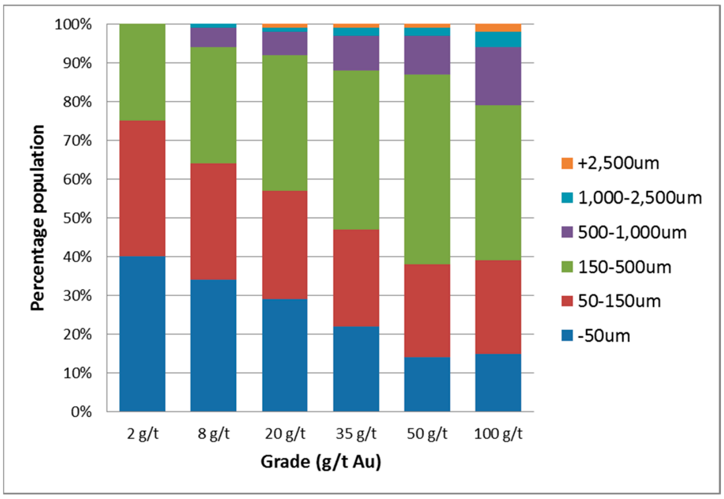


Figure 7. Gold particle size distribution by mass across different grades for the Target Zone Main vein.

4.4. Theoretical Sample Mass

The theoretical field sample mass was estimated using Poisson statistics to achieve a ±15% precision at 90% reliability [31]. The range of mass values for different grade scenarios is given in Table 28. An individual sample masse in range 15 kg to 25 t is not practical to collect but provides an indication of the nature of the mineralisation. A given ore zone probably requires around 20 spatially distributed grade control samples at approx. 6 kg each. These numbers are not unrealistic, for example a standard stope block of approx. 700 t would be informed peripherally by 16–20 face samples (96–120 kg).

Table 28. Range of theoretical sample mass values.

Grade (g/t Au)	$d_{95\mathrm{Au}}$ (µm) [d_{Auclus} (µm)]	Sampling Constant (K)	Optimum Mass
8 (BCOG)	400	2400	15 kg
20 (ROM)	850	2700	30 kg
20 (ROM)	[3000–5000]	13,600–38,700	1–5 t
30 (VHG)	[5000–10,000]	25,800–73,000	3–25 t

VHG: very-high grade reflecting some of the early Target Block Main vein stopes (diluted grades).

4.5. Sampling Protocol Development

4.5.1. Exploration Phase Development Sampling

During the 1998–2001 exploration/evaluation phase, underground face sampling and mapping was undertaken to gain a better understanding of grade continuity and distribution and for comparison to bulk samples [13]. Three channel samples were collected from each drive face and two from each raise face. The drive face channel samples were collected using a diamond saw (Figure 8). In raises, they were collected with a lump hammer and moil due to safety issues with using a diamond saw.

Figure 8. Diamond saw in operation during the exploration/evaluation phase.

All faces were cleaned and subsequently mapped and photographed. A minimum sample width of 0.5 m was used and in many cases required some wallrock to be collected to make up the interval. Where the vein was wider than 0.5 m, the interval was extended 5–10 cm across the contacts. The saw-cut channel samples were generally considered to be better quality than chip samples, though in places where the quartz vein was friable, the cutting and cooling water action promoted quartz fines loss (e.g., EE).

An individual sample was placed into a plastic pail containing a tag, sealed and stacked ready for transport to an off-site laboratory. Each face chip or channel (≈4 kg) sample was crushed to −3 mm in its entirety and 1 kg split off for SFA. After each sample, 0.5 kg of coarse silica sand was pulverised to clean the equipment.

4.5.2. Exploration Drill Core Sampling

Since 1993, over 30 km of NQ surface diamond drilling has been undertaken at Nalunaq. After logging and digital photography, half core samples were taken weighing approximately 2–2.5 kg. An individual sample was placed into a plastic pail containing a tag, sealed and stacked ready for transport to an off-site laboratory. Each sample was crushed to −3 mm in its entirety and 1 kg split for SFA. After each sample, 0.5 kg of coarse silica sand was pulverised to clean the equipment. Early (2000–2002) exploration core was sent off-site for assay, whilst core drilled during production was assayed on site.

4.5.3. Grade Control Sampling

Production drives at Nalunaq were developed with a "shanty" profile, approximately 3.4 m wide, by 4.3 m high and 2.4 m on the low side. As a result of production constraints, face samples were not routinely taken. If the geologist is in doubt as to the 'pay' of the face (i.e., ore or waste), a sample of one or two chip lines was taken across the face. All faces were mapped and recorded on pro-forma face sheets.

All drive samples were taken from the vein exposure in the 'short' sidewall by chipping three parallel vertical lines 1 m apart (Figure 9). A minimum sample width of 0.5 m was used and in many cases required wallrock to be taken to make up the interval. Each individual channel produced 1–2 kg of rock, with the final composite weighing 3–6 kg. A sample was collected every 3 m along the drive.

Figure 9. Chip sampling at Nalunaq mine. Each sample comprises three cuts, with a central cut (see blue lines on both pictures) and a separate cut 1 m either side of the centre cut.

Each sample was chipped as close to the reference line as possible, in practice within about 15 cm either side of the line. A lump hammer and chisel or geological pick was used as required. During sampling it was apparent that an extraction bias towards vein material was likely. The vein quartz appears generally less resistant than the hosting amphibolite rocks (Figure 9). A bias towards quartz material in a sample is a possibility, therefore biasing grade upwards. Such issues are typical of chip samples and relate to high DE and EE.

A key issue with chip samples at Nalunaq was the non-uniform sample support. Whatever length of sample was required (e.g., >0.5 m to 1.5 m), a 1–2 kg sample was collected from each sample site (Figure 9; blue lines). Even if the samples were collected perfectly (e.g., zero DE and EE), this resulted in a mass error (WE). For example, a 0.5 m sample should relate to a 2–4 kg/m mass, where a 1 m sample would have a 1–2 kg/m mass. The smaller sample lengths were biased with respect to the longer ones, add to this the DE and EE noted previously, this compounds the grade variability.

An experiment was undertaken where staff were given training to optimise sample collection to reduce DE and EE. Subsequently, five sample sites were chosen, where a 0.7 m sample was appropriate. Each chip sample was based on chipping a zone of 0.7 m long by 0.1 m wide by 0.01 m depth to yield approx. 1.9 kg of sample. As per normal practice, each site comprised three samples (Figure 9) to provide 15 samples in all. The collected sample masses ranged from 1 kg to 3.8 kg, with a mean of 2.4 kg. As a bias check, each sample was visually split into quartz vein and wallrock and compared to the measured quantity of each from each sample site. Overall, the samples contained 9–45% more quartz than they should - this was an indication of high EE.

4.5.4. Sample Preparation and Assay

Samples were processed at an on-site laboratory, which had a capacity of 25 samples per day and routinely processed 150–200 chip samples per month. In addition, it would also process drill core and grab samples (Figure 10).

Figure 10. Mine laboratory with the LM5 pulveriser in the foreground.

Each combined channel sample (approximately 3–6 kg) was dried and crushed in its entirety and then pulverised. Between samples, a vacuum head and compressed air blast were used to clean out the pulveriser bowl and subsequently a barren sand charge was run for 30 seconds. The pulverised sample was placed in a plastic tub. Five hundred grams were then scooped off for LW500 and leached for 5 h. After gold extraction by solvent, analysis was by atomic absorption spectrometry. Nalunaq mineralisation responded well to assay by LW, generally achieving a 98% recovery for grades >0.5 g/t Au. Table 29 provides a summary of the sampling protocols and analysis of FSE.

Table 29. FSE for exploration/evaluation and production protocols at Nalunaq. Calculations based on the BCOG and run of mine grade (non-clustered) scenarios in Table 28. FSE calculations assume coarse gold in the pulp.

Stage	Type	Protocol	[1] FSE	Comment
Exploration	Face chip	4 kg crush to −3 mm Riffle split 1 kg Pulverise in LM5 to P_{90} −75 µm SFA1000	±55% ±55%	Substantial FSE at post-crusher split
Exploration	Face channel	4 kg crush to −3 mm Riffle split 1 kg Pulverise in LM5 to P_{90} −75 µm SFA1000	±55% ±55%	Substantial FSE at post-crusher split
Exploration	Core	2–2.5 kg crush to −3 mm Riffle split 1 kg Pulverise in LM5 to P_{90} −75 µm SFA1000	±50% ±50%	Substantial FSE at post-crusher split
Production	Sidewall chip	5 kg crush to −8 mm Pulverise in LM5 to P_{90} −75 µm Riffle split 0.5 kg LW500	±5% ±5%	The 0.5 kg assay charge was scooped from the pulp, which is likely to yield high GSE, DE and EE
Production	Core	2–2.5 kg crush to −8 mm Pulverise in LM5 to P_{90} −75 µm Riffle split 0.5 kg LW500	±5% ±5%	The 0.5 kg assay charge was scooped from the pulp, which is likely to yield high GSE, DE and EE
Production	Grab	Collect 4 × 5 kg −8 cm development or stope material 4 × 5 kg crush to −8 mm Pulverise in LM5 to P_{90} −75 µm Riffle split 4 × 0.5 kg 4 × LW500	±140% ±165%	The 0.5 kg assay charge was scooped from the pulp, which is likely to yield high GSE, DE and EE

[1] FSE error definition: red: high (>±30%); orange: moderate (±20–30%); green: low (<±20%).

The FSE values display contrasting results, where the exploration protocol included a 1 kg coarse split, which yields a high FSE. Conversely during production, the entire sample was crushed and pulverised, followed by a split to provide a 500 g assay charge. This yields a low FSE, though there was potential for high GSE, given that the 0.5 kg assay charge was scooped from the pulverised lot. A riffle splitter was introduced to improve split quality.

QC included; pulp quality tests (1 in 50), CRMs (1 in 10–15), blanks (1 in 50) and pulp duplicates (1 in 10–15 sent to an external laboratory). In addition, 1 in 20 LW residues were sent to an external laboratory for FA to monitor recovery.

4.6. Face Sample Variability

Resource estimation and grade control was dominantly based on the face sample data, though drill holes were used to inform some resources. Table 30 shows the statistics for each of the three sample types.

Table 30. Statistics of different sample types within the Nalunaq database (as of 2006).

Sample Type	No. of Samples	Mean Grade (g/t Au)	Maximum Grade (g/t Au)	RSV	Field Sample Mass (kg)	Assay Type and Mass	Nugget Effect
Exploration face chip	856	62.1	2268	250%	3–5	SFA1000	81%
Exploration face channel	721	56.3	2831	270%	3–5	SFA1000	69%
Production sidewall chip	1321	50.7	1361	195%	4–6	LW500	93%
All samples	2898	55.5	-	230%	-	-	75%

The exploration channel samples showed less short-range variability with a nugget effect of 69%, whereas the production chip samples gave the highest value of 93%. The exploration chip samples yielded a nugget effect of 81%. The overall difference can be explained by the fact that the exploration channel samples were of a higher quality (e.g., diamond saw cut), as opposed to the chip-based samples. In addition, the production samples are based on a different assay method and smaller assay sample size (Table 28).

4.7. Determining Sampling Variability

Duplicates Pair Analysis—Channel and Core Samples

Duplicate pair analysis was undertaken on data from different stages of the project life (Tables 31 and 32).

Table 31. Exploration face channel sample duplicate pair analysis.

Sample Type/Error (Preparation Route)	Sampling (%)	Preparation (%)	Analytical (%)	Total (%)
	Duplicate 4 kg face	**Split at −3 mm**	**[1] SFA500**	**-**
Exploration saw-cut channel	48	33	21	62
Number of pairs	50	50	50	-

Pairs cover range of mineralisation grades from low to high. Based on saw-cut face channel samples collected during the exploration/evaluation programme. [1] Protocol used SFA1000 on full coarse split; for analytical error, 1 kg pulp was riffle split for two SFA500 to investigate pulp variability.

Table 32. Production sidewall chip sample duplicate pair analysis.

Sample Type/Error (Preparation Route)	Sampling (%)	Preparation (%)	Analytical (%)	Total (%)
	Duplicate 5 kg Face	**No Split**	**Split at −75 µm LW500**	-
Production sidewall chip	84	-	25	88
Number of pairs	65	-	65	-

Pairs cover range of mineralisation grades from low to high.

The exploration channel samples provide the highest relative quality sample, with a sampling error component of 48% compared to 84% for the production chip samples. This is also borne out in the nugget effects of 69% and 93% respectively (Table 30). There is a high preparation error with the channel samples related to sample splitting (33% error), which is predicted by the FSE calculations (Table 29). The production chip samples are split at the pulp stage, therefore there is no preparation splitting error per se. The analytical error component for the channel and chip samples of 21% and 25% respectively indicates residual coarse gold in the pulps. At this stage the FSE is expected to be relatively low (Table 29), with the error dominantly related to splitting (e.g., GSE, DE and EE).

4.8. Discussion

Nalunaq was a relatively small high-grade, coarse-gold bearing narrow-vein operation. Sidewall-chip and face-channel and -chip samples were used for both publicly reported Mineral Resources and the internal stope estimates (Table 30). The mineralisation displayed a high level of variability through a 75% total nugget effect.

The mountainous topography of the mine area negated the use of extensive diamond drilling for resource estimation, generally only permitted drilling on a 80 m to 100 m grid. Therefore the focus during exploration/evaluation and production was sidewall chip and/or channel sampling. During exploration/evaluation effort was spent on cutting high quality channel samples with a diamond saw. These generally produced a higher quality sample, as evidenced by duplicate pair (sampling error component: 40%) and spatial analysis (nugget effect: 69%). During production reliance was placed on hammered sidewall-chip samples due to time constraints. These were of a poorer quality, as evidenced by duplicate pair (sampling error component: 85%) and spatial analysis (total nugget effect 93%). These samples suffered from high DE and EE, as well as WE due to their variable support per unit length.

During the exploration/evaluation period, an underground bulk sampling programme was undertaken to verify grades [13]. The mean bulk sample grade diluted to 1.5 m along the drives was 18.4 g/t Au (based on 1350 m of development and 11,000 t of bulk sample material), which was in close agreement with the 2002 resource that provides a diluted (1.5 m) mean grade of 19.5 g/t Au proximal to development. This estimate was based dominantly on saw-cut channel samples and indicates that they provided a reasonable representation of the mineralisation.

Nalunaq did not have a processing plant on site and shipped its ore in approx. 32,000 t parcels every three months to a plant in Spain and subsequently in Canada. Reconciliation stope-by-stope was impossible and related to groups of stopes and development feeding into a parcel. Three-month parcel reconciliation ranged between −42% to +2% for grade. The tonnage reconciliation was generally within ±5% of that expected. The likely cause for the grade under-call was initially believed to be related to: (1) a loss of gold within the stopes and elsewhere and (2) the nature of the estimation method and the high-nugget effect (i.e., poor local estimation). After further investigation it was found that the stope estimates were based on weighted averages of the bounding grade control samples, which provided a consistently high estimate. The overcall related to both the averaging technique and bias of chip samples. Gold loss in fines was shown to occur in stopes, confirmed by grab sampling of stope remnants which yielded high grades (up to 60 g/t Au). Table 33 provides a risk review of the production chip samples at Nalunaq. Overall the samples have a marginal fit-for-purpose rating, with the majority of error originating from the sample collection component.

Table 33. Risk review of the Nalunaq sampling programme used during production for resource definition.

	Key Parameter	Comment	[1] Component Error	TOS Error	[2] Error Rating
1	Spatial distribution and number of samples	Samples collected at approx. 3 m intervals along drives and raises Each stope block informed by around 15–20 samples	84%	GNE	Mod.
2	Sample mass (representa-tivity)	Each composite sample approx. 3–6 kg; total sample mass collected around a stope block approx. 85–110 kg Indicated optimum mass around 120 kg to 1 t to achieve 90% ±15%			Mod.
3	Collection and handling	Samples collected by hammer chipping All samples placed into plastic bags and sealed		DE, EE, WE	High
4	Transport and security	Samples delivered directly from the mine to site laboratory		-	Low
5	Preparation	Entire sample lot crushed and pulverised to P_{85} −75 µm, with 0.5 kg split for assay Error from pulp splitting Equipment cleaned between sample	25%	GSE, DE, EE, PE	Mod.
6	Assay	LW500 process undertaken correctly		-	Low
7	QAQC	Duplicates, CRM and blanks within expectation Written protocols for the sampling-assaying process	-	-	Low
8	Validation/variability indicators	Nugget effect 93% RSV 195% Process parcel (±32 kt) grade reconciliation generally under-called grade	Total 88%	GNE, GSE, DE, EE, WE	High
		Summary			
		Representativity (1)–(3)			High
		Preparation and assaying (4)–(7)			Low
		Fit-for-purpose			Marginal

[1] Sampling component errors from duplicate sample analysis; [2] Indicative total error rating; red: high (>±50%); orange: moderate (±25–50%); green: low (<±25%).

The sampling regime at Nalunaq is deemed to be marginal, bordering on being not fit-for-purpose given the associated resource classification of Indicated was inappropriate. An Inferred classification would have been more reasonable given the reconciliation issues. Improvements to mining practice related to stope cleaning and blast design, led to the reduction of "lost" gold. In addition, grade interpolation by kriging was optimised to blocks that better mirrored stope panels and thus improved stope grade estimation [34].

More effort could have been put into the sample collection stage via staff training and mentoring. Sampling error could be reduced by re-introducing saw-cut channel samples and whole sample assaying. Whilst advantageous, the time cost would have been relatively high, though not impossible given sampling of sidewalls behind the advancing face (e.g., saw cutting would not delay development). More drilling was the best option but would have required foot-wall cross-cuts to establish underground positions. A geostatistical conditional simulation study indicated that a 15 m by 15 m grid could have provided a monthly precision on the estimate of around ±25%, with a 10 m grid reducing this to ±15% [34]. Implementing an improved channel sample and assaying strategy would have been a cheaper option than underground drilling; no new grade control strategy was implemented prior to mine closure in 2009.

5. Overview

The case studies presented discuss three characteristic though contrasting types of gold ore (Table 34). Common features are that they are all coarse gold-bearing, with >50% gold above 100 µm in size. Ballarat provides for the coarsest gold up to 5 mm. San Christina is the most challenging ore type due to extensive gold particle clustering.

Table 34. Summary of key factors and performance of grade control samples for each case study.

Key Factors	Case Study Mine	#1 Ballarat	#2 San Christina	#3 Nalunaq
Project status		Operating	Operating	Evaluation
Mining	Method	Longhole	Shrinkage	Longhole
	Annual production	270,000 t	15,000 t	160,000 t
Geology	Host rocks	Black shales	Volcanics	Metavolcanics
	Mineralisation style	Vein and stockworks	Vein	Vein
	Dip	>45°	75–90°	35–50°
	Width	1–5 m to <20 m	0.5–1.5 m	0.2–1.5 m
Current resource base		415,000 at 10.2 g/t Au	55,000 at 9.5 g/t Au [16,500 at 25.7 g/t Au]	445,000 t at 18.7 g/t Au
		Inferred	Inferred/[Indicated]	Inferred
Ore/sampling characteristics	ROM (g/t Au)	6	23	20
	[1] BCOG (g/t Au)	4	6	8
	Coarse gold (at mine grade)	60% >100 µm 35% >1000 µm 20% >2000 µm	70% >100 µm 9% >1000 µm <1% >2000 µm	50% >100 µm 5% >1000 µm <1% >2000 µm
	Cluster effects	Minor	Major	Minor
	d_{95Au}	500–3000	1500	400
	d_{maxAu}	5000	2500	4000
	d_{Auclus}	2500–3000	5000–10,000	1000–2000
	Sampling constant (K)	15,000–60,000	28,000-81,000	7000–22,000
	Opt. field mass	350 kg–5 t	30 kg–35 t	120 kg–1 t
Sampling and assaying	Sample type	NQ whole core	Bulk	Chip
	Field sample mass	1.4–3.4 kg	30 t	3–6 kg
	Assay mass	2.3 kg	30 t	0.5 kg
	Assay method	LW2300	Pilot plant	LW500
TOS errors	Predicted FSE	±10%	±5%	±5%
	Other TOS errors	DE, EE, GSE	DE, EE	DE, EE, GSE
Nugget effect	Total nugget effect	65–85%	49%	75%
Sampling errors (duplicate pair analysis)	Sampling error	N.D.	40%	84%
	Prep/anal error	20%	18%	25%
	Total error	N.D.	44%	88%
Reconciliation	Annual	[3] ±45% [4] ±20%	±20%	±30%
	[2] Quarterly	[3] ±70% [4] ±45%	±20%	±40%
	Monthly	[3] ±90% [4] ±60%	±20%	N.D.
	Stope-by-stope	N.D.	±25%	N.D.
Risk rating	[1] Resource class	Inferred	Indicated	Indicated
	Representativity	Moderate	Moderate	High
	Preparation and assay	Low	Low	Low
	Fit-for-purpose	Yes	Yes	Marginal

[1] BCOG based on current operation or last period of operation. [2] Reconciliation expectation based on resource classification. [3] Values reported over a six year period (2012–2018), [4] The last four years (2014–2018) show better reconciliation; N.D.; not determined.

The sampling approaches across the three mines vary from traditional small volume drill core and face chip samples at Ballarat and Nalunaq, to large development bulk samples at San Christina. These are driven by necessity and practicality, where underground drill access at Ballarat is relatively easy compared to Nalunaq where it is more difficult. The San Christina bulk sampling is practical as the mine is small, allowing development rounds to be batched through a purposefully configured pilot and production plant.

For Ballarat and San Christina, the TOS errors are well-controlled and in particular the FSE is very low. At Ballarat, the core samples are likely too small to resolve the potentially high geological nugget effect. During preparation and assay there is potential for DE, EE and GSE during pulp splitting (preparation/analytical component 20%), though this considered acceptable.

At San Christina errors during sample collection principally relate to the gold-cluster enhanced geological nugget effect. During preparation and assay there is some potential for DE, EE and GSE during tails and pulp splitting (preparation/analytical component 20%), though this not considered to be problematic.

At Nalunaq, whist the FSE of the protocol is low, the critical issue relates to high DE, EE and WE during sample collection (sampling error component 84%) and potentially high DE, EE and GSE during the pulp splitting (preparation/analytical component 25%).

6. Conclusions

This contribution demonstrates that effective sampling is critical to grade control. Grade control is about adding value by delivering quality tonnes to the mill via the accurate definition of ore and waste. The magnitude of measurement error (e.g., the sum of the sampling, preparation and analytical relative errors) during grade control is a critical consideration, as it can undermine the quality of resource/reserve estimates and any decisions made thereon.

The case studies present an analysis which commences with evaluation of ore characteristics (e.g., $d_{95\mathrm{Au}}$), duplicate sample pairs (e.g., relative error determination), sampling protocols in the context of TOS, and programme performance via reconciliation (Table 35). A table-based method is presented to evaluate the fit-for-purpose nature of the programmes (e.g., Tables 14, 27 and 33). The approach presented is also applicable to the analysis of fine gold-dominated mineralisation and open pit grade control programmes.

Table 35. Stages in the design of a new grade control programme and for the review of an existing programme. Detail may differ depending upon circumstances.

Stage	New Programme	Existing Programme
Example	Case study 2: San Christina	Case Study 1 and 3: Ballarat and Nalunaq
1: Overview	Set programme goals and data quality objectives	Review programme goals and data quality objectives Review resource/reserve reconciliation
2: Characterise	Review existing characterisation data and determine grade-liberation diameter relationships and critical optimisation grade Plan and undertake addition testwork if required	Review existing characterisation data and determine grade-liberation diameter relationships and critical optimisation grade Plan and undertake addition testwork if required
3: Design or review	Apply Stage (2) data to design protocols, including TOS-FSE analysis Undertake duplicate pair analysis (if possible)	Apply Stage (2) data to review protocols, including TOS-FSE analysis Undertake duplicate pair analysis
4: Implement	Set-up systems and written codes of practice Training of mine geology and production staff	Alter or set-up systems and written codes of practice Training of mine geology and production staff
5: Monitor	On-going QAQC programme with timely review and action as required Annual internal and/or external peer review Review resource/reserve reconciliation Risk analysis	On-going QAQC programme with timely review and action as required Annual internal and/or external peer review Review resource/reserve reconciliation Risk analysis
6: Update	On-going training Revision of protocols if required, return to Stage 2 or 3 as required	On-going training Revision of protocols if required, return to Stage 2 or 3 as required

Case specific conclusions are:

- At Ballarat core drilling is undertaken to define resources ahead of mining in orebodies that display both geometric and grade variability. The coarse gold nature of the ore, drives the whole core and assay approach. The sampling and resource risk is recognised, where development and stoping commence on Inferred Mineral Resources.
- At San Christina, small-sample based assays understate grade in geometrically simple, coarse gold-dominated veins. Drilling with whole core sampling and assaying is applied to estimate Inferred Mineral Resources that are accepted to understate grade. Dominant gold particle clustering drives the application of bulk sampling. Development drives are sampled round by round and processed via an on-site plant. Upper and lower development drive grades are assigned to stope blocks and reported in the Indicated Mineral Resource category.
- At Nalunaq sidewall chip samples were used to define resources in geometrically-simple coarse gold-dominated vein. Chip sampling generally imparted a positive bias that contributed to poor reconciliation. Samples were used to inform a resource estimate, which was reported in the Indicated Mineral Resource category. Such a classification was inappropriate given poor reconciliation.

General conclusions include:

- A range of sampling methods are available for underground grade control, all of which require evaluation before routine application. The highest error is generally introduced during sample collection. A reduction in the need for chip or channel samples will only come from the use of more pre-development drilling at a spacing to allow local estimation.
- Application of TOS enables sampling programme design and practice to be optimised. All errors along the sampling value chain are additive and impart variability making local estimation less reliable. Estimation must take into account the sampling strategy, with sample quality reflected in the resource classification.
- There is a greater need towards the quantification of sampling errors to better communicate resource/reserve uncertainty and risk. Sampling relative error can be estimated using duplicate pairs. The error components reflect the ore characteristics, sample type, and collection, preparation and analysis. The COV calculated from paired data produces an estimate of sampling relative error that can be used as a basis for risk evaluation within the framework of the JORC or other international reporting codes.
- QA/QC cannot be divorced from the TOS and is a mandatory step for fit-for-purpose sample evaluation.

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Abbreviations

The following abbreviations are used in this manuscript:

AE	Analytical error
BCOG	Breakeven cut-off grade
CRM	Certified reference material
DE	Delimitation error
d_{95Au}/d_{Auclus}	Liberation diameter for sampling purposes, individual particle vs. clustered value
EE	Extraction error
FA	Fire assay (assay charge size 30 g; FA30)
FSE	Fundamental sampling error
GNE	Geological (or in-situ) nugget effect
GSE	Grouping and Segregation error
LM5	Ring pulverising unit with approx. 2.5 kg capacity
LW	LeachWELL assay (assay charge size 500 g; LW500)
P_{80} or P_{90}	Percent passing (e.g., P_{90}; 90% passing a given screen size)
PAL	Pulverise and leach (assay charge size 500 g; PAL500)
PE	Preparation error
ROM	Run of mine grade
RSD	Rotary sample divider
RSV	Relative sampling variability
SFA	Screen fire assay (assay charge size 500 g; SFA500)
TOS	Theory of Sampling
QA/QC	Quality assurance/quality control
WE	Weighting error

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