Report for Inventus Mining

Pardo Project - Geostatistical Review and Drillhole Spacing Analysis

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1 Introduction

Inventus Mining (Inventus) engaged Snowden Optiro to undertake a geostatistical review, sampling analysis and drillhole spacing analysis on its Pardo Gold Project, located in Ontario, Canada. Snowden Optiro has experience across similar conglomerate-style gold mineralisation in Western Australia (Beatons Creek and Comet Well-Purdy's) and South Africa (Witwatersrand). The task included a high-level review of sampling procedures and advice on appropriate sample sizes for the Pardo Project. This report provides a summary of the work undertaken, results, findings, and recommendations.

2 Geostatistical review

2.1 Data – drilling

The initial drill database provided did not match the reef wireframes. A second database was provided, which had collars pressed to topography and an updated reef wireframe (15 March 2023).

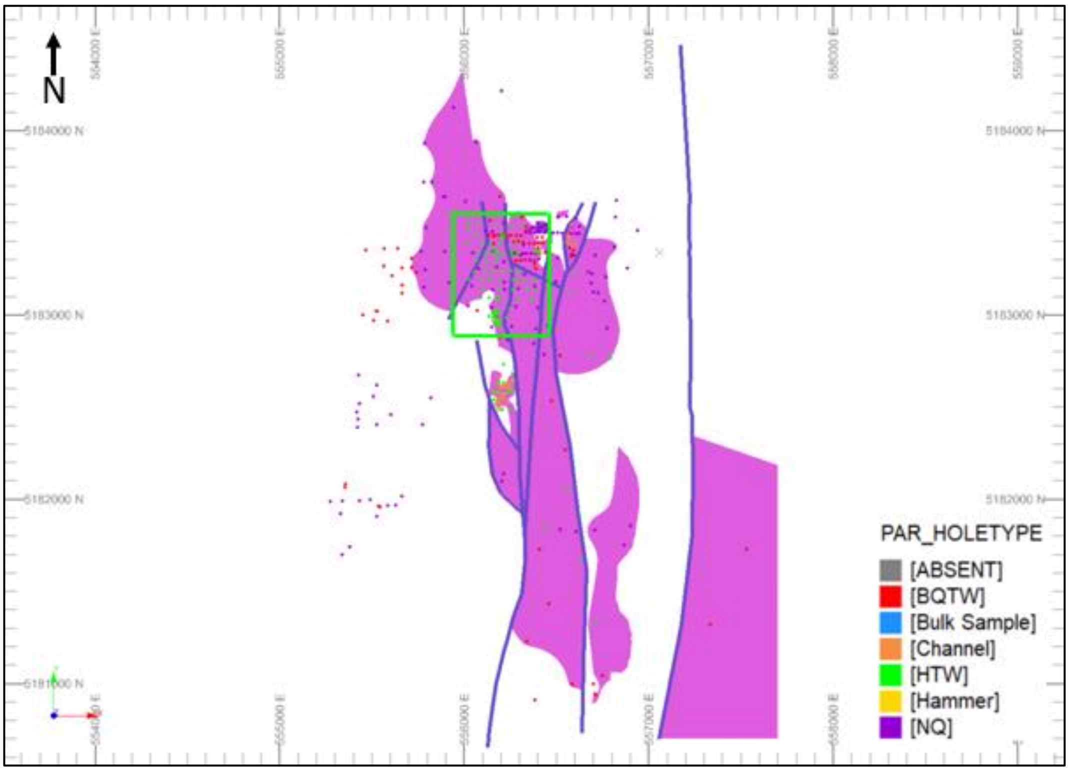
The drill database contained data for 459 drillholes in total, summarised by drillhole type in Table 2.1.

Table 2.1 Full drillhole database summary by hole type

Hole type	No. of holes
Channel	69
NQ	154
BQTW	83
Hammer	12
HTW	133
Bulk sample	8
Total	**459**

The data analysed as part of this review was restricted to an area 525 m x 660 m, shown by the green outline in Figure 2.1.

Figure 2.1 Plan view of Pardo data provided (green outline shows project study area)



Drilling and sampling data within the project area reviewed in this study is summarised by hole type and year in Table 2.2. Figure 2.2 and Figure 2.3 show the location by hole type and year, respectively.

Table 2.2 Drillholes within the project area summarised by hole type and year

Hole type	Year	No. of holes	Metres
BQTW	2012	35	503.62
	2014	6	52.81
Bulk sample	2009	2	2
	2021	6	6
Channel	2013	5	64
	2016	11	34.75
Hammer	2014	12	62.72
HTW	2015	5	93.3
	2017	71	1,314.26
	2018	34	166.28
NQ	2007	48	565.58
	2010	29	746.46
Total		**264**	**3,611.78**

Figure 2.2 Plan view of drillholes in study area coloured by hole type

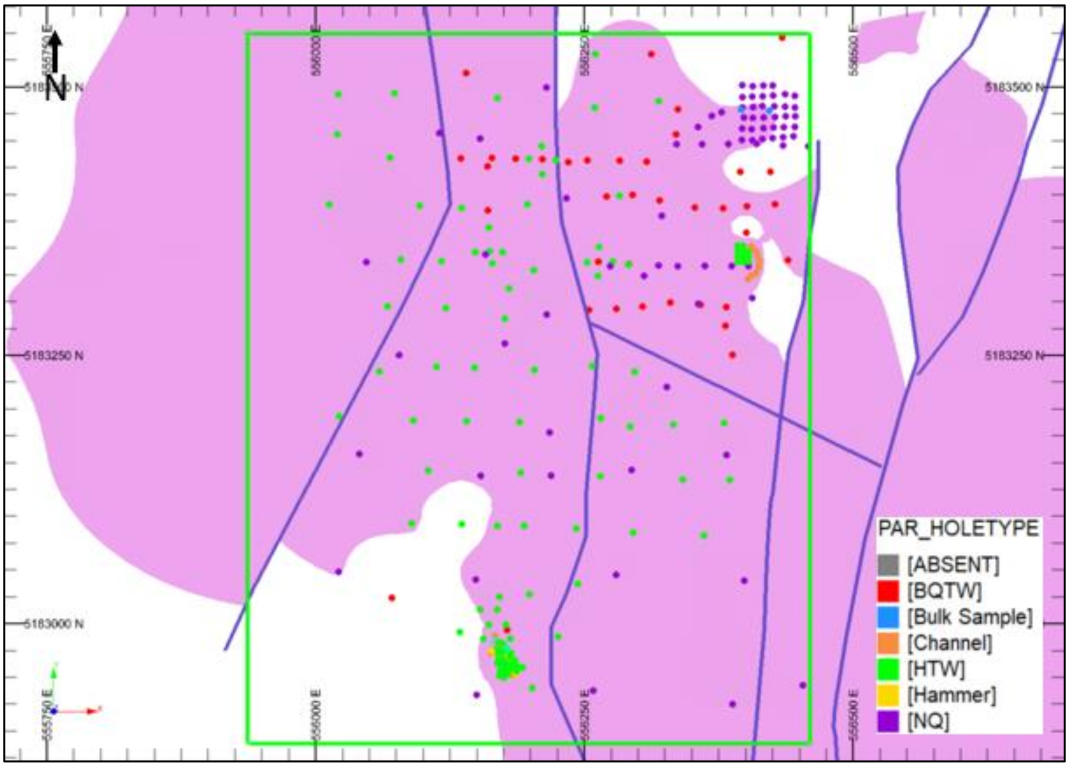
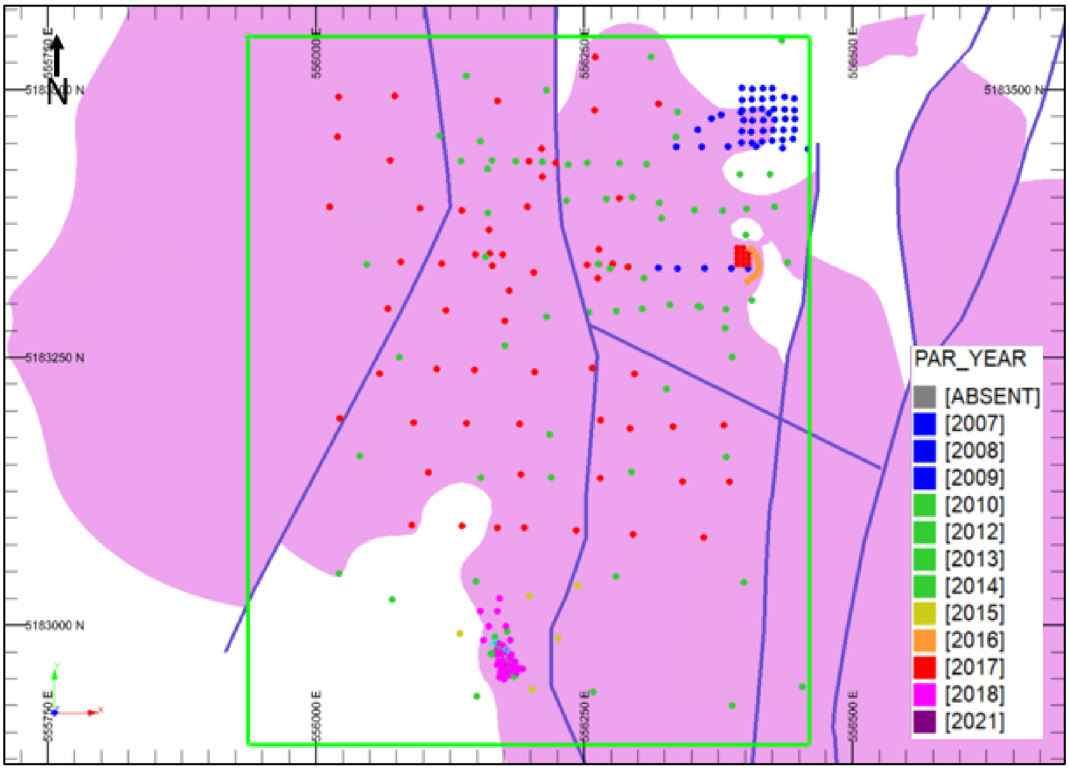


Figure 2.3 Plan view of drillholes in study area coloured by year



It was noted that pre-2017 drilling had been sampled across the lithological boundaries, in particular the A_M reef, which is not recommended as it is not clear where the gold is located.

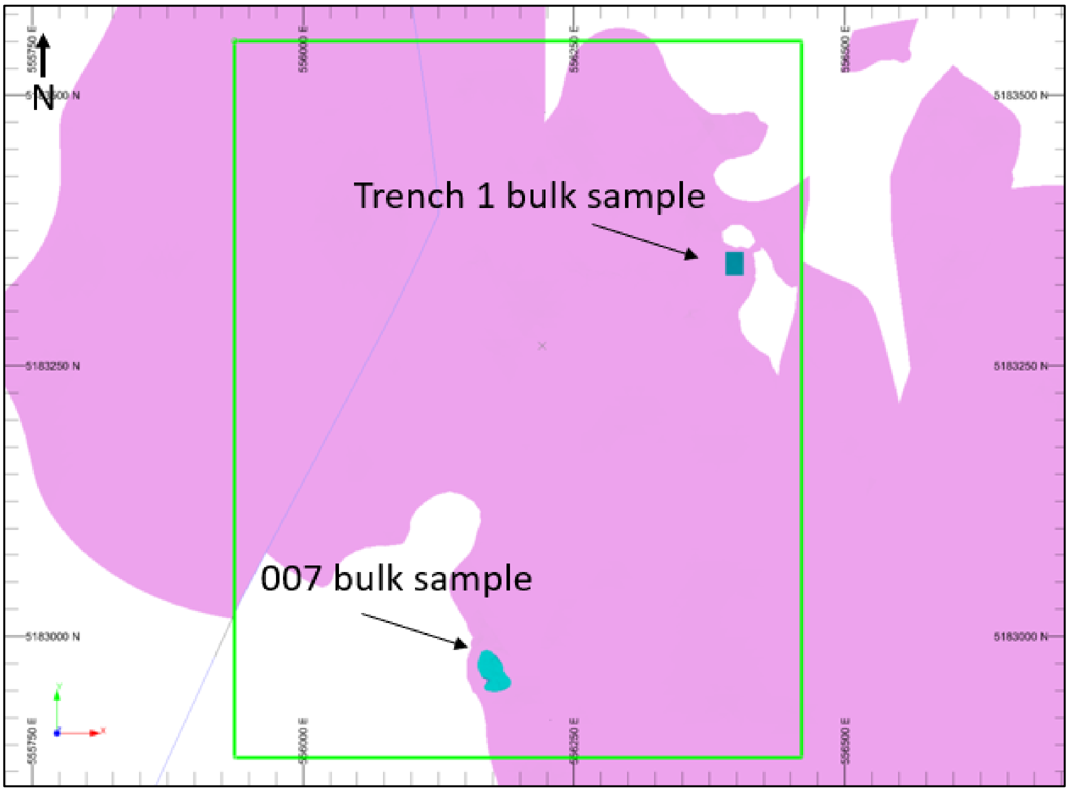
Snowden Optiro recommends using picked-up (surveyed) collars and modelling geology, mineralisation, and bulk samples within the same project to ensure consistency in future modelling.

2.2 Data – wireframes

In total, 102 wireframes were provided. These comprised lithological wireframes for 14 fault blocks, a topographic wireframe, bulk sample wireframes, faults, and the reef wireframe. The lithological wireframes were combined by lithology type. The reef wireframe is modelled on the lithological unit A_M (mineralised conglomerate) and is not based on grade. The reef wireframe is the basis for the study.

Bulk sample wireframes were provided for 007 and Trench 1 (Figure 2.4). Bulk sample 007 was modified to match the drill data (translated vertically by 1.3 m) for flagging purposes.

Figure 2.4 Plan view showing location of bulk sample wireframes, with reef shown in pink



The topography, lithology, reef, and bulk sample wireframes used for flagging are summarised in Table 2.3.

Table 2.3 Summary of wireframes used to code drill data

Wireframe name	Type	Code	Description
reef_model_a_m_v2	REEF	1	Reef
all_abs	LITH_WF	1	Combined Archaean basement
ALL_MaC	LITH_WF	2	Combined Matinenda Conglomerate
ALL_MaS	LITH_WF	3	Combined Matinenda Sandstone
ALL_MiBC	LITH_WF	4	Combined Mississagi Boulder Conglomerate
ALL_MiBS	LITH_WF	5	Combined Mississagi Lower Sandstone
ALL_MiPC	LITH_WF	6	Combined Mississagi Upper Conglomerate
ALL_MiPS	LITH_WF	7	Combined Mississagi Upper Sandstone
ALL_OVB	LITH_WF	8	Combined Overburden
ALL_UNKNOWN	LITH_WF	9	Combined Unknown lithologies
007_bulk_sample	BULK	2	007 bulk sample wireframe
trench1_bulk_sample	BULK	1	Trench 1 bulk sample wireframe
topography_	TOPO	1	Topographic surface

2.3 Raw data analysis

All data within the comparison area was coded using the lithological and reef wireframes in Datamine RM Pro. The raw data was imported into Supervisor for analysis.

The samples within the reef were reviewed by year and by hole type.

2.3.1 Reef by year

The coded reef samples were analysed by year; the results are displayed in a box-and-whisker plot (Figure 2.5) and the statistics presented in Table 2.4.

Figure 2.5 Box-and-whisker plot of gold grades by year for reef samples

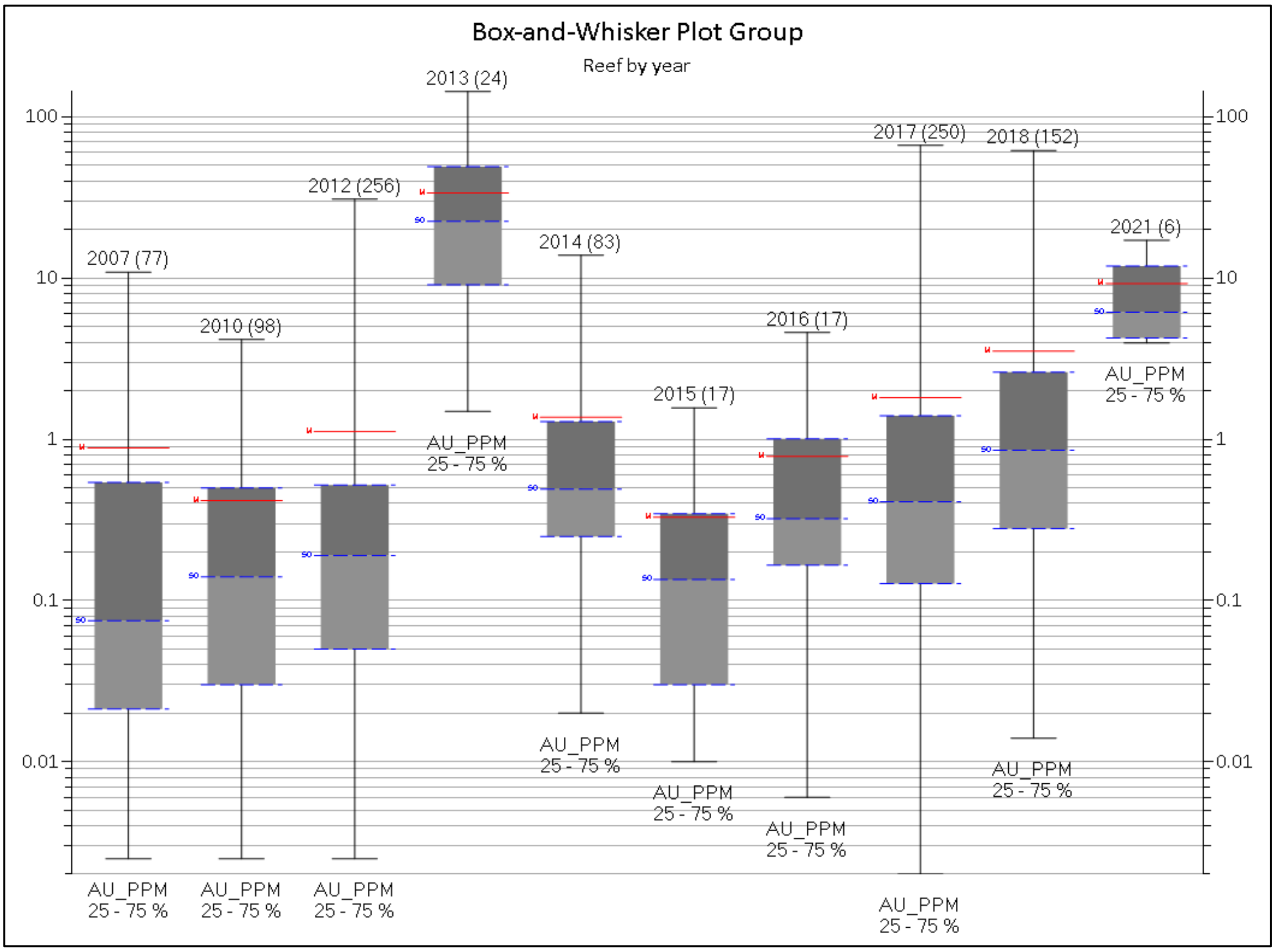


Table 2.4 Reef summary statistics for gold by year

Year	No. of samples	Mean grade (Au ppm)	Minimum grade (Au ppm)	Maximum grade (Au ppm)	Standard deviation	Coefficient of variation
2007	77	1.38	0.003	10.90	2.27	1.65
2010	98	0.51	0.003	4.16	0.77	1.51
2012	256	1.15	0.003	31.00	3.65	3.17
2013	24	33.78	1.490	144.00	31.74	0.94
2014	83	1.59	0.020	13.90	2.34	1.47
2015	17	0.38	0.010	1.57	0.48	1.24
2016	17	0.80	0.006	4.61	1.15	1.43
2017	250	1.82	0.002	66.78	5.58	3.07
2018	152	3.29	0.014	62.00	6.42	1.95
2021	6	9.27	3.960	17.20	4.79	0.52

The results show that there is a lot of variation in the gold grades by year (drill program). There appears to be a steady increase in grade between 2015 and 2021. Some years' drilling comprises multiple different hole types, and these have been analysed in the next section.

2.3.2 Reef by hole type

The coded reef samples were analysed by hole type; the results are presented in a box-and-whisker plot (Figure 2.6) and statistics have been tabulated in Table 2.5. Channel and Hammer sampling types would not be used in a resource estimate, and so have been excluded from Figure 2.6 for clarity.

Figure 2.6 **Box-and-whisker plot of gold grades by hole type for reef samples**

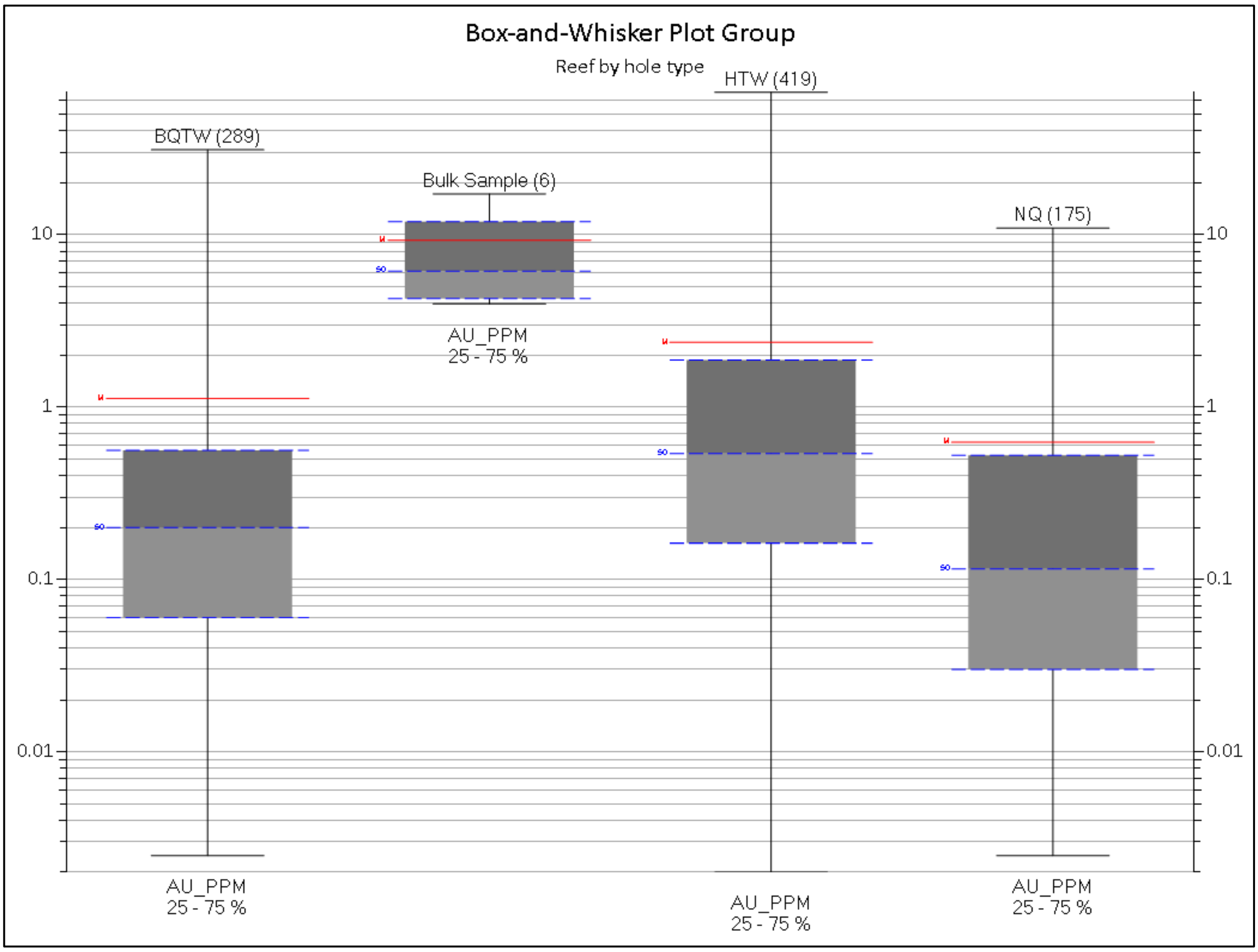


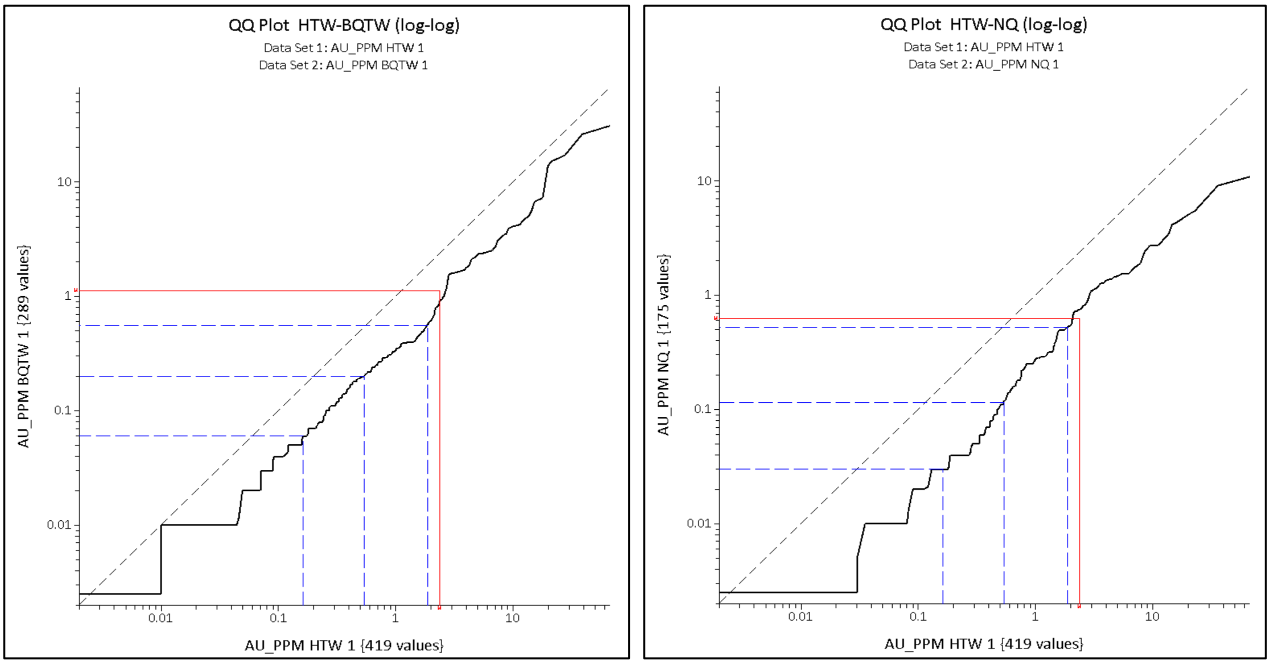
Table 2.5 **Reef summary statistics by hole type**

Hole type	No. of samples	Mean grade (Au ppm)	Minimum grade (Au ppm)	Maximum grade (Au ppm)	Standard deviation	Coefficient of variation
BQTW core	289	1.15	0.003	31.00	3.53	3.07
Bulk sample	6	9.27	3.960	17.20	4.79	0.52
HTW core	419	2.32	0.002	66.78	5.89	2.54
NQ core	175	0.95	0.003	10.90	1.76	1.85
Channel	41	21.36	0.006	144.00	29.73	1.39
Hammer	50	1.79	0.020	10.56	2.27	1.27

The results show that there appears to be a bias between the hole types. The bulk sample type comprises six samples and has the highest grade. The larger diameter core (HTW) has a higher mean grade than both the NQ and BQTW core samples. The full sampling protocols have not been reviewed and so there may be other factors influencing the higher grades through biases in the sampling methods or assaying.

The HTW hole type was compared to the other drill core methods (BQTW and NQ) and quantile-quantile (Q-Q) plots generated (Figure 2.7).

Figure 2.7 Q-Q plot showing HTW vs BQTW (left) and HTW vs NQ (right)



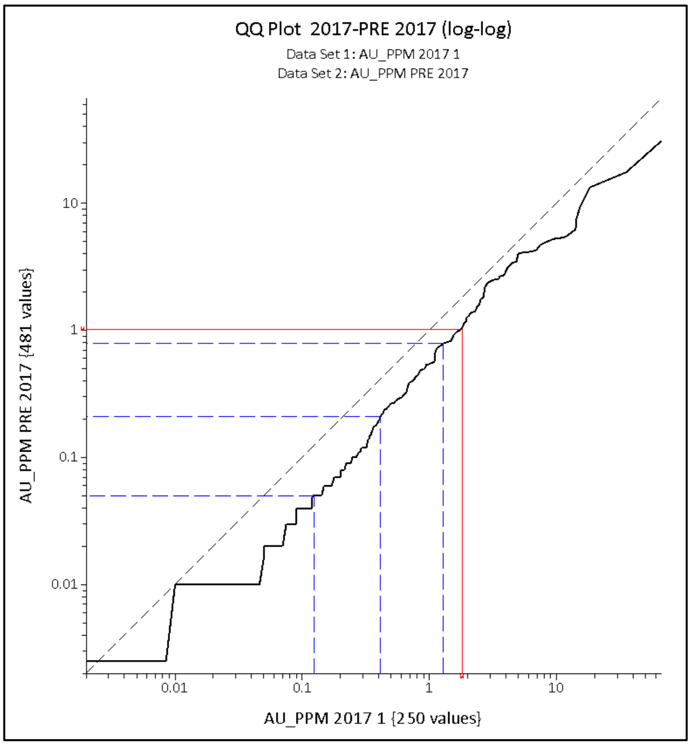
Both Q-Q plots show a constant bias towards the HTW core sampled holes. It is understood that the HTW holes have been sampled carefully, optimising the sampling, and ensuring that no lithological boundaries were crossed in the sampling process. The bias may be a function of volume (support) but needs to be investigated further.

2.3.3 Reef by drill core pre-2017 and 2017

Inventus indicated that a bias between holes was suspected for holes drilled prior to 2017 and the 2017 HTW drilling; furthermore that the older holes may not have been sampled optimally.

Snowden Optiro grouped all pre-2017 drill core reef samples and compared them to the 2017 drill core reef samples (Figure 2.8).

Figure 2.8 Q-Q plot showing drill core reef samples for 2017 vs pre-2017 drilling



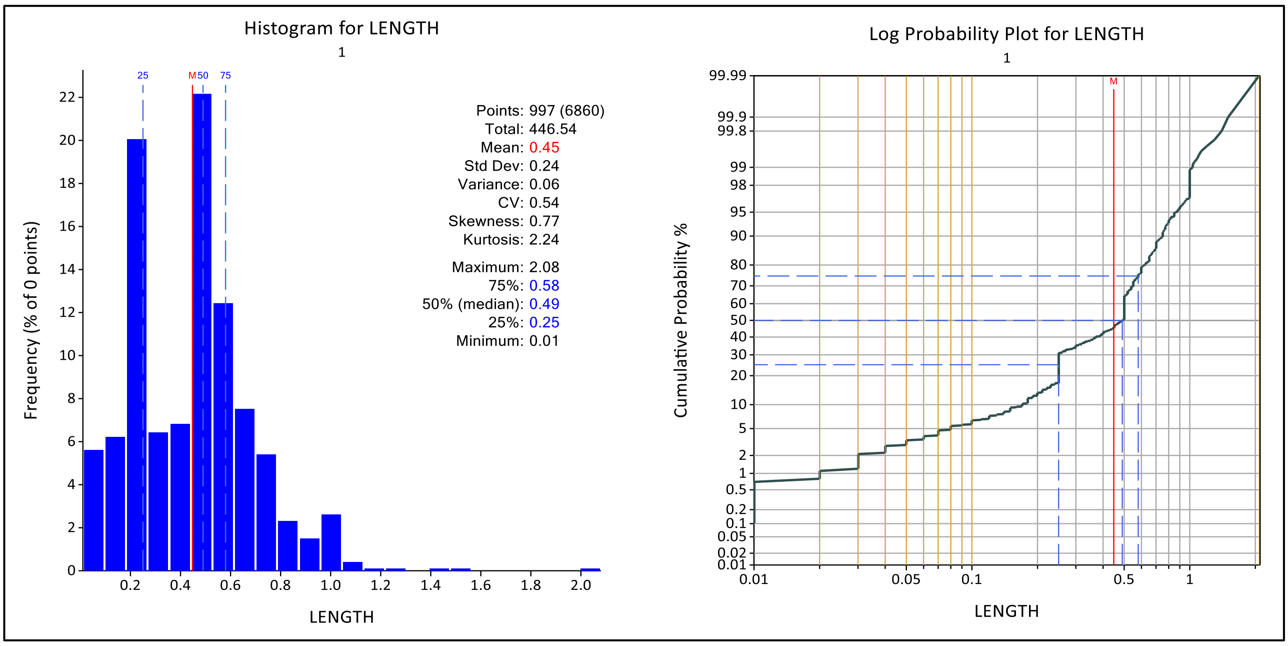
The Q-Q plot shows that there is a positive bias towards the 2017 drilling. It is understood that the 2017 HTW drilling and sampling procedure was the most carefully managed drill program, was believed to be the best and had the greatest sample mass due to the core diameter. Inventus indicated that holes drilled prior to 2017 would likely be superseded by the next drill program and therefore should be excluded from the Mineral Resource estimate.

Snowden Optiro cannot pinpoint the reason for the bias (there are many possible factors), however, endorses the approach of using larger core with careful sampling and sample preparation.

2.3.4 Length

A review of the reef drilling sample lengths showed the sampling to have variable sample lengths; however, a composite length of 1 m was selected as 99% of the data is at or below 1 m (Figure 2.9).

Figure 2.9 Histogram and distribution of sample lengths



Samples were composited within the reef to 1 m lengths, using downhole compositing with the parameter MODE = 1 set in Studio RM Pro to ensure there were no residuals.

2.4 Composite analysis

The 1 m composites were imported into Supervisor for analysis. Given the bias noted in Section 2.3, only the 2017 and later data was used for the analysis. Composite statistics are presented in Table 2.6.

Table 2.6 Reef composite statistics for gold

Hole type	Year	No. of composites	Mean grade (Au ppm)	Minimum grade (Au ppm)	Maximum grade (Au ppm)	Standard deviation	Coefficient of variation
HTW	2017	144	1.82	0.01	36.75	3.78	2.08
	2018	81	3.28	0.03	23.58	4.23	1.29
	2017 + 2018	225	2.34	0.01	36.75	4.01	1.71
BULK	2021	6	9.27	3.96	17.20	4.79	0.52

The 2018 composites have a higher mean grade than the 2017 composites. The 2018 drilling was almost entirely within the 007 bulk sample and are representing a local high grade reef concentration. Given the potential bias, variography was undertaken on the 2017 drilling separately to the 2018 drilling.

2.5 Variographic analysis

Normal scores variography was undertaken on the 2017 and the 2018 composite data.

2.5.1 2017 – HTW drillholes across project area

The 2017 data is distributed across the area of comparison at variable spacing, generally 50 m or greater. The continuity orientations selected are presented in Figure 2.10 (plan at top and dip plane at bottom), with the modelled variograms and back-transformed variograms displayed in Figure 2.11 and Figure 2.12, respectively.

Figure 2.10 Continuity analysis for 2017 reef data

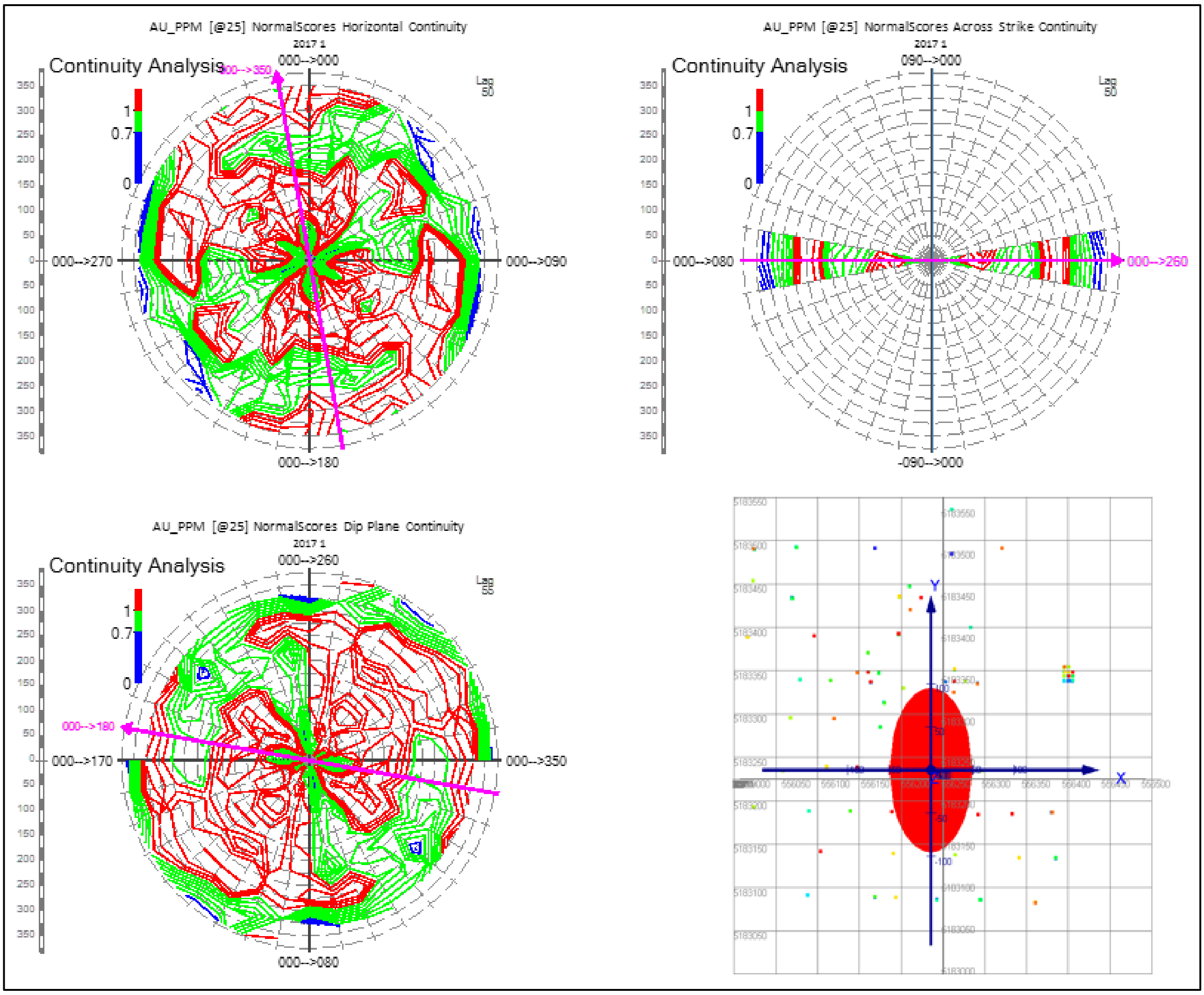


Figure 2.11 Variography on reef – 2017 data

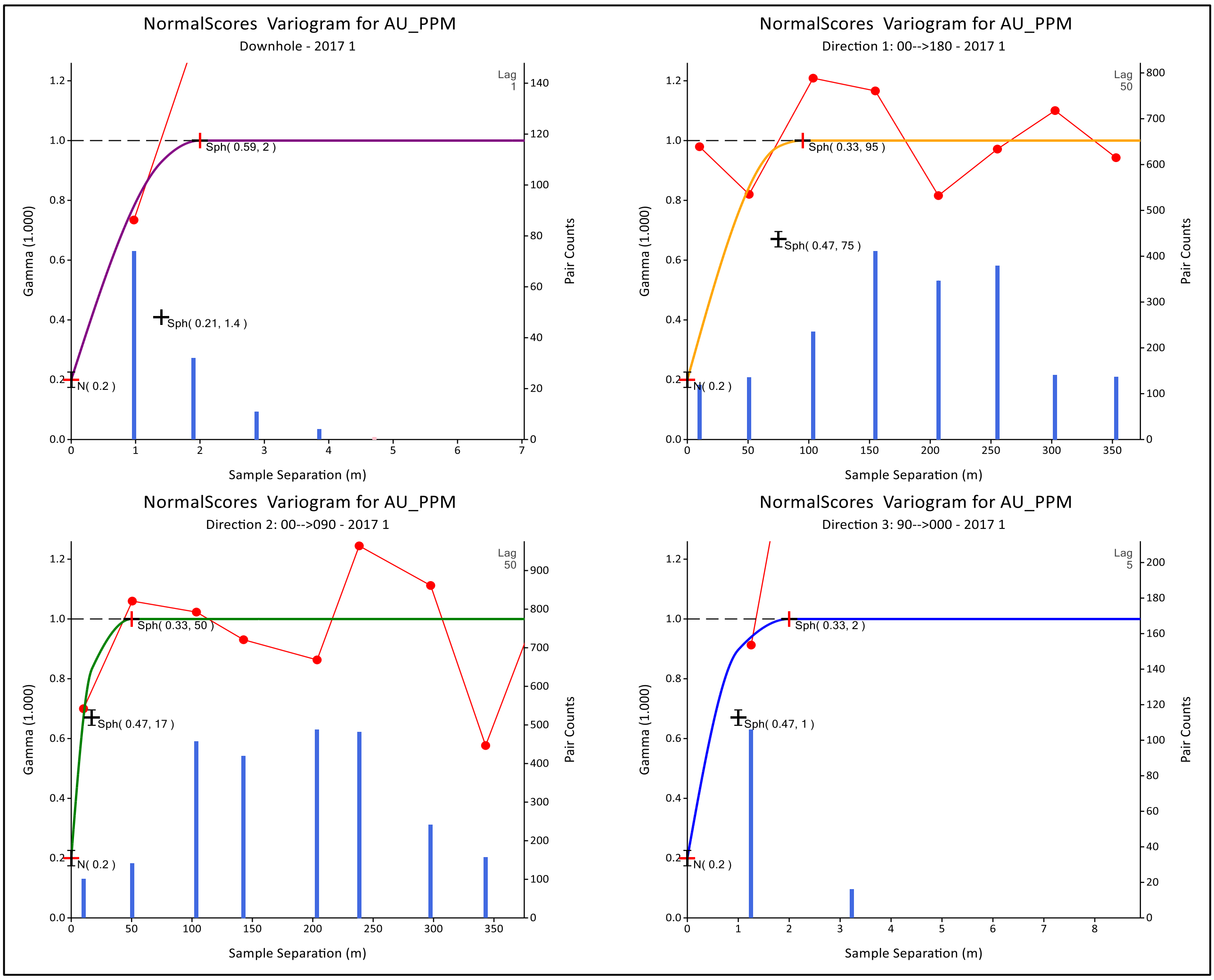
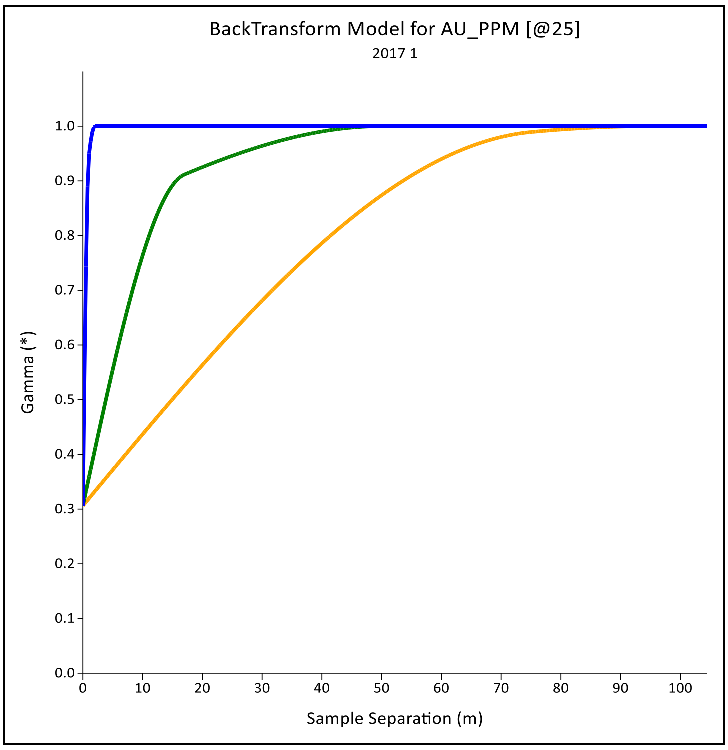


Figure 2.12 Back-transformed variography – 2017 data



The downhole variography indicated a low nugget, which back transformed to 30% of the sill. The major range is around 95 m directly north-south, with the semi-major range at 50 m (east-west).

2.5.2 2018 – 007 bulk sample area drillholes

The 2018 data is on a close-spaced grid approximately 5 m x 5 m (within the bulk sample 007 area). The purpose of reviewing the variography was to review the orientations, the nugget, and structures of the close spaced data. As with the wide-spaced drilling the continuity analysis is depicted in Figure 2.13, the resultant modelled variograms in Figure 2.14 and the back-transformed variogram summary in Figure 2.15.

Figure 2.13 007 continuity analysis

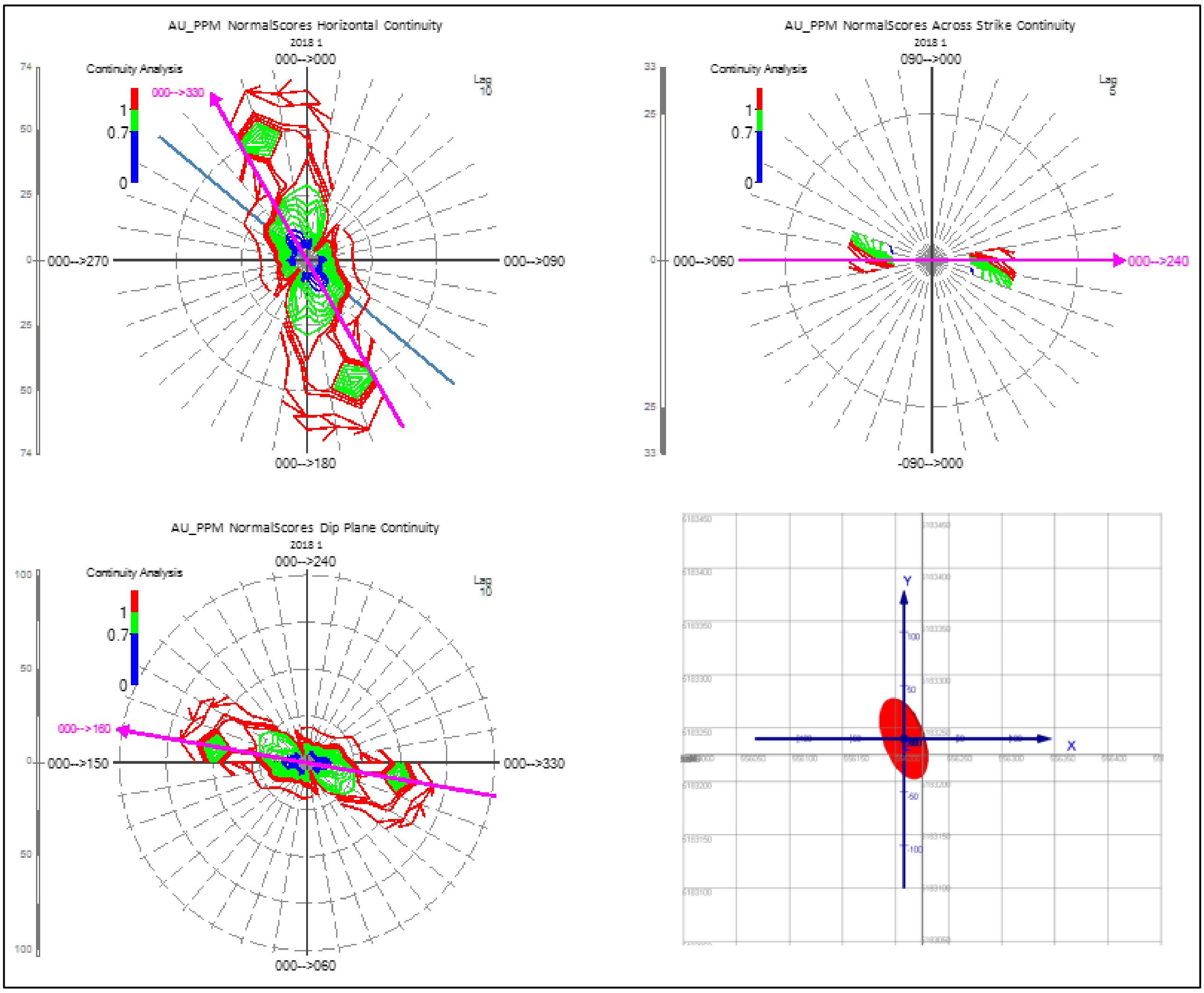


Figure 2.14 Variography on reef – 2018 close spaced data

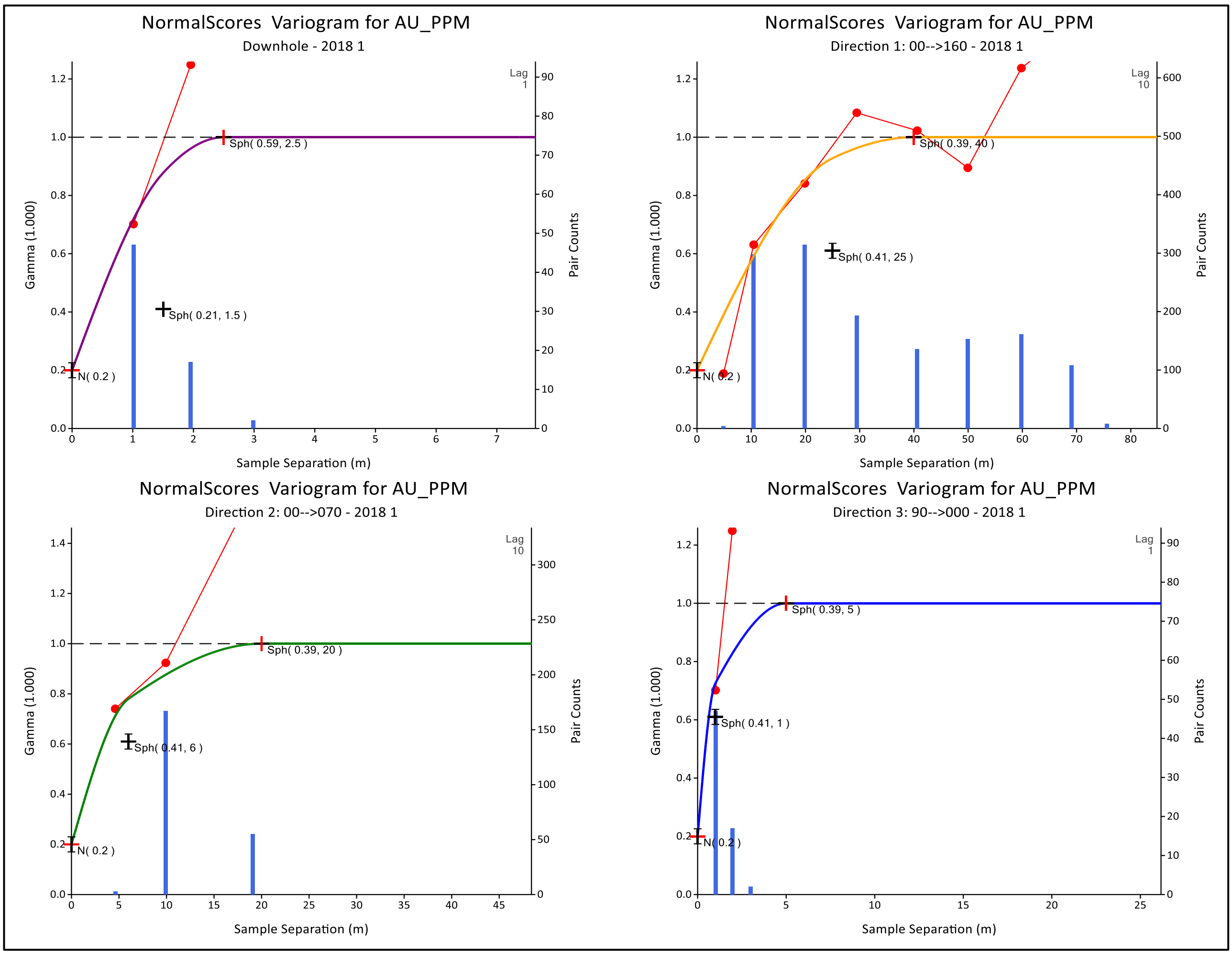
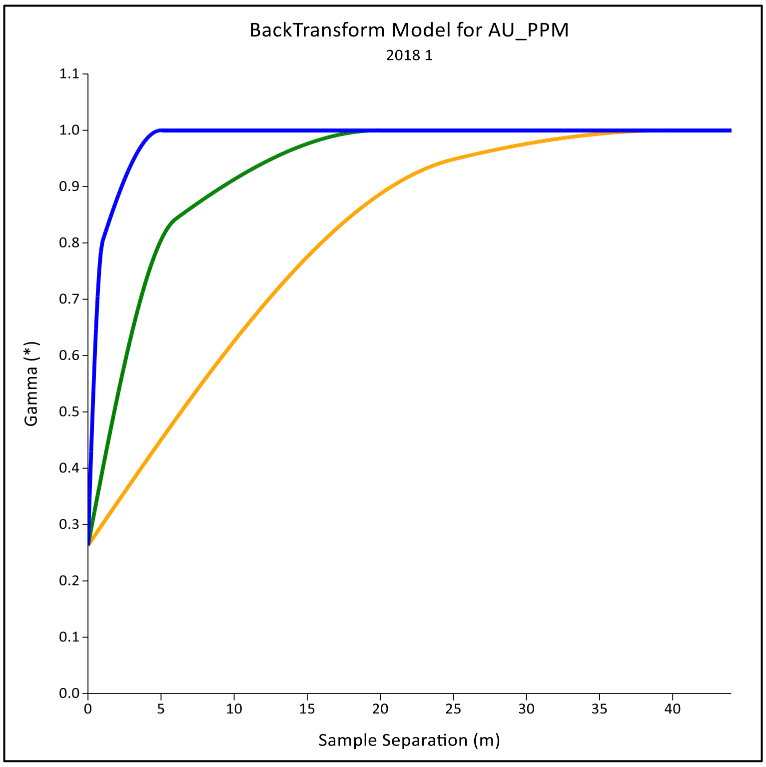


Figure 2.15 Back-transformed variography – 2018 data



The downhole variography again indicated a low nugget, which back transformed to just below 30%. There is still a north-south trend but the orientation for this data was more to the north-northwest to south-southeast. The ranges were shorter, however, this is most likely due to the data being restricted to the bulk sample area (40 m x 20 m). The 007 data confirms the relatively low nugget derived from the wide-spaced data.

3 Sampling analysis

3.1 Gold particle size

A key input into sampling optimisation calculations is the gold particle size, known as the "liberation diameter". Not to be confused with any metallurgical parameter, the liberation diameter – $d_{\ell Au}$ – is effectively the screen size that retains 5% of gold given a theoretical lot (or sample volume) of liberated gold. The $d_{\ell Au}$ value may vary across different mineralised domains and with grade. If gold particle clustering is observed, the combined clustered-particle liberation diameter ($d_{\ell clus}$) needs to be defined (Dominy and Platten, 2007). The $d_{\ell clus}$ is effectively a maximum or coarsest gold particle size present. Unless the given domain contains large masses of gold, it is unlikely that an individual gold particle size would be greater than $d_{\ell clus}$. Parameters $d_{\ell Au}$ and $d_{\ell clus}$ vary depending upon the comminution state of the lot: as a sample is crushed or ground, the clusters progressively break down into single gold particles. Cluster size and/or abundance may vary across different mineralised domains.

Inventus has not undertaken any targeted evaluation of gold particle sizing, beyond a limited XCT study of gold in core/rock samples (Whymark, 2023). This study of material from conglomerate mineralisation in the basal Mississagi Formation identified clusters of gold particles with a range of sizes. The composite cluster volumes ranged from 0.02 mm^3 to 236 mm^3 (N = 13) with maximum cluster length dimensions ranging from 200 µm to 10,500 µm. The volumes provide equivalent spherical diameters (ESDs) ranging from 300 µm to 7,700 µm. The mean ESD is 2,000 µm, reducing to 1,500 µm if the maximum value of 7,700 µm is removed. A lower ESD of 650 µm is achieved based on the lower values (N = 7; <3,000 µm).

The largest cluster from Sample Mi_01 was cut by drilling, so was larger in situ. If an assumption is made that one-third of the cluster was lost, the original volume could have been ca. 354 mm^3, giving an ESD of ca. 8,800 µm.

The composite cluster volumes from 0.02 mm^3 to 354 mm^3 provide cluster masses of <0.1 g to 4.0 g, assuming that the clusters are 60% gold.

Previous work reported in Tilsley (2014) notes that the maximum observed gold particle sizing was 800 µm, with most gold being <180 µm. Tilsley (op. cit.) noted the presence of gold particle clustering but provided no further information.

Given the relative lack of gold particle size data, Snowden Optiro has opted to undertake calculations at selected grades (COG, ROM, HROM and VHG) with a range of $d_{\ell Au}$ and $d_{\ell clus}$ values used for each. The grade categories applied were:

- Nominal cut-off (COG): 0.5 g/t Au
- Nominal run of mine (ROM): 2.0 g/t Au
- High run of mine (HROM): 3.5 g/t Au
- Very high (VHG): 10.0 g/t Au.

Sampling calculations undertaken were:

- Representative in-situ sample mass (RSM)
- Fundamental sampling error (FSE) of sample-assay protocols.

3.2 Representative sample mass and sample protocol optimisation

3.2.1 Representative in situ sample mass

Background

Optimum primary sample masses can be estimated using Poisson statistics. A range of grade-particle size scenarios were used to estimate the RSM to achieve a ±20% precision at 90% confidence limits. Poisson distributions are characterised by their high probability of drawing zeros, so if the sampling distribution of the coarse gold particles is Poisson, many of the samples will contain no gold. Taking large numbers of samples will rectify this, with the result approaching the mean grade. Samples of too small a volume have the same effect as too few samples. The degree of under- or over-valuation of grade is conditioned by the way in which the gold particles are partitioned between the fine-background gold vs the coarse-high grade population.

Poisson probability theory can adequately model the frequency distribution of the rare grains (nuggets) in idealised samples. As a result, the probabilities of obtaining a specific number of grains in a sample of a geological material containing a given number of nuggets can be determined. Although samples under consideration do not exhibit these ideal equant grain model characteristics, this model may be used to determine the magnitude of the nugget effect in real samples if only a few modifications to theory are considered.

The Poisson approach is not a panacea and has limitations in the interpretation of the RSM value. The value is dependent upon inputs based on the nature of the mineralisation (e.g. the mean grade and d_L) and the number of particles deemed to be required in the sample to be significant. The definition of the liberation diameter is difficult, due to the complex relationships that exist between gold particles and grade. The method is based on an ideal statistical distribution and does not account for geological complexities, such as spatial distribution and natural segregation (e.g. clustering) at different scales.

RSM calculations

RSM calculations were undertaken based on given grade-gold particle size scenarios (Table 3.1). RSM values have been tabulated for a precision of ±20% at the 1.6 and 1 standard deviation (confidence limits or CLs). Based on Poisson theory, the number of gold particles captured in the masses at the given CLs is 64 and 25.

Table 3.1 RSM calculations across various grade-gold particle size scenarios; RSM values provided at the 90% and 68% confidence limits

Grade scenarios		Microns (dl ESD)	Particle type (Single / Clustered)	90% CL mass	68% CL mass
COG	0.5 g/t Au	50	S	<5 kg	<5 kg
		100	S	<5 kg	<5 kg
		200	S	10 kg	5 kg
		500	C	150 kg	60 kg
		1,000	C	1 t	470 kg
ROM	2 g/t Au	50	S	<5 kg	<5 kg
		100	S	<5 kg	<5 kg
		200	S	<5 kg	<5 kg
		1,000	S / C	300 kg	100 kg
		2,500	S / C	5 t	2 t
		5,000	C	38 t	15 t
		7,500	C	125 t	50 t
HROM	3.5 g/t Au	50	S	<5 kg	<5 kg
		100	S	<5 kg	<5 kg
		200	S	<5 kg	<5 kg
		1,000	S / C	175 kg	70 kg

Grade scenarios		Microns (dl ESD)	Particle type (Single / Clustered)	90% CL mass	68% CL mass
		2,500	S / C	3 t	1 t
		5,000	C	22 t	8 t
		7,500	C	70 t	30 t
VHG	10 g/t Au	50	S	<5 kg	<5 kg
		100	S	<5 kg	<5 kg
		200	S	<5 kg	<5 kg
		1,000	S / C	60 kg	25 kg
		2,500	S / C	1 t	400 kg
		5,000	C	8 t	3 t
		7,500	C	25 t	10 t

It can be seen in Table 3.1 that the clustered scenarios have a significant impact on the RSM value, pushing it to the tonnes scale of mass. These figures are **maximum** theoretical values, given that the average number of grains in the samples is assumed to be all at the input value size (e.g. d_L). There will naturally be a range in particle sizes and not several equally sized particles.

A view of representativity can also be taken from the mass of samples included in the search neighbourhood during estimation. In general terms, between 10 (minimum) to 25 (maximum) samples will be used based on 1 m composites (requires assessment by the Mineral Resource estimate Qualified Person, assumed for illustrative purposes only here). If it is assumed that 1 m of HTW core has a mass of 11 kg (assuming 100% recovery), then 110 kg to 275 kg of sample informs the estimate (per discretisation point). RSM values of <275 kg is flagged green in Table 3.1. It is dominantly the clustered scenarios that fail this test. The distribution of clusters is unknown, so their effect cannot be assumed to be present all the time. It is also worth noting that gold particle populations can usually be partitioned into size ranges with given grades, each requiring an RSM value.

If PQ core were to be used, 1 m of whole core has a mass of 15.9 kg (assuming 100% recovery), then 159 kg to 398 kg of sample informs the estimate (per discretisation point). Again, it is dominantly the clustered scenarios that fail this test.

The key conclusion here is that for cluster-dominated mineralisation, core drilling will always understate the grade. The limitation of this finding is that we do not understand the distribution or probability of cluster occurrences. Using whole core samples is preferred and will be discussed further in the next section.

If further bulk sampling is undertaken, then each bulk sample should be no less than 200 t. This is based on the maximum required mass for ±20% precision at the 90% CL (see Table 2.1) and allowing for a 50% factor of safety.

3.2.2 Sample protocol optimisation

Sample protocol

The 2017 and 2018 sampling protocol is based on whole core (HTW) samples, crushed to P_{75} -2 mm. Thereafter 1 kg was split for pulverisation to P_{85} -150 µm. Two 30 g pulp splits were taken for fire assay (FA). The final assay is based on the mean of the two FAs.

Duplicate pulp (FA) analysis

The precision of the FA30 + FA30 (FA30 = 30 g FA) pairs and, where available, the original 2x FA30 versus duplicate 2FA30 was calculated for the 2017 and 2018 data. To ensure the largest dataset, both datasets were combined.

The FA30 vs FA30 precision is ±26% based on N = 325 analysed pairs. All data <0.1 g/t Au (10x DL) was filtered out.

The 2FA30 vs 2FA30 precision is ±22% based on N = 76 analysed pairs. All data <0.1 g/t Au (10x DL) was filtered out.

The most relevant figure is the 2FA30 vs 2FA30 duplicate precision of ±22%. For pulp duplicates this is high, where better than ±10% or ±15% (if some coarse gold was present) would be expected. The conclusion here is that the final 2FA30 assay is not optimal.

In addition, if the preceding protocol actions were not optimal then the pulp precision is understated. The splitting of pulps containing liberated gold is always high risk and will generate a high grouping and segregation error (GSE) (Minnitt, Esbensen and Dominy, 2022).

FSE equation

The FSE equation can be used to calculate what subsample size should be used to obtain a specified variance at a given reliability. The FSE is one of several sampling errors that are defined in the Theory of Sampling (TOS).

The FSE is dependent upon the Constitution Heterogeneity, which relates to sample weight, mineral fragment size and shape, liberation stage of the gold, gold grade, and gold and gangue density. It is the smallest residual error that can be achieved even after homogenisation of a sample lot is attempted. When FSE is not optimised for each subsampling stage, it becomes a major component of the Sampling Nugget Effect or SNE (Dominy, 2014, 2016).

The FSE can be theoretically estimated before the material is sampled, provided that the sampling characteristics embedded in the FSE equation are determined or assumed (Gy, 1982; Pitard, 2019). The “FSE equation” can be used to optimise sampling protocols (Gy, 1982; Pitard, 2019), where it addresses key questions for the sampling of broken rock:

- What weight of sample should be taken from a larger mass of mineralisation so that the FSE will not exceed a specified variance?
- What is the possible FSE when a sample of a given weight is obtained from a larger lot?
- Before a sample of given weight is drawn from a larger lot, what is the degree of crushing or grinding required to lower the error to a specified FSE?.

The FSE can be modelled before material is sampled, provided certain characteristics are determined, specifically grade, liberation diameter, and nominal material size. The François-Bongarçon modified FSE equation is given as follows (François-Bongarçon, 1998):

- $FSE_{(\text{rel var})} = f\ g\ c\ (d_L)^b\ d_n^{\alpha}\ (1/M_S - 1/M_L)$.

where f = shape factor; g = granulometric factor; d_n = nominal material size (95% passing 5% retained value); c = mineralogical factor; d_L = liberation diameter; b = (3 − α), where α is determined experimentally from duplicate series analysis tests or a default value of α = 1 to 1.5 is applied; M_S = sample mass; and M_L = lot mass.

Protocol FSE analysis

The François-Bongarçon modified FSE equation has been applied to the Inventus 2017–2018 sampling protocol. As no calibration work has been undertaken, the grade scenarios applied in the RSM calculations were used. Similarly, the sample gold particle size (e.g. d_L) size ranges were also used. Note that as the whole sample was crushed to P_{75} -2 mm, most clusters would have been broken down, or would be at 1,000 µm or less.

Table 3.2 shows the FSE outputs for all scenarios, referred to as the Stage 1, Stage 2 and Total FSE columns, along with the estimated Quality Fluctuation Error (e.QFE, see below) which is an estimate of the sum of the FSE and GSE. The general recommendation is that the total FSE for a protocol should be less than ±20% (Pitard, 2013). For a coarse gold mineralisation, better than ±30% is reasonable.

Table 3.2 FSE and e.QFE values for the 2017–2018 sampling protocol across various grade-liberation diameter scenarios

Grade scenarios		Microns (dl ESD)	Stage 1 FSE	Stage 2 FSE	Total FSE	Stage 1 e.QFE	Stage 2 e.QFE	Total e.QFE
COG	0.5 g/t Au	50	10%	8%	**13%**	14%	11%	**18%**
		100	21%	15%	**25%**	30%	21%	**36%**
		200	41%	22%	**47%**	58%	31%	**66%**
		500	104%	22%	**106%**	147%	31%	**150%**
		750	155%	22%	**157%**	219%	31%	**221%**
		1,000	207%	22%	**208%**	293%	31%	**294%**
ROM	2 g/t Au	50	5%	4%	**6%**	7%	6%	**9%**
		100	10%	7%	**13%**	14%	10%	**17%**
		200	21%	7%	**22%**	30%	10%	**31%**
		500	52%	11%	**53%**	74%	16%	**75%**
		750	78%	11%	**78%**	110%	16%	**111%**
		1,000	104%	11%	**104%**	147%	16%	**148%**
HROM	3.5 g/t Au	50	4%	3%	**5%**	6%	4%	**7%**
		100	8%	6%	**10%**	11%	8%	**14%**
		200	16%	6%	**17%**	23%	8%	**24%**
		500	39%	6%	**40%**	55%	8%	**56%**
		750	59%	8%	**59%**	83%	11%	**84%**
		1,000	78%	11%	**79%**	110%	16%	**111%**
VHG	10 g/t Au	50	2%	2%	**3%**	3%	3%	**4%**
		100	5%	3%	**6%**	7%	4%	**8%**
		200	9%	3%	**10%**	13%	4%	**13%**
		500	23%	5%	**24%**	33%	7%	**33%**
		750	35%	5%	**35%**	49%	7%	**50%**
		1,000	46%	7%	**47%**	65%	10%	**66%**

The dominant error comes from Stage 1 of the protocol, where 1 kg is split from 11 kg at a size of P_{75} -2 mm. The actual crush product value required for the FSE calculation is P_{95}, which has been assigned at -2.5 mm in this case. Most FSE values for Stage 1 fail, being >30%. The Stage 2 (2 x 30 g from 1 kg of pulp) values are acceptable from an FSE perspective.

A second effect is that of the GSE which is present at both stages, but which becomes material during Stage 2. In Stage 2, 60 g is split from 1 kg of pulp. This pulp will contain liberated gold, which cannot be homogenised. Therefore, any splitting method will give rise to some GSE. General understanding in the TOS community is that the GSE can easily be equal to the FSE. Similarly, the GSE is generally a transient error dependent upon material characteristics, comminution state (d_N and d_L), and handling. Note that where delayed comminution is marked, with higher grades and coarser coarse gold, the FSE will enhance and the GSE will be higher (Dominy, 2016, 2017; Minnitt, Esbensen and Dominy, 2022).

The GSE can be further promoted by bad sampling practice (e.g. through mat rolling or scooping).

The sum of the FSE and GSE is the Quality Fluctuation Error (QFE1). Table 3.3 shows an estimate of the QFE (e.QFE; estimated QFE) for Stage 1, Stage 2 and the Total FSE. In this case, the GSE has been set to equal the FSE according to convention.

Substantive delayed comminution, that is, splitting substantially before pulverising (assuming a grade of 0.5 g/t Au and liberation diameter of 1,000 µm) could increase the Total e.QFE from ±294% to ±360%.

Given the high FSE and e.QFE in the 2017–2018 protocol, revised protocols are recommended. These are based on HTW whole core samples, with a mass of 11 kg, assuming 100% recovery over 1 m.

- **Protocol 1:** Take 1 m composite at 11 kg whole core, crush to P_{90} -2 mm and split off 2 kg for assay via Photon Assay (PA) (ca. four PA jars assuming 500 g per jar).

- **Protocol 2:** Take 0.5 m composite at 5.5 kg whole core, crush to P_{90} -2 mm and split off 2 kg for assay via PA (ca. four PA jars assuming 500 g per jar).
- **Protocol 3**: Effectively the compositing of Protocol 2, thus 11 kg is assayed via two sets of 2 kg, thus 11 kg to 4 kg split.

Table 3.3 FSE and e.QFE values for the suggested sampling protocols across various grade-liberation diameter scenarios; the total e.QFE of the original 2017–2018 protocol is provided for comparison

Grade scenarios		Microns (dl ESD)	Protocol 1 11–2 kg at 2.5 mm	Protocol 2 5.5–2.0 kg at 2.5 mm	Protocol 3 11–4 kg at 2.5 mm	Protocol 1 e.QFE	Protocol 2 e.QFE	Protocol 3 e.QFE	Original total e.QFE
COG	0.5 g/t Au	50	7%	6%	4%	10%	8%	6%	**18%**
		100	14%	12%	9%	20%	17%	13%	**36%**
		200	28%	24%	17%	40%	34%	24%	**66%**
		500	69%	61%	43%	98%	86%	61%	**150%**
		750	104%	92%	65%	147%	130%	92%	**221%**
		1,000	139%	122%	87%	197%	173%	123%	**294%**
ROM	2 g/t Au	50	3%	3%	2%	4%	4%	3%	**9%**
		100	7%	6%	4%	10%	8%	6%	**17%**
		200	14%	12%	8%	20%	17%	11%	**31%**
		500	34%	30%	19%	48%	42%	27%	**75%**
		750	52%	46%	29%	74%	65%	41%	**111%**
		1,000	69%	61%	39%	98%	86%	55%	**148%**
HROM	3.5 g/t Au	50	3%	2%	2%	4%	3%	3%	**7%**
		100	5%	5%	3%	7%	7%	4%	**14%**
		200	10%	9%	7%	14%	13%	10%	**24%**
		500	25%	23%	16%	35%	33%	23%	**56%**
		750	39%	35%	25%	55%	49%	33%	**84%**
		1,000	52%	46%	33%	74%	65%	47%	**111%**
VHG	10 g/t Au	50	2%	1%	1%	3%	1%	1%	**4%**
		100	3%	3%	2%	4%	4%	3%	**8%**
		200	6%	5%	4%	8%	7%	6%	**13%**
		500	16%	14%	10%	23%	20%	14%	**33%**
		750	23%	21%	15%	33%	30%	21%	**50%**
		1,000	30%	27%	19%	42%	38%	27%	**66%**

For the revised protocols, all FSE and e.QFE values are lower than the 2017–2018 protocol, but not all are below ±30%. The optimum mass of the crushed material in most cases is >11 kg, effectively indicating that the entire core should be assayed to extinction (noting that PA is a non-destructive method and that all sample material can be retained after assaying). Whilst this is possible with PA, it is routinely impractical through expense (ca. C$23 per PA jar assay, giving C$506 per sample for whole core assay not including sample preparation).

Recommendations have been provided in Section 5.

Limitations

Sampling protocol and optimisation has been reviewed based on the data provided to Snowden Optiro. No dedicated characterisation or heterogeneity calibration has been undertaken. Snowden Optiro has undertaken its best efforts to provide appropriate protocols based on the data provided and experience. Analysis of the QFE1 is based on the GSE having the same value as the FSE. This is a general assumption, where the GSE could have a greater or lesser value than the FSE. The true nature of the gold particle sizing within a given sample, the protocol applied and sample handling will impact on the QFE1.

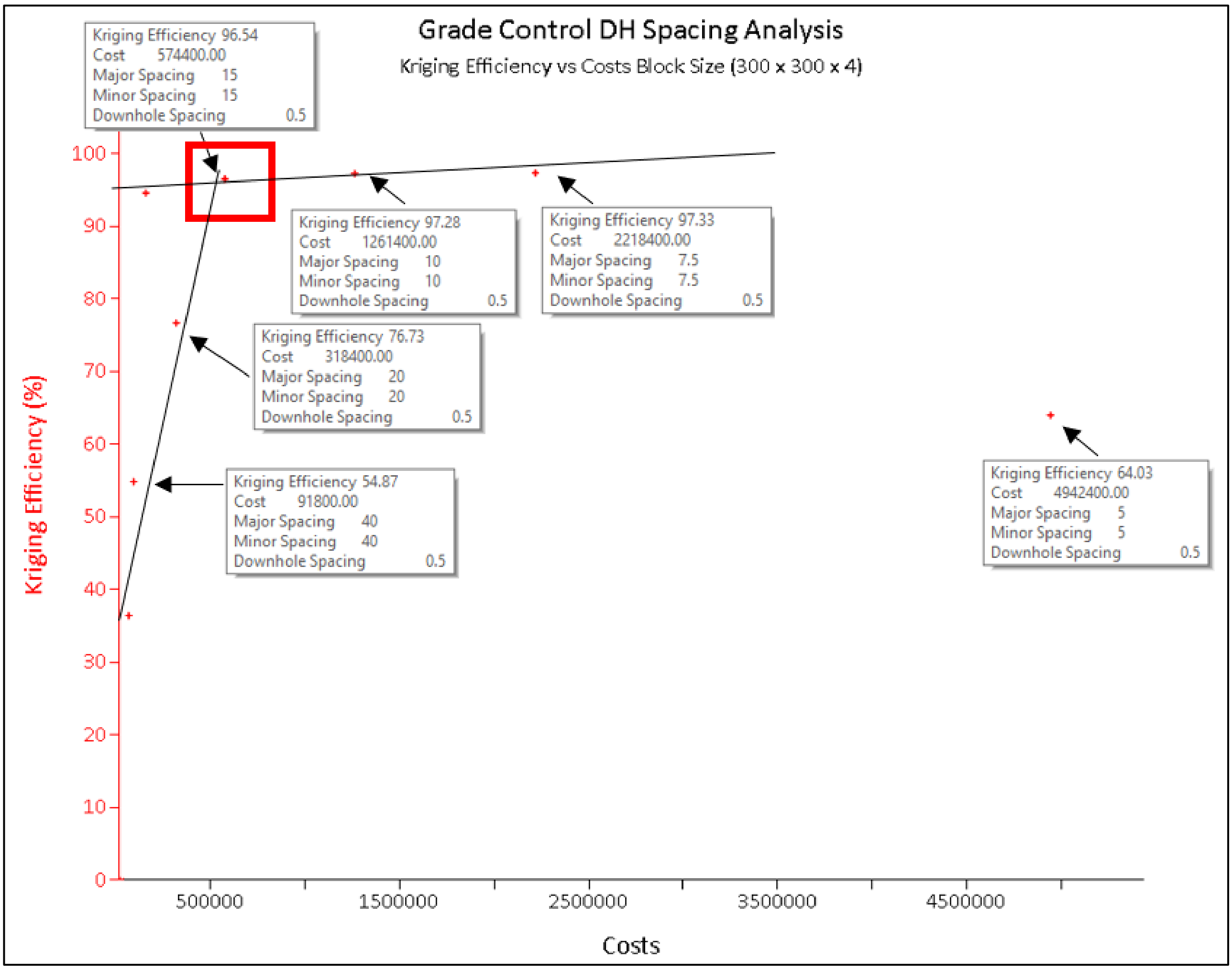
4 Drillhole spacing analysis

Drillhole spacing analysis (DHSA) was undertaken using three methods, summarised in the following sections. The purposes of the DHSA are to recommend a spacing which would reasonably generate Indicated resources, and to investigate a likely grade control (final) drill spacing.

4.1 DHSA high-level analysis

A high-level DHSA analysis using assumed drilling costs was undertaken in Supervisor. Nominal costs were used to assess the impact on the kriging efficiency with drillhole spacing, (Figure 4.1) it should be noted that the costs are not based upon quoted figures; however, the **relative** behaviour of the various drill spacings is relevant.

Figure 4.1 DHSA using kriging efficiency



The results of the DHSA show that there is an inflection at the 15 m x 15 m drill spacing, whereby beyond these costs increase significantly for relatively little improvement in the kriging efficiency or the quality of the estimation.

This rather simplistic analysis indicates that a 15 m x 15 m spacing may be the optimal drill spacing.

4.2 DHSA – 90:15

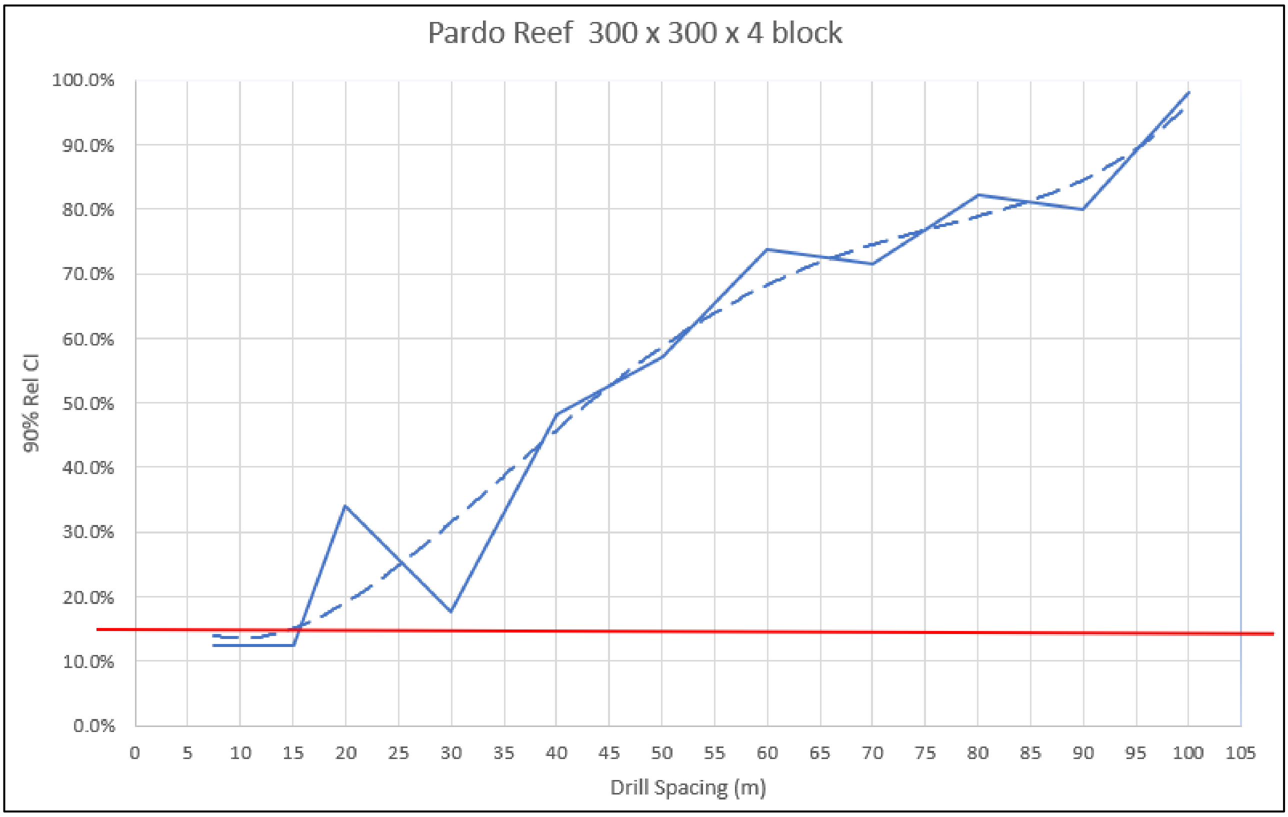
One recognised and widely used industry practice is that for an Indicated Resource, the drillhole spacing should be sufficient to predict tonnage, grade, and metal on annual production parcels with ±15% relative precision at the 90% confidence level. Snowden Optiro has used the 90:15 method as part of the DHSA at Pardo. The 90:15 method uses an assumed annual production tonnage as the basis to assess appropriate drill spacing for an Indicated Resource (Verly, Postolski and Parker, 2014). Given the exploration status of the Pardo Project, a production tonnage is unknown at this stage.

For the purposes of this study, Snowden Optiro has assumed a 1 million tonne per annum operation. A block size representing the 1 million tonne annual production was calculated for the analysis as 300 m(X) x 300 m(Y) x 4 m(Z) and uses a density of 2.8 g/cm^3.

The density value was taken from the 2017 NI 43-101 report, where density values by rock type have been summarised by Inventus, and the average density for the A_M boulder conglomerate was 2.82 g/cm^3. The reef varies between 1 m and 4 m thick, so a thickness/Z component of 4 m was selected for this purpose.

The variography determined in Section 2.5.1 was used to calculate the estimation variance for the block at different drill spacings. The 90% confidence level was then calculated and plotted against each drill spacing, as shown in Figure 4.2. A line of best fit has been applied to smooth out local variations. The line was assessed against a 15% relative error (red line).

Figure 4.2 Graph showing 90% CI plotted against the drill spacing



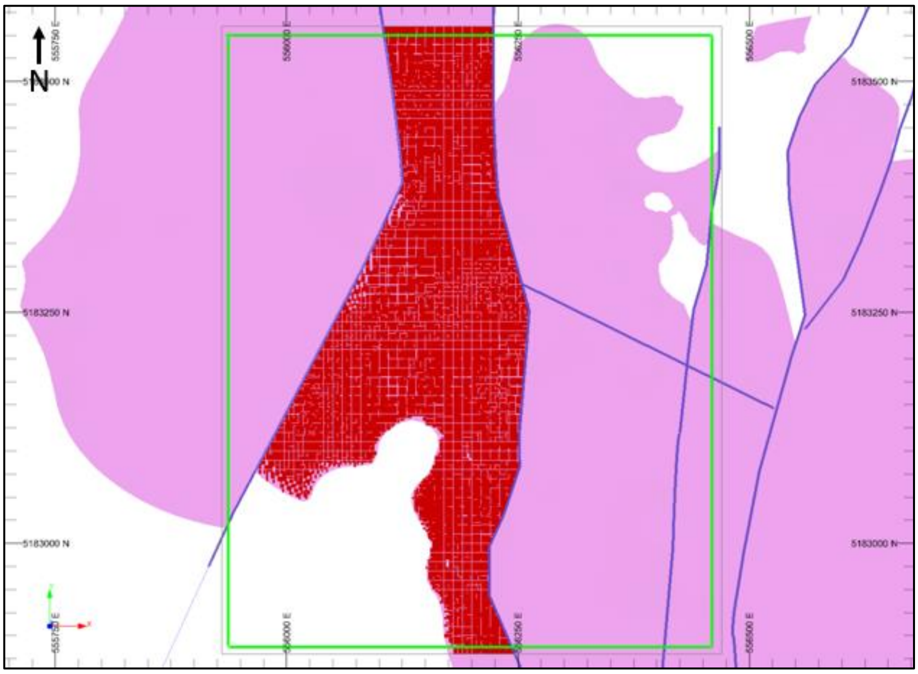
The graph in Figure 4.2 shows that at 90% confidence, 15% relative precision is at approximately 15 m x 15 m spacing. The trendline is crossing 15% line between 10 m and 15 m spacing; however, 15 m is reasonable.

For the Inferred category, data are inadequate for assessing confidence intervals. Geological understanding and experience on similar deposits should be considered instead (Verly et al., 2014).

4.3 DHSA – Simulation study

A conditional simulation study was undertaken on the data, within a defined area within fault block 8, (Figure 4.3). Only this fault block was considered in the study, as the reef is offset at the faults and it would not be appropriate to combine data across the faults. A key assumption, therefore, is that fault block 8 represents the deposit.

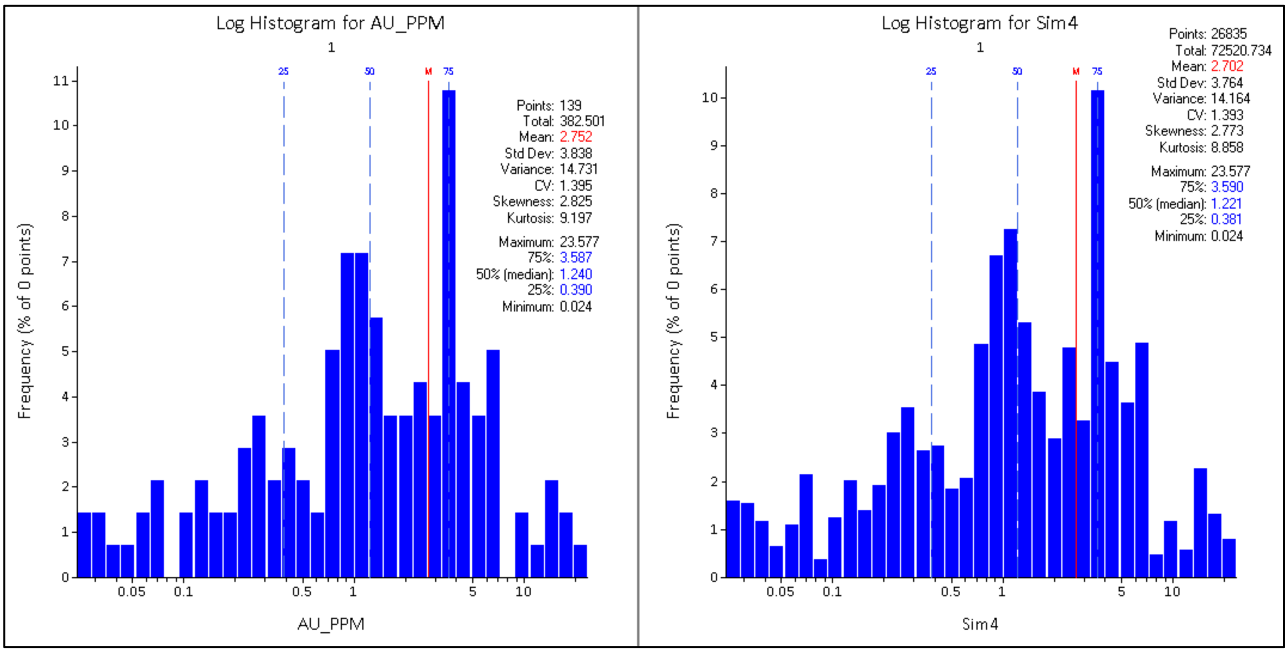
Figure 4.3 Plan view of simulation area – fault block 8



An empty block model with 10 m x 10 m x 1 m cells was created within the reef wireframe for this area and imported into Supervisor to define the area for the simulations.

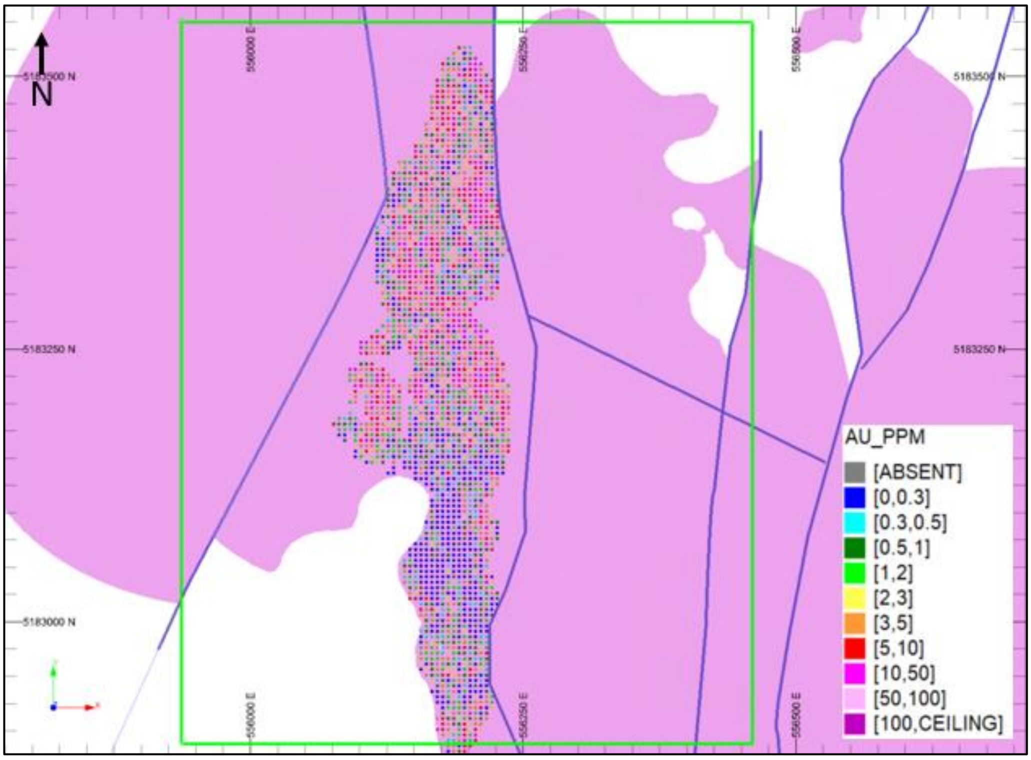
Twenty (20) simulations (equiprobable realisations of gold grade) were run on a 2.5 x 2.5 x 0.5 grid using the variography determined in Section 2.5.1. The simulations were validated. The simulation that best reflected the input data in terms of mean and variance was Sim4, shown in Figure 4.4. This simulation was used as the "truth". It is noted that the reproduction of the input data is almost perfect.

Figure 4.4 Log histogram of input data (left) and simulation (right)



Pseudo-drillholes were generated by sampling the simulation at different drill spacings; 5 m x 5 m, 7.5 m x 7.5 m, 10 m x 10 m, 15 m x 15 m, 20 m x 20 m, and 25 m x 25 m (Figure 4.5 shows the grid selected at 5 m x 5 m, coloured on gold grade).

Figure 4.5 Plan view displaying drillholes generated on 5 m x 5 m grid



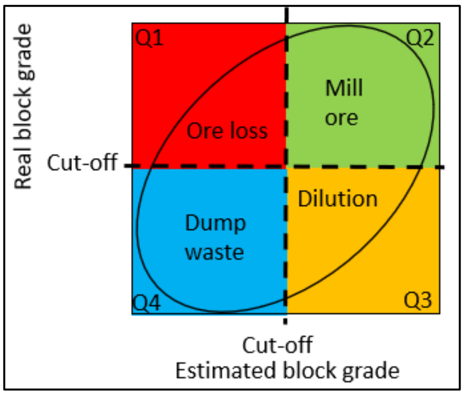
Variography was reviewed and modelled for each dataset, based upon the actual set of pseudo-drillholes. An ordinary kriged estimate was undertaken for each dataset into the 10 x 10 x 1 block model and each model estimate validated against the informing pseudo-drillholes.

The different scenarios were reviewed in terms of value optimisation, with misclassification of material analysed. Each scenario was compared to the “truth” (i.e. the original simulation) and classified into four quadrants using a cut-off of 0.5 g/t Au. The quadrants and classification description are summarised in Table 4.1, with a descriptive image shown in Figure 4.6.

Table 4.1 Misclassification quadrant summary

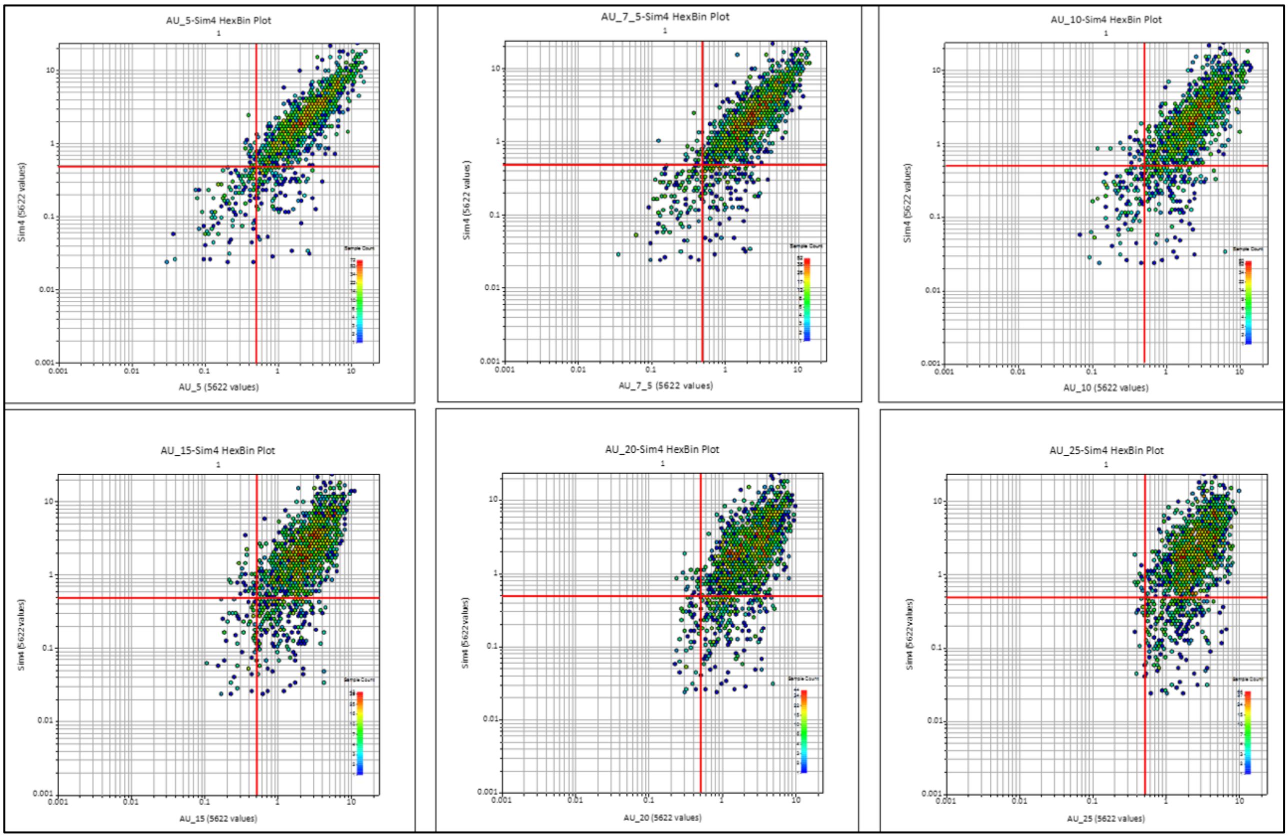
Quadrant	Location	Classification	Description
1	Dump	Ore loss	Block classed as waste and it is ore
2	Plant	Revenue	Block classed as ore and it is ore
3	Plant	Dilution	Block classed as ore and it is waste
4	Dump	Planned mining cost	Block classed as waste and it is waste

Figure 4.6 Misclassification plot



The misclassification plots for each of the drill scenarios are presented in Figure 4.7, with the estimated grades on the X axis and the “true” (i.e. Sim4) grade on the Y axis.

Figure 4.7 Misclassification plots for each drill spacing (5 x 5, 7.5 x 7.5, 10 x 10 m, top row left to right, 15 x 15 20 x 20, 25 x 25 m bottom row left to right)



The results show that there is very little ore loss (quadrant 1) in each scenario; however, the dilution (i.e. waste misclassified as ore) increases as the drill spacing increases.

A review of the optimal spacing with respect to net revenue was undertaken reporting the models by quartile and calculating costs associated with the misclassification as well as the drilling and processing costs incurred.

The following assumptions used in this analysis have been summarised in Table 4.2. Calculations have been undertaken in Australian dollars, but approximate US dollar prices are included for reference. In many cases, these costs are relative so are not accurate. It is assumed that all blocks would have to be mined and so no mining cost has been applied. An average overburden thickness has been assumed, and this will incur a per meter drilling (but not an assaying) cost.

Table 4.2 Assumptions applied to net revenue calculations

Metric	A$	US$ (approx. equivalent)
oz conversion	31.1035	
Gold price ($/oz)	2,600	1,750
Reef drill and sample cost ($/m)	150	100
Processing cost ($/t)	37	25
Overburden drill cost ($/m)	100	67
Average overburden (m)	12	

Using the metrics in Table 4.2, for each of the drill spacing scenarios generated, the total costs and net revenue have been calculated and summarised in Table 4.3. The drill spacing against the number of holes required to meet the spacing has been plotted in Figure 4.8, and the results of the spacing versus the net revenue are displayed in Figure 4.9.

Table 4.3 Summary of costings (Australian dollars)

DH spacing (m)	No. of holes	No. of overburden drill metres	No. of reef drill metres	Overburden drill cost ($)	Reef drill total cost (A$)	Total drill cost (A$)	Net revenue (A$)
5 x 5	2,212	26,544	3,342.5	$2,654,400	$501,375	$3,155,775	$45,596,206
7.5 x 7.5	990	11,880	1,490.5	$1,188,000	$223,575	$1,411,575	$47,095,692
10 x 10	546	6,552	831.5	$655,200	$124,725	$779,925	$46,059,508
15 x 15	248	2,976	367.5	$297,600	$55,125	$352,725	$42,425,140
20 x 20	136	1,632	205	$163,200	$30,750	$193,950	$44,071,778
25 x 25	93	1,116	134.5	$111,600	$20,175	$131,775	$44,882,879

Figure 4.8 Graph displaying the number of holes required at each drill spacing for the test area

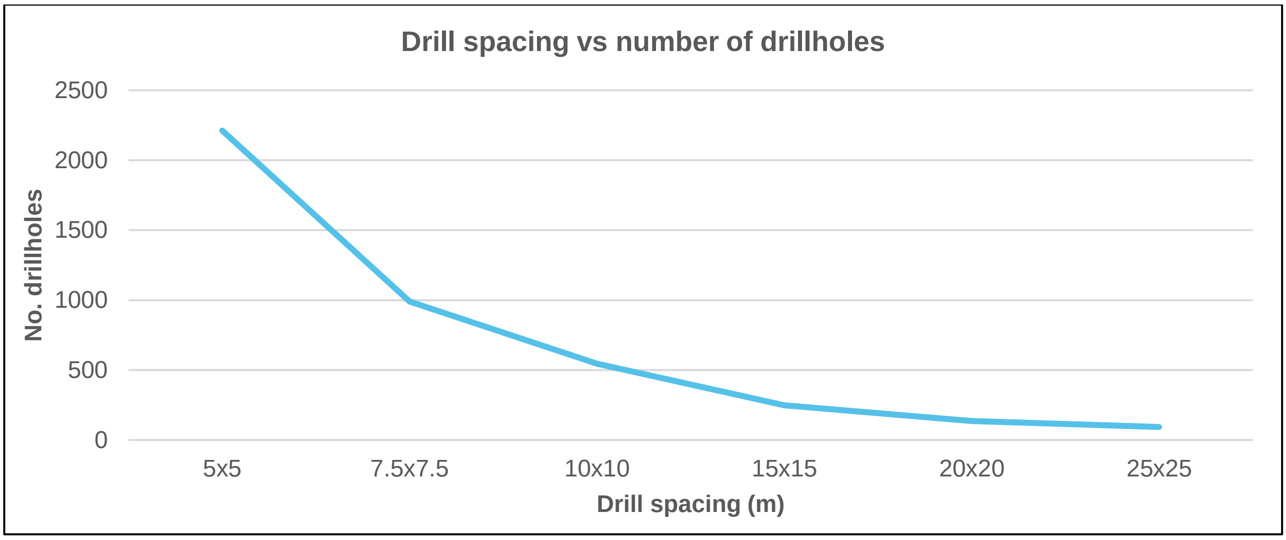
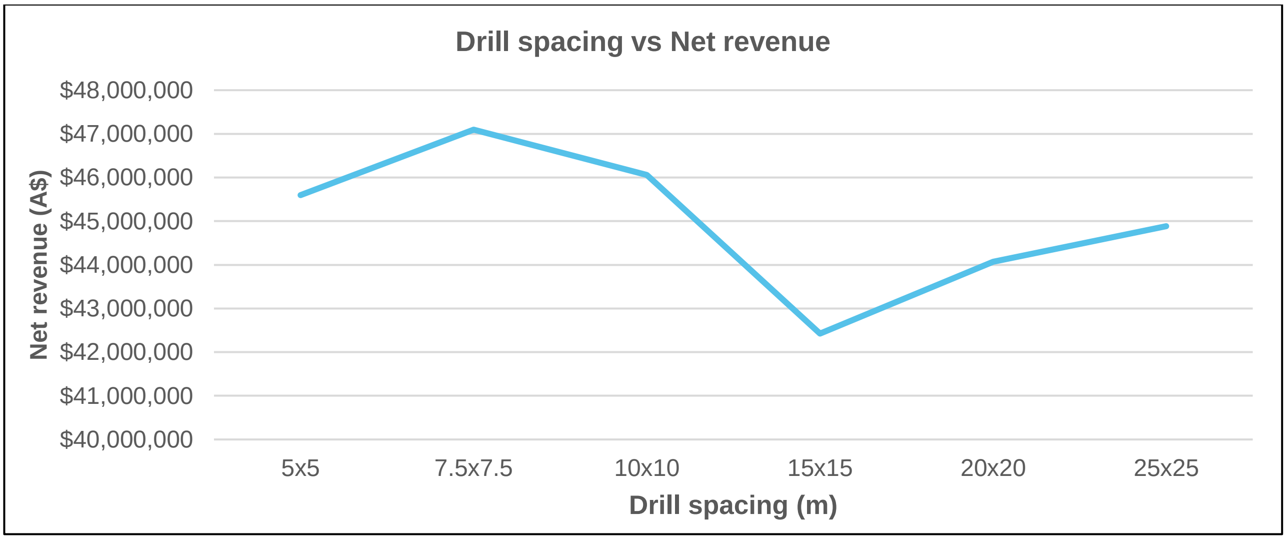


Figure 4.9 Graph displaying net revenue for each drill spacing



The number of drillholes increases exponentially with the closer spacings (Figure 4.8), adding significant time spent drilling, sampling, and awaiting assays. Note that a very large number of holes (e.g. at a 5 m x 5 m spacing) may impose additional non-financial penalties in terms of scheduling and delays.

The results presented in Figure 4.9 show that the 7.5 m x 7.5 m drilling is giving the highest net revenue return. Having tested sensitivities of increasing the drill costs and gold price, it seems that the revenue generated from the correctly classified material has a greater impact than the cost of drilling at all but the very closest spacing. From a grade control perspective, the 7.5 m x 7.5 m drill spacing would provide the best return, with less than half the drilling required compared to the 5 m x 5 m scenario.

4.4 DHSA limitations

All the above methods assume that the variogram will not change with additional drilling. If the variography changes with further drilling, then the results may change, and additional work will be required.

This spacing study is limited to the grade; however, there may be risk elsewhere potentially in the geology, the thickness of the reef or in high grade portions of the deposit.

The method also assumes that the grade characteristics of the test area are applicable over the other portions of the reef, which may not be the case.

4.5 DHSA conclusions

Both the high level and 90:15 drill spacing analysis methods indicate that 15 m x 15 m is an appropriate spacing for Indicated resources.

The simulation showed the 7.5 m x 7.5 m spacing to be most cost effective and reasonable from a grade control perspective using the data available at this stage.

The author's experience with similar deposits supports these findings, with a 15 m x 15 m spacing considered reasonable for Indicated resources, closing to 7.5 m x 7.5 m for grade control.

Inferred resources require either geological continuity or grade continuity to be demonstrated. The geological continuity appears to be reasonable at Pardo; however, the thickness of the reef may not be resolved at a wide spacing. The author's experience with similar deposits suggests that for Inferred resource classification a drill spacing of 50 m to 60 m would be reasonable.

5 Recommendations

5.1 Drillhole spacing recommendations

From the work undertaken, Snowden Optiro recommends the following for drillhole spacing:

- Drill spacing of 15 m x 15 m is reasonable for Indicated resources
- Drill spacing of 60 m x 60 m is reasonable for Inferred resources
- A drill spacing of 7.5 m x 7.5 m is appropriate for grade control.

5.2 Sampling recommendations

From the work undertaken, Snowden Optiro recommends the following for sampling:

- If any additional bulk sampling is undertaken, ensure that the minimum individual sample mass is 200 t.
- Continue to use whole diamond core at HTW size, though increase to PQ size if available (note current calculations have not been undertaken on PQ).
- Use primary sample composites of 0.5 m ±0.1 m.
- Apply Protocol 2 where the 0.5 m composite (5.5 kg) is crushed to P_{90} -2 mm and 2 kg split off for PhotonAssay.
- When drilling commences implement a quality control program that includes a crush duplicates (e.g. take 2 x 2 kg splits for assay), which will allow the protocol to be monitored and reduced if appropriate. More than 200 pairs are required for analysis.
- Once a reasonable number of assay results are returned (suggest >200), review the grade distribution and select several residues across a given intersection (e.g. if intersection is 3 m, then select six sets of residues) and assay by PhotonAssay so the data for full core is available for heterogeneity analysis.

5.3 Additional recommendations

- Exclusion of pre-2017 holes for the MRE is appropriate, given the bias and sampling across lithological boundaries, particularly if the area is drilled using updated sample protocol.
- Use picked-up (surveyed) collars and model geology, mineralisation, and bulk samples within the same project to ensure consistency in future modelling.
- Undertake a gold deportment study on selected drill intersections and intervals post-PhotonAssay. Use methodology optimised to sequentially liberate gold fractions so that particle size and distribution can be evaluated. Snowden Optiro will provide further details on this option.

6 References

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