Monument Mining Limited

Selinsing Gold Sulphide Project - NI 43-101 Technical Report

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CONSOLIDATED REPORT

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1 SUMMARY

1.1 Introduction

This NI43-101 Technical Report describes the Buffalo Reef Property (“Buffalo Reef”) and Selinsing Property (“Selinsing”), a mineral exploration, development, and production area of the Selinsing Gold Mine (SGM) Project (“Selinsing Gold Mine Project”), located in the region of Selinsing, Pahang State, Malaysia. The Buffalo Reef and Selinsing Property is 100% owned by Monument Mining Limited (TSX-V: MMY and FSE: D7Q1). Monument Mining Limited (Monument) is a mining and exploration company based in Vancouver, British Columbia.

The report focus is on feasibility study evaluation of the operation with a BIOX® and flotation ore treatment of identified sulphide ore which is planned to be added to the existing oxide process plant at Selinsing. To this end, prefeasibility studies (PFS) were undertaken in 2016 (refer Snowden, Dec 2016) to assess the viability of the combined Buffalo Reef and Selinsing Property using a similar methodology but now updated studies to feasibility study (FS) standard have been carried out demonstrating a BIOX® and flotation ore treatment is the preferred method.

This report summarises the detailed FS carried out for final front end engineering design (FEED) and plant construction detail. This FS includes updated sulphide and oxide Mineral Resource and Mineral Reserve estimates reported in accordance with CIM guidelines. Except for mining and stockpile depletions to March 2018, the Mineral Resource model has not been updated since the 2016 study, but revised reporting of the Mineral Resource has presented a significant increase in reported Mineral Resource from the previous Snowden (Dec 2016) PFS due to changes in assumed resource optimisation pit shell constraints and a revised cut-off grade.

As well as detailed metallurgical and engineering studies, the mining aspects of the Snowden (Dec 2016) study were reviewed and optimised by Monument with updated cost and revenue information. The reserve pit mine design was largely unchanged except for revision in the Selinsing Pit area from an updated geotechnical study in 2018 by Peter O’Bryan and Associates (POB).

The FS presents the economic viability of Selinsing Gold Mine Project and is based on mineral reserve estimation. There were also investigations for inclusion of Inferred material in mine planning as well as underground mining potential,

The discussion of opportunities in converting Inferred Mineral Resources to Indicated Mineral Resources, subject to future exploration success, include the following scenarios:

- Opportunity Case One – FS open pit designs inclusive of Inferred Mineral Resource material
- Opportunity Case Two – All open pit potential material including Inferred Mineral Resource material

The report was compiled by Monument with external review and final reporting by Snowden Mining Industry Consultants Pty Ltd (Snowden). Other external peer review was completed by Orway Mineral Consultants (OMC) and Mike Kitney, Independent Technical Director of Monument.

Qualified Persons for geology, mining, metallurgy and for engineering and costs are provided for the Technical Report.

The Qualified Person for the entire report has ensured review of all the FS items to ensure they are current and complete as required by the NI 43-101 instrument.

1.2 Geology and mineralisation

The gold mineralisation at Selinsing is hosted within a shear zone that strikes at 350 degrees (°) and dips 60° to 70° to the east, with the higher-grade mineralised shoots within the main mineralised shear plunging to the southeast. The main shear zone is hosted within a sequence of felsic tuff and very fine clastic argillite with calcareous material and limestone in the hangingwall. High-grade mineralisation is often associated with quartz stockworks and quartz-carbonate veins within highly deformed sedimentary rocks. The gold at Selinsing is generally in the form of fine-grained gold particles (<20 µm) commonly associated with pyrite and arsenopyrite and rarely with chalcopyrite. Visible (mm-scale) gold, although not common, occurs in quartz veins within the shear zone. The higher-grade quartz veins can be over a metre in true thickness and have been traced up to 300 m along strike and 200 m down dip. Lower grade gold mineralisation occurs as finely disseminated gold within intensely deformed envelopes around the quartz veins within the shear zone.

Gold mineralisation at Buffalo Reef is structurally controlled and associated with Permian sediments within a 200 m wide shear zone that parallels the north-south trending Raub-Bentong suture. Mineralisation occurs over a total strike length of approximately 2.6 km. Rocks within the Buffalo Reef shear zone have typically undergone silica-sericite-pyrite alteration to varying degrees.

The gold occurs within moderately to steeply east-dipping veins and fracture zones, which range in thickness from 1 m up to 15 m in thickness (average thickness is approximately 10 m in the main mineralised veins), although local flexures in the veins can host mineralisation up to 25 m in thickness. Veins, which are boudinaged in some areas, are generally composed of massive quartz with 1% to 5% (by volume) sulphide minerals, namely pyrite and arsenopyrite, along with varying amounts of stibnite.

1.3 Mineral Resource estimates

The Mineral Resource estimate (MRE) for the Selinsing deposit is provided in Table 1.1, and for Buffalo Reef in Table 1.2. The Mineral Resource is limited to a pit shell based on a long-term perspective potential gold price of US$2,400/oz. The pit shell was used by Monument to define the likely limits of potential open pit mining. The mining and cost parameters and assumptions used by Monument to generate the resource pit shell are discussed in Sections 14.4.7 and 14.5.7. The Mineral Resource estimates for both Selinsing and Buffalo Reef have been depleted for all mining to end of March 2018.

Table 1.1 Selinsing Mineral Resource statement, inclusive of Mineral Reserves, depleted for mining to end of March 2018

Classification	Oxidation	Cut-off (g/t Au)	Tonnes (kt)	Au (g/t)	S (%)	Ounces (koz Au)
Indicated	Oxide	0.3	64	0.62	0.03	1
	Transitional	0.5	100	1.16	0.38	4
	Fresh	0.5	5,007	1.51	0.47	243
Indicated – Total			**5,171**	**1.49**	**0.46**	**248**
Inferred	Oxide	0.3	8	0.98	0.03	0.3
	Transitional	0.5	3	1.14	0.17	0.1
	Fresh	0.5	1,680	2.02	0.52	109
Inferred – Total			**1,691**	**2.02**	**0.51**	**110**

Notes: Small discrepancies may occur due to rounding. The classification applies to the Au grades only; S grades are considered indicative only. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 1.2 Buffalo Reef Mineral Resource statement, inclusive of Mineral Reserves, depleted for mining to end of March 2018

Classification	Oxidation	Zone	Cut-off (g/t Au)	Tonnes (kt)	Au (g/t)	As (ppm)	Sb (ppm)	S (%)	Ounces (koz)
Indicated	Oxide	BRN	0.3	121	0.97	1,843	185	0.35	4
		BRC	0.3	158	0.81	1,565	141	0.50	4
		Felda	0.3	160	1.11	2,273	232	0.08	6
		BRS	0.3	55	2.06	3,076	837	0.15	4
	Oxide – Total			**494**	**1.08**	**2,031**	**258**	**0.29**	**18**
	Transitional	BRN	0.5	151	1.16	2,132	196	0.57	6
		BRC	0.5	382	1.06	2,167	96	0.40	13
		Felda	0.5	219	1.48	2,776	291	0.33	10
		BRS	0.5	234	2.57	2,895	3,143	0.59	19
	Transitional – Total			**986**	**1.53**	**2,470**	**878**	**0.45**	**48**
	Fresh	BRN	0.5	86	1.05	2,122	83	0.71	3
		BRC	0.5	1,106	1.56	3,157	1,796	0.88	55
		Felda	0.5	686	1.66	2,711	878	0.71	37
		BRS	0.5	1,167	2.06	2,732	1,111	0.87	77
	Fresh – Total			**3,045**	**1.76**	**2,864**	**1,278**	**0.83**	**172**
INDICATED – TOTAL				**4,525**	**1.63**	**2,687**	**1,080**	**0.69**	**238**
Inferred	Oxide	BRN	0.3	72	0.87	1,646	89	0.10	2
		BRC	0.3	114	1.10	1,459	60	0.20	4
		Felda	0.3	66	1.03	1,424	141	0.31	2
		BRS	0.3	89	1.14	1,296	180	0.05	3
	Oxide – Total			**341**	**1.05**	**1,453**	**113**	**0.16**	**11**
	Transitional	BRN	0.5	127	1.12	1,854	88	0.72	5
		BRC	0.5	179	1.18	1,891	156	0.28	7
		Felda	0.5	68	1.24	1,731	143	0.46	3
		BRS	0.5	108	1.37	1,708	673	0.49	5
	Transitional – Total			**482**	**1.22**	**1,818**	**252**	**0.47**	**20**
	Fresh	BRN	0.5	102	1.18	2,761	52	0.71	4
		BRC	0.5	1,851	1.71	2,592	1,708	0.87	102
		Felda	0.5	1,263	1.80	3,104	1,066	0.82	73
		BRS	0.5	668	1.46	2,353	516	0.54	31
	Fresh – Total			**3,883**	**1.68**	**2,722**	**1,251**	**0.80**	**210**
INFERRED – TOTAL				**4,706**	**1.59**	**2,537**	**1,066**	**0.72**	**241**

Notes: Small discrepancies may occur due to rounding. The classification applies to the Au grades only; As, Sb and S grades are considered indicative only. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

An increase of 48 koz or 24% for deposit Indicated Resources and 45 koz or 69% for Inferred Resources is observed for the Selinsing deposit, compared to the previous resource estimate for the 2016 PFS (Snowden, 2016). For the Buffalo Reef deposit, Indicated Resources have decreased 4 koz or 2%, and Inferred Resources increased 31 koz or 15%. The main factors driving this overall positive difference, even considering the mining depletion in the period, are the larger constraining pit shell and a revised cut-off grade for resource reporting. A breakdown of the differences is shown in Table 1.3 for Selinsing and Table 1.4 for Buffalo Reef.

Table 1.3 Differences between the current and previous (Snowden, 2016) Selinsing deposit Mineral Resources

Classification	Oxidation	Cut-off (g/t Au)	Tonnes (kt)	Ounces (koz Au)
Indicated	Oxide	0.3	-26	-1
	Transitional	0.5	10	0
	Fresh	0.5	1,967	50
Indicated – Total			**1,951**	**48**
Inferred	Oxide	0.3	-2	0
	Transitional	0.5	0	0
	Fresh	0.5	1,140	44
Inferred – Total			**1,138**	**45**

Notes: Small discrepancies may occur due to rounding.

Table 1.4 Differences between the current and previous (Snowden, 2016) Buffalo Reef deposit Mineral Resources

Classification	Oxidation	Zone	Cut-off (g/t Au)	Tonnes (kt)	Ounces (koz)
Indicated	Oxide	BRN	0.3	-59	-2
		BRC	0.3	-12	0
		Felda	0.3	-100	-5
		BRS	0.3	-45	-3
	Oxide – Total			**-216**	**-10**
	Transitional	BRN	0.5	1	0
		BRC	0.5	72	1
		Felda	0.5	29	0
		BRS	0.5	4	0
	Transitional – Total			**106**	**1**
	Fresh	BRN	0.5	16	1
		BRC	0.5	116	2
		Felda	0.5	66	2
		BRS	0.5	37	0
	Fresh – Total			**235**	**5**
INDICATED – TOTAL				**125**	**-4**
Inferred	Oxide	BRN	0.3	-28	0
		BRC	0.3	-6	0
		Felda	0.3	-4	0
		BRS	0.3	-1	0
	Oxide – Total			**-39**	**0**
	Transitional	BRN	0.5	37	1
		BRC	0.5	39	1
		Felda	0.5	18	1
		BRS	0.5	18	1
	Transitional – Total			**112**	**4**
	Fresh	BRN	0.5	72	3
		BRC	0.5	351	13
		Felda	0.5	223	7
		BRS	0.5	138	4
	Fresh – Total			**783**	**27**
INFERRED – TOTAL				**856**	**31**

1.3.1 Stockpile Mineral Resources

Stockpiles at the Selinsing Gold Mine Project comprise ore mined from the Selinsing and Buffalo Reef pits. Ore is stockpiled according to the source (Selinsing or Buffalo Reef) and oxidation state (oxide and sulphide ore) along with the gold grade. For the sulphide ore, the stockpiles are further subdivided based on the leachability of the ore (designated as “leachable” and “non-leachable”). The leachability designation refers to the current processing plant configuration.

Stockpile volumes are surveyed by the Selinsing mine survey department on a monthly basis. The bulk density of the stockpiles is based on applying a 25% swell factor to the density.

The grade of each stockpile is primarily based on grade control estimates of the source ore blocks during mining. The grade of the stockpiles is then adjusted each month according to the opening balance, material added through mining (from grade control and haulage estimates) and material sent to the crusher.

The Mineral Resources contained in the stockpiles at the Selinsing Gold Mine Project are classified as Measured Mineral Resources in accordance with CIM guidelines. Snowden believes that a Measured classification is appropriate for the stockpile resources, based on the following:

- High confidence in the stockpile volumes which are surveyed on a monthly basis
- Stockpile grade estimates are based on grade control of ore blocks during mining
- Reconciliation of tonnes and grade with plant production.

Mineral Resources for the stockpiles at the Selinsing Project, as at the end of March 2018, are summarised in Table 1.5.

Table 1.5 Stockpile Mineral Resources as at end of March 2018

Stockpile name	Stockpile ID	lcm	Tonnes (kt)	Au (g/t)	Ounces (koz)
Oxide Stockpiles					
High Grade 1 (Oxide)	SEL HG1 O	-	-	-	-
Low Grade 1 (Oxide)	SEL LG1 O	6,081	12	1.1	0.4
Low Grade 2 (Oxide)	SEL LG2 O	3,189	6	0.73	0.2
Super Low Grade 1 (Oxide)	SEL SLG1 O	2,191	4	0.44	0.1
Super Low Grade 2 (Oxide)	SEL SLG2 O	72,179	991	0.43	13.9
Super Low Grade 4 (Oxide)	SEL SLG 4	36,293	78	0.50	1.3
BR High Grade 1 (Oxide)	BR HG1 O	566	1	3.28	0.1
BR Low Grade 1 (Oxide)	BR LG1 O	928	2	1.02	0.1
BR Super Low Grade 1 (Oxide)	BR SLG1 O	84,030	168	0.55	3.0
B7 High Grade 1 (Oxide)	B7 HG1 O	1,767	3	2.12	0.2
B7 Low Grade 1 (Oxide)	B7 LG1 O	77	0.13	1.02	0.01
B7 Super Low Grade 1 (Oxide)	B7 SLG1 O	449	0.73	0.69	0.01
Oxide – Total		**607,750**	**1,265**	**0.47**	**19.1**
Leachable Sulphide Stockpiles					
High Grade 1 (Leachable Sulphide)	SEL HG1 S	-	-	-	-
Low Grade 1 (Leachable Sulphide)	SEL LG1 S	-	-	-	-
BR High Grade 1 (Leachable Sulphide)	BR HG1 S	879	1.776	1.86	0.1
BR Low Grade 1 (Leachable Sulphide)	BR LG1 S	1,237	2.499	0.97	0.1
High Grade 2 (Leachable Sulphide)	SEL HG2 S	-	-	-	-
Low Grade 3 (Leachable Sulphide)	SEL LG3 S	213	0.46	0.68	0.01
Low Grade 4 (Leachable Sulphide)	SEL LG4 S	-	-	-	-
BR High Grade 2 (Leachable Sulphide)	BR HG2 S	5,186	11.399	2.72	1.0
BR Super Low Grade 1 (Leachable Sulphide)	BR SLG1 S	2,145	4.333	0.55	0.1
B7 High Grade 1 (Leachable Sulphide)	B7 HG1 S	-	-	-	-
B7 Low Grade 1 (Leachable Sulphide)	B7 LG1 S	-	-	-	-
B7 Super Low Grade 1 (Leachable Sulphide)	B7 SLG1 S	-	-	-	-
Leachable Sulphide – Total		**9,660**	**20.5**	**1.92**	**1.3**
Non-Leachable Sulphide Stockpiles		0	0		0
BR Low Grade 2 (Non-Leachable Sulphide)	BR LG2 S	11,570	23.115	1.25	0.9
Super Low Grade 3 (Non-Leachable Sulphide)	SEL SLG3 S	748	1.511	0.6	0.03
B7 High Grade 2 (Non-Leachable Sulphide)	B7 HG2 S	-	-	-	-
B7 Low Grade 2 (Non-Leachable Sulphide)	B7 LG2 S	-	-	-	-
Non-Leachable Sulphide – Total		**12,318**	**24.626**	**1.21**	**0.9**

Notes:
- *All stockpiles classified as Measured Resources with 100% conversion to Proven Reserves*
- *lcm = loose cubic metres; stockpile volume and tonnes are not rounded as based on surveyed volumes*
- *BR = Buffalo Reef stockpile; SEL = Selinsing stockpile*
- *SLG = super low grade (0.30–0.75 g/t Au); LG = low grade (0.75–1.50 g/t Au); HG = high grade (1.50–3.50 g/t Au).*

1.3.2 Old tailings resource

A significant volume of old tailings is located on the property. Old tailings at the Selinsing project include the balance of used tailings dams located next to Pit V and Pit VI Selinsing Pits. Most of the current tailings material originated from the oxide mining operation by the local operator, Tshu Lian Seng Mining since 1987 until the operation ceased in late 1995. There are small amounts of tailings deposited from previous underground mining in the early 20th century but boundaries from underground tailing and tailings from the oxide operation are not clearly discernible.

A total of 201 holes for 1,503 m of drilling were available and used for estimating the gold grades in blocks coded as tailings in the Selinsing resource block model.

Drilling of the old tailings was completed in 1996 to 1997 using air-core, auger and reverse circulation (RC) drilling, in an incomplete grid of 20 m x 20 m spacing. More recent Monument diamond drilling intercepted the old tailings close to surface. This drilling was used in 2013 to define a wireframe surface representing the volume of tailings. This was checked visually on section against the drillhole positions, verifying the grades and logging information.

The blocks coded as tailings in the Selinsing block model were estimated for gold using inverse distance squared (ID^2), from 1.5 m composite samples.

The Mineral Resources contained in the old tailings at the Selinsing Project are classified as Indicated Resources in accordance with CIM guidelines. The Indicated classification was considered appropriate for the old tailings resources based on the following:

- Moderate to high confidence in the tailings volumes currently processed, which are surveyed monthly as part of the routine end of month surveying
- Relatively low variability of the grade distribution of the tailings samples
- Most of the tailings grade estimates are based on a drillhole grid spacing of up to 20 m x 20 m
- Reconciliation of tonnes and grade with plant production.

The Mineral Resources estimated for old tailings at the Selinsing Project, as at the end of March 2018, are summarised in Table 1.6. The same lower cut-off of 0.30 g/t Au as used for the oxide Mineral Resource. A mining recovery factor of 80% was applied for the likely practical limits of reclaim mining, accounting for losses associated with scattered waste material in the tailings, such as material used to construct bunds/walls between the ponds and the dam retaining wall on the west side.

Table 1.6 Selinsing Old Tailings Mineral Resource, inclusive of Mineral Reserves, depleted for reclaiming to end of March 2018

Classification	Cut-off (g/t Au)	Tonnes (kt)	Au (g/t)	Ounces (koz Au)
Indicated Total	0.30	975	0.75	24

Notes: Small discrepancies may occur due to rounding.

1.4 Mining

The Selinsing and Buffalo Reef deposits have been mined using open pit methods for nine years and the mining contractor for the mining activities is Minetech Construction Sdn Bhd (Minetech). The contract is current, and Minetech is responsible for the mining of ore and waste including:

- Drill and blast activities
- Load and haul activities
- Rehabilitation activities.

All mining functions relating to health and safety, mine planning and mining technical support are provided by Monument. Currently, there is no plan for any underground operations.

1.4.1 Geotechnical

Geotechnical studies were undertaken by Snowden as a geotechnical review of the Selinsing and Buffalo Reef mines in 2016 (Geotechnical Review of Selinsing and Buffalo Reef). In 2018, another review was carried out by Peter OBryan ans Associates (POB (2018)) confirming much of the findings by Snowden for Buffalo Reef but revised the design guidelines for the Selinsing pit area.

The Selinsing and Buffalo Reef pits have the following design considerations:

- Maximum recommended batter height: 10 m
- Minimum berm width: 4 m (increased where necessary to suit maximum inter-ramp angle).

Recommendations for pit wall design parameters have been reviewed by POB (2018); these are summarised in Table 1.7. The guidelines for Buffalo Reef as proposed by Snowden (Dec 2016) shown in Table 1.8.

Table 1.7 Recommended pit wall design parameters for Selinsing Pit area by POB 2018:

Wall position	Geotechnical domain	Bench height	Batter angle	Berm width	Inter-ramp angle
West	Completely to highly weathered phyllites	5 m	45°	4 m	29°
	Moderate to fresh phyllites	5 m	60°	6 m	29°
East	Completely to highly weathered rocks	10 m	50°	5 m	37°
	Moderate to fresh limestones	10 m	65°	5 m	46°
North and south	Completely to highly weathered sedimentary rocks	10 m	50°	5 m	37°
	Moderate to fresh sedimentary rocks	10 m	60°	5 m	43°

Table 1.8 Snowden slope design recommendations for Buffalo Reef

Deposit	Geotechnical domain	Batter face angle	Inter-ramp slopes	
			Face angle	Maximum bench stack height
Buffalo Reef	Saprolite/highly weathered rock	45°	30°	30 m
	Footwall	50°	40°	80 m
	Hangingwall	50°	40°	80 m

1.4.2 Mine planning

Monument undertook mine planning for the FS by undertaking pit optimisation of the Indicated Resources in the block model using the Pseudo Flow algorithm in the DeswikCAD software. Pit Optimisation was also undertaken to assess whether there is a reasonable prospect of extraction of Mineral Resources and does not include economic analysis. The model was not diluted prior to optimisation. Optimisation results were then compared to Snowden (Dec 2016) results. With very similar results obtained for optimisation in Buffalo Reef, the pits and waste dumps as designed by Snowden (Dec 2016) for Buffalo Reef were left unchanged.

With revised pit wall guidance for Selinsing pit areas as proposed by POB (2018) a revised design for Selinsing Pit 4 and 5/6 was carried out based on pit optimisation revised for this area.

Resultant pit inventories updated for mining depletion, including the stockpiles at end of March 2018, are summarised in Table 1.9 and open pit location and haulage routes in Figure 1.1.

Figure 1.1 Overall mining layout with haulage paths

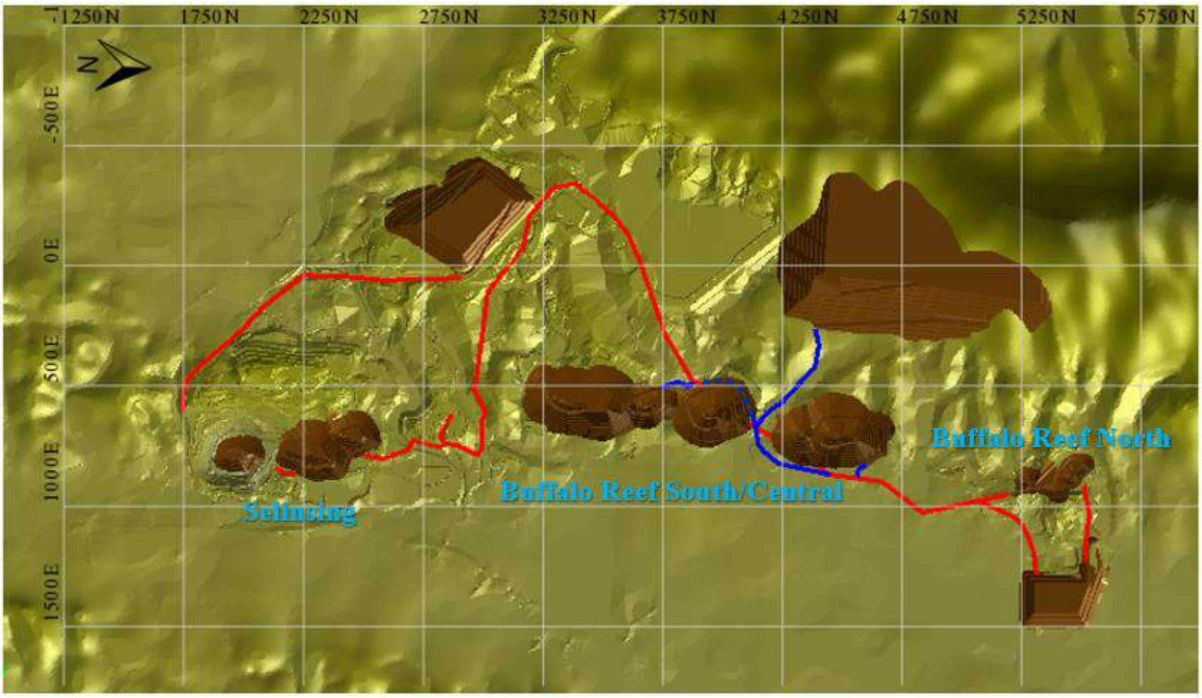


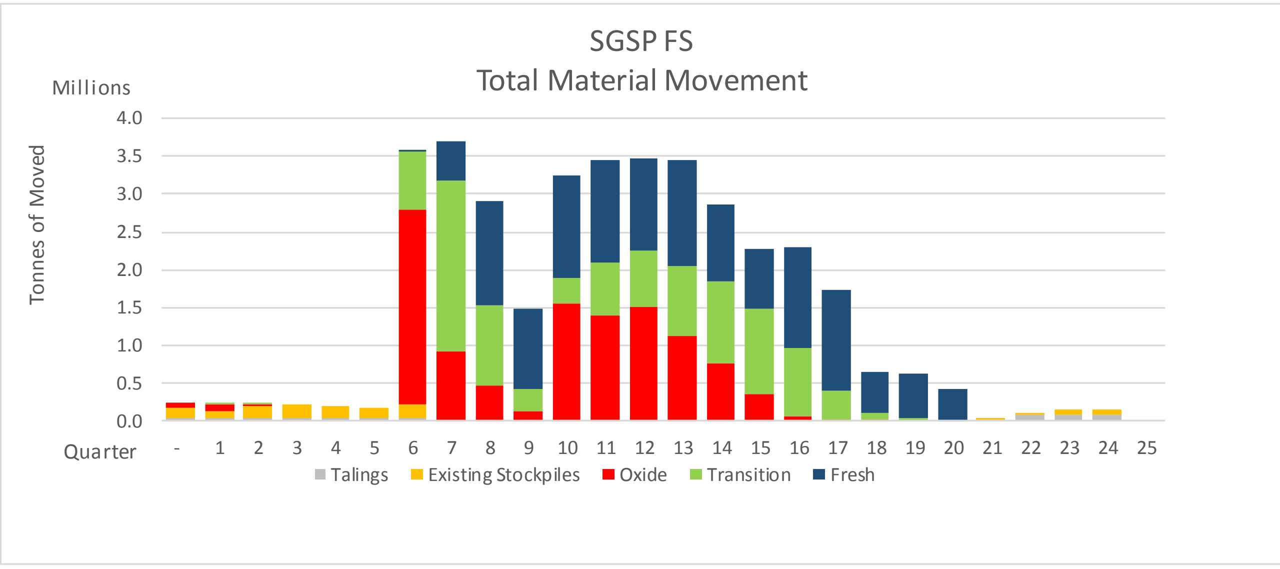
Table 1.9 Total mining inventory for scheduling (all Indicated Resources)

Item	Selinsing	Buffalo Reef South/Central	Buffalo Reef North	Stockpiles	Total
Oxide ore (kt)	51	351	80	1,287	**1,770**
Au (g/t)	0.66	1.20	1.14	0.49	**0.68**
Trans ore (kt)	43	641	73		**757**
Au (g/t)	1.61	1.78	1.31		**1.77**
Fresh ore (kt)	578	2,088	13	25	**2,704**
Au (g/t)	2.28	1.97	1.31	1.21	**1.96**
Total ore (kt)	672	3,080	167	1,312	**5,231**
Au (g/t)	2.11	1.84	1.23	0.51	**1.38**
Waste (kt)	7,125	28,409	468	-	**36,003**
Strip ratio (w:o)	10.60	9.22	2.81		**9.19**

1.4.3 Production scheduling

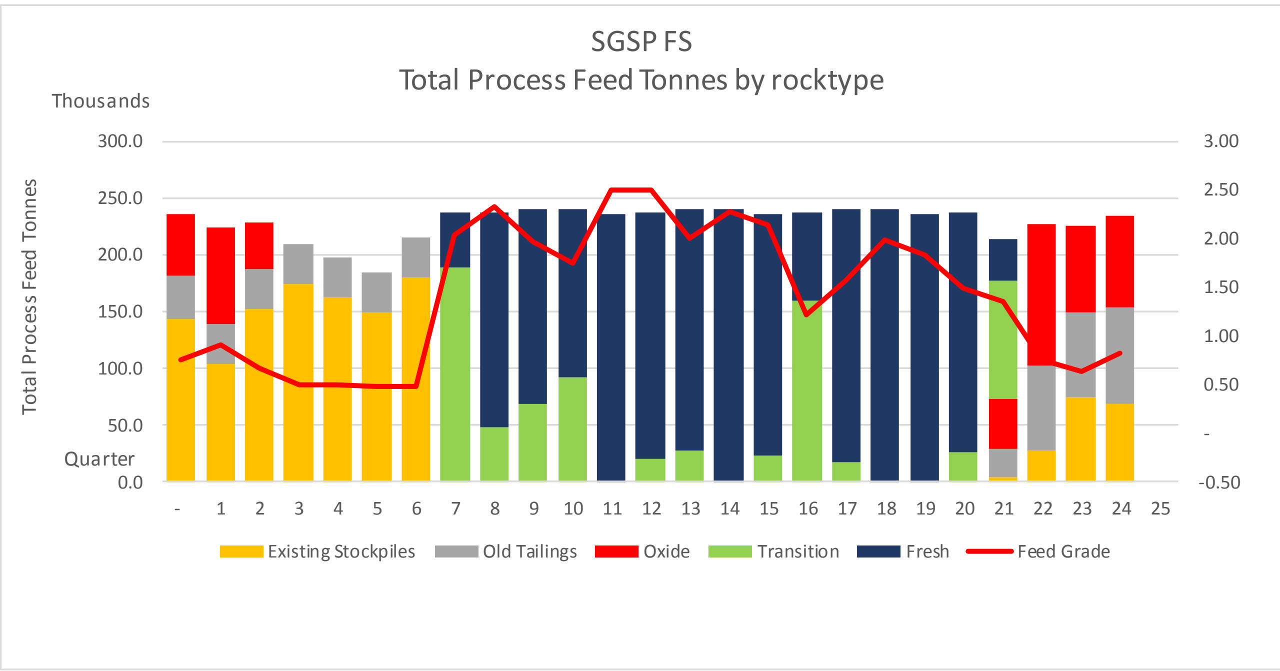
The first year of mining, prior to the sulphide circuit commencing, mines only a small amount of pit material, sourced mostly from the Buffalo Reef surface deposits. The total mining rate in this period is the equivalent of 1.5 million tonnes per annum (Mt/a). Just prior to the sulphide circuit commencing the mining rate ramps up to 5–10 Mt/a, with the initial focus of mining on the fresh portions of the Selinsing pits and the C2 Buffalo Reef pit. The more marginal pits (C3 and C4) are mined towards the end of the mine schedule. The total life of mining is approximately five years; stockpiles are then depleted to provide feed to the plant for the remaining project life. The movement schedule is shown in Figure 1.2.

Figure 1.2 Total SGM movement schedule by area



The process feed schedule is shown in Figure 1.3 for feed tonnes and grade. The first 1.5 years of production mines lower grade oxide resources from deposits and existing stockpiles. The sulphide circuit is commissioned and ramped up in Q9, prior to this point a dip in production occurs. During this time, high grade sulphide ore is stockpiled to avoid losing excessive gold to tails during commissioning. Following the ramp up, the grade increases significantly to bring forward cash flow. High grade (>2.0 g/t Au) feed is achieved for about nine quarters before dropping off to approximately 1.2 g/t Au. When the fresh and transition ore is exhausted in Q21, sulphide production is stopped, and oxide processing recommences from the remaining stockpiles.

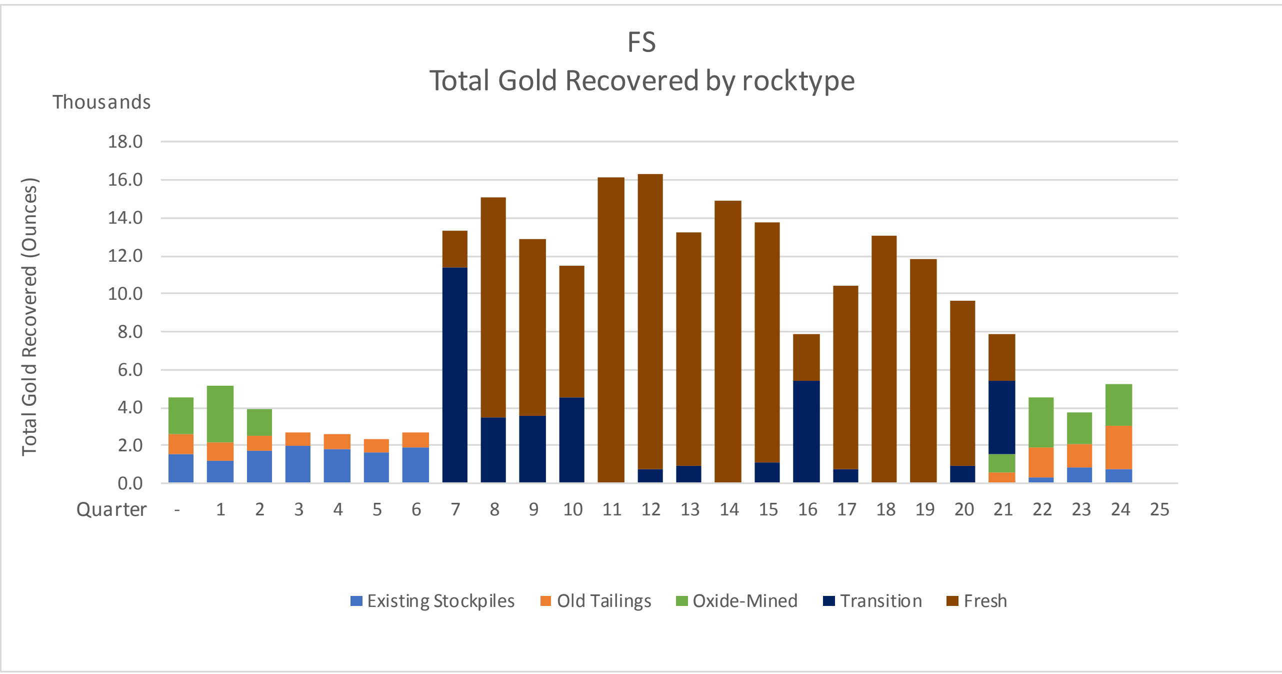
Figure 1.3 FS Process feed schedule



Production schedule

The gold production schedule shown in Figure 1.4 is quite variable. During initial oxide processing, the production rate varies approximately 5–15 thousand ounces per annum (koz/a). During the first phase of sulphide processing, the production increases to 60 koz/a before settling at approximately 30 koz/a. When sulphide processing stops, the production rate for remaining oxide stockpiles is approximately 20 koz/a.

Figure 1.4 FS Recovered gold schedule



1.5 Mineral Reserve estimates

Monument assessed the issues that affect the conversion of Mineral Resources to Mineral Reserves as modifying factors that are summarised in Table 1.10.

Table 1.10 Qualified Person's assessment of Mineral Reserve estimation for Selinsing and Buffalo Reef deposits and Selinsing stockpile

Item	Comment
Mineral Resource for conversion to Ore Reserve	Monument Mining Limited (Monument) prepared the updated Selinsing Mineral Resource estimate in March 2018. The Mineral Resource estimate was classified using CIM guidelines and a summary is provided below. No planned dilution was applied to these estimates. The Selinsing Mineral Resources comprise the Selinsing and Buffalo Reef deposits as well as existing stockpiles and are inclusive of Mineral Reserves. The Selinsing and Buffalo Reef Indicated Mineral and Tailings Resources and Measured Stockpile Resources used as a basis for the Mineral Reserves are summarised in Table 1.1 to Table 1.6. The Selinsing and Buffalo Reef Indicated Mineral Resources, measured stockpile and Indicated tailings Mineral Resources are inclusive of Selinsing and Buffalo Reef Probable Mineral Reserves and Proven stockpile and Probable tailings Mineral Reserves.
Site visits	A site visit to the Selinsing project site was undertaken by Mr Frank Blanchfield in March 2016. Mr Frank Blanchfield is the Mineral Reserves Qualified Person for the current NI 43-101 Technical Report. Mike Kitney is the Principal Consultant of Metallurgical Design and visited the Selinsing Property on multiple occasions between mid-2008 and mid-2016 inclusive and is Qualified Person for engineering design and process capital and operating cost estimation. Mike is also an independent Techical director of Monument Mining Ltd.

Item	Comment
Study status	The current NI 43-101 Technical Report is for a feasibility study (FS) to establish the viability of sulphide ore extraction through the extension of the existing oxide plant to incorporate additional sulphide ore extraction. This FS study includes: • Metallurgical review by Orway Mineral Consultants (OMC). • Upgraded processing plant for sulphide ore treatment using flotation and BIOX®; review by PIE and CES. • Geotechnical review by Peter O'Bryan and Associates (POB) (2018). • A review by Snowden of mining and geological aspects of this study. Previous studies include a prefeasibility study (PFS) by Snowden Mining Industry Consultants Pty Ltd (Snowden) and another study completed by Practical Mining LLC in 2012 for the extraction of sulphides from the Selinsing and Buffalo Reef deposits. Snowden re-evaluated this work in the PFS using reports from Lycopodium that updated the metallurgy costs and recoveries in 2016.
Cut-off parameters	A nominal cut-off grade of 0.40 g/t Au was applied to oxides and 0.75 g/t Au for sulphides and 0.35 g/t Au for tailings when developing the Mineral Reserve estimate, based on the economic cut-off grade.
Mining factors and assumptions	To identify the Selinsing and Buffalo Reef Mineral Reserve, a process of optimisation using the Deswik Pseudo Flow, staged pit design, production scheduling and mine cost modelling was undertaken by Monument. The mining method is conventional open pit drill and blast, load and haul on a 2.5 m mining flitch with a 10 m high blasting bench, reflective of semi-selective mining. The maximum excavator bucket size of 2.3 m^3 is matched to this selectivity. A stripping ratio of approximately 6 was identified. Overall, dilution assumption used has reduced the recovered ounces by approximately 2% and marginally increased the ore tonnage processed by 2%.
Metallurgical factors and assumptions	The Selinsing Gold Mine was originally developed on the basis of treating oxide ore via conventional crushing and ball milling followed by gravity recovery of free gold and cyanidation of gravity concentrate. Gravity tails are subjected to conventional carbon-in-leach (CIL). Final gold recovery from carbon strip solution and gravity concentrate leach solution is by electrowinning onto stainless steel cathodes. In 2009, mining operations commenced at Selinsing. Since then, Monument developed an open pit mine and construction of 1,200 tonnes per day gold treatment plant in three phases. The 2016 PFS proposed conventional CIL extraction from oxide ores and bio-leach for transition and fresh ores. Since that time, review as part of this study has indicated treatment of transition and fresh ores by floatation and then bio oxidation BIOX® followed by CIL. All the oxide unit processes included in the design are standard and common to many current gold operations, including: • Crushing • Grinding and classification • Gravity concentration (Knelson centrifugal concentrator) • Intense leaching (Acacia reactor) of gravity concentrate • CIL with cyanidation and carbon adsorption • Carbon desorption • Electrowinning • Smelting • Tailings disposal and effluent reclaim • Cyanide detoxification. For the sulphide fresh and transit ore treatment, the following has been done as part of the FS: • Process design criteria • Process design and flow diagrams • Engineering design criteria • Mechanical and electrical equipment lists • Process plant layout • Capital cost estimates. The metallurgical factors for sulphide were developed by Monument and in-house and independent testwork by Outotec and reviewed by OMC. The oxide metallurgical factors are from site data.

Item	Comment
	The metallurgical recovery parameters applied are: **Material treated / Deposit / Gold grade (g/t Au) / Recovery (%)** Old Tailings — Selinsing, Buffalo Reef — All — 74 Oxide — Selinsing, Buffalo Reef — All — 75 Transition — Selinsing: All, 85; Buffalo Reef: All, 85 Fresh/Sulphides — Selinsing: All, 85; Buffalo Reef: All, 85 It is the Qualified Person's opinion that the plant production numbers are accurate and correct. Given superior flotation response and bioleach response from metallurgical testwork conducted for Phase IV of the project, it is reasonable to assume that the results obtained and design criteria and process flowsheet adapted for Phase IV are reasonable and adequate for a FS level of accuracy.
Environmental	Rock characterisation was completed in Malaysia and potentially acid-forming acid rock drainage items were identified. The waste dumps are recommended to be designed at a final angle of 18° but the final landform designs will require completion prior to mining; however, Monument has verified that there is enough space for these designs. A cost provision has been made for the construction of the final land forms. Currently, an exploration licence is approved. An MLA (mining application licence) will be submitted for approval in October. The MLA allows provision for tailings dams and waste dumps.
Infrastructure	After extensive negotiating with the local authority for power purchase from the electricity grid, Monument has decided to construct a new powerline form Kuala Lipis as the only option for the increased reliable power requirement. Monument has indicated the bio-oxidation plant build will be an EPC execution with Monument providing the management. Accommodation will be in surrounding communities.
Cost and revenue factors	Process costs were estimated from historical oxide production costs at the Selinsing gold processing plant and sulphide operating costs are estimated primarily from Selinsing historical production data and market estimates, based on OMC and BIOX® processing design and CES engineering design. The estimated process unit costs are as follows: **Material treated / Deposit / Gold grade (g/t Au) / Process operating cost (US$/t)** Oxide — Selinsing, Buffalo Reef — All — 8.68^ Old Tailings — Selinsing, Buffalo Reef — All — 5.41* Transition* — Selinsing: All, 17.26; Buffalo Reef: All, 17.26 Fresh/Sulphides — Selinsing: All, 17.26; Buffalo Reef: All, 17.26 *^ 2018/19 FY planned oxide cost of US$8.68/t* ** 2018/19 FY planned oxide cost of US$5.41/t* Mining costs are based on historical data during high mining activity developed from the existing contract. The all-up mining operating cost was estimated to be US$1.99/t mined. The mining capital cost was absorbed by contract mining. Administration costs are fixed at US$1.72 million per year and are inclusive of mining, plant and all other administration costs including refining costs. Capital costs of US$52.93 million were estimated by Monument and others as follows: • Process capital costs: US$34.9 million • Power upgrade costs US$4.9 million • Mine rehabilitation totallingUS$0.6 million • Sustaining costs: totalling US$2.5 million • Mining capitalised costs including waste cutback and TSF construction, access roads and River diversion for Buffalo Reef totalling US$9.2 million • Old tailings process upgrade US$0.1 million • Communications and training US$0.8 million.

<table>
<tr><th>Item</th><th>Comment</th></tr>
<tr><td></td><td>All costs in US$.
Royalties averaging 10% plus tenement fees were applied to all gold produced, at expiry of each current tenement.</td></tr>
<tr><td>Revenue factors</td><td>Monument supplied a gold price of US$1,300/oz. This was applied as real and flat forward in the financial model.</td></tr>
<tr><td>Market assessment</td><td>Monument has completed comprehensive market studies, including likely refiners.
Gold is freely traded, and the price is set by the LME. A comprehensive marketing study was completed as part of the PM LLC 2013 NI 43-101 Technical Report. The selling of gold is straight forward.</td></tr>
<tr><td>Economic</td><td>The discount rate in the Monument financial model was set at 8%.
A financial sensitivity study was undertaken to evaluate capital expenditure, operating costs and gold price. The project was found to be most sensitive to changes in gold price.
The key performance indicators from the Monument model are summarised below:
<table>
<tr><th>Key performance indicator</th><th>Units</th><th>Value</th></tr>
<tr><td>All-in cash cost (including royalty)</td><td>US$/oz produced</td><td>863.67</td></tr>
<tr><td>IRR ungeared</td><td>%</td><td>49</td></tr>
<tr><td>NPV (at 8%)</td><td>US$M</td><td>27.56</td></tr>
<tr><td>Net cash flow</td><td>US$M</td><td>44.55</td></tr>
<tr><td>Initial capital cost[a]</td><td>US$M</td><td>39.77</td></tr>
</table>
[a] Excludes working capital</td></tr>
<tr><td>Social</td><td>A socio-economic study was prepared by Monument. The commentary provides a summary of the socio-economic characteristics of the area at a household level. Monument has a full-time Community Relations Officer engaged in maintaining open communications with the local communities. Monument has advised there are no community or social encumbrances that could obstruct the provision of an MLA from the Malaysian government.</td></tr>
<tr><td>Classification</td><td>The Mineral Reserve is classified as Proven and Probable in accordance with the CIM Code, corresponding to the Mineral Resource classification of Measured for stockpiles and Indicated for ore sources from pit ore material.
No Inferred Resources are included in the Mineral Reserve estimate.</td></tr>
<tr><td>Audits or reviews</td><td>Snowden has completed an internal peer review of the Mineral Reserve estimate.</td></tr>
<tr><td>Relative accuracy/ confidence</td><td>It is Snowden’s opinion that the Mineral Reserve classification of “Probable” for the deposits and Proven for the stockpiles is reasonable. The lower Probable confidence in this estimate is attributed to the use of Indicated Resources.</td></tr>
</table>

1.5.1 Mineral Reserves reporting

Selinsing and Buffalo Reef deposits – Mineral Reserve statement

The Mineral Reserve estimate for the Selinsing and Buffalo Reef deposits, as at the end of March 2018, is provided in Table 1.11. Note that tonnes and ounces have been rounded and this may have resulted in minor discrepancies.

Table 1.11 Mineral Reserves as at end of March 2018

Classification	Oxidation	Zone	Approximate cut-off (g/t Au)	Tonnes (kt)	Au (g/t)	Ounces (koz)
Probable	Oxide	Selinsing	0.4	51	0.66	1.1
		BRN	0.4	80	1.14	3.0
		BRC	0.4	77	0.90	2.2
		Felda	0.4	233	1.34	10.0
		BRS	0.4	41	0.93	1.2
	Oxide – Total			**483**	**1.13**	**17.5**
	Transitional	Selinsing	0.75	43	1.61	2.2
		BRN	0.75	73	1.31	3.1
		BRC	0.75	171	1.11	6.1
		Felda	0.75	171	1.66	9.1
		BRS	0.75	299	2.23	21.4
	Transitional – Total			**757**	**1.72**	**41.9**
	Fresh	Selinsing	0.75	578	2.28	42.4
		BRN	0.75	13	1.31	0.6
		BRC	0.75	699	1.78	40.0
		Felda	0.75	476	1.79	27.5
		BRS	0.75	913	2.21	64.8
	Fresh – Total			**2,680**	**2.03**	**175.1**
PROBABLE – TOTAL				**3,919**	**1.86**	**234.6**

Selinsing property stockpile – Mineral Reserves statement

Mineral Reserves for the stockpiles at the Selinsing project, as at the end of March 2018, are summarised in Table 1.12. Note that tonnes and ounces have been rounded and this may have resulted in minor discrepancies.

Table 1.12 Stockpile Mineral Reserves as at end of March 2018

Stockpile name	Stockpile ID	lcm	Tonnes (kt)	Au (g/t)	Ounces (koz)
Oxide Stockpiles					
High Grade 1 (Oxide)	SEL HG1 O	-	-	-	-
Low Grade 1 (Oxide)	SEL LG1 O	6,081	12	1.1	0.4
Low Grade 2 (Oxide)	SEL LG2 O	3,189	6	0.73	0.2
Super Low Grade 1 (Oxide)	SEL SLG1 O	2,191	4	0.44	0.1
Super Low Grade 2 (Oxide)	SEL SLG2 O	72,179	991	0.43	13.9
Super Low Grade 4 (Oxide)	SEL SLG 4	36,293	78	0.50	1.3
BR High Grade 1 (Oxide)	BR HG1 O	566	1	3.28	0.1
BR Low Grade 1 (Oxide)	BR LG1 O	928	2	1.02	0.1
BR Super Low Grade 1 (Oxide)	BR SLG1 O	84,030	168	0.55	3.0
B7 High Grade 1 (Oxide)	B7 HG1 O	1,767	3	2.12	0.2
B7 Low Grade 1 (Oxide)	B7 LG1 O	77	0.13	1.02	0.01
B7 Super Low Grade 1 (Oxide)	B7 SLG1 O	449	0.73	0.69	0.01
Oxide – Total		**607,750**	**1,265**	**0.47**	**19.1**
Leachable Sulphide Stockpiles					
High Grade 1 (Leachable Sulphide)	SEL HG1 S	-	-	-	-
Low Grade 1 (Leachable Sulphide)	SEL LG1 S	-	-	-	-
BR High Grade 1 (Leachable Sulphide)	BR HG1 S	879	1.776	1.86	0.1
BR Low Grade 1 (Leachable Sulphide)	BR LG1 S	1237	2.499	0.97	0.1
High Grade 2 (Leachable Sulphide)	SEL HG2 S	-	-	-	-
Low Grade 3 (Leachable Sulphide)	SEL LG3 S	213	0.46	0.68	0.01
Low Grade 4 (Leachable Sulphide)	SEL LG4 S	-	-	-	-
BR High Grade 2 (Leachable Sulphide)	BR HG2 S	5186	11.399	2.72	1.0
BR Super Low Grade 1 (Leachable Sulphide)	BR SLG1 S	2145	4.333	0.55	0.1
B7 High Grade 1 (Leachable Sulphide)	B7 HG1 S	-	-	-	-
B7 Low Grade 1 (Leachable Sulphide)	B7 LG1 S	-	-	-	-
B7 Super Low Grade 1 (Leachable Sulphide)	B7 SLG1 S	-	-	-	-
Leachable Sulphide – Total		**9,660**	**20.5**	**1.92**	**1.3**
Non-Leachable Sulphide Stockpiles		0	0		0
BR Low Grade 2 (Non-Leachable Sulphide)	BR LG2 S	11570	23.115	1.25	0.9
Super Low Grade 3 (Non-Leachable Sulphide)	SEL SLG3 S	748	1.511	0.6	0.03
B7 High Grade 2 (Non-Leachable Sulphide)	B7 HG2 S	-	-	-	-
B7 Low Grade 2 (Non-Leachable Sulphide)	B7 LG2 S	-	-	-	-
Non-Leachable Sulphide – Total		**12318**	**24.626**	**1.21**	**0.9**
GRAND TOTAL		**630,000**	**1,312**	**0.51**	**21.3**

Note: All stockpiles classified as Proven Mineral Reserves; lcm = loose cubic metres

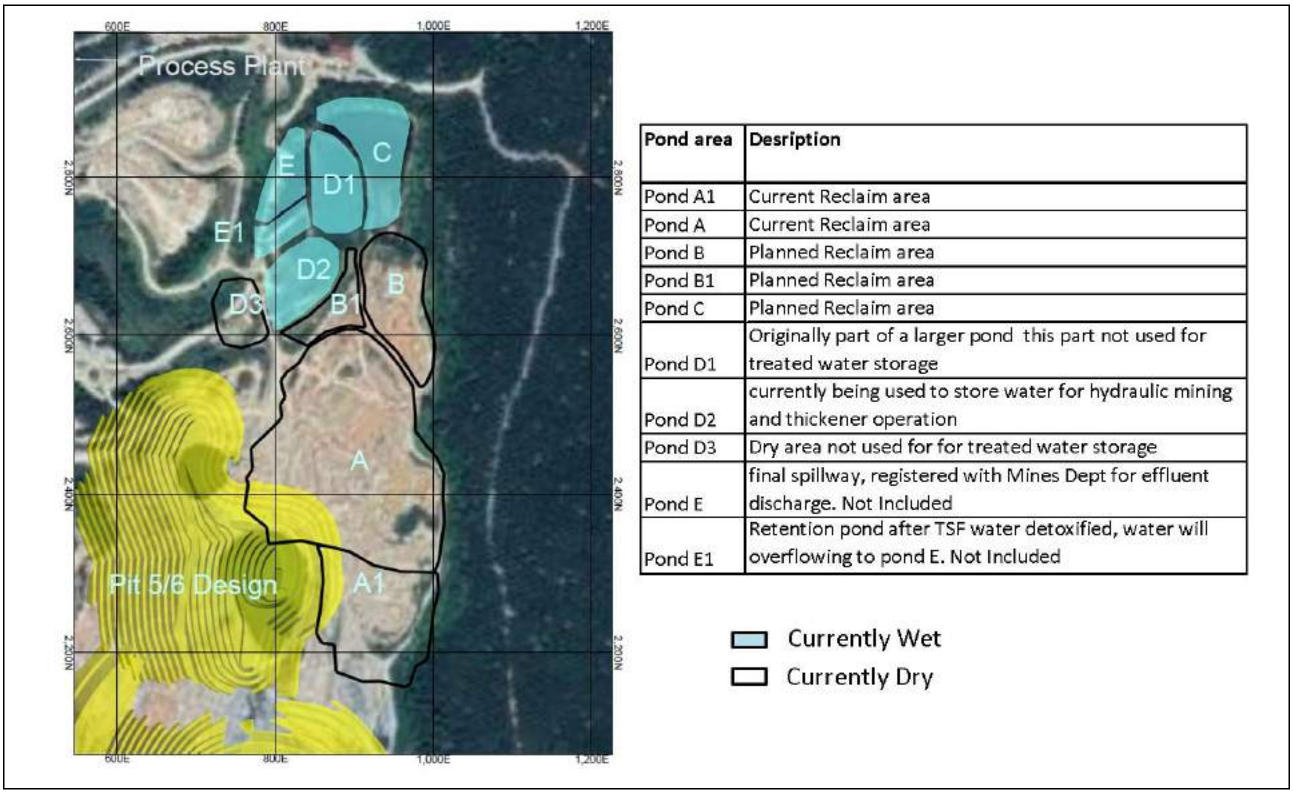
Selinsing property tailings – Mineral Reserve statement

The reclaim tailings potential at the SGM is referred to as “Old Tailings”. Current tailings are deposited in the tailings storage facility (TSF) are discussed in Section 1.7.7.

The “Old Tailings” are processed by excavation and haulage to a dry stockpile, then transferred to a wet pond area for separation and slurry formation by water cannon. The material is then classified and fed to the CIL plant. Figure 1.5 showing the Old Tailings area and pond formation.

Modifying factors include historical reconciliation to plant, pond bund depletion and removal of areas below designated treated water storage areas required for environmental management and legal responsibility. The modifying factors are applied to the resource model used for the Old Tailings Indicated Resource are described in Section 1.3.2. Table 1.13 shows the Probable Old Tailings Reserve.

Figure 1.5 Selinsing Old Tailings area



Pond area	Desription
Pond A1	Current Reclaim area
Pond A	Current Reclaim area
Pond B	Planned Reclaim area
Pond B1	Planned Reclaim area
Pond C	Planned Reclaim area
Pond D1	Originally part of a larger pond this part not used for treated water storage
Pond D2	currently being used to store water for hydraulic mining and thickener operation
Pond D3	Dry area not used for for treated water storage
Pond E	final spillway, registered with Mines Dept for effluent discharge. Not Included
Pond E1	Retention pond after TSF water detoxified, water will overflowing to pond E. Not Included

Table 1.13 Old Tailings Mineral Reserves as at end of March 2018

Classification	Volume (m^3)	Tonnes (kt)	Au (g/t)	Ounces (koz)
Probable	353,000	508	0.71	12

1.6 Metallurgical testwork and design

The SGM was originally developed on the basis of treating oxide ore via conventional crushing and ball milling followed by gravity recovery of free gold and cyanidation of gravity concentrate. Gravity tails are subjected to conventional carbon-in-leach (CIL). Final gold recovery from carbon strip solution and gravity concentrate leach solution is by electrowinning onto stainless steel cathodes. In 2009, mining operations commenced at Selinsing. Since then, Monument developed an open pit mine and construction of a 1,200 tonnes per day (t/d) gold treatment plant in three phases:

- Phase I: Process plant consisting of a gravity circuit which commenced in August 2009 with the first gold pour in October 2009. The gold treatment plant with 400,000 t annual capacity was fully completed in December 2009.
- Phase II: The CIL circuit was fully commissioned by August 2010. Total capital cost for Phase I and Phase II was US$18.2 million.
- Phase III: This expansion began on 6 September 2011 with a budget of US$8.1 million and was completed in June 2012 on time and at a cost of US$8.6 million. The Phase III expansion increased the capacity of the gold treatment plant from 400,000 tonnes per annum (t/a) to approximately 1,000,000 t/a. The 2012 plant expansion followed the identification of further Mineral Reserves in three pits at Selinsing, being the southernmost Pit 4, Pit 6 which is the northernmost pit, and Pit 5 between Pit 4 and Pit 6. The Selinsing pits were designed by consultants with Snowden and Monument staff mine engineers.

- FS work of 2018. This study replaces the previous Phase IV Study of Snowden (Dec 2016) NI 43 101 Technical Report.

From 2011, Monument has been engaged in Phase IV of the expansion, with key areas of evaluation being:

- Inspectorate of Vancouver, metallurgical test program on a selection of diamond drill core material collected from the Buffalo Reef deposit.
- Engineering study by Lycopodium of Brisbane, Australia, and reported by Lycopodium in "Selinsing Phase IV Study" (Feb 2013). This study is summarised in the Snowden (Dec 2016) PFS.
- FS work of 2018. This study replaces the previous Phase IV Study of Snowden (Dec 2016) NI 43-101 Technical Report.

The Qualified Person has provided the summary and write-up for this section based on previous NI 43-101 reports for oxide processing only. Additionally, the sulphide processing utilising the BIOX® and flotation processes is evaluated in this Technical Report.

Testwork was carried during the FS to enable both BIOX® and flotation process designs to be completed. This testwork including:

- Detailed testwork was carried out by Outotec at the SGS laboratories in Johannesburg, South Africa, during 2017 to establish the suitability of the BIOX® process for subsequent gold recovery from the refractory Selinsing sulphide ores. This testwork including variability testwork formed the basis of the BIOX® process design developed by Outotec as part of the FS.
- Detailed testwork both in-house and by Bureau Veritas Laboratory in Perth supervised by OMC to enable flotation design

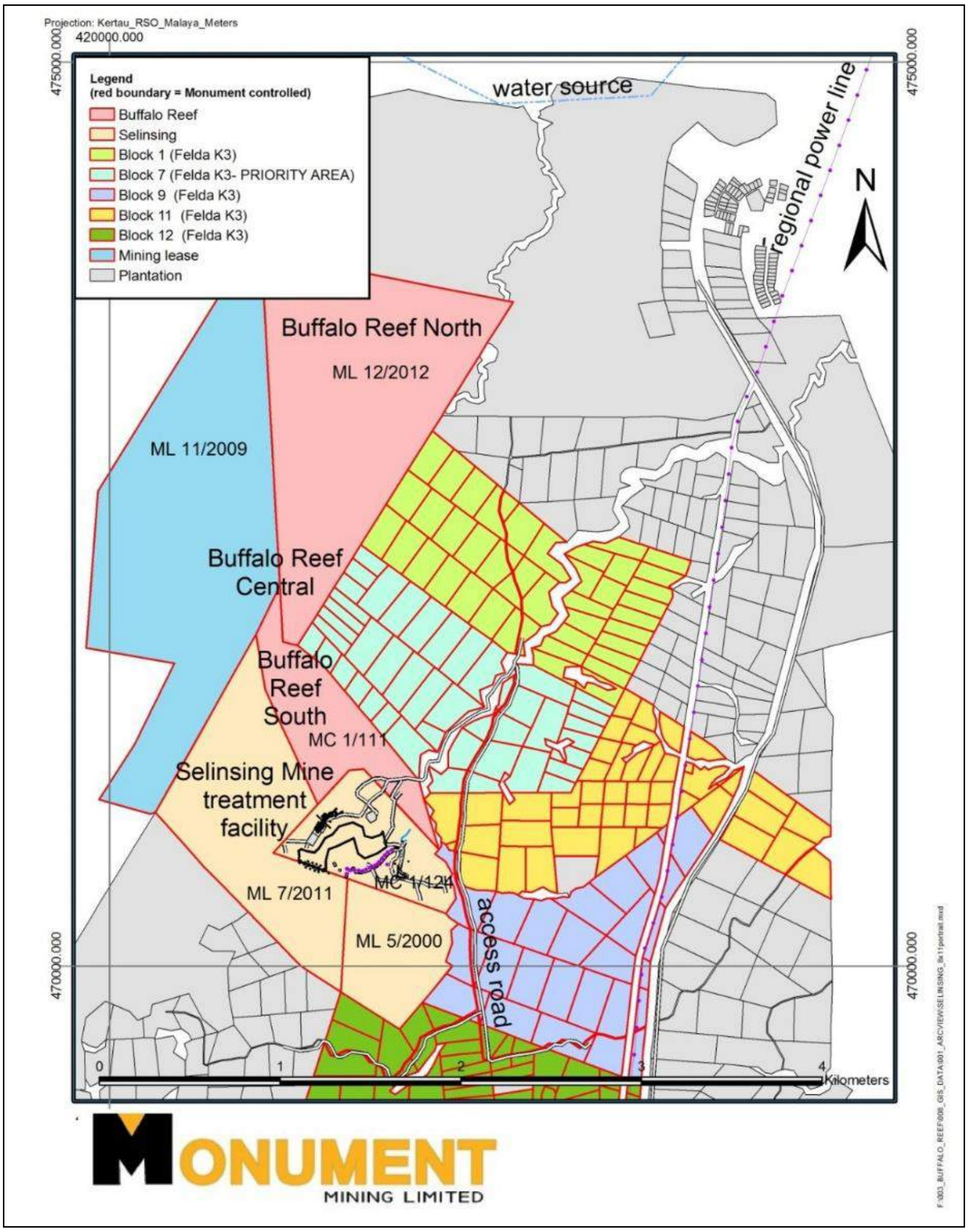
The above enabling process design criteria to be developed by OMC and plant engineering designs largely completed by CES.

It is the Qualified Person's opinion that the plant production numbers are accurate and correct. Given superior flotation response and bioleach response from metallurgical testwork conducted for Phase IV of the project, it is reasonable to assume that the results obtained, and design criteria and process flowsheet adapted for Phase IV are reasonable and adequate for a FS level of accuracy.

1.7 Project infrastructure

A map of the mine area showing current infrastructure is provided in Figure 1.6.

Figure 1.6 Location map showing roads, water and power



1.7.1 Electrical power

A 33 kV national grid powerline runs past the leases. Power is allocated by the provider and the agreed offtake is currently 3.8 MW. The new expected demand for the plant with a bio-oxidation plant is 10.3 MW. Hence additional power is required.

A new 33 kV dedicated powerline will be installed by Tenaga Nasional Berhad (TNB) from Kuala Lipis to Selinsing site. SGM will construct the switch room at the site main gate to TNB design. The switch room will house the TNB 33 kV metering equipment and the 33 kV SSU high voltage equipment.

This project with anticipated cost of US$4.87 million will take some 18 months to complete from inception. A 1.5 MW diesel generator set will be used temporarily to power Project Office, Site Accommodation, Main Administration Building as well as construction equipment (i.e. welding sets, blasting and painting machine etc.).

1.7.2 Water

During initial plant operation, water from Selinsing Pit 4 will be pumped out and used for plant usage. The water recovery thickener will be used to recover high proportion of process water which will be pumped to the Process Water Tank. Raw water will still be pumped from Sungai Kermoi. Tailings decant return water will be discharged to the environment through the existing detoxification system.

1.7.3 Disposal and drainage

Sewerage disposal with an existing arrangement will be used for waste disposal by local contractors. The existing TSF will be used for the joint flotation tailings, neutralisation tailings and ASTER™ solid tailings as water recovery thickener underflow. Stormwater drainage will be designed according to Malaysian Department of Irrigation and Drainage (DID) standards for 100-year events.

1.7.4 Transport infrastructure

The road from the main entrance leading to the plant area need to be upgraded with crushed limestone and soil.

1.7.5 Communications

Upgrades for internet connectivity to 50 MB/s during construction and 20 MB/s post construction are planned together with a site radio system for construction.

1.7.6 Mining personnel

Mine personnel consist of expatriates and in-country professionals. Labour employment covenants requiring 50% of all employees to be Bumiputra (Malays and other indigenous peoples). There are many universities throughout Malaysia including Kuala Lumpur, Selangor, Penang and Sabah where many of the site professionals have sourced tertiary training in metallurgy, engineering and geology. There has been an employment survey conducted by Monument. There is currently a good availability of professional and non-professional roles for the Selinsing gold operation.

1.7.7 Tailings storage areas

SRK Consulting (SRK) was appointed to undertake a scoping level design to increase the capacity of the existing TSF for the remaining LOM of seven years. A centreline raise of 9 m to a final elevation of 540 m RL was recommended. The embankment zoning will comprise an upstream low permeability zone (Zone A), an internal drainage zone (Zone F) and a downstream structural fill zone (Zone C) – refer SRK (Feb 2018).

A program of geotechnical investigation was recommended to address stability analysis gaps including the testing of tailings, borrow material, embankment fill and waste dumps (buttress at saddle dam and south dam).

SRK will design the first raise of the TSF to an elevation of 533 m RL or similar. The initial design will include new drainage works to tie in with existing system. To minimise construction cost in any given year successive TSF lifts will likely be carried in small increments of 2 m or so to stay ahead of the rate of rise of the tailings beach and provide enough freeboard in line with safe practice and to satisfy regulatory compliance.

1.7.8 Waste disposal areas

The pit oxide waste as waste rock is all considered to be inert. Initial characterisation testwork shows that there are potentially acid forming (PAF) waste rocks in the transition and sulphide rocks, so for this study all the PAF waste is encapsulated in oxide with the expectation there will be acid forming minerals. The management of these dumps will need to be further identified prior to production.

1.8 Process plant construction

The layout of the new flotation plant and BIOX® facilities shown in Figure 1.7 has been designed for the minimum of disruption to ongoing operations. Most of the BIOX® facilities will be located over an open area to the north and west of the existing facilities.

Earthwork requirements for the plant expansion will include the excavation of the area to be occupied by the limestone mill, the crushed limestone stockpile area and the new lime silo. A new access road to the TSF and cooling towers will be excavated alongside the pipeline corridor. Excavated material will be used to backfill the fish pond behind the plant office and to raise this area to the same elevation as the existing process plant.

Figure 1.7 Overall plant layout

1.9 Environmental studies, permitting, and social or community impact

As part of the Mineral Reserve assessment, Snowden (Dec 2016) assessed key items as part of an environmental review of the SGM, based on information provided by Monument. The following items were reviewed:

- Environmental approvals and permits (existing and any further requirements)
- Key environmental impacts associated with existing operations and the planned project expansion
- Social and community impacts.

Snowden (Dec 2016) mentioned *"based on the information provided for this environmental review, it is not anticipated that the Selinsing Gold Mine Project for Phase IV, including Buffalo Reef project development and bio-oxidation processing, will be significantly delayed or impacted by environmental approval or permitting concerns, or significant environmental compliance issues. This expectation is underpinned by solid environmental compliance and performance to date, and the lack of significant environmental incidents or regulatory and community concern over the mining operations to date. The non-compliances against Environmental Impact Assessment (EIA) approval conditions identified during past site audits have been or are being addressed.*"

Current information available for the FS indicates the aforementioned environmental summary is still current.

The current Environmental Management Plan, as required by the Environmental Impact Assessment, was approved by the Department of Environment in March 2016. Snowden (Dec 2016) recommended the EMP should be updated to include environmental compliance for Phase IV technology. They also recommended an entire Environmental Management System is established for the project in line with a recognised standard such as ISO14001 Standard for Environmental Management Systems. This will facilitate the identification and management of environmental risks and help to ensure environmental compliance management requirements are achieved.

One important risk is the inadvertent spill of BIOX® bacteria. The updated Environmental Management System includes a spill monitoring and management plant for this risk.

1.10 Capital and operating costs

1.10.1 Cost estimation battery limits

The capital and operating costs for the Project are based on the mining, processing and sale of gold doré. The process rates of 0.95 Mt/a for both oxide and sulphide for CIL and BIOX® sulphide process have been used. All costs in this section are in US dollars (US$).

1.10.2 Operating cost sources

The mining operating costs are from the mining contract and the actual mining costs are used from production or mining costs are applied going forward from the contract schedule of rates. The process operating costs are based on plant actuals for the oxide operation as there are several years of production history and costs. The process operating costs for sulphide were estimated by Monument in 2018 with review by Mike Kitney. Administration costs were compiled by Monument from the 2019 budget.

All cost estimates assumed the MYR/US$ exchange rate assumed was 4.1037 based on the previous six months average based on Bank Negara Malaysia foreign exchange (FX) rates.

1.10.3 Capital cost sources

The oxide processing plant will continue processing stockpiled oxide materials until the BIOX® plant is commissioned in Q8. Capital costs from the 2017 budget were used for the sustaining capital and the capital cost estimate for the plant was estimated by Monument and CES with review by Plant and Infrastructure Engineering (PIE).

The sulphide plant pre-production capital costs in the cash flow model have been allocated 8% in Year 1 and 92% in Year 2 of the one year of construction.

The capital and operating cost estimates were prepared or advised by the following groups:

- Sulphide process plant: Monument (reviewed by Mike Kitney and OMC)
- EPC management: Monument (reviewed by Mike Kitney)
- Mining: Monument (reviewed by Snowden)
- Oxide process plant: Monument (reviewed by Mike Kitney and OMC)
- On-site infrastructure: Monument (reviewed by Mike Kitney)
- Off-site infrastructure: Monument (reviewed by Mike Kitney)
- Environmental: Monument
- Social: Monument
- Corporate G&A: Monument
- Royalties: Monument
- Taxation: Monument.

1.10.4 Capital costs

The capital costs have been estimated following a series of indicative prices received from major equipment manufacturers and suppliers. All capital costs have been reviewed by Mike Kitney who is the Qualified Person for the capital cost estimates.

Import duties on capital equipment not sourced but manufactured in Malaysia, are high and as such, work has been done to minimise or where possible eliminate importing items not supplied originally or manufactured in Malaysia. On advice from Monument, no import duties have been applied on capital equipment. No escalation has been applied to the capital cost estimate.

Plant construction estimate

The plant construction cost has been estimated by Monument with review by Mike Kitney and shown in Table 1.14.

Table 1.14 Summary of plant construction capital cost estimate

Major Component	Total/Area (US$) initial	Ongoing (US$)	Contingency (US$)	Contingency %	TOTAL (US$)
Flotation	3,650,297		365,030	10%	4,015,326
BIOX®	13,934,059	600,000	1,393,406	10%	15,927,465
BIOX® – CIL	4,020,542		402,054	10%	4,422,596
Detoxification	50,000		5,000	10%	55,000
ASTER™ process	2,016,853		201,685	10%	2,218,538
Air/Water/Plant services	280,931		28,093	10%	309,024
Piping	866,743		86,674	10%	953,417
Other plant upgrades	898,825		26,407	2.94%	925,232
Detailed engineering	2,820,000			nil	2,820,000
First fill	1,630,307		163,031	10%	1,793,337
Owner cost/EPCM	1,298,315		129,832	10%	1,428,147
Plant Construction – Total	**31,466,671**	**600,000**	**2,801,212**	**8.9%**	**34,868,083**

Total capital cost estimates

The other most significant capital cost is the electrical power cost upgrade at US$4.87 million. Table 1.15 shows the total capital cost estimate.

Table 1.15 Total capital cost estimate

Item	Total/Area (US$M) Initial	Ongoing (US$M)	Contingency (US$M)	TOTAL (US$M)	Comments
FLOTATION					Estimates by CES and Monument reviewed by Mike Kitney
320 and 330 – Flotation and conc. thickener area	3.65		0.37	4.02	
BIOX®					Estimates by CES and Monument reviewed by Mike Kitney
340 – BIOX®	5.16		0.52	5.68	
350 – CCD	2.11		0.21	2.32	
360 – Neutralisation	1.39		0.14	1.53	
370 – Water recovery thickener	0.46		0.05	0.51	
380 – BIOX® services	4.49		0.45	4.94	
410 – pH adjustment	0.22		0.02	0.24	
Infrastructure BIOX® water	0.09		0.01	0.10	

Item	Total/Area (US$M) Initial	Ongoing (US$M)	Contingency (US$M)	TOTAL (US$M)	Comments
supply					
BIOX® licensing		0.60		0.60	Estimates by Outotec reviewed by Mike Kitney
420 – BIOX® CIL	1.63		0.16	1.79	
820 – Reagents	2.35		0.23	2.58	
900 – Tailings	0.04		0.01	0.05	
DETOXIFICATION					Estimates by CES and Monument reviewed by Mike Kitney
901 – Detoxification	0.05		0.01	0.06	
ASTER PROCESS					Estimates by CES and Monument reviewed by Mike Kitney
920 – ASTER™ (use existing tanks and equipment)	0.20		0.02	0.22	
Infrastructure water services	1.82		0.18	2.00	
AIR SERVICES					Estimates by CES and Monument reviewed by Mike Kitney
Air services	0.16		0.02	0.17	
Instrument air	0.12		0.01	0.13	
Piping rack	0.87		0.09	0.95	
DETAILED ENGINEERING					Estimates by Monument reviewed by Mike Kitney
Plant detailed design	2.82			2.82	
OTHER PLANT UPGRADES					Estimates by CES and Monument reviewed by Mike Kitney
Upgrade existing crushing and milling circuits	0.12			0.12	
Infrastructure Control Room + power distribution	0.01		0.00	0.02	
Process plant amenities	0.05			0.05	
Bulk earthworks – plant site filled	0.41			0.41	
Warehouse (estimated, see Structural Worksheet)	0.25		0.03	0.28	
Workshop	0.05			0.05	
TRAINING					Estimates by Monument reviewed by Mike Kitney
BIOX® test facility for operator and technical team training (Pilot Plant)	0.75			0.75	
Flotation cells training	0.00			0.00	
CAPITAL SUSTAINING COSTS					Estimates by Monument reviewed by Mike Kitney
Capital sustaining cost		2.50		2.50	Assumed US$600,00 per annum full year production + US$100,000 per annum first year
Mine rehabilitation cost		0.60		0.60	Estimation by Monument US$100,000 per annum
TAILING STORAGE FACILITIES					
TSF construction		0.9		0.9	Estimation by Monument – costed against capital waste
POWER UPGRADE					

Item	Total/Area (US$M) Initial	Ongoing (US$M)	Contingency (US$M)	TOTAL (US$M)	Comments
Infrastructure HV power supply	4.43		0.44	4.87	Estimation by Monument reviewed by Mike Kitney
COMMUNICATION					
Internet connectivity					
CCTV					
To install CCTVs in process plant	0.04			0.04	Estimation by Monument
STRIPPING AND CUTBACK					
Selinsing and Buffalo Reef pit waste mining capital		7.66		7.66	Estimation by Monument
RIVER DIVERSION (BR)					
River diversion BRC3		0.50		0.50	Estimation by Monument
OTHER INFRASTRUCTURE					
Upgrade mine access road for project construction and plant operation	0.12			0.12	Estimation by Monument
FIRST FILL					
First fill and spares	1.63		0.16	1.79	Estimates by Monument reviewed by Mike Kitney
OWNER COST					
Owners costs	0.81		0.08	0.89	Estimates by Monument reviewed by Mike Kitney
Contractors overheads	0.23		0.02	0.26	Estimates by Monument reviewed by Mike Kitney
Engineering by CES					
Insurances and statutory fees (0.48% of total project value; suggested by PIE)	0.26		0.03	0.28	Estimates by Monument reviewed by Mike Kitney
Old Tailings upgrade	0.12			0.12	Estimation by Monument
TOTAL (US$)	**36.80**	**12.76**	**3.24**	**52.81**	

1.10.5 Operating costs

Process operating costs

For the processing of oxides and sulphides, Table 1.16 lists the unit costs as determined by Monument and reviewed by Mike Kitney.

Table 1.16 Operating costs for oxide and sulphide material treated (after Phase IV, BIOX® expansion)

Material treated	Deposit	Gold grade (g/t Au)	Process operating cost (US$/t)
Old Tailings	Selinsing	All	5.41
Oxide (treated by CIL only after sulphide processing completion)		<1.0	8.68
		1.0 to 1.5	8.68
		1.5 to 2.5	8.68
		>2.5	8.68
Transition	Selinsing	All	17.26
	Buffalo Reef	All	17.26
Fresh/Sulphides	Selinsing	All	17.26
	Buffalo Reef	All	17.26

Mining operating costs

The mining operating costs were developed by Monument for the project. These are largely based on historical costs for 2017 as they represented historical periods of high mining activity including fresh rock mining in the Selinsing pit area. These base costs were then used with an additional vertical increment haulage cost of US$0.0006/t/m and for drill and blast in fresh rock of US$0.2594/t for waste and US$0.2529/t for ore. These are based on estimates from the Minetech mining contract.

Based on the study FX assumptions, the overall mining cost was US$1.99/t mined.

1.10.6 Administration costs

The 2018/2019 budgeted administration cost of US$1.715 million per year was used as an annual fixed cost for site administration. This including all processing, mining and selling costs.

1.10.7 Royalty

Pursuant to an amendment to the Mineral Regulations 2005, which comes to effect on June 18, 2015, the royalty rate of gold has been increased from 5% to 10%, applying to any newly granted and renewed tenements after the effective date. The industry has appealled against this increase and it is now under review by the Pahang State government. Monument has used a conservative scenario of 10% royalty rate as well as associated fees in its cost evaluation.

The Buffalo Reef leases carry an additional 2% royalty payable to Pahang State Development Corporation (or in Malay language, “Perbadanan Kemajuan Negeri Pahang (PKNP))”.

1.11 Economic evaluation

Snowden relied on Monument for the economic evaluation outputs, which is based on estimated mineral reserves.Table 1.17 shows a full project summary for the FS including financial evaluation.

Table 1.17 FS Economic evaluation and project summary based on mineral reserve estimation inclusive of Oxide and Tailing material

Financial parameter	Value	Units
Costs		
Capital cost		
Plant construction	34.87	US$M
Other initial capital areas	4.91	US$M
Mining capital	7.66	US$M
Other capital	5.49	US$M
Total capital	**52.93**	**US$M**
Operating costs		
Mining operational	71.98	US$M
Processing operational	78.03	US$M
Site administration	10.31	US$M
Royalty	31.22	US$M
Total operating	**191.54**	**US$M**
Corporate tax	**1.28**	**US$M**
Unit costs		
Mining	1.99	US$/t mined
Processing	13.60	US$/t processed
Cash cost/ounce	**863.67**	**US$/oz**
Revenue	**290.31**	**US$M**
Operating expenditure	**191.54**	**US$M**
Cash flow from operations (EBIT)	**98.77**	**US$M**
Corporate tax	**11.73**	**US$M**
Capital investment tax credit	**10.44**	**US$M**
Operating cash flow after tax	**97.48**	**US$M**
Capital expenditure	**52.93**	**US$M**
Cash flow, net	**44.55**	**US$M**
Discount rate	8	%
NPV	**27.56**	**US$M**
IRR	49	%
Payback	2.5	years
ROIC	1.11	ratio

1.11.1 Sensitivity analysis

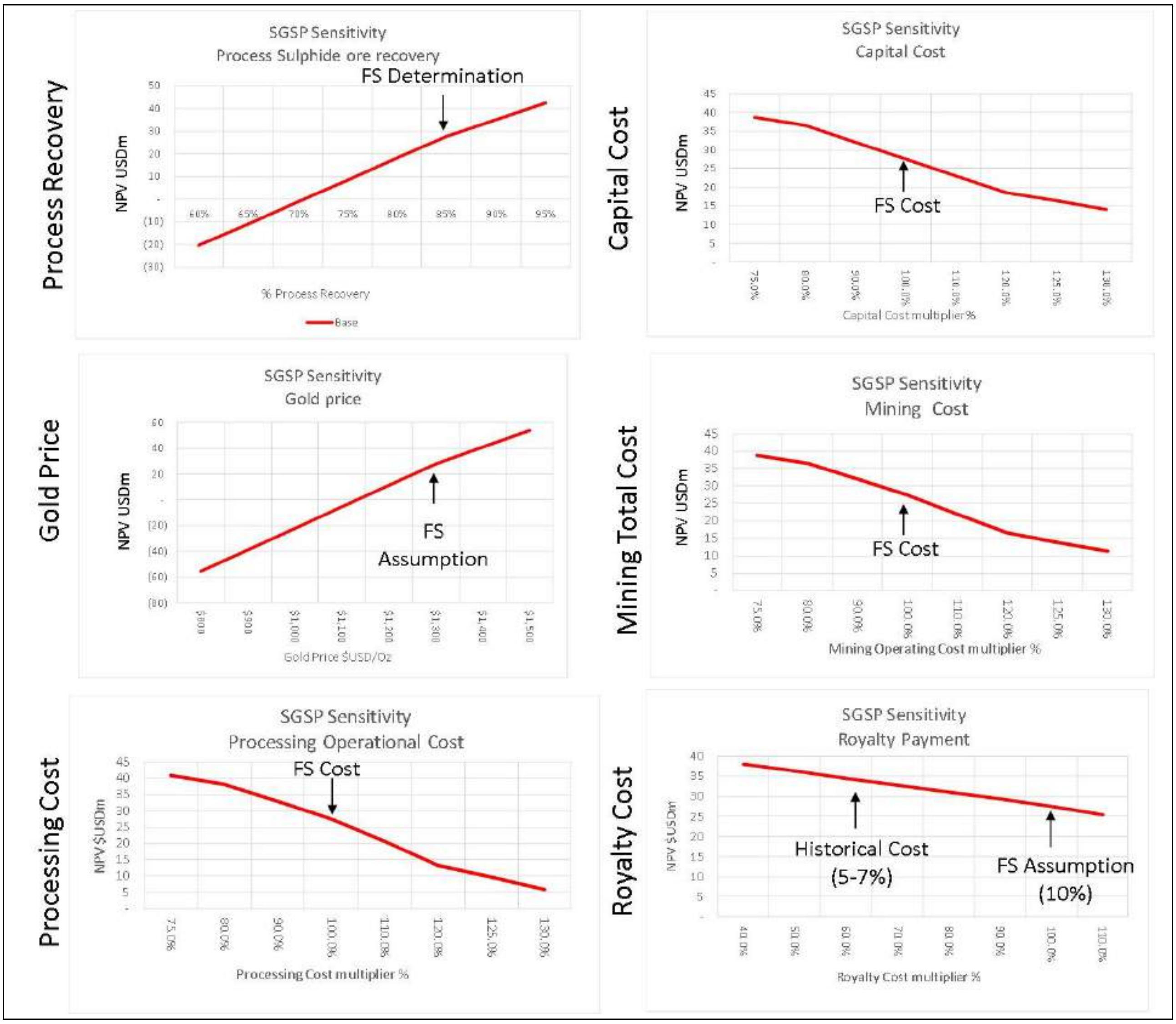
The economic cash flow model was used to prepare a sensitivity analysis for the net present value (NPV) (8%) for the project before taxation. The sensitivity analysis was completed on the following variables:

- Gold price
- Process recovery of gold
- Capital cost
- Mining operational cost
- Processing operational cost
- Royalty payment.

The sensitivities are shown in Figure 1.8.

The Project is sensitive to gold price and process recovery but less sensitive to cost areas. The sensitivities for capital and operating costs indicate a 60% increase is possible before a negative NPV is obtained for capital costs and 50% for mining cost and 35% for processing costs. For royalty cost (10% average assumed), a return to historical values (5–7%) would increase project value by US$10 million.

Figure 1.8 Sensitivities for gold price, process recovery, capital, processing, mining and royalty costs



1.12 Conclusion and recommendations

Economic viability has been demonstrated for the FS Reserve for all treatable materials processed with the Selinsing Processing Plant with a NPV (8%) of US$27.56 million. Sensitivities show the project can with stand up to a 35% increase in costs but is sensitive to gold price (US$1,100/oz breakeven) and process recovery (70% breakeven).

Detailed studies were carried out in many areas as part of the FS. With the geological and mine planning detail largely unchanged from the Snowden (Dec 2016) PFS, most of the detailed work has been associated with metallurgical study and evaluation as well as plant design and construction. This work demonstrated the feasibility of a plant capable of processing the sulphide ores at Selinsing using a flotation and BIOX® process. It is Monuments opinion that the study work includes enough detail to proceed to final plant design stages.

1.12.1 Other relevant information

The opportunity studies for the upgrading of Inferred Mineral Resources have shown significant increases in available process inventory, and these resources should be improved in confidence to allow the consideration for Mineral Reserves.

A potential underground desktop study was also carried out indicating potential in the Selinsing area but more inventory over 3 g/t Au cut-off grade and sourcing an available and cost-effective mining contractor arrangement is required.

Other opportunities described in the FS include the optional early mining of Selinsing Pit 5. Preliminary testwork indicates of ore is treatable by the current oxide processing plant.

1.12.2 Recommendations

The major recommendation is for further resource definition and conversion of the Inferred open pit potential as well as extensions for underground mining. The major emphasis will be in the Buffalo Reef (BRF) area with following aims:

- Converting the Opportunity Case One Mineral Resource to Indicated at BRF
- Converting the Opportunity Case Two Mineral Resource to Indicated at BRF.

There is significant exploration potential at the Selinsing Gold Mine Project with the sulphide orebodies open at all directions allowing for potential in both open pit and underground mining. To date Monument Mining Limited has demonstrated historical evidence of mineral resource being converted to reserve.

Construction and operation of the Sulphide processing plant may also allow treatment of ores form other sources in Malaysia and the Asian region. This depending on treatment compatibility and economic distance and other factors for ore and concentrate transport.

Metallurgical specific recommendations are:

- BRF Transition ore variant Gangue Depressant Tests.
- Additional flotation testwork is recommended to confirm the recovery at the target concentrate grade for lower head grade samples.
- Further refinement of the use of Cu SO_4 addition in flotation.
- Further refinement and optimisation of the Sulphide ore treatment process with further ongoing metallurgical testwork.

2 INTRODUCTION

2.1 Overview

This Technical Report has been prepared by Snowden Mining Industry Consultants Pty Ltd (Snowden) for Monument Mining Limited (Monument) in compliance with the disclosure requirements of the Canadian National Instrument 43-101 (NI 43-101). The trigger for preparation of this report is the press release Monument made on 26 May 2017 “Monument Commenced FEED on Selinsing Sulphide Project” (Release #08- 2017), disclosing a front-end engineering design (FEED)/ feasibility study (FS) had begun.

The FS is reported in the NI 43-101 Technical Report format.

Unless otherwise stated, information and data contained in this report or used in its preparation has been provided by Monument.

The Monument Qualified Persons for preparation of the report and the status of project site visits are shown in Table 2.1, as are Section responsibilities of each author.

Table 2.1 Qualified Persons

Author	Responsible for section(s)	Site visit
Frank Blanchfield	15, 16	Selinsing and Buffalo Reef areas – February 2016
John Graindorge	6 to 12, 14, 16	Selinsing and surrounding tenements – August 2010
F. Kock	13, 17	Co-author and QP is an employee of Orway Mineral Consultants and has not visited the site
M. Kitney	18, 21	Co-author and QP, Independent Technical Director of Monument overseeing Metallurgical Design and Process cost estimation; visited the site on numerous occasions since 2008.

Frank Blanchfield is an employee of Snowden and an independent Qualifed Person for Buffalo Reef and Selinsing Mineral Reserve estimates. Frank Blanchfield is also the Qualified Person responsible for currency and completion of Technical Report.

John Graindorge is an employee of Snowden and an independent Qualifed Person for Buffalo Reef and Selinsing Mineral Resource estimates.

Fred Kock is and employee of Orway Mineral Consultants and an independent Qualified Person for metallurgical aspects of the FS including testwork review and process design criteria but has not visited the site.

Mike Kitney is the independent technical director of Monument, and Qualified Person for engineering design and process capital and operating cost estimation aspects of the FS, including the testwork conducted in relation to the Selinsing flotation plant design.

Unless otherwise stated, all currencies are expressed in US dollars (US$).

The effective date of the report is 1 February 2019.

2.2 Basis of Technical Report

In order to progress the Selinsing Gold Mine (SGM) operation in Malaysia from an oxide through to sulphide ore processing, the Selinsing Gold Sulphide Processing Project (SGSP) has been initiated. The SGSP is to include an FS and a FEED study with a project charter to develop the geological, mining and metallurgical models to support a project incorporating flotation and bio-oxidation facilities to the existing plant, development of the plant process design and completion and documentation of the basic engineering as a basis for preparation of the capital and operating cost estimates together with the other supporting documentation required for an FS. The FS Technical Report represents a detailed summary of the FS.

A key to progressing the SGSP is the investment required to finance the upgrade of the Selinsing Processing Plant to enable the processing of the sulphide ore contained in the Selinsing and Buffalo Reef deposits. An FS is required to document the business case for the plant upgrade as well as document the current mineral resource and reserves. The FS document to be the major reference for potential investors.

The FS describes the full SGSP from geological interpretation though mining and metallurgical evaluation to engineering design to final Project Execution Plan (PEP) and FEED study input. As such, the FS is not just a processing plant design and plant construction study, it needs to consider the full site-wide implications and all opportunities.

This FS is a natural continuation of previous prefeasibility study (PFS) and evaluation PFS work as follows:

- Monument Mining Limited – *NI 43-101 Technical Report Selinsing Gold Mine and Buffalo Reef project by Snowden, December 2016* (Snowden, Dec 2016)
- Monument Mining Limited – *Selinsing Gold Mine Manager SDN BHD Selinsing Phase IV Pre-Feasibility Study by Lycopodium Minerals QLD Pty Ltd* (Lycopodium, 2013).

The PFS followed another NI 43-101 Technical Report *"NI 43-101 Technical Report Selinsing Gold Mine and Buffalo Reef Project Expansion Selinsing Gold Mine and Buffalo Reef Project by Practical Mining Ltd, May 2013"*. This report was carried out essentially to a scoping level of detail, demonstrating a pathway of evaluation of this FS.

In terms of geological Mineral Resource and Mineral Reserve updates for this FS, these are largely based on depletions for mining of those reported in Snowden (Dec 2016). Snowden (Dec 2016) presented the Mineral Resource and Mineral Reserves as at June 2016 completed a NI 43-101 Technical Report. With no major geological drilling and/or mineral resource updates since that time, except for depletions for mining, the June 2016 Mineral Resource and Mineral Reserves are considered as a basis. Some adjustment to potential reserves for updated Selinsing pit design parameters was however required.

The Mineral Resources and Mineral Reserves were assessed and presented, and Mineral Reserves include all economic material including oxide and tailings carbon-in-leach (CIL) treatable material.

The FS report is aimed to evaluate economic vailibility of the SGSP on a mineral reserve basis; however, as part of the SGSP, inclusion of Inferred a LOM were studied as opportunities. Two opportunity scenario cases have been documented in this study:

- Opportunity Case One: Inclusion of Inferred material in the reserve open pit designs also inclusive of oxide and tailings material
- Opportunity Case Two: Inclusion of all Indicated and Inferred open pit Mineral Resource potential also inclusive of oxide and tailings material.

Much of this study will concentrate on metallurgical and associated engineering design and project execution of an operation with a flotation and bioxidation upgrade to the processing plant. Other areas of infrastructure such as utilities with electrical power and water supply are also considered.

The Selinsing operation has now limited supply of gold-bearing material that can be processed by the current oxide plant. Currently, the plant processes oxide ore, tailings and leachable sulphide ore. Progressing to a stage of sulphide ore process capability in a timely manner is now of the utmost concern. Snowden (Dec 2016) estimated the plant would be operational in 2018 but recent estimates, particularly with required metallurgical testwork and full plant design, indicate that 2020 is more likely.

The FS presents the final business case for Monument for the SGSP in terms of whether to proceed to full completion of the FEED study.

2.3 Scope of work

The FS is scoped to report economic viability of Selinsing Gold Mine Project including all mineral reserves: ofoxide and sulphidemaaterial, either to be mined, or being stockpiled and in the Old Tailings, which would be mined, reclaimed to appropriate treatment plants for process according to the Life of Mine (LOM) plan.

In particular, this FS is primarily driven by adding flotation and BIOX® processes to the current Selinsing Gold Processing Plant to treat sulphide ore. It has focused on metallurgical testwork, metallurgical processing design and Front-End Engineering Design, along with geological review and mining optimization. In order to sustain the Selinsing Gold Mine production from oxide through to sulphide ore operations, a NI43-101 pre-feasibility study on Selinsing Gold Sulphide Processing Project (SGSP) was filed in December 2016 (Snowden (2016)), supporting bio-leaching process can be used to treat sulphide materials at an economic level. As a follow-up, the Feasibility Study (FS) was initiated in May 2017 to study and demonstreat the BIOX ® sulphide treatment technology is the most preferred bio-leaching method to achieve best economic outcomes.

Figure 2.1 shows a summary view of the study process for the FS including that done by external consultants and providers shown and summarised in this FS.

The major components of the FS are as follows:

Introduction, Property Description and Location, Accessibility, Climate, Local Resources, Infrastructure and Physiography

- Largely based on Snowden (Dec 2016) with some elaboration in the environmental area required.

Geological Study

- The mineral resource models remaining largely unchanged except for mining depletion and the addition of sulphur; however, geometallurgical study work is included.

Mining Study

- Geotechnical evaluation:
  - Updates based on recent external study on Selinsing and Buffalo Reef wall stability.
- Mine design:
  - Conducted largely by Monument personnel in the Perth office
  - Re-optimisation of the Snowden 2016 work based on updated mining, processing costs and process recoveries.
- Reserve-only Case:
  - Separate case for reserves of all economic material including oxide and tailings.
- SGSP opportunity cases
  - Opportunity Case One:
    - Reserve pit design inclusive of Inferred potential.
  - Opportunity Case Two:
    - All open pit Indicated and Inferred potential
  - Underground mining study:
    - Completed as an internal opportunity case to desktop study level only with possible later progression to PEA and FS.

Metallurgy

- This section was updated from the Snowden (Dec 2016) Technical Report, as that study assumed biological leaching and lacked the detailed testwork for flotation and that is required for BIOX®. The new studies include:
  - Ore characteristics.

  - Testwork and treatment history – this section is a major update to Snowden (Dec 2016) detailing previous testwork and treatment history including Intec pilot plant results.
  - Process selection – this section justifies process selection and include results for the previous PFS findings.
  - The overall metallurgical process design and overall study work was reviewed by Orway Mineral Consultants (OMC) and includes preliminary engineering design criteria; electrical power study; and metallurgical study work including plant process design.

Plant Design

- Full description of process design criteria.
- Facility description, including:
  - Block process diagrams
  - Detailed process flow diagrams
  - Waste management facilities layouts and designs including tailings dams.
- Outotec through its South African subsidiary, Outotec (RSA) Pty Ltd, to design the BIOX® as part of a licence agreement with Monument.
- Overall plant design including existing plant interface to be completed by OMC.
- Recovery:
  - Full description of recoveries required including grade vs recovery relationships.
- Treatment cost.
- Tailings disposal.

Mineral Balance Sheet

- Updated reserves and Mineral Resource with depletion to March 2018 inclusive of oxide and tailings.

Project Infrastructure

- This was included in the engineering study for review by PIE for its study on utilities and other infrastructure required for the SGSP.

Capital Cost, Operating Cost and Financial Evaluation

- This was carried out to FS level of accuracy for teh FS reserve only evaluation
- Used corporate guidance on taxation and key financial assumptions
- LOM scenarios were evaluated
- Sensitivity and financial risk modelling to be carried out.

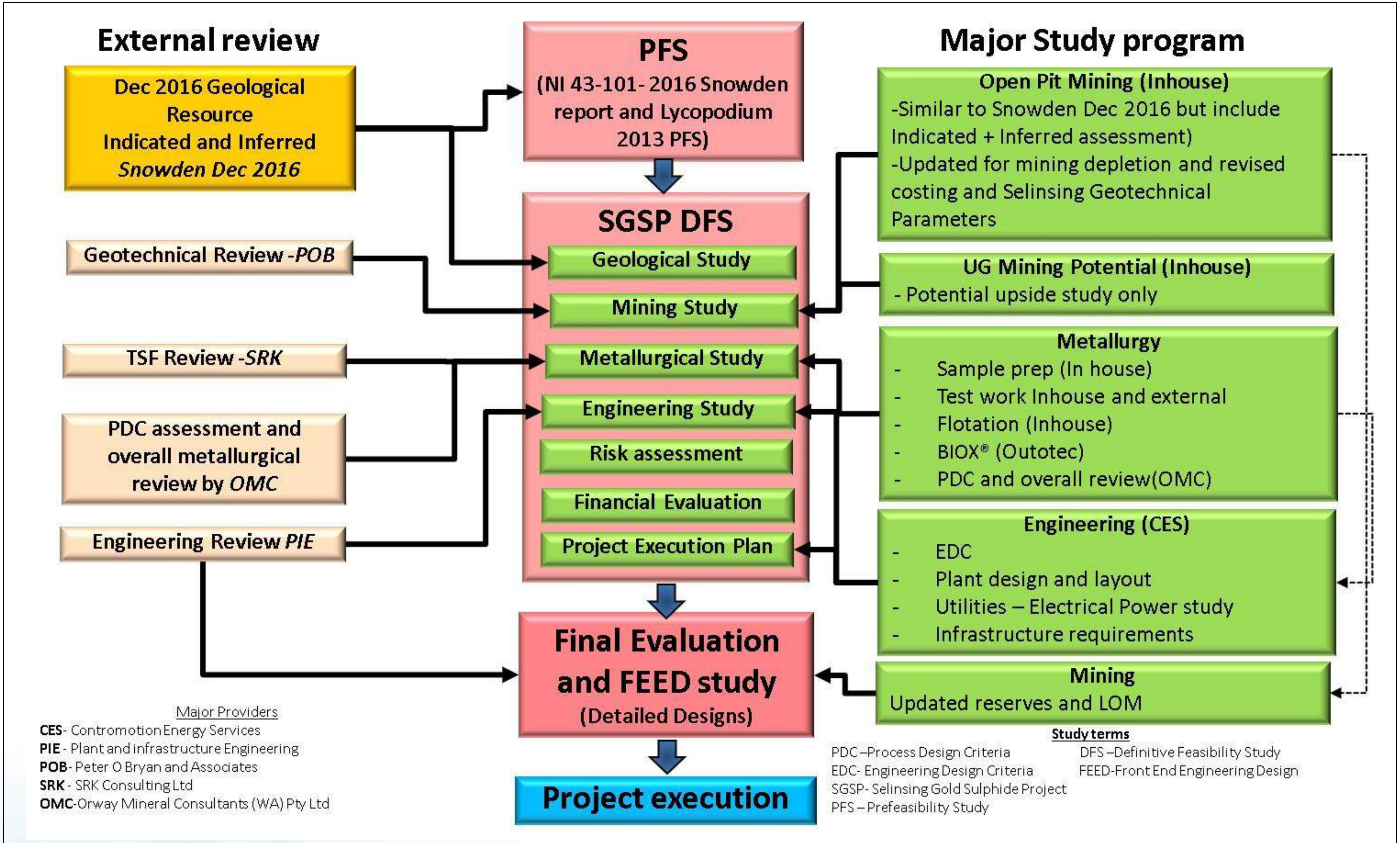
Project Risk Analysis

- A summary of completed risk assessment SGSP May 2017 and treatment of associated control plan in the FS
- Summary of revised Risk Assessment as required

Conclusion and Recommendations

- Detail conclusions and future FEED work plan.

Figure 2.1 SGSP FS program summary



2.4 Assumptions

The assumptions on which the SGSP and this FS is based on are:

- Water, power and other supporting services are to be made available in accordance with the project baseline schedule and will have sufficient capacity for the new plant
- Logistics, materials handling and the site infrastructure and services to be supplied by the Project Owner to support the construction activities are to be made available in accordance with schedule and agreed rates as set out in the Project Owner's Supply Schedule
- There will be no significant delay due to environmental licences or permitting concerns or significant environmental compliance issues, and is based on environmental compliance and performance to date
- Noise level evaluation or provision of noise barriers at the plant boundary fence for sound attenuation is not required
- The existing process plant area tower crane will be available for construction and maintenance of process plant equipment
- The process flow diagrams and piping and instrument diagrams depicting the existing plant are accurate for the purpose of the engineer progressing with the design
- Process control system will patch into the existing system
- Selinsing Operations will provide the resources to undertake the wet and ore commissioning of the plant with support from the Project Team and specialists.

Specific to this FS, all economic material including mined oxide, stockpiles and Old Tailings will be included as well as the sulphide only plant treatable SGSP material.

2.5 Sources of information

The key references and previous work sources are as follows:

- PFS related work:
  - Monument Mining Limited – NI 43-101 Technical Report Selinsing Gold Mine and Buffalo Reef project by Snowden, December 2016 (Snowden, Dec 2016)
  - Monument Mining Limited – Selinsing Gold Mine Manager SDN BHD Selinsing Phase IV Pre-Feasibility Study by Lycopodium Minerals QLD Pty Ltd, February 2013 (Lycopodium, 2013)
  - Monument Mining Limited – Selinsing Gold Mine Manager SDN BHD Selinsing Phase IV Pre-Feasibility Study Capex and Opex Revision by Lycopodium Minerals QLD Pty Ltd, September 2016 (Lycopodium, 2016)
  - Buffalo Reef Concentrate Test Work Review 7614 by Orway Mineral Consultants, February 2015 (OMC, 2015).
- Scoping study related work:
  - NI 43-101 Technical Report – Selinsing Gold Mine and Buffalo Reef Project Expansion Selinsing Gold Mine and Buffalo Reef Project by Practical Mining Ltd, May 2013 (Practical Mining, 2013)
  - Monument Mining Ltd – Buffalo Reef Gold Deposit, Malaysia NI43-101 Technical Report Project No. 2015 by Snowden, May 2011 (Snowden, 2011).
- Before Monument acquisition:
  - June 2006. Selinsing Gold Mining Project, Malaysia NI 43-101 Technical Report
  - Damar Prospects, Malaysia, Information Memorandum, November 2006, unpublished internal report for Avocet Mining PLC, November 2006 (Avocet, 2006).

2.6 List of abbreviations, acronyms and definitions

Units of measurement used in this report conform to the metric system. All currency in this report is US$ unless otherwise noted.

%	percentage	L	litre
°	degrees	lb	pound
°C	degrees Celsius	L/s	litres per second
µm	micron	m	metre(s)
µg	microgram	M	mega (million)
a	annum	m^2	square metre
cal	calorie	m^3	cubic metre
cm	centimetre	m^3/hr	cubic metres per hour
cm^2	square centimetre	min	minute
dia.	diameter	mm	millimetre
dmt	dry metric tonne	Mt	million tonnes
dwt	dead-weight ton	Mt/a	million tonnes per annum
ft	foot	MVA	megavolt-amperes
g	gram	MW	megawatt
G	giga (billion)	MWh	megawatt-hour
g/L	gram per litre	oz	Troy ounce (31.1035g)
g/t	gram per tonne	ppm	part per million
hr	hour	psia	pound per square inch absolute
ha	hectare	psig	pound per square inch gauge
hp	horsepower	RL	relative elevation
in	inch	s	second
in^2	square inch	t	metric tonne
J	joule	t/a	metric tonne per year
k	kilo (thousand)	t/d	metric tonnes per day
kcal	kilocalorie	t/hr	metric tonnes per hour
kg	kilogram(s)	US$	United States dollar
km	kilometre(s)	V	volt
km/hr	kilometres per hour	W	watt
km^2	square kilometre	wmt	wet metric tonne
koz	kilo (thousand) ounces	yd^3	cubic yard
koz/a	kilo (thousand) ounces per annum	yr	Year
kPa	kilopascal		
kVA	kilovolt-amperes		
kW	kilowatt		
kWh	kilowatt-hour		

Abbreviations and acronyms for the FS are as follows:

2D	two-dimensional
3D	three-dimensional
AC	air-core (drilling)
Amelda	Amelda and Partners
Avocet	Avocet Mining plc
BAT	batch amenability test
BIOX®	bio-oxidation process registered trademark
BOCO	base of complete oxidation
BRC	Buffalo Reef Central
BRG	Buffalo Reef Gap
BRN	Buffalo Reef North

BRS	Buffalo Reef South
CCD	counter-current decantation
CIL	carbon-in-leach
COS	crushed ore stockpile
CRM	certified reference material
Damar	Damar Consolidated Exploration Sdn Bhd
DFS	definitive feasibility study
DID	Department of Irrigation and Drainage
DOE	Department of Environment
ECER	East Coast Economic Region
EIA	environmental impact assessment
EMP	environmental management plan
EMS	environmental management system
EPCM	engineering, procurement, construction management
FEED	front-end engineering design
FELDA	Federal Land Development Authority
FS	feasibility study
FX	foreign exchange
GPS	global positioning system
HG	high grade
ICP	inductively coupled plasma
ICP-MS	inductively coupled plasma mass spectroscopy
ICP-OES	inductively coupled plasma optical emission spectroscopy
IRR	internal rate of return
ISD	inverse of square distance
JMG	Jabatan Mineral Geosciences
KNA	kriging neighbourhood analysis
LG	low grade
LOM	life of mine
MC	Mining Certificate
MIK	multiple indicator kriging
Minetech	Minetech Construction Sdn Bhd
ML	Mining Lease
Monument	Monument Mining Limited
NATA	National Association of Testing Authorities, Australia
NPV	net present value
OK	ordinary kriging
OMC	Orway Mineral Consultants
OP	open pit
OT	Owner’s Team
PAX	potassium amylxanthate
PCP	Project Controls Plan
PEG	Project Estimation Guide
PEP	Project Execution Plan
PFS	prefeasibility study
PIE	Plant and Infrastructure Engineering
PML	Proprietary Mining Lease
POB	Peter O’Bryan and Associates
POX	pressure oxidation
QA	quartz ankenite
QAQC	quality assurance/quality control
Q-Q	quantile-quantile
QS	quartz stibnite

RC	reverse circulation (drilling)
ROM	run of mine
RQD	rock quality designation
SEP	Study Execution Plan
SGM	Selinsing Gold Mine
SGSP	Selinsing Gold Sulphide Project
SLG	super low grade
SMBS	sodium metabisulphite
Snowden	Snowden Mining Industry Consultants Pty Ltd
SPT	standard penetration test
SRK	SRK Consulting
TNB	Tenaga Nasional Berhad
TOFR	top of fresh rock
TRA	Target Resources Australia NL
TSF	tailings storage facility
UG	underground
WBS	Work Breakdown Structure
XRD	x-ray diffraction

3 RELIANCE ON OTHER EXPERTS

This Technical Report has been prepared by Monument for review by Snowden. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to Monument at the time of preparing this Technical Report including previous technical reports prepared on the Project and associated licences within the Project
- Assumptions, conditions, and qualifications as set forth in this Technical Report
- Data, reports, and other information supplied by Monument and other third-party sources.

Snowden and the Qualified Persons are reliant on Monument for the financial model estimates and results and the disclosed gold price of US$1,300/oz. The risks associated with the gold pricing are analysed in Section 24 in the price sensitivity discussion. Metallurgy QPs relied on Dr Mike Wort who is an employee of Monument Mining, and co-author for the metallurgical aspects of the FS, including the test work conducted relating to the Selinsing floatation plant design.

Except for the purposes legislated under provincial securities laws, any use of this report by a third party is at that party's sole risk.

Information sources and other parties relied upon to provide technical content and review are shown in Table 3.1.

Table 3.1 Other parties relied upon to provide technical content and review

Information supplied	Other parties	Sections
Ownership, title, social and environmental studies and information	Monument	4 to 12, 14 to 16, 20, 23, 24
Infrastructure capital and OPEX estimates	CES, Outotec, Afrima, Monument	18, 21, 22
Marketing report	Monument	19, 22
Financial modelling	Monument	22
Taxation and royalties	Monument	21, 22

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Area

The property related to the Selinsing and Buffalo Reef deposits and controlled by Monument consists of four land types as:

- The Selinsing mineral titles
- The Buffalo Reef mineral titles
- Exploration leases and blocks
- Prospecting leases.

Note that lease applications are currently under review by regulatory aiuthorities.

Pahang State Government via the State Land and Mine Department issued Mining Certificate Leases prior to 2005 and thereafter replaced these by the issue of Mineral Leases.

The Selinsing deposit currently comprises wholly owned active mining areas covered by a Mining Certificate (MC1/124) and two Mining Leases (ML7/2000 and ML5/2000) and the total coverage of concessions and leases is 125 hectares (ha).

The Buffalo Reef deposit is covered by Mining Certificate Leases MC12/2012 in the north and MC1/111 in the south. The land areas for the two Mining Leases that cover Buffalo Reef are 180 ha.

Monument controls exploration access to Mining Lease ML11/2009 (189 ha to the west of the mine). There is also exploration access to 14 blocks in Felda Sungai Koyan 3 totalling 1,587 ha, granted by the Federal Land Development Authority (FELDA) in 2013. Of the 14 blocks, Monument also acquired exploration and mining rights for blocks 7, 9, 11 and 12 directly from 60 individual settlers (totalling 340 ha) within the same settlement scheme.

4.2 Location and history

The Selinsing Gold Mine Project is located at Bukit Selinsing, approximately 65 km north of Raub and 30 km west of Kuala Lipis on the lineament known as the Raub Bentong Suture. Selinsing is located approximately two hours' drive from Kuala Lumpur, the capital of Malaysia, on a sealed highway in Pahang State, in Malaysia (as shown in Figure 4.1). The location of the Selinsing mine is approximately 4°15'00"N latitude and 101°47'10"E longitude, or 421500 mE and 470500 mN in the "Kertau_RSO_Malaya_Meters" projected coordinate system. Figure 4.2 shows the mine position in relation to infrastructure, town and state.

Figure 4.1 Selinsing Gold Mine and Buffalo Reef Project regional location in Malaysia

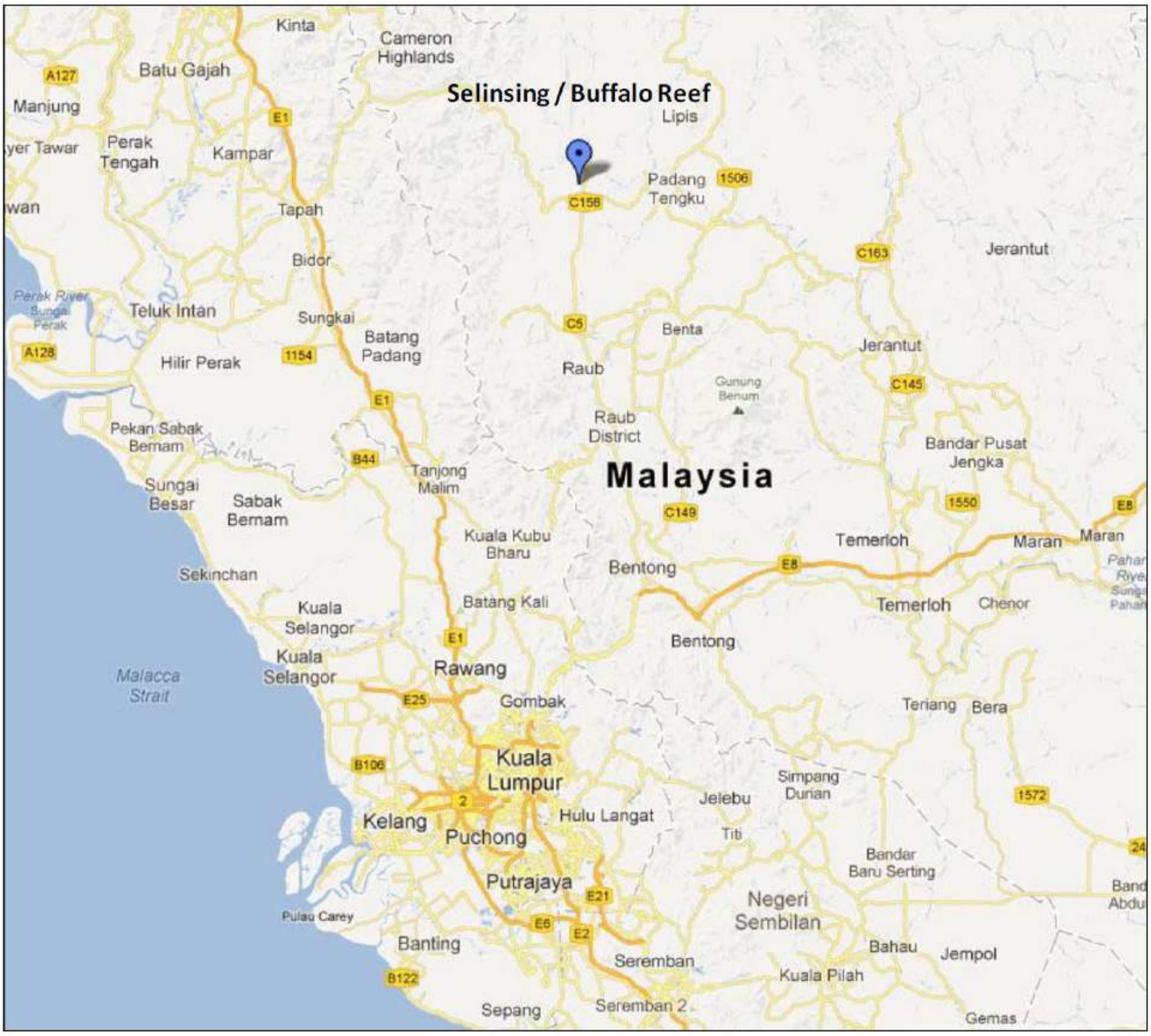
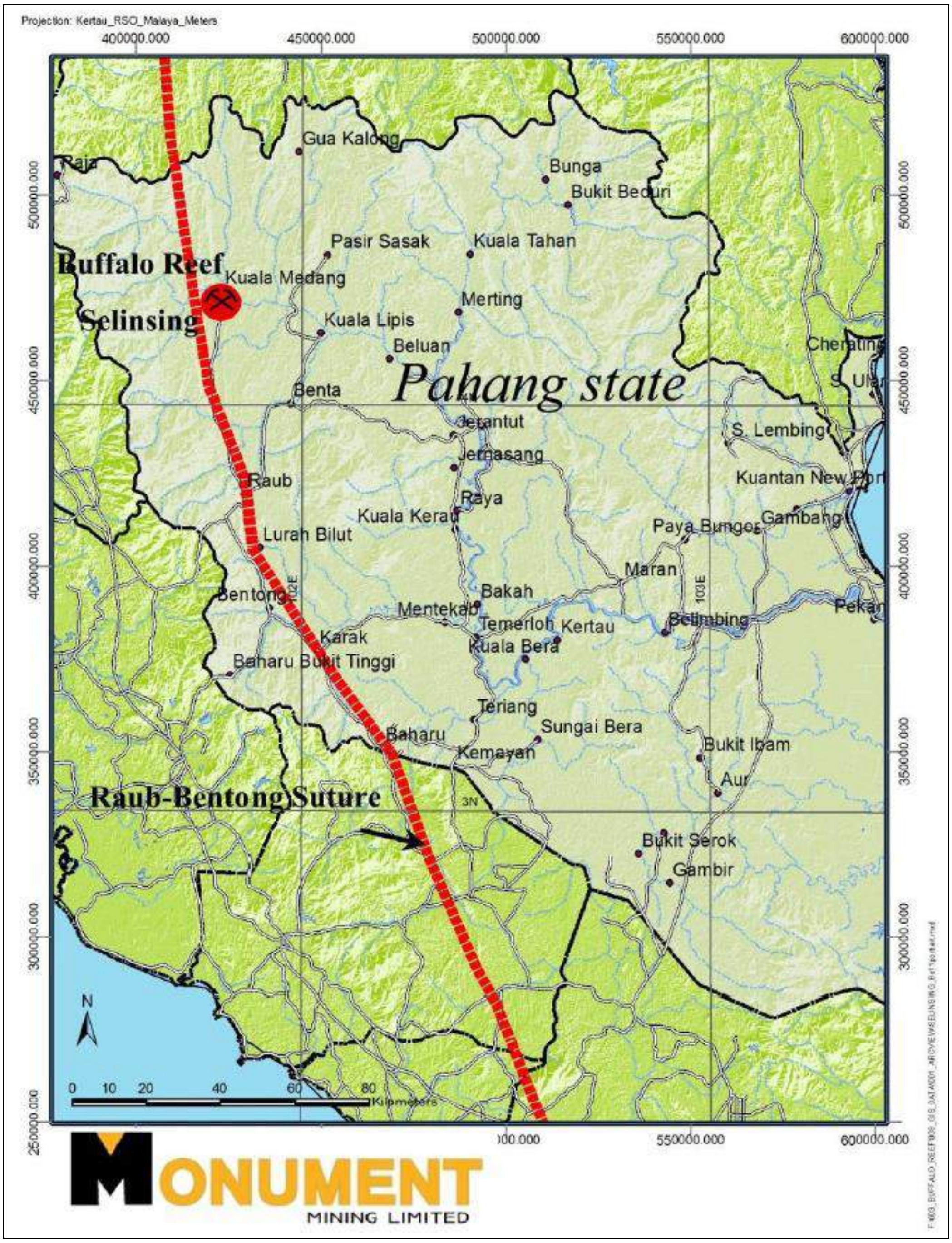


Figure 4.2 Selinsing Gold Mine and Buffalo Reef Project regional location in Pahang State, Malaysia



Since 25 June 2007, through its wholly-owned Malaysian subsidiary, Polar Potential Sdn Bhd, Monument has held 100% of the Selinsing property. The Selinsing operation consists of:

- Selinsing Gold Mine
- Buffalo Reef Project
- Exploration leases and blocks.

Historical mining of visible gold at Selinsing by crude means probably occurred for centuries prior to 1888, when British companies began production on a larger scale utilising machinery for primary processing treatment, with intermittent mining continuing at Selinsing up until June 2007 when Monument acquired the Selinsing Project. Mining was restarted in July 2009 with a planned production rate of 40,000 oz of gold per year. Since mining was restarted, 278,784 oz of gold was produced as of the end of June 2018.

Mining at Selinsing is carried out by open pit methods utilising a local mining contractor Minetech Construction Sdn Bhd (18457-W) (Minetech).

Small-scale mining at Buffalo Reef dates back to the early 1900s with 1,000 m of underground workings developed in 1934, including adits, drifts, crosscuts, winzes and shafts. Production details from that time are not available and assumed to be relatively small.

Antimony in the form of stibnite occurs in the gold-bearing veins at Buffalo Reef. Stibnite was mined in the 1970s and that amount of material is also assumed to be small.

The Buffalo Reef gold deposit was acquired by Monument in July 2007 through its acquisition of Damar, a wholly-owned subsidiary of Avocet (Cavey & Gunning, 2007). Damar (and Avocet) owned the project from 1993 to 2007, when initial exploration commenced.

Open pit mining commenced at Buffalo Reef in 2012 and uses the same mining contractor as for the Selinsing Pit.

To date the majority of mining and ore treatment history has been associated with oxide material but at the Selinsing and Buffalo Reef deposits an extensive gold sulphide ore deposit remains in fresh and transitional material yet to be mined and treated.

The Selinsing Gold Mine processing plant has been operating since 2009. The plant was constructed in three phases, being:

- Phase I: Process plant consisting of a gravity circuit which commenced in August 2009 with the first gold pour in October 2009. The gold treatment plant with 400,000 t annual capacity was fully completed in December 2009.
- Phase II: Addition of the CIL circuit fully commissioned by August 2010. Total capital cost for Phase I and Phase II was US$18.2 million.
- Phase III: This expansion began on 6 September 2011 with a budget of US$8.1 million and was completed in June 2012 on time and at a cost of US$8.6 million. The Phase III expansion increased the capacity of the gold treatment plant from 400,000 t/a to approximately 1,000,000 t/a. The 2012 plant expansion followed the identification of further Mineral Reserves in three pits at Selinsing, being the southernmost Pit 4, Pit 6 which is the northernmost pit, and Pit 5 between Pit 4 and Pit 6. The Phase III Selinsing pits were designed by consultants with Snowden and Monument staff mine engineers.

The current plant consists of three stages of crushing, with two stages of ball milling operating in closed circuit with current throughput of approximately 3,000 t/d. A gravity recovery circuit is also used, consisting of a Knelson centrifugal concentrator operating on a split from the mill cyclone underflow. The Knelson concentrate is subjected to an Acacia high intensity leach with the leached concentrate returned to the ball mill. The mill cyclone overflow discharges to a three-stage leach and six-stage carbon-in-leach (CIL) cyanidation circuit, with a targeted grind of 80% passing 75 µm and a 36-hour retention time. Loaded carbon is advanced through the leach circuit, collected, then stripped of precious metals with hot caustic, reactivated and recycled. The pregnant solution from the Acacia reactor and from the stripped carbon is sent to the refinery for electrowinning and subsequent production of doré. The leached CIL slurry is discharged to the TSF.

Phase IV of the process plant upgrade is planned for the treatment of the extensive sulphide refractory ore remaining to be mined. This is the subject of this FS.

In the last few years, extensive sulphide ore processing evaluation has occurred at PFS level by:

- Intec
- Biological leaching
- Acid leaching
- BIOX®.

The BIOX® process has now been determined as the most suitable process to be used.

In order to progress the operation through to sulphide ore production the Selinsing Gold Sulphide Processing Project (SGSP) has been initiated. The SGSP is to include this FS and a FEED study with a project charter to develop the geological, mining and metallurgical models to support a project incorporating flotation and bio-oxidation facilities to the existing plant, development of the plant process design and completion and documentation of the basic engineering as a basis for preparation of the capital and operating cost estimates together with the other supporting documentation required for a FS.

4.3 Land tenure

The mineral tenures for the Buffalo Reef/Felda and Selinsing properties consist of:

- Mining leases
- Mining certificates
- proprietary mining leases (formally Felda Block 7).

4.3.1 Issuer's interest

Monument has taken steps to verify the title to its mineral property interests in accordance with industry standards for the current stage of exploration of such properties; however, these procedures do not guarantee Monument's title. Property title may be subject to unregistered prior agreements or transfers and title may be affected by undetected defects. To the best of Monument's knowledge, title to its properties is in good standing. The author has reviewed the land tenure situation but has not independently verified the legal status or documents of ownership of the properties or any contractual agreements that pertain to the Selinsing or Buffalo Reef gold deposit project area.

Monument engaged Malaysian solicitors, Amelda and Partners (Amelda), to complete a review on the title of the Felda lands acquired through the acquisition of Able Return Sdn Bhd. Amelda confirms Monument's right to explore and mine on the Felda blocks shown in Figure 4.2 (Amelda and Partners, 4 April 2013).

4.3.2 Selinsing land rights

On 25 June 2007, through its wholly-owned Malaysian subsidiary Able Return Sdn Bhd, Polar Potential Sdn Bhd, Monument acquired 100% of the Selinsing Gold Project with two mining concessions (MC1/124 and ML5/2000), the total coverage of concessions and leases is 68.76 ha. The tenement owner held Malaysian Pioneer status which among other benefits provides a five-year tax break from Malaysian Federal and other taxes, which was fully applied against the first five year's taxable income of Able Return Sdn Bhd.

4.3.3 Buffalo Reef land rights

Concurrent with the acquisition of the Selinsing Gold Project, the Company acquired 100% of the shares of Damar Consolidated Exploration Sdn Bhd (Damar), a company incorporated in Malaysia, from Avocet Mining plc (Avocet), which is the parent company of Damar; thereby acquiring the Buffalo Reef property, which is contiguous and continuous with the Selinsing Gold Project for an approximate 4.2 km of controlled property along the regional gold trend.

The Buffalo Reef gold deposit is covered by Mining Leases ML12/2012 in the north and MC1/111 in the south, and two other mining leases that hosting the waste dump and tailing storage facilities are ML 7/2011 and ML11/2009 respectively. The land areas for the four mining leases that cover Buffalo Reef properties are 478.4 ha.

Damar Consolidated Exploration Sdn Bhd is a benefitiary owner under a Consolidated Agreement with Pahang State Development Corporation ("PKNP"), Purswant to which, the Company is obligated to pay land fees and 2% royalty of gold prodution to PKNP.

4.3.4 Block 7 (Felda) tenements

In October 2017, the Company through its Malaysia subsidiary acquired a 100% of proprietary mining leases ("PML") at Felda Block 7, which is adjacent east of Buffalo Reef Central ("BRC"), It contains the extension of the BRC oxide ore body which have been mined and processed through the nearby existing gold process plant. Felda Block 7 allows the pit shell to be fully developed across the boundary to access the entire sulphide ore beneath the BRC once the Sulphide Gold Project is in production. Figure 4.3 shows the Block 7 (Felda) tenement which encroaches on the Buffalo Reef South (BRS) and Buffalo Reef Central (BRC) areas.

4.3.5 Summary of mineral tenure

The Selinsing Gold Mine Project encompasses the Selinsing and BR/Felda Block 7 deposits and the Selinsing plant. Mining Certificates (MCs) were issued under the old enactment and subsequent renewals of the MC are called Mining Leases (MLs), following the new enactment. For the Felda blocks, the state government has issued Proprietary Mining Lease (PML) on the settler's private landholding. Under the MC, ML and PML, right to mine was given by the state government to the holder of the title.

The mineral tenure for the Selinsing Gold Mine Project and obligations for mineral tenure retention are shown in Table 4.1, and Figure 4.3 shows a plan of the Selinsing and Buffalo Reef tenure.

Table 4.1 Mineral tenure retention and expiry dates obligations

Deposit	Lease	Area (ha)	Expiry
Selinsing active	MC1/124	40.4	27-Mar-2017*
	ML5/2000	28.3	4-May-2020
	Subtotal	**68.7**	
Buffalo Reef	ML12/2012	157	11-Feb-2021
	MC1/111	43	31-Oct-2016*
	ML 7/2011	91.5	28-Mar-2021
	ML 11/2009	186.9	28-Jun-2019
	Subtotal	**478.4**	
Felda Blok 7	PML 1-10/2017	15.83	9-Aug-2019
	Subtotal	**15.83**	
TOTAL		**566.83**	

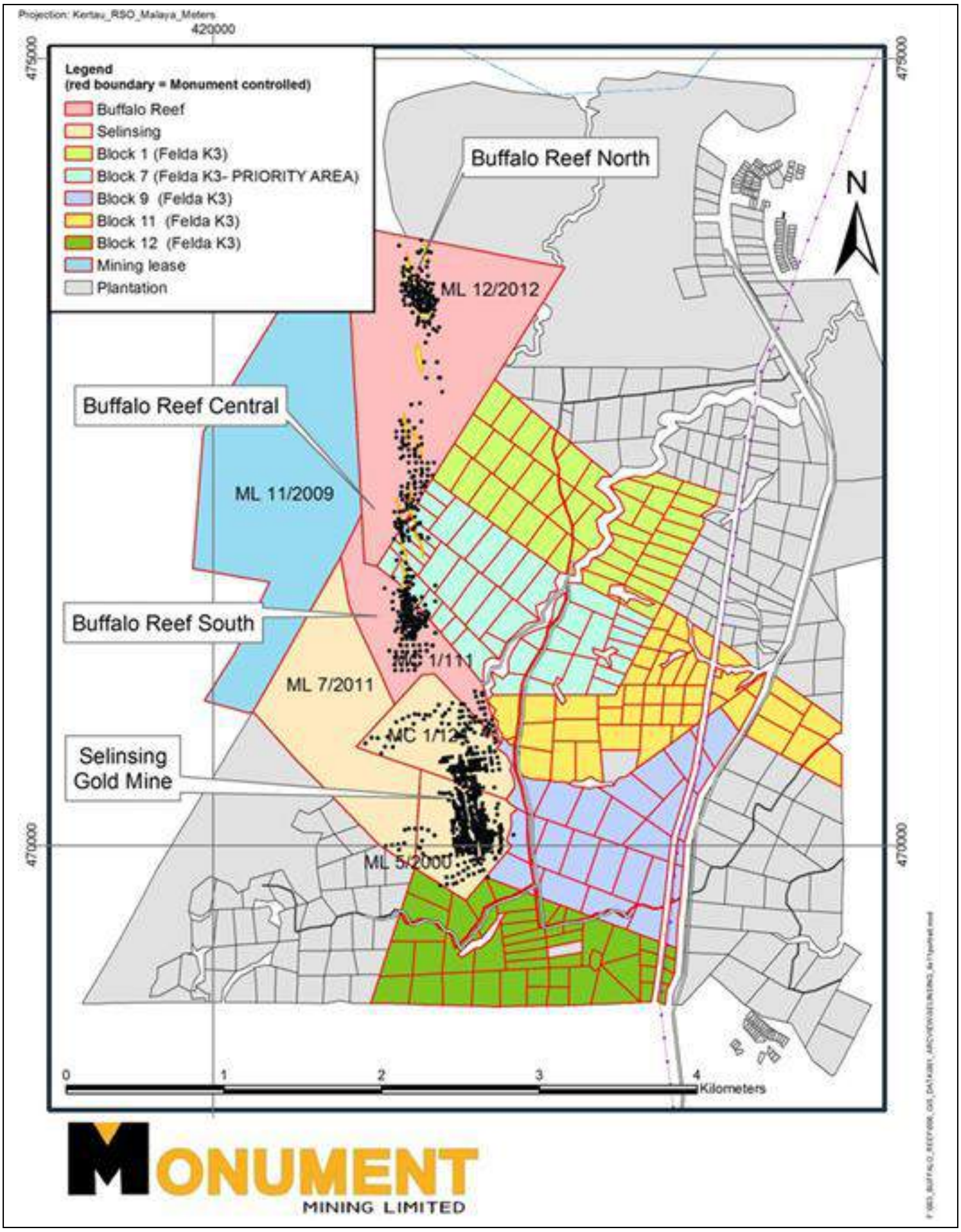
**The renewal applications for two Mining Leases have been submitted by PKNP and currently under Authority's review for approval.*

The risk associated with MC1/111 and MC1/124 is an increase of royalty to the 10% assumed in the FS. Historically approvals for renewing mining leases in Malaysia is slow however has not affected project operations to date.

The leases are subject to the provisions and conditions of the *Pahang Mineral Enactment 2001*, as obligations governing occupation and retention the Selinsing property and these include:

- Approval of the Director of Forests to remove timber (granted)
- State Government approval of any mine development (granted)
- Mines Department approval of any mine development (granted)
- Labour employment covenants requiring 50% of all employees to be Bumiputra (Malay or indigenous people).

Figure 4.3 Selinsing and Buffalo Reef land tenure



4.4 Underlying agreements

4.4.1 Royalties

Previously as reported in Snowden (Dec 2016) *"All mining leases in Malaysia carry a 5% royalty payable to the Pahang State government. The Buffalo Reef leases carry an additional 2% royalty payable to Pahang State Development Corporation (or in Malay language, "Perbadanan Kemajuan Negeri Pahang (PKNP))".*

Since that time, revised tenement fees and royalties are to apply as tenements expire. These royalties are increasing to a 10% royalty payable to Pahang State Government.

Tenements granted to Monument have no encumbrances or liabilities with them at the time of this report.

4.5 Permits and authorisation

There are no further permits that need acquisition prior to the proposed mining or exploration activities.

4.6 Environmental considerations

There are no environmental liabilities known to Monument.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Topography, elevation and vegetation

The Selinsing and Buffalo Reef properties are approximately 100 m above sea level and the surrounding area has relatively moderate to gentle relief. Land use around the site is primarily agricultural with palm oil the principal crop.

The southern portion of the Buffalo Reef deposit is situated within palm oil plantations with secondary jungle occurring in the northern portions of the deposit. Prior to clearing of any plantation trees (e.g. to establish drill sites), agreement with the local landholders, and possibly compensation, is required.

5.2 Accessibility

The Selinsing Project is accessed by sealed roads from the regional centres of Kuala Lipis 30 km to the east and Raub 65 km to the south. Figure 5.1 shows the location of the mine relative to the access road located to the east.

The Buffalo Reef area is primarily accessed via unsealed roads from the Selinsing Gold Mine, which can be accessed via sealed highway number C5. Alternative access includes tracks (unsealed) off highway number C5 leading to the northern parts of the Buffalo Reef area.

5.3 Climate

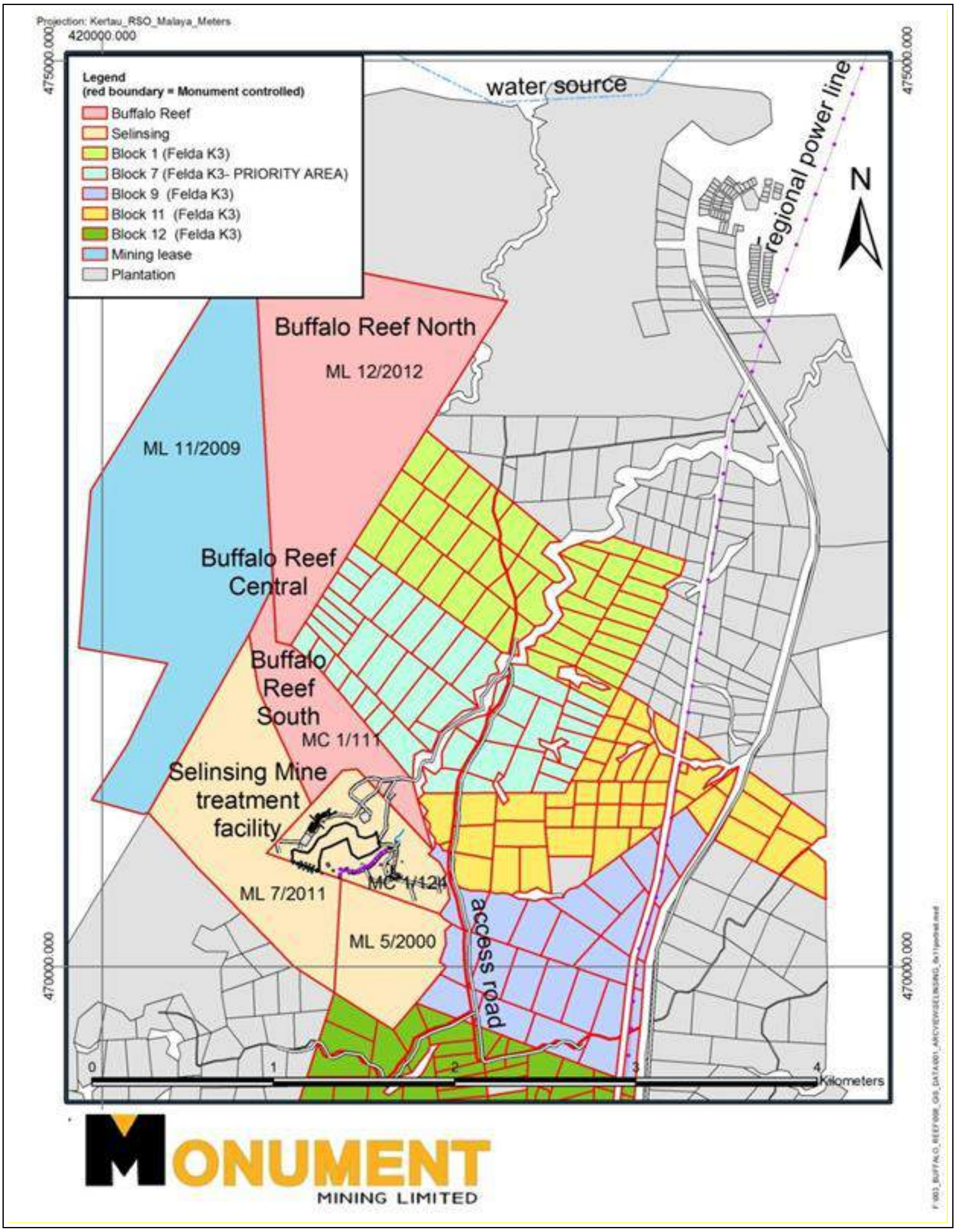
The central Malaysian Peninsula has a tropical climate, with the annual temperature ranging between 23°C and 36°C. Annual rainfall averages approximately 2,300 mm per annum. Peak rainfall periods are September through to December and March through to May. Operational activities able to be conducted across the full year.

5.4 Local resources and infrastructure

The main accommodation centre for the Selinsing Project is Koyan and Kuala Medang which are 3.5 km and 10 km from the process plant respectively and have accommodation for 60% of the mine workforce; and the towns of Kuala Lipis and Raub that have accommodation for approximately 15% of the mine workforce. Either private motor vehicles or a company bus is used to transport employees to and from the mine site.

A map of the mine area showing current infrastructure is provided in Figure 5.1.

Figure 5.1 Location map showing roads, water and power



5.4.1 Electrical power

A 33 KV national grid powerline runs past the leases. Power is allocated by the electricity provider and the agreed offtake is currently 3.8 MW.

With total installed power requirement of 10.3 MW for the upgraded plant with BIOX® and flotation, the current arrangement is deemed insufficient. Hence a revised power supply is being proposed as part of this FS.

5.4.2 Water

The site water supply is drawn from a local river, from which there is no abstraction limit.

5.4.3 Mining personnel

Mine personnel consist of expatriates and in-country professionals. Labour employment covenants require 50% of all employees to be Bumiputra (Malay or indigenous people). There are many universities throughout Malaysia including Kuala Lumpur, Selangor, Penang and Sabah, where many of the site professionals have sourced tertiary training in metallurgy, engineering and geology. There has been an employment survey conducted by Monument. There is currently a good availability of professional and non-professional roles for the Selinsing gold operation.

5.4.4 Tailings storage areas

Snowden (Dec 2016) mentioned a tails storage design was completed by (Knight Piesold) capacity of the current facility is adequate based on 5,434,174 m^3 up to the 530 mRL final crest design.

This design work has now been reviewed and a revised design study with staged lifts is currently underway with SRK. A full description of the revised TSF design is given in Section 18.

5.4.5 Waste disposal areas

The pit oxide waste as waste rock is all considered to be inert. Initial characterisation testwork shows that there are PAF waste rocks in the transition and sulphide rocks, so for this study all the PAF waste is encapsulated in oxide with the expectation that there will be acid forming minerals. The management of these dumps will need to be further identified prior to production.

5.4.6 Processing plant sites

Currently there is an existing oxide plant. It is envisaged that the sulphide ore will be processed through a bio-oxidation leach plant that will be added to existing plant infrastructure. The planned bio-oxidation processing facilities are current based on work completed to a PFS standard by Lycopodium in 2013 and updated for costing in 2016.

There is a design capacity of the bio-oxidation plant of 0.95 Mt/a and this capacity is not breached in the ore process rate identified in the mine production schedule.

6 HISTORY

6.1 Prior ownership and ownership changes

6.1.1 Selinsing

Historical mining of visible gold at Selinsing by crude means probably occurred for centuries prior to 1888, when British companies began production on a larger scale utilising machinery for primary processing treatment, with intermittent mining continuing at Selinsing up until June 2007 when Monument acquired the Selinsing Project. Mining was restarted in July 2009 with a planned production rate of 40,000 oz of gold per year.

The Selinsing Gold Mine (SGM) hosted an Indicated Resource as of November 2007 of 4.82 Mt grading at 1.49 g/t Au (230,000 oz Au) above a 0.59 g/t Au cut-off, with an additional Inferred Resource of 10.32 Mt grading at 1.17 g/t Au (388,000 oz Au) (Snowden, 2007, Addendum to the Technical Report).

The Selinsing gold processing plant began full operation in September 2010. The plant consists of two stages of crushing, with a single-stage ball mill operating in closed circuit, having a throughput of approximately 1,000 t/d. A gravity recovery circuit is used, consisting of a Knelson centrifugal concentrator that operates on a split from the mill cyclone underflow. The Knelson concentrate is subjected to an Acacia high intensity leach with the leached concentrate returned to the ball mill. The mill cyclone overflow discharges to a six-stage CIL cyanidation circuit, with a targeted grind of 80% passing 75 µm and a 36-hour retention time. Loaded carbon is advanced through the leach circuit, collected, then stripped of precious metals with hot caustic, reactivated and recycled. The pregnant solution from the Acacia reactor and from the stripped carbon is sent to the refinery for electrowinning and subsequent production of doré. The leached CIL slurry is discharged to the tailing facility.

Monument completed the Phase III plant expansion in June 2012 increasing production capacity to 1,000,000 t/a from 400,000 t/a. A Phase IV addition of bio-oxidation leach to the existing circuit is outlined in this PFS and technical report, which increases the gold Mineral Resource and Mineral Reserve estimates outlined in this report.

6.1.2 Buffalo Reef

Small-scale mining at Buffalo Reef dates back to the early 1900s with 1,000 m of underground workings developed to 1934, including adits, drifts, crosscuts, winzes and shafts. Production details from that time are not available and are assumed to be relatively small.

Antimony in the form of stibnite occurs in the gold-bearing veins at Buffalo Reef. Stibnite was mined in the 1970s and that amount of material is also assumed to be small.

The Buffalo Reef gold deposit was acquired by Monument in July 2007 through its acquisition of Damar, a wholly owned subsidiary of Avocet (Cavey & Gunning, 2007). Damar (and Avocet) owned the project from 1993 to 2007, when initial exploration commenced.

The major recent drilling program at Buffalo Reef was conducted in phases by Monument from 2010 to May 2012.

6.2 Previous exploration and development work

The information for previous exploration is presented in Section 9 (Exploration), whereas previous development work is discussed in Section 6.1.1 above.

6.3 Historical Mineral Resource and Mineral Reserve estimates

No historical estimates for Mineral Resources and Mineral Reserves are used in this report and all estimates have been revised and are complete and current for this report.

6.4 Production history

The first full year commercial operation in fiscal 2011 produced 44,438 oz of gold, 11% higher than projected mainly due to higher feed grade and higher recovery of the ore materials compared to the budget in fiscal 2011. Subsequent production is in line with projection with increased reliance on lower grade stockpiles with lower mill feed and recovery. A summary of the production history is provided in Table 6.1.

Table 6.1 Production history for the Selinsing Gold Mine

Selinsing/Buffalo Reef	Unit	Year ended 30 June							
		2017	2016	2015	2014	2013	2012	2011	2010
Operating results									
Ore mined	t	179,351	423,011	421,845	494,141	882,159	501,881	740,909	662,330
Waste removed	t	1,208,069	2,489,500	3,196,553	4,245,156	2,831,755	2,770,491	2,707,598	2,326,502
Stripping ratio		6.74	5.89	7.58	8.59	3.21	5.52	3.65	3.51
Ore stockpiled	t	1,736,201	2,335,250	2,755,981	558,252	947,794	920,633	773,432	387,545
Ore processed	t	847,416	992,070	954,165	1,018,972	938,598	364,680	351,999	272,120
Average ore head grade	g/t Au	0.80	0.88	1.45	1.31	2.07	4.24	4.31	3.08
Process recovery rate	%	58.80	67.40	82.40	75.87	86.97	93.70	92.90	58.70
Gold production	oz	12,845	18,155	36,473	35,983	52,982	44,585	44,438	13,793
Gold sold	oz	12,700	23,150	36,500	37,670	57,905	36,938	40,438	13,793
Financial results									
Gold sales	US$'000	15.72	23,595	44,838	48,583	91,275	61,709	56,627	16,316
Cash costs									
Mining	US$/oz	197	114	214	219	112	54	53	64
Processing	US$/oz	667	444	313	326	207	140	120	90
Royalties	US$/oz	68	51	63	66	78	107	69	62
Operations, net of silver recovery	US$/oz	4	3	(3)	2	3	5	-	-
Total cash cost per ounce	US$/oz	936	612	587	613	400	306	242	216

6.5 Brief description of the Project

The SGM will continue to mine and process oxide material as well as process stockpile and Old Tailings material until the Selinsing Gold Sulphide Project (SGSP) production commences in January 2020 when extraction of the available gold sulphide economic Mineral Resource at the SGM will commence. All remaining CIL only treatable stockpiles and Old Tailings will then be treated at the end of mine life.

Mining will involve extensions of the current oxide only open pit mining areas to extract the deeper gold sulphide refractory ore zones. Sulphide ore is then hauled to the existing run of mine (ROM) and stockpile areas for treatment.

A major component of the project is an upgrade of the existing processing plant to enable refractory ore treatment as part of the SGSP.

This upgrade will include:

- Addition of a flotation plant.
- Addition of a BIOX® treatment facility.
- Upgrades to the existing facility, including:
  - ASTER™ process for thiocyanate and cyanide removal

  - New control room and upgrade amenities
  - Upgrade of existing crushing and grinding circuits.
- Closed circuit TV monitoring.
- Additional infrastructure, including:
  - Upgraded HV power supply
  - Improved water and air services in the processing plant
  - Upgraded mine access road
  - Improved data communications with the site
  - Additional TSF capacity.

The SGSP allows for an extension to the mine life of five to seven years, based on current reserves and inventory assessment.

Construction of the process plant upgrade will commence upon funding approval and is anticipated to take one year.

Other key features of the operation in production with the SGSP in progress include:

- Continuous mine production
- Enlarged waste dumps
- Increased manning particularly by the mining contractor
- Mining on nightshift to enable productivities to be achieved.

Other opportunities to be discussed in this report will include underground mining and LOM open pit mining inventory (inclusive of Inferred Resource).

7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional geology

The regional geological setting of the Selinsing and Buffalo Reef gold mineralisation is detailed in Yeap (1993), and was presented by Snowden (2016) as follows:

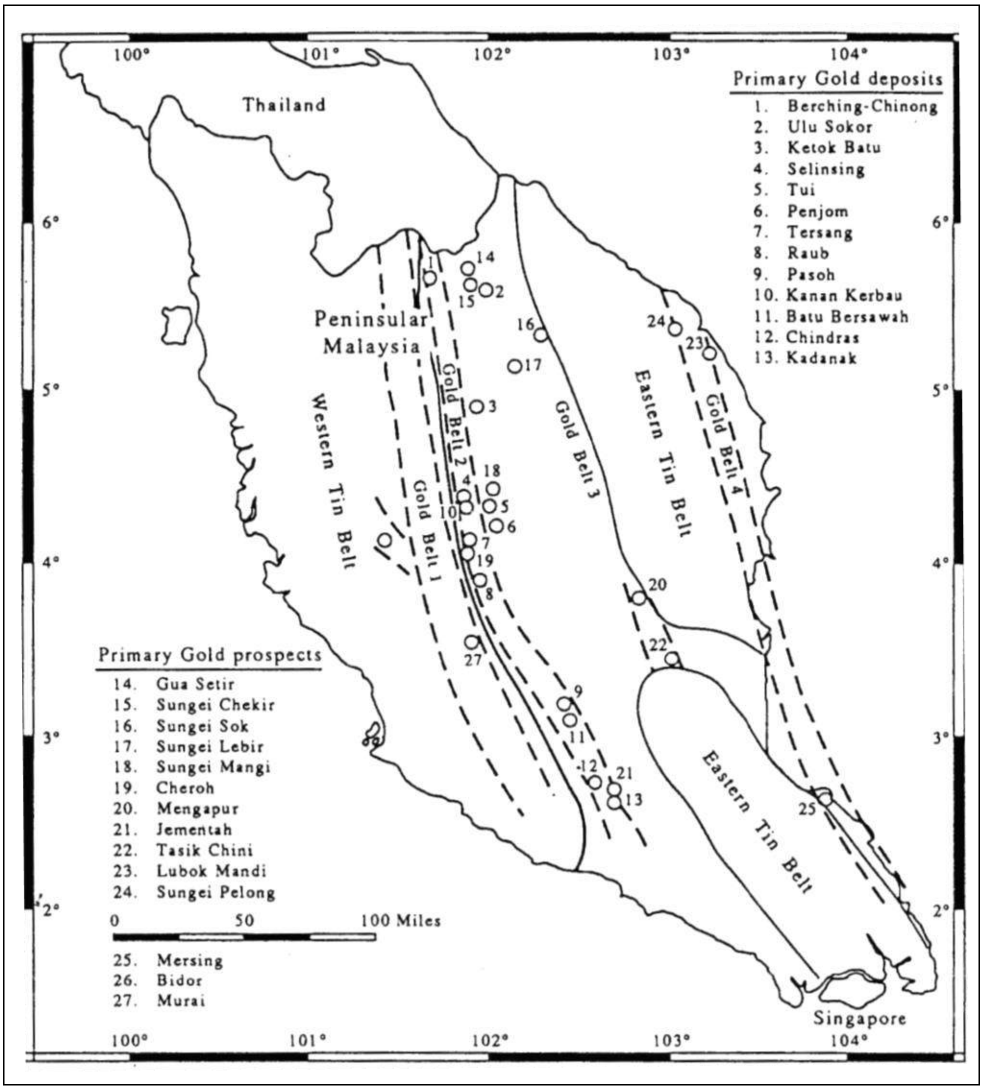
Peninsular Malaysia can be divided into two main regional blocks separated by the Raub-Bentong Line which is a major suture zone. This shear zone divides the Sibumasu Block (Western Block) in the west from the Manabor Block (Eastern Block) in the east (Yeap, 1993). By the late Carboniferous, the Western Block was attached to a continent, possibly Gondwana, and the eastern margin of this was occupied by a shelf which quickly gave way to open ocean.

By the Late Carboniferous to Early Permian, westward subduction of oceanic lithosphere beneath the Western Block, close to the Raub-Bentong suture, was initiated. Riding on this oceanic lithosphere were many continental fragments which were accreted onto the Eastern Block to form the Timur and Tengarra Foreign Terranes. This subduction led to the granitic intrusion that now makes up the Western Tin Belt.

Subduction ceased temporarily and the subduction zone shifted to the east. By the Early Triassic, subduction was re-initiated along a new zone to the east of the earlier zone. With time, gold-bearing fluids are believed to have been released as oceanic lithosphere was subducted beneath the newly accreted wedges of shelf carbonates and marine sediments. These fluids migrated upwards along large regional fractures, cutting the sediments that were newly accreted onto the eastern margins of the Western Block and deposited the gold deposits which constitute Yeap's "Gold Belt 2". Gold Belt 2 (Figure 7.1), or the Berching-Raub-Bersawah Gold Belt, is the best defined of the four gold belts defined by Yeap (1993). The gold mineralisation typically takes the form of veins, reefs and lodes striking from north-northwest to north in moderately to strongly metamorphosed sediments.

In terms of historical gold production, this belt is the most significant as the Raub Australian Gold Mine produced an estimated one million troy ounces of gold bullion between 1889 and the 1960s. Yeap (1993) gives further details of the primary gold occurrences within this belt.

Figure 7.1 Peninsular Malaysia mineral occurrences



Source: Yeap, 1993

7.2 Local geology

The local geology for Selinsing and Buffalo Reef deposits is detailed is discussed in Snowden (2016) as follows:

7.2.1 Selinsing

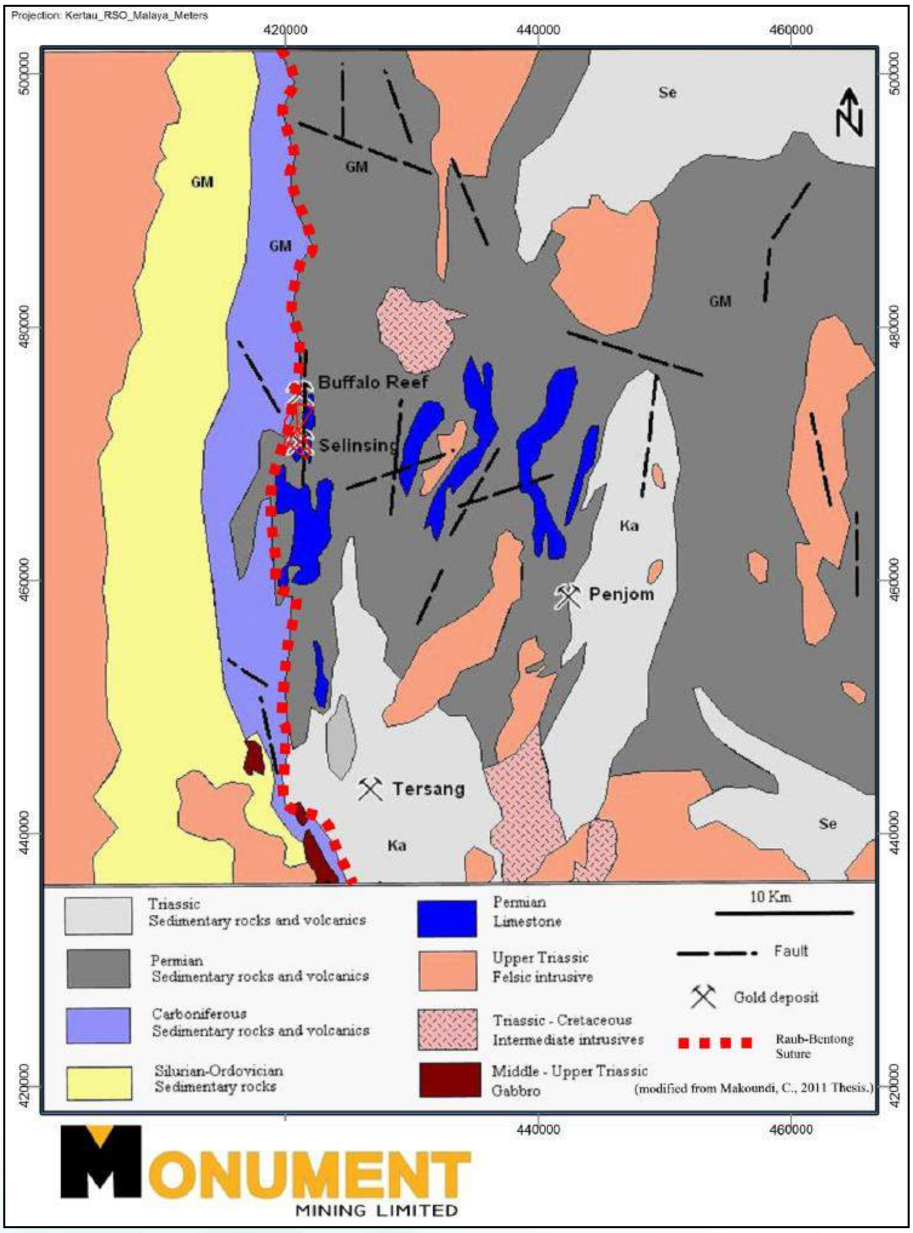
The Selinsing gold deposit is hosted by a 20 m to 100 m thick shear zone that dips 55° to 75° towards mine grid east (082° true grid). This zone or “envelope” of sheared rocks has been variably mineralised and intruded by gold-bearing quartz veins and stockworks. The quartz veins are likely to have been emplaced along fault surfaces, which are thought to be reverse thrusts caused by compression from the east. Strike-slip movement is not thought to be significant; however, a north-westerly trending structure, which post-dates the gold mineralisation, has been identified and may have a strike-slip component. The host rocks for the Selinsing shear zone consist of a series of finely interbedded argillites and very fine-grained arenites, along with sequences of quartz-rich, variably silicified sediments of likely tuffaceous origin, which are referred to as “felsic tuff” and a few thin beds of quartzite conglomerate. These host rocks are collectively known as “the mine sequence series”.

The mine sequence sediments are deep marine epiclastic sediments, originally laid down in low-energy conditions and are thought likely to be of volcanogenic origin. The mine sequence has undergone low-grade regional Greenschist facies metamorphism, which is seen by the development of chlorite in some of the host rocks, more notably the felsic volcanic rocks.

The true thickness of the mine sequence is not well understood. One interpretation is that the mine sequence has a true thickness of about 200 m; however, the footwall contact is not well defined, and it is difficult to distinguish between the mine sequences in the field due to the fine-grained nature of the host rocks. A second interpretation is that within the shear zone, repetition of these units by shearing creates a structural thickening of the sequence.

The hangingwall rocks at Selinsing are a distinctive sequence of predominantly competent, well-bedded, dark-coloured limestones (Figure 7.2). Towards the base of the limestones is a narrow unit of black, well-bedded carbonaceous shales, which may be calcareous in places. The contact of these hangingwall units with the mine sequence below is thought to be a tectonic or faulted contact due to the unconformable nature of the bedding on either side of the contact. The contact itself is characterised by large water-filled clay-lined cavities. The footwall contact of the mine sequence is poorly understood as the base of the mine sequence has not been extensively explored. However, the footwall rocks consist of the same type of grey-black limestone as found in the hangingwall and it is thought that the footwall rocks are the same as the hangingwall, repeated due to faulting. This would mean that the less competent mine sequence rocks were more deformed by shearing due to rheological contrasts between the limestones and the argillites and arenites. The hangingwall limestones have locally-developed folding resulting from easterly compression.

Figure 7.2 Local geology of the Selinsing area



Within the shear zone, there are distinctive tectonic-deformed rock types, the most noticeable of which are cataclasites and mylonites. Local variation in the degree of deformation has resulted in both brittle deformation (cataclasites) and ductile deformation (protomylonites or foliated cataclasites, through to recrystallised mylonites).

Gold and sulphide mineralisation are associated with these rock types as well as intensive replacement by quartz and calcite gangue minerals.

Mineralisation

The gold mineralisation at Selinsing is hosted within a shear zone that strikes at 350° and dips 60–70° to the east, with the higher-grade mineralised shoots within the main mineralised shear plunging to the southeast. The main shear zone is hosted within a sequence of felsic tuff and very fine clastic argillite with calcareous material and limestone in the hangingwall. High grade mineralisation is often associated with quartz stockworks and quartz-carbonate veins within highly deformed sedimentary rocks. Pressure and temperature studies on fluid inclusions in quartz from veins suggest that the mineralisation formed at a temperature in the order of 200–350°C and at a depth of between 2 km and 5 km (Makoundi, 2011).

The gold at Selinsing is generally in the form of fine-grained gold particles (<20 µm) commonly associated with pyrite and arsenopyrite and rarely with chalcopyrite. Visible (mm-scale) gold, although not common, occurs in quartz veins within the shear zone. The higher-grade quartz veins can be over a metre in true thickness and have been traced up to 300 m along strike and 200 m down dip. Lower grade gold mineralisation occurs as finely disseminated gold within intensely deformed envelopes around the quartz veins within the shear zone. Disseminated pyritisation also occurs within the deformed country rock within the shear zone, with the presence of euhedral arsenopyrite as a good indicator of elevated gold grades.

7.2.2 Buffalo Reef

The Buffalo Reef deposit occurs approximately 1 km to the east of the Raub-Bentong suture. The area is dominated by argillite and limestone of Permian age to the east, with conglomerates and sandstones of Devonian age to the west (Figure 7.2). Low grade regional metamorphism up to Greenschist facies (locally up to Amphibolite facies) occurs throughout the area (Naidu, 2005). The sediments have subsequently been intruded by granitic bodies of approximately Jurassic age. These intrusive bodies occur to the east of Buffalo Reef and generally form elevation highs.

The dominant structural feature present is a 200 m wide, north-south striking shear zone, with an apparent sinistral sense of displacement, which parallels the tectonic Raub-Bentong suture to the west. The shear zone is composed of graphitic shale with minor interbedded fine-grained sandstone and tuffaceous rock (Naidu, 2005). Bedding within the sediments typically dips 65–75° to the east and strikes towards a bearing of 330–360° (Flindell *et al.,* 2003).

Mineralisation

Gold mineralisation at Buffalo Reef is structurally controlled and associated with Permian sediments within a 200 m wide shear zone that parallels the north-south trending Raub-Bentong suture. Mineralisation occurs over a total strike length of approximately 2.6 km. Rocks within the Buffalo Reef shear zone have typically undergone silica-sericite-pyrite alteration to varying degrees (Flindell *et al.,* 2003).

The gold occurs within moderately to steeply east-dipping veins and fracture zones, which range in thickness from 1 m up to 15 m in thickness (average thickness is approximately 10 m in the main mineralised veins), although local flexures in the veins can host mineralisation up to 25 m in thickness. Veins, which are boudinaged in some areas, are generally composed of massive quartz with 1–5% (by volume) sulphide minerals, namely pyrite and arsenopyrite, along with varying amounts of stibnite. The stibnite generally occurs in association with elevated gold grades; however, the presence of gold does not necessarily indicate high stibnite levels (i.e. the stibnite tends to be associated with gold, rather than the gold being associated with stibnite).

8 DEPOSIT TYPES

The Buffalo Reef and Selinsing gold deposits are thought to be mesothermal lode gold deposits, with auriferous quartz-pyrite-arsenopyrite±stibnite veining, with associated hydrothermal alteration. The two deposits are structurally controlled and thought to be part of the same structural trend, with the Buffalo Reef deposit occurring along strike and to the north of the Selinsing deposit.

9 EXPLORATION

Exploration at the Selinsing and Buffalo Reef deposits primarily comprises drilling, with additional surface sampling used to assist with targeting of drilling. Exploration prior to June 2012 is documented in the May 2013 NI 43-101 Technical Report on the property (Odell *et al.,* 2013), and a summary of the exploration, other than drilling, was provided by Snowden 2016 and is presented below.

9.1 Pre-2012

Initial exploration at the Selinsing and Buffalo Reef deposits commenced in 1993 and consisted of regional mapping, along with rock chip/float sampling and soil sampling.

Damar and Avocet excavated some 139 trenches at Buffalo Reef between 1993 and 2003, totalling approximately 6,800 m (Flindell *et al.,* 2003); however, majority of this data was either not recorded or the data has since been lost. Data is only available for 34 trenches at Buffalo Reef, totalling 1,345.8 m in length.

In 1999, a VLF-EM (very low frequency-electromagnetic) geophysical survey was conducted at Buffalo Reef over an area measuring 1 km wide x 2.8 km long. The results of the survey concluded that the technique was not able to map the quartz veins and gold mineralisation, although it was useful for mapping the geological fabric (Flindell *et al.,* 2003).

No systematic recording of the exploration data (especially trench sampling) occurred until 2002 (Flindell *et al.,* 2003), when Avocet combined the available data into an exploration database, including grid soil sampling. As such, some of the initial exploration data has been lost.

9.2 Post June 2012

As discussed by Snowden 2016, during the period between June 2012 and February 2016, a total of 141 drillholes were completed at the Selinsing deposit for 20,017.8 m. In the same period, a total of 526 drillholes were completed for 48,119.1 m at the Buffalo Reef deposit, including within the Block 7 area. Monument successfully negotiated with the local landowners to gain access to the Felda Block 7 settler land between the Buffalo Reef South and Central deposits, enabling drilling to commence in May 2013. From January 2016 to December 2016, a total of 6,342 m were drilled at the Selinsing and Buffalo Reef Deposit, mostly for metallurgical drilling, not included in the Snowden model, but considered for S% estimation.

The drilling is discussed further in Section 10.

At the Bukit Ribu road-cut and ridge, Buffalo Reef Gap area and Buffalo Reef North spur (Figure 9.1), a total of 1,525 horizontal channel and trench samples were taken. Additional 1,776 channel and trench samples were taken in the remaining of 2016, in Buffalo Reef North, Buffalo Reef Gap, Buffalo Reef South deposits. Also, pit mapping, grab and trench sampling was undertaken to the west of Selinsing Pit 4. The exploration channel and trench samples are primarily used for the planning and targeting of exploration drilling.

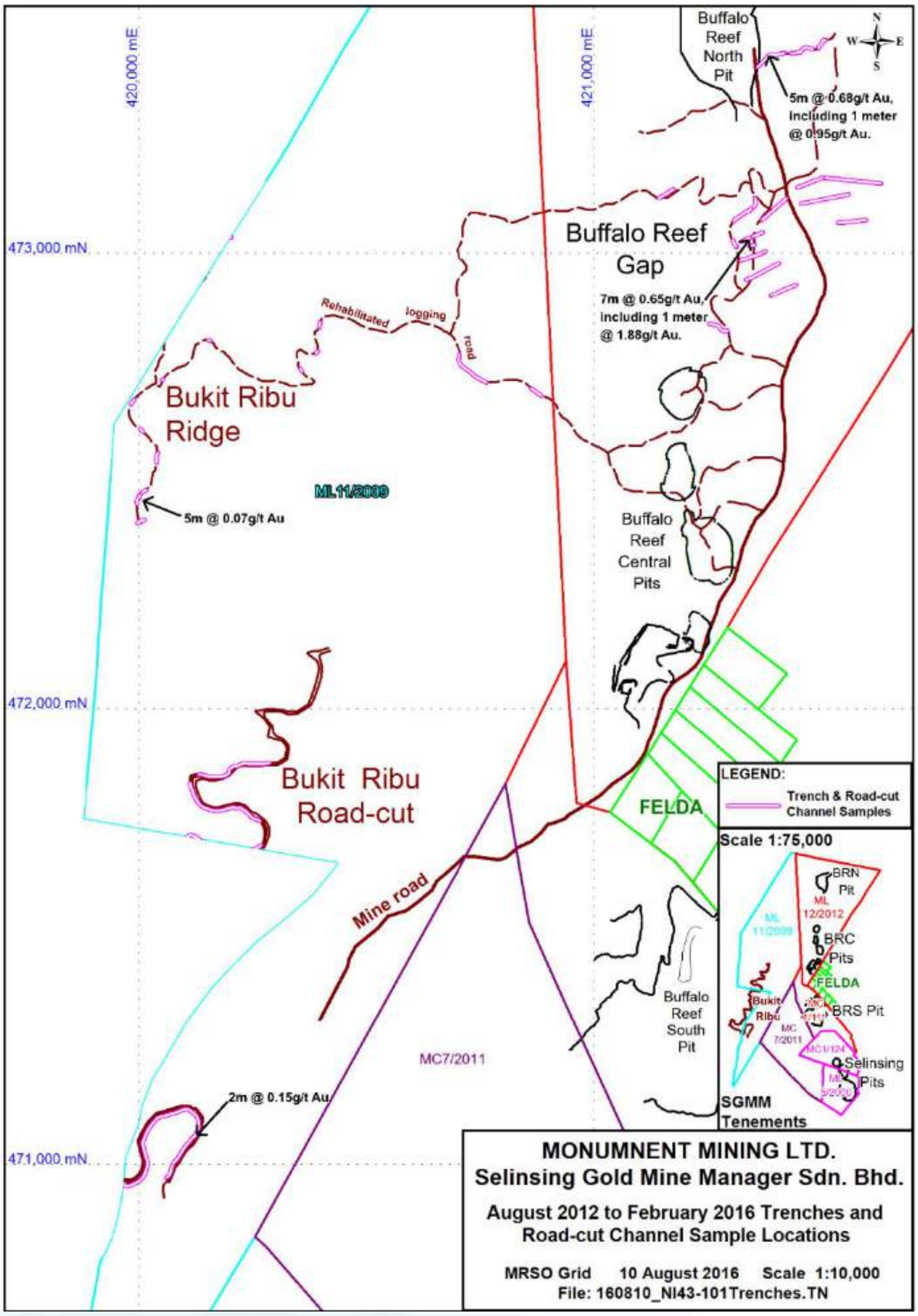
At the Bukit Ribu road-cut, a total of 2,194 m of road side slopes were sampled, with 222 samples collected from three cleared slopes along the road cut. Initially, 5 m reconnaissance-style channel samples were collected, which were later resampled at 1 m intervals where anomalous gold values of >0.1 g/t Au were encountered. A 2 m sample returned 0.15 g/t Au from an altered tuff which is locally brecciated.

At the Bukit Ribu ridge, along an old logging road and at altered portions towards a landslide on the ridge (west of tenement ML5/2009), a total of 13 slopes and one trench were sampled, with a total of 289 m of 1 m channel samples collected from the altered tuff. A 5 m zone returned slightly anomalous grade of 0.07 g/t Au, which Monument indicated requires detailed mapping for further assessment.

At Buffalo Reef Gap, a total of five slopes were channel sampled along the rehabilitated logging road to the west, investigating the historical soil geochemical survey. In addition, 13 trenches were dug at the eastern flats with a total of 870 m of horizontal channel samples, at 1 m intervals, collected. North-northwest trending quartz-vein remnants were mapped in the five westernmost trenches and returned anomalous values of 7 m at 0.65 g/t Au, including 1 m at 1.88 g/t Au. Trenches north and south along this mineralised trend also identified quartz veins returning 13 m at 0.15 g/t Au, 7 m at 0.14 g/t Au and 14 m at 0.21 g/t Au. This mineralised trend confirms the historical soil geochemical anomalies.

At Buffalo Reef North, three road-cut slopes, located at the south flank of the ridge to the east of the pit, were sampled with 1 m channel samples and a total of 155 samples were collected. A 5 m intersection (not true width) grading at 0.68 g/t Au (including 1 m at 0.95 g/t Au) was returned which is interpreted to be the southern continuation of weak mineralisation identified in limited historical drilling. Also, as a north extension of the Buffalo Reef North mineralisation, a total of 162 samples collected from two trenches to the north of Buffalo Reef North pit returned a 3 m intersection (not true width) grading at 1.73 g/t Au (including 1 m at 2.44 g/t Au), and several other intersections above 0.2 g/t Au.

Figure 9.1 Exploration channel and trench sample location plan (2012 to 2016)



Horizontal channel samples were taken from selected slopes and trenches, sampled at 1 m intervals for detailed areas, 2 m for semi-detailed and 5 m for reconnaissance sampling. In near-mine areas, a reference point was established and from the reference point, sample points and intervals were allocated using a compass and tape measure. The tape measure was laid out on the ground below the face or outcrop being channel sampled. Once the channel had been mapped, the geologist marked out the intervals to be sampled using flagging tape, recording the “from” and “to” meterage and sample number. For vertical channel samples, the “depth from” point was considered as the top of the channel and the “depth to” point was considered the bottom.

Sample intervals were marked on a base map, and the sample number, geological description, and global positioning system (GPS) coordinates for samples recorded. Sample points were later surveyed by the Monument mine surveying team using differential GPS or Total Station, depending on the density of the vegetation.

Once sample markers were in place, the surface was cleaned of loose debris to expose the fresh rock surface at chest level. A sample pick was used for soft rock, and chisel and hammer for hard rock. For soil-rich or soft rock with 1 m interval, a sample bag was held with one hand and another holding the sample pick chipping the sample into the sample bag taking care to dig lightly along the interval and collecting the sample as evenly as possible from an approximately 10 cm wide x 2.5 cm deep channel. For longer intervals, chips/chunks were collected with rice sacks spread on the ground with quartering of the collected sample after the channelling, to obtain an appropriate sample weight (less than 4 kg).

9.2.1 Pit mapping

Pit mapping was undertaken in all Selinsing and Buffalo Reef pits, including the collection of local point geology data, to identify areas for grab or channel sampling and understand the ore shoot characteristics. In the Selinsing pits, five groups of 1 m channel samples located to the west of Selinsing Pit 4 were conducted. In the Buffalo Reef Pits, 1 m channel samples were collected from three (slope) road-cuts at the southeast ridge of the Buffalo Reef North Pit.

Critical geological features were mapped by a geologist along the benches or ramps and recorded onto a base map (which included the topography contour lines provided by the mine surveyor with suitable intervals based on the scale appropriate for mapping using the MRSO grid to allow a handheld GPS to locate the point, along with other geological information). Mapped points such as lithological contacts, structural bedding and faults were plotted on the base map directly or onto transparent paper. The mapped points were plotted using the handheld GPS coordinates (ensuring sufficient satellite coverage to achieve an accuracy of at least 3 m) and measured out using the protractor-scale ruler. Where this accuracy could not be obtained, reference points were established, which were later surveyed by the mine surveyors using a Total Station or differential GPS. Planar and linear data were collected as required using a Brunton or Suunto compass and inclinometer. Mapping data was digitised using MapInfo software and imported into Surpac software for visualisation.

As part of ongoing geotechnical evaluation in 2018, geotechnical mapping was conducted together with mapping of major structures at the Selinsing pits, plus geological descriptions of exposures in the BRC2 and BRC3 pits at Buffalo Reef.

10 DRILLING

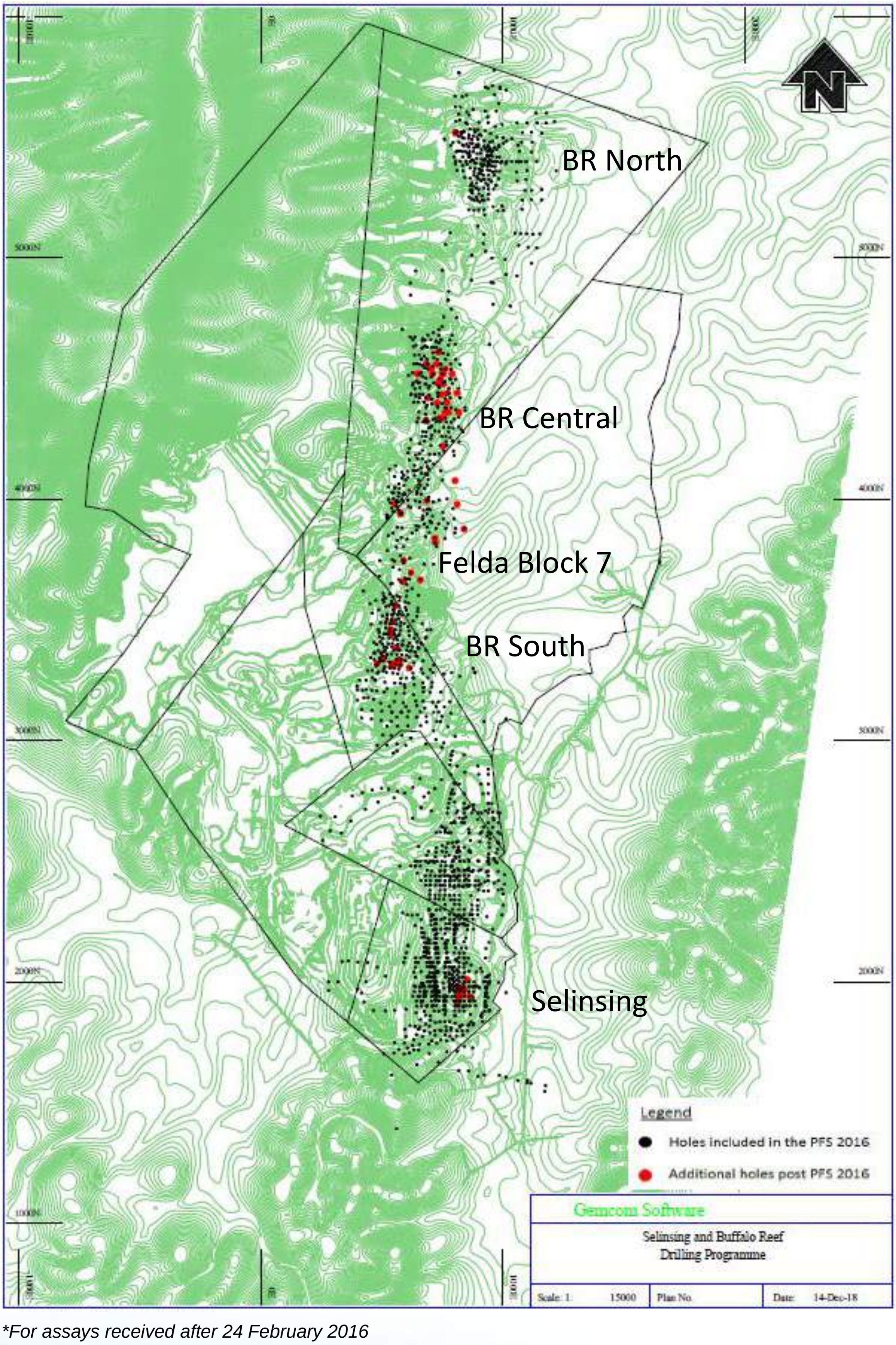
Approximately 145,217 m of drilling, with assays received up to the 24 February 2016, were completed across the Selinsing and Buffalo Reef deposits, comprising predominately reverse circulation (RC) (50.9%) and diamond drilling (41.5%). This drilling was used for the Au, Sb and As estimation for the 2016 resource model compiled by Snowden for. RC and diamond drilling account for approximately 92% of the drilling at the two projects. The drilling up to the end of February 2016 is summarised in Table 10.1 and a collar location plan is presented in Figure 10.1.

Table 10.1 Summary of Buffalo Reef and Selinsing drilling to end February 2016*

Hole type	No. of holes	Length (m)	Proportion of project
Buffalo Reef			
Air-core	-	-	-
Air-core + DD	-	-	-
Auger	21	42	0.1%
Banka	7	54	0.1%
Diamond (DD)	344	36,820	50.2%
DD (metallurgical)	46	3,905	5.3%
Exploration grade control	25	250	0.3%
Reverse circulation (RC)	454	31,636	43.2%
RC + DD	6	601	0.8%
Subtotal – Buffalo Reef	**903**	**73,307**	**100.0%**
Selinsing			
Air-core	56	871	1.2%
Air-core + DD	5	1,504	2.1%
Auger	12	24	<0.1%
Banka	46	422	0.5%
Diamond (DD)	151	23,522	32.7%
Diamond (metallurgical)	4	623	0.8%
Exploration grade control	-	-	-
Reverse circulation (RC)	542	42,353	58.9%
RC + DD (metallurgical)	18	2,591	3.6%
Subtotal – Selinsing	**834**	**71,910**	**100.0%**
TOTAL	**1,737**	**145,217**	

** Cut-off date for assays received up to 24 February 2016*

Figure 10.1 Collar location plan of all drilling up to 2016 with 2 m topographic contours, showing additional drilling assayed after end February 2016*



*For assays received after 24 February 2016

An additional 6,342 m of drilling, which was not included in the 2016 PFS (Snowden, 2016), has been completed by Monument (Table 10.2). Most of the additional drilling at Buffalo Reef comprises diamond core drilling for metallurgical testwork (3,401 m or 60% of the total additional). The remaining 2,293 m of additional infill and extensional drilling at Buffalo Reef do not materially impact the global resource. At Selinsing, the additional drilling since 2016 comprises only 648 m and has confirmed the continuity of a high-grade zone beneath the Pit 4 floor.

Table 10.2 Summary of Buffalo Reef and Selinsing drilling after end February 2016*

Hole type	No. of holes	Length (m)
Diamond (DD)	13	1,612
DD (metallurgical)	16	1,452
Reverse circulation (RC)	10	681
RC + DD (metallurgical)	20	1,949
Subtotal – Buffalo Reef	**59**	**5,694**
Diamond (DD)	9	648
Subtotal – Selinsing	**9**	**648**
TOTAL	**68**	**6,342**

** For assays received after 24 February 2016*

10.1 Drilling history

Drilling prior to June 2012 is detailed in the May 2013 NI 43-101 Technical Report (Odell, 2013). Drilling comprised RC and diamond drilling targeting the Selinsing and Buffalo Reef deposits (excluding the Felda area due to access restrictions at the time).

10.1.1 Selinsing

A six-month exploration program was run from June 2012 to December 2012 within the Selinsing pits areas (Snowden, 2016). This program comprised 45 diamond drillholes totalling 8,450.2 m and 17 RC drillholes totalling 1,065 m. The drilling was designed primarily as infill to the existing holes, with the diamond drillholes testing the deeper down dip extensions of the sulphide mineralisation.

In 2013, a six-month drilling program was run from January 2013 through to May 2013 within Selinsing Pit 4 and Pit 5 and Selinsing Pit 5 and Pit 6. This program comprised: 19 diamond drillholes for 2,596.6 m at Selinsing Pit 5 and Pit 6, and nine diamond holes for 1,441 m at Selinsing Pit 4 and Pit 5. Four RC drillholes were drilled for 402 m at Selinsing Pit 4 and 5. Additionally, a total of 18 diamond drillholes with RC pre-collars were drilled for a total of 2,591.4 m, comprising nine holes at Selinsing Pit 4 and Pit 5 for 1,269.6 m and nine holes at Selinsing Pit 5 and Pit 6 for 1,321.8 m. The purpose of the 2013 drilling was to test the continuation of mineralisation at the north of Pit 6, along with infill drilling. A further four diamond holes were drilled for a total of 623.3 m, for geotechnical purposes.

In January 2014, six diamond drillholes were drilled for 678.5 m, with five of these drillholes at Selinsing Pit 4 and Pit 5 and the other at Selinsing Pit 5 and Pit 6. This program was mainly designed to test the southward extension of mineralisation to the southwest of Pit 4. A second program commenced in May 2014 with four diamond drillholes drilled for 406.7 m, to follow-up the northward extension of mineralisation encountered in the holes drilled in the January 2014 program. The results indicated northward and up-dip continuation of mineralisation determined by the earlier holes drilled to the south and east of these holes. This area is known as “Bamboo Shoot” and has since been mined.

In March and April 2015, seven diamond drillholes were drilled at Selinsing Pit 5 and Pit 6 for 1,166.6 m. This program was designed to infill possible oxide mineralisation to the east of Pit 5 and northwest of Pit 6, as well as to test the deeper sulphide mineralisation.

In November and December 2016, exploration drilling conducted to the east of Selinsing Pit 4 aimed to test the continuity of a high-grade zone beneath the Pit 4 floor, verifying the down-dip extension of the sulphide mineralisation. Nine holes with a total length of 647.9 m were drilled.

The drilling conducted at Selinsing between 2012 and 2016 is summarised in Table 10.3.

Table 10.3 Drilling summary – Selinsing 2012 to 2015

Year	Hole numbers	Prospect	Drilling type	No. of holes	Total metres	Average metres per hole
2012	MSMDD053 to 095	Selinsing Pit 4 and 5	DD	26	4,896.4	188.3
	MSMDD062 to 096	Selinsing Pit 5 and 6	DD	19	3,553.8	187.0
	MSMRC017 to 023	Selinsing Pit 4 and 5	RC	7	455.0	65.0
	MSMRC015 to 031	Selinsing Pit 5 and 6	RC	10	610.0	61.0
2013	MSMDD105 to 165	Selinsing Pit 5 and 6	DD	19	2,596.6	136.6
	MSMDD123 to 153	Selinsing Pit 4 and 5	DD	9	1,441.0	160.1
	MSMRC129 to 166	Selinsing Pit 4 and 5	RC	4	402.0	100.5
	MSMDM154 to 164	Selinsing Pit 4 and 5	DM	4	623.3	155.7
	MSMDD140A to 138	Selinsing Pit 4 and 5	RD	9	1,269.6	144.1
	MSMRD132 to 157	Selinsing Pit 5 and 6	RD	9	1,321.8	146.9
2014	MSMDD168 to 175	Selinsing Pit 4 and 5	DD	8	920.4	115.1
	MSMDD167 and 178	Selinsing Pit 5 and 6	DD	2	164.8	82.4
2015	MSMDD179 to 185	Selinsing Pit 5 and 6	DD	7	1,166.6	166.7
2016	MSMDD196 to 204	Selinsing Pit 4 and 5	DD	9	647.9	72.0

DD = diamond; RC = reverse circulation; DM = metallurgical diamond; RD = RC with diamond tail

10.1.2 Buffalo Reef

Between June 2012 and 2016, extensive exploration was undertaken at the Buffalo Reef projects including Buffalo Reef South (BRS), Buffalo Reef Central (BRC), Buffalo Reef North (BRN), Buffalo Reef Gap (BRG), Selinsing Pit 5 and Pit 6 and Felda. Selinsing Pit 5 and Pit 6 are included as there is some overlap between the boundary of Selinsing and Buffalo Reef South.

In 2012, at BRS a six-month drilling program was carried out consisting of 27 diamond holes for 3,687.6 m and 46 RC holes for 2,552 m. The purpose of this program was to better define future targets for infill drilling. In 2013, exploration at BRS consisted of 23 infill and extension diamond holes for 2,680.4 m, along with two additional diamond holes for 120.4 m to obtain sufficient samples to conduct metallurgical testwork. In 2014, drilling continued at BRS with 13 diamond holes drilled for 1,447.2 m between January and April 2014. In June to July 2014, a further three diamond holes were drilled at BRS for 337.6 m to follow up on results produced from the 2013 drilling. Four metallurgical diamond holes were also drilled in 2014 at BRS for 304 m.

The 2012 drilling program included five RC drill holes of 70 m each, which were drilled at the Selinsing Pit 5 and Pit 6. In 2013, a 100 m diamond hole was drilled at Selinsing Pit 5 and Pit 6 as part of a program of holes mentioned in Section 10.1.1 above and in June 2014, two diamond holes were drilled for 146.5 m.

Between September and December 2012, exploration at BRC consisted of 20 diamond drillholes for 1,507 m, and 26 RC holes for 1,241 m. The purpose of this program was to delineate the main trends of the mineralisation and define future targets. In 2013, exploration at BRC consisted of 32 diamond holes drilled between 40 m and 110 m, totalling 1,992.6 m in length, and one metallurgical diamond hole of 60.5 m. In 2014, 12 diamond holes were drilled at BRC to provide samples of sulphide mineralisation for metallurgical testwork. A further six metallurgical holes were drilled at BRC in January 2015. Between March 2015 and December 2015, 45 diamond holes were drilled for 6,074.3 m and 13 RC holes were drilled for 1,159 m. These holes were designed as infill to improve the resource confidence and to test down-dip mineralisation extensions.

In September 2012, 28 RC holes were drilled for 1,374 m at BRN to delineate the main trends of the mineralisation. This was followed up in December 2012 with seven diamond holes totalling 567 m. In 2013, a total of 29 diamond holes were then drilled for 2,364.3 m, five RC holes (each 80 m deep) totalling 400 m, five diamond holes with RC pre-collars for 470.5 m and one metallurgical diamond hole. This program was designed as a follow up of the previous program. Four diamond holes were drilled in 2014 at BRN for a total of 391.1 m, averaging approximately 100 m each. An additional 11 diamond holes were drilled (totalling 947.6 m) to produce sulphide samples for metallurgical testwork. The metallurgical drilling program continued into February 2015, with a further nine metallurgical holes being drilled for 854.3 m.

Five exploratory shallow RC holes, totalling 250 m, were drilled at BRG in September 2012. Between January and July 2013, a further 21 diamond holes (for 2,998.6 m) were drilled at BRG as an extension of the initial exploration program.

In May 2013, after successful negotiation with settlers, Monument commenced exploration drilling at Felda. The initial program consisted of 69 diamond drillholes between 40 m and 180 m depth, for a total of 5,682.7 m. This program was mainly designed to verify the continuity and extension of BRS and BRC mineralisation. Between September 2015 and December 2015, 13 diamond holes were drilled for 2,217 m and 16 RC holes for 1,927 m, at Felda, predominantly as infill drilling for resource definition and verification of down dip extensions of the mineralisation.

In 2016, diamond core drilling was conducted to collect samples for metallurgical testwork, specifically for the sulphide gold recovery test work. A total of 38 diamond holes were drilled at BRS, BRC, BRN and Felda areas, for a total of 3,659.8 m.

The drilling in 2016 also included a continuation of the infill and extension drilling program at Felda Block 7 and BRC area. This drilling successfully confirmed the continuity of the mineralisation in these areas. A total of 14 RC holes were drilled for a total of 1,143.0 m, along with 15 diamond holes for a total of 1,908.4 m.

The drilling conducted at Buffalo Reef/Felda between 2012 and 2016 is summarised in Table 10.4.

Table 10.4 Drilling summary – Buffalo Reef 2012 to 2016

Year	Hole numbers	Prospect	Drilling type	No. of holes	Total metres	Average metres per hole
2012	MBRDD022 to 098	BRS	DD	27	3,687.6	136.6
	MBRRC169 to 259; MSMRC001 to 034		RC	46	2,552.0	55.5
	MBRDD047 to 065	BRC	DD	20	1,507.0	75.4
	MBRRC167 to 258		RC	26	1,241.0	62.1
	MBRDD066 to 072	BRN	DD	7	567.0	80.9
	MBRRC202 to 252		RC	28	1,374.0	49.1
	MBRRC226 to 230	BRG	RC	5	250.0	50.0
	MSMRC009 to 014	Selinsing Pit 5 and 6	RC	5	350.0	70.0
2013	MBRDD073 to 339	BRN	DD	29	2,364.3	81.5
	MBRRC264 to 273		RC	5	400.0	80.0
	MBRRD265 to 272		RD	5	470.5	94.1
	MBRDM320		DM	1	113.4	113.4
	MBRDD153 to 349; MSMDD099 to 117	BRS	DD	23	2,680.4	116.5
	MBRDM313 to 315		DM	2	120.4	60.2
	MBRDD088 to 350	BRC	DD	32	1,992.6	62.3
	MBRDM317		DM	1	60.5	60.5
	MBRDD081 to 143	BRG	DD	21	2,998.6	142.8
	MBRDD116 to 348	Felda	DD	69	5,682.7	82.4
	MSMDD109	Selinsing Pit 5 and 6	DD	1	100.0	100.0
2014	MBRDD351 to 385	BRS	DD	19	1,784.8	93.9
	MBRDM386 to 389		DM	4	304.0	76.0
	MBRDM367 to 400	BRC	DM	12	915.3	76.3
	MBRDD374 to 379	BRN	DD	4	391.1	97.8
	MBRDM371 to 396		DM	11	947.6	86.1
	MSMDD176 to 177	Selinsing Pit 5 and 6	DD	2	146.5	73.3
2015	MBRDD416 to 480	BRC	DD	45	6,074.3	135.0
	MBRRC460 to 478		RC	13	1,159.0	89.2
	MBRDM401 to 408		DM	6	589.5	98.3
	MBRDM406 to 415	BRN	DM	9	854.3	94.9
	MBRDD455 to 498	Felda	DD	13	2,217.0	170.5
	MBRRC479 to 503		RC	16	1,927.0	120.4
2016	MBRRC505 to 561; *MBRDM511A to 523	Felda, BRC	RC	14	1,143.0	81.6
	MBRDD507 to 555		DD	15	1,908.4	127.2
	MBRDM511 to 570	Felda, BRN, BRC, BRS	DM, RD	38	3,679.8	96.8

*DD = diamond; RC = reverse circulation; DM = metallurgical diamond; RD = RC with diamond tail; *pre-collar RC only*

10.2 Collar surveying

Avocet identified errors in the early drillhole collar surveys (prior to 1996) which were resurveyed; however, not all drillhole collars were able to be located for resurveying (Flindell *et al.,* 2003). Moreover, the relative accuracy of the collar surveys was not recorded. Whilst the quality of the collar coordinates of the early drilling is not ideal, this drilling accounts for less than 10% of the drilled metres and is not considered material to the resource estimate.

Since 2010, the vast majority of drillholes were surveyed by the Selinsing mine surveyors using a Topcon GPS GR-5/GR3 system. Where the holes are located in forested areas and there is a thick tree canopy, collars were surveyed using a Total Station QS1AC. A numbered steel peg was placed at each drillhole collar.

The accuracy of the Topcon GPS GR5/GR3 is ±5 mm horizontally and vertically, while the accuracy of the Total Station QS1AC is ±3 mm in both horizontal and vertical directions.

Collars were surveyed using the local mine grid and then a grid transformation was applied using Surpac Software to the MRSO grid system. The grid transformation was undertaken by the Selinsing mine surveyors using a two-dimensional (2D) transformation with two known coordinate parameters. The local mine grid Reduced Level was established for Monument's mining activity purposes and the reference bench mark was assumed at level 500 m, which is equivalent to an International Reduced Level of 108.19 m. This was adjusted during the grid transformation in Surpac.

The surveyed data was forwarded to the project geologist who then verified the locations in Surpac, before sending the collar location data to the database manager to be loaded into the primary database. Drillhole locations were recorded in the database using both the local mine grid and the MRSO grid.

10.3 Downhole surveying

The Damar, Avocet and Monument RC drilling at Buffalo Reef prior to 2011 had no downhole surveys conducted. In Damar (2002), there is a mention that all diamond drillholes at Buffalo Reef were surveyed downhole after the end of hole had been reached. Downhole survey measurements were taken at depth intervals of either 30 m or 50 m. The downhole surveying methodology for the diamond drillholes is not stated and as such the accuracy of the measurement technique cannot be evaluated.

At Selinsing, holes prior to 1996 were not surveyed; however, given that this accounts for a very small proportion of the drilling and predominantly is in the upper portions of the deposit which has been mined out, this is considered material to the resource estimate. Starting in 1996, drillholes were surveyed downhole using an Eastman single shot wire-line camera, with all diamond drillholes surveyed.

All Monument holes drilled at Selinsing and Buffalo Reef since June 2012 (including the additional holes drilled in 2016) were downhole surveyed by the drillers, who are employees of Monument. The drillers use a Camteq Proshot downhole survey instrument which measures the azimuth to an accuracy of ±0.5°RMS, and the inclination (i.e. dip) with an accuracy of ±0.2° RMS. The Camteq Proshot instrument records azimuth, inclination, magnetic field, roll face, temperature, date and time. Azimuths were recorded using the local mine grid and then transformed in a spreadsheet to the MRSO grid by the project geologist. A gyro survey tool was used to downhole survey a small number of holes when the Camteq instrument was unavailable.

The data from the Camteq instrument was transferred wirelessly from the probe to a handheld control unit. Raw data was saved in a CSV format and transferred via email to the project geologist to be validated in Surpac. The validated downhole survey measurements were then sent to the database manager to be loaded into the primary database. Measurements flagged as "unaccepted" were excluded from the resource modelling, with only "accepted" measurements used. The following criteria for drillhole deviation must be met for the dip and azimuth to be accepted:

- <1° deviation over 9 m
- <3° deviation over 30 m
- <5° deviation over 60 m
- <7° deviation over 90 m
- <9° deviation over 120 m
- <11° deviation over 150 m.

In some cases, the deviation was flagged as "accepted" or "unaccepted" at the discretion of the project geologist, with agreement from the exploration manager.

10.4 Sample recovery

The core recovery for diamond holes drilled by Monument at Selinsing averages approximately 94%. This figure includes intervals from the drill hole collar where recovery is typically expected to be poor, along with some logged cavities. Approximately 75% of intervals at Selinsing recorded a core recovery of 100%. Similarly, for diamond holes drilled by Monument at Buffalo Reef, the core recovery averages 88% (and also includes intervals close to the collar) with 69% of intervals recording a recovery of 100%. The lower core recovery at Buffalo Reef may be partly due to wet ground conditions encountered below the water table, which occurs at approximately 60 m downhole, along with increased shearing/faulting compared to Selinsing.

Snowden considers the sample recovery for diamond drilling conducted by Monument between June 2012 and February 2016, at both Selinsing and Buffalo Reef, is good and should provide samples suitable for resource estimation. The additional drilling completed in 2016 used the same protocols with the majority drilled in fresh rock for metallurgical testwork of sulphide mineralisation. However, that sample recovery has not been measured for any of the RC drilling at Selinsing or Buffalo Reef.

10.5 Geological logging

Drilling information has been recorded using industry standard logging conventions from the drilling at both Selinsing and Buffalo Reef, with the logging procedures properly documented.

All intervals for RC and diamond drilling are geologically logged, with information such as lithology, oxidation, alteration, main minerals and veining recorded. In addition, geotechnical information is recorded for diamond holes, such as recovery, rock quality designation (RQD), rock strength and structural data.

Observations are recorded onto paper log sheets in the field (RC drilling) or in the core shed (diamond drilling). The data is then entered in formatted data-entry templates by the geologist before being emailed to the database manager to be loaded into the primary database.

The core is photographed before being sampled.

10.6 Sampling method and approach

10.6.1 Diamond core

The diamond drilling at Selinsing and Buffalo Reef was undertaken by Monument after 2012 using two Desco SP6500SA rigs, which are owned by Monument. Diamond drilling included both PQ (used mostly for metallurgical drilling) and HQ (63.5 mm core diameter) diameter.

The diamond core was placed by the drillers into plastic core trays at the rig, where the geologist undertook a "quick log" before the core boxes were taken to the core shed for detailed logging and sampling. Samples were collected using a nominal 1.5 m sample length, with a minimum length of 0.5 m. Sample intervals were designated according to geological features such as mineralisation, lithology, alteration and structure.

The core was cut in half using a diamond saw, with one half of the core placed in numbered sample bags which were subsequently submitted to the assay laboratory in batches, with blind certified reference materials and uncertified blanks inserted into the sample sequence. Quarter core samples were taken from a number of sections of metallurgical holes.

Damar (2002) indicates that early diamond drilling at Buffalo Reef was HQ diameter triple tube drilling. The diamond core was placed in wooden core boxes and, after geological logging, the core was cut in half using a diamond saw. One half of the core was then placed in numbered sample bags which were subsequently submitted to the assay laboratory in batches. Sample intervals were based on the lithological contacts with a minimum sample length of 0.8 m and a maximum length of 2.0 m (Damar, 2002).

10.6.2 Reverse circulation

RC drilling conducted by Monument since approximately 2007, at both Selinsing and Buffalo Reef, was primarily drilled using a Monument-owned rig using a 4½-inch face sampling bit. The drill rig utilises a 350-psi ELGI Air compressor, with an additional 650-psi booster compressor available. The drill hole was flushed with compressed air at the end of each rod run (i.e. every 6 m). Drill cuttings were collected at 1 m intervals downhole via a cyclone in marked plastic sample bags. Bulk samples were subsequently split using a tiered riffle splitter to obtain a 25% split (refer Figure 10.2) which was collected in pre-numbered calico sample bags. The rejects were retained in plastic sample bags and stored at the drill site.

Figure 10.2 Three-tiered riffle splitter (photo taken during 2016 Snowden site visit)

For wet or damp samples, the sample was dried in an oven at the Selinsing mine laboratory for approximately six to 10 hours. Once dry, the samples were split using a tiered riffle splitter to obtain a 25% split which was collected in a pre-numbered calico sample bag.

Sample condition is poorly recorded in the database for RC drill holes drilled by Monument, with only 30% of the sample intervals recording the sample condition. For the samples that have the sample condition recorded, 97% of the mineralised samples (>0.5 g/t Au) are recorded as dry samples, with the remaining 3% recorded as wet samples.

RC drilling within the Block 7 area in 2015 and 2016 was conducted by a local drilling contractor, DRC Drilling Services S.B., using a Desco SP700SA-RC rig following the same sampling method described above (Figure 10.3).

At Buffalo Reef, RC drilling completed by Damar utilised a 4½-inch face sampling drill bit (Damar, 2002). Cuttings were collected from the cyclone at 1 m intervals and passed through a “standard riffle splitter” which collected a sample of approximately 2 kg (Damar, 2002). Damar (2002) notes that where wet samples were encountered, the sample was left to settle and then split “manually”; however, details of this process are not documented. Snowden notes that the number of wet samples is not recorded for the Damar drilling.

Figure 10.3 RC drilling by contractor at Buffalo Reef (photo taken during 2016 Snowden site visit)

Snowden considers the sampling of the diamond drilling follows standard industry practices, with half-core samples collected based on a 1.5 m sample interval, adjusted to suit the geology. In Snowden’s opinion, the diamond core samples should be representative and are considered suitable for resource estimation.

For the RC drilling, a three-tiered riffle splitter is used to collect the sample from the RC cuttings. Tiered riffle splitters, whilst reasonable, do not provide a truly representative sample due to a delimitation error. Between each tier, the assumption is that the material from the slot above will spread evenly across the next tier; however, in practice the material tends to fall down the sides of each chute and does not mix.

Whilst overall the RC sampling is, in Snowden’s opinion reasonable, it is recommended that Monument investigate alternative splitting techniques for the RC drilling, such as the use of a cone splitter mounted below a collecting-box below the cyclone, to improve the quality of the RC samples. Moreover, riffle splitting is a labour intensive, manual process and relies on correct operation of the splitter by the sampling personnel (e.g. correctly pouring the lot into the top of the splitter). Additionally, Snowden notes that the condition of the spitter in Figure 10.2 is not ideal.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

The primary laboratory used to prepare and analyse samples from Selinsing and Buffalo Reef was SGS (Malaysia) in Mengapur.

11.1 Laboratories used

Several laboratories have been used to prepare and analyse samples from Selinsing and Buffalo Reef over the history of the project, including:

- SGS (Malaysia), Mengapur
- SGS (Malaysia), Port Klang
- Inspectorate, Vancouver
- PT Intertek, Jakarta
- Selinsing (SGM) on-site laboratory
- ALS, Malaga, Western Australia.

11.1.1 SGS (Malaysia), Mengapur

The majority of samples included in the 2016 resource estimate by Snowden and the 2017 update by Monument to add estimates of S, were analysed by the SGS on site laboratory in Mengapur. The SGS Mengapur laboratory falls under the jurisdiction of the SGS Port Klang laboratory (which complies with ISO 17025:2005), with final assay reports signed off by SGS Port Klang. The SGS laboratory in Mengapur ceased operating in March 2017.

Samples submitted to the SGS Mengapur laboratory were analysed for gold, arsenic, silver and antimony, also sulphur, calcium, manganese, lead and iron. Gold was analysed by fire assay using a 50 g charge with an atomic absorption spectroscopy (AAS) finish. A number of samples were also analysed for gold by screen fire assay with an AAS finish.

11.1.2 SGS (Malaysia), Port Klang

In 2012, drill samples were submitted to SGS (Malaysia) in Port Klang, an accredited laboratory that complies with ISO 17025:2005 requirements for sample preparation and analysis.

The samples were analysed for gold by fire assay with a 50 g charge and an AAS finish. Silver, arsenic and antimony were analysed ICP (inductively coupled plasma).

11.1.3 Inspectorate, Vancouver

Samples from two RC drillholes at BRC in September 2012 were analysed by the Inspectorate (now Bureau Veritas) laboratory in Vancouver for multi-element analysis. The Inspectorate laboratory in Vancouver is accredited as complying with the requirements of ISO 17025:2005 for gold analysis.

Samples were analysed for gold by fire assay (unknown charge weight) with an AAS finish.

11.1.4 PT Intertek, Jakarta

In 2012 and 2013, samples from 41 drillholes were submitted to PT Intertek in Jakarta, Indonesia to overcome a backlog of samples being submitted to SGS, Mengapur. The PT Intertek laboratory in Jakarta is accredited as complying with ISO 17025:2005 requirements with respect to gold analysis.

Samples submitted to Intertek were analysed for gold by fire assay using a 30 g charge with an AAS finish.

11.1.5 Selinsing (SGM) on-site laboratory

Samples from four drillholes at Selinsing were submitted in mid-2013 to the Selinsing (SGM) on-site laboratory due to a backlog of samples being submitted to SGS Mengapur.

These samples were analysed for gold by fire assay; however, the charge weight and finish are not documented and Snowden notes that metadata for these analyses have not been captured in the database.

11.1.6 ALS, Malaga, Western Australia

Between March 2014 and March 2015, approximately 728 check samples, including coarse crush samples (i.e. duplicate coarse rejects) and pulp samples, from 256 diamond drillholes, were dispatched from SGS (Mengapur) and Intertek (Jakarta) under instruction from Monument to the ALS laboratory in Malaga, Western Australia for umpire analysis as part of Monument's quality assurance/quality control (QAQC) protocols. The ALS laboratory in Malaga is accredited by the National Association of Testing Authorities, Australia (NATA) as meeting the requirements of ISO 17025:2005.

Samples were analysed at ALS by fire assay with an AAS finish.

11.2 Laboratory sample preparation and analysis

11.2.1 2010 to 2016

Samples for Monument drilling between 2010 and 2016 were submitted to SGS (Malaysia) in Mengapur for gold analysis by fire assay using a 50 g charge with AAS finish. Multi-element analysis was carried out using four-acid digest (comprising HNO_3, $HClO_4$, HF and HCl) with an ICP-OES (inductively coupled plasma – optical emission spectroscopy) finish.

Samples are dried and then crushed using a jaw crusher. The crushed samples were riffle split twice and then pulverised to P_{90} 75 µm. Results of grind size testing, along with sample weights, were reported for every batch of samples.

The laboratory process effective 10 March 2014 followed by SGS in Mengapur is as follows:

1) Sample received and checked against sample submission sheets.
2) Sample weighed (wet weight).
3) Oven drying at approximately 105°C for six to eight hours.
4) Primary jaw crushing to <10 mm.
5) Secondary crushing by jaw crusher or Boyd crusher to P_{90} 4 mm:
   a) One in 20 samples are dry sieved as part of internal laboratory QAQC.
   b) Coarse reject is put back into original plastic bag and tied with cable tie. The coarse rejects are returned to Selinsing.
   c) 250 g of coarse reject put in paper craft packet and sealed with a sticker for retention by SGS. All coarse residues are returned to Monument.
6) Riffle split to approximately 1.5 kg and pulverised to P_{90} 75 µm:
   a) One in 20 pulps wet screened as part of internal laboratory QAQC.
7) Subsample of pulp by riffle splitter:
   a) Second split (duplicate) one in 20 for internal laboratory quality control.
   b) 250 g of pulp placed in paper craft packet and sealed with a sticker for retention by SGS. All pulp residues are returned to Monument.
   c) Pulp reject is placed into a plastic bag, sealed with a cable tie and returned to Monument.

8) 250 g subsample of pulp put into paper craft packet and used for multi-element analysis (including silver, arsenic and antimony) by ICP-OES (this sample is retained by SGS).
9) A further 200 g subsample of the pulp is collected and analysed for gold by fire assay using a 50 g charge with the gold content determined by AAS.
10) Assay results are emailed by the laboratory to Monument in PDF and CSV format. The Monument database manager uploads the csv files, including all metadata, into the primary database.

11.2.2 Pre-2010

Sample preparation procedures for samples from drilling before 2010 are detailed in the May 2013 NI 43-101 Technical Report (Odell, 2013) and summarised as follows.

Selinsing

Sample preparation at Selinsing for samples collected up to drillhole SELRC280, was undertaken by Target Resources Australia NL (TRA) at a sample preparation facility at Kuala Lipis. This facility was inspected (company not documented) in March 1997 and problems with equipment were identified. A new sample preparation facility was subsequently commissioned at the Selinsing Project site in April 1997.

RC samples at the Selinsing laboratory were dried and then split through a 50:50 bench scale riffle splitter prior to pulverising. Half of the original 2 kg sample was discarded due to a maximum pulveriser bowl capacity of approximately 800 g. Diamond core samples were crushed using an Essa jaw crusher and then sampled as per the RC samples. If the sample was wet the sample was returned to its bag and dried prior to splitting. Each sample was pulverised in Essa RM2000 pulverisers for four minutes to P_{95} 75 µm. A 250 g subsample was then collected and dispatched. The site laboratory prepared all 2010 to 2012 RC and diamond core samples from the Phase 2 Selinsing drilling program, (SELRC281 to 509, SELDD0001 and SELDD0003 to SELDD0013).

Prior to 2007, TRA resubmitted samples from the Selinsing Phase 1 drilling program to the AssayCorp laboratory in Kuching for regrind and re-assaying, resulting in duplicate certificates for the same sample number, but using different analysis techniques. This second pulverisation was due to the coarse nature of the samples that TRA had previously prepared. The pulverising was conducted using either a disc grinder or a Keegor Mill. 250 g to 300 g samples were reground to P_{90} 100 µm and then a 50 g charge was taken for fire assaying.

For TRA's Phase 2 drilling program at Selinsing, samples did not require regrind and a 50 g charge was split out immediately for fire assaying with the gold determined by AAS. Assay precision (i.e. repeatability) was quoted at ±15% with a minimum detection limit of 0.01 ppm Au. AssayCorp was instructed to re-assay Selinsing samples with results >1 g/t Au, until another result within 15% was obtained.

Buffalo Reef

RC and trench samples collected by Damar at Buffalo Reef prior to 2007 were initially analysed for gold at an on-site laboratory facility which determined the gold concentration by titration, following aqua regia digestion of a 20 g subsample ground to 150 µm (Flindell *et al.,* 2003). Flindell *et al* (2003) notes that this technique is prone to errors and inaccuracy due to the coarseness of the gold, the association of the gold with sulphides (refractory nature) and encapsulation within quartz grains. These factors typically result in an underestimation of the gold concentration in the sample. Damar subsequently re-assayed 528 RC samples by fire assay at the MMC laboratory in Kuala Lumpur and at Analabs in Kuching. Flindell *et al* (2003) suggests that any titration assays remaining in the drillhole database are limited to trench samples only.

All samples collected by Avocet at Buffalo Reef were analysed for gold by fire assay at either the Penjom Mine site laboratory or at Analabs in Kuching (Flindell *et al.,* 2003). These laboratories are not designated in the drillhole database and as such, a comparison of results cannot be made between the two laboratories.

Samples collected during the October 2007 to September 2008 Monument drilling campaigns at Buffalo Reef were submitted to the Ultra Trace Pty Ltd (Ultra Trace) laboratory in Perth, Western Australia, for sample preparation and assaying (Harun, 2011). The samples, which weighed up to 5 kg, were dried and pulverised to a nominal P_{95} 75 µm. Silica sand was used by Ultra Trace between sample batches to clean pulverisers. The samples were fused using fire assaying techniques followed by a four-acid digest, consisting of hydrochloric acid, hydrofluoric acid, nitric acid and perchloric acid. The gold concentration was determined by ICP-OES, while arsenic and antimony were determined by inductively coupled plasma mass spectroscopy (ICP-MS).

11.3 Sample security

Security procedures used to ensure the integrity of sampling at Selinsing prior to 2007 are not documented and as such, security measures employed are unknown.

Core is stored on racks in a core shed located at the Selinsing Project, with core collected for both Selinsing and Buffalo Reef stored at the Selinsing core facilities. The main core storage facility is maintained by Monument staff and kept secure (Figure 11.1). Some of the core is kept temporarily at the core processing facility.

Figure 11.1 Core storage area (top) and core racks in the core processing facility (bottom)

11.4 Quality assurance and quality control

A systematic or independent QAQC program was not applied during the Damar and Avocet drilling and sampling campaigns at Buffalo Reef.

For the drilling conducted by Monument, Monument included certified commercial reference materials (CRMs), uncertified blanks and field duplicates in the sample batches as part of their QAQC protocols. The CRMs are sourced primarily from Geostats Pty Ltd (Geostats).

11.4.1 Pre-2012

The results from the QAQC samples for drilling prior to 2012 were assessed by Snowden in the 2011 NI 43-101 Technical Report on the Buffalo Reef project and in the May 2013 NI 43-101 Technical Report (Odell, 2013). A summary of the results is provided below.

Odell (2013) concluded that the CRM assay results from 2010 to 2012 show acceptable analytical accuracy was achieved and that no significant analytical bias is present. No outliers were identified by Odell (2013) in the CRM results. Similarly, control charts from CRMs inserted into the 2007 to 2011 RC sample batches show that good accuracy was achieved in the laboratory assaying and that no significant analytical bias is present.

Results of 40 blank samples inserted into the sample batches from 2010 through to 2012 show grades on or around the analytical detection limit. The limited blanks from this period show no evidence for contamination during laboratory sample preparation or analysis.

In 2007, a total of 6,982 repeat gold analyses from Selinsing were performed by Monument on pulps in their primary laboratory. The results demonstrated acceptable correlation between the original and repeat analysis with less than 10% of the duplicate analyses considered to be poorly matched.

Field duplicate samples were collected by Monument during the RC drilling programs of 2008, up to drillhole MBRRC0166 as at November 2010. The results show the overall precision of the field duplicates, whilst not optimal, is reasonable and that no systematic bias is present. Analysis of duplicated sample assays derived from both Selinsing and Buffalo Reef datasets during 2010 to 2012 show acceptable results from repeat gold samples performed on pulps.

11.4.2 2012 to 2016

QAQC reports produced by Monument were provided to Snowden in 2016, covering the period June 2012 to February 2016. QAQC reports for the period covering the drilling between February 2016 and December 2016 have also been compiled by Monument.

In 2013, SGS Mengapur implemented a number of different routine laboratory checks to assess accuracy and repeatability at all levels, including sample preparation and analysis. These measures included:

- For assay batches, which comprised 25 samples for fire assay and 22 samples for ICP, one replicate (random pulp), one standard (SGS CRM) and one blank was included.
- Wet sizing analysis (90% passing 0.075 mm) for pulps at a rate of approximately one sizing analysis for every 18 or 19 samples, set randomly by the LIMS system.
- Inclusion of a duplicate every 18 or 19 samples, set randomly by the LIMS system. Duplicate samples comprised a second split of the coarse reject (i.e. after crushing).
- All duplicates, replicates and internal lab QAQC samples were analysed by CCLASS (the LIMS system used by SGS Mengapur). The data was provided to Monument for analysis and included in the Monument's internal QAQC reports.

Certified reference materials

CRMs were included in the sample batches submitted to the laboratories by Monument at a rate of approximately 1:10. The CRMs were sourced from Geostats and range from 0.49 g/t Au to 16.92 g/t Au. The CRMs and their expected values used during the reporting period are listed in Table 11.1.

Table 11.1 CRMs and expected values used between June 2012 and December 2016

Reference material	Expected value (g/t Au)
G901_5	1.65
G901_7	1.52
G909_1	1.02
G909_7	0.49
G910_4	16.92
G910_7	0.51
GBMS304-1	3.06
GBMS304-4	5.67
GBMS911-1	1.04

CRM GBMS304-4 was introduced by Monument in mid-September 2015; however, this CRM was discontinued in December 2015, as Monument found that the CRM consistently failed to produce results within two standard deviations of the expected value. After investigation, the CRM was reintroduced into the QAQC program in June 2016, and performed well until the end of 2016, reporting well within two standard deviations of the recommended expected value.

Control charts for CRMs for the period June 2012 to February 2016 (Snowden, 2016) show that reasonable accuracy was achieved by the laboratories with a slight negative bias (<5%) present. This trend continued through the rest of 2016. A small number of the total number of CRM assays (<5%) reported outside of three standard deviations from the expected value. The majority of these are likely due to data entry errors or mislabelling in the field.

In the fourth quarter of FY2016 some issues with CRMs being mislabelled on site were observed, with over 14% of all the CRMs submitted to the laboratories for analysis reporting well outside the recommended expected values. Most of the outliers are thought to be caused by mislabelling or placing the CRMs into the incorrect sample bag. The procedure for inserting CRMs was reviewed on site, and the revised procedure was implemented in mid-June, with closer supervision by the responsible geologist. In the first and second quarter of FY2017, following the implementation of the improved procedure on site for inserting blanks and CRMs in the sample sequence, there was a marked improvement in the quality of this data, with very few (<1%) CRMs submitted reporting outside the recommended expected values.

Example control charts are provided for CRMs G910_4 and G901_7 in Figure 11.2 and Figure 11.3 respectively. The outliers in these control charts are primarily due to mislabelling.

Figure 11.2 **Control charts for CRM G910_4 (16.92 g/t Au) for 2012 to 2014 (top), 2014 to 2015 (middle) and 2015 to 2016 (bottom)**

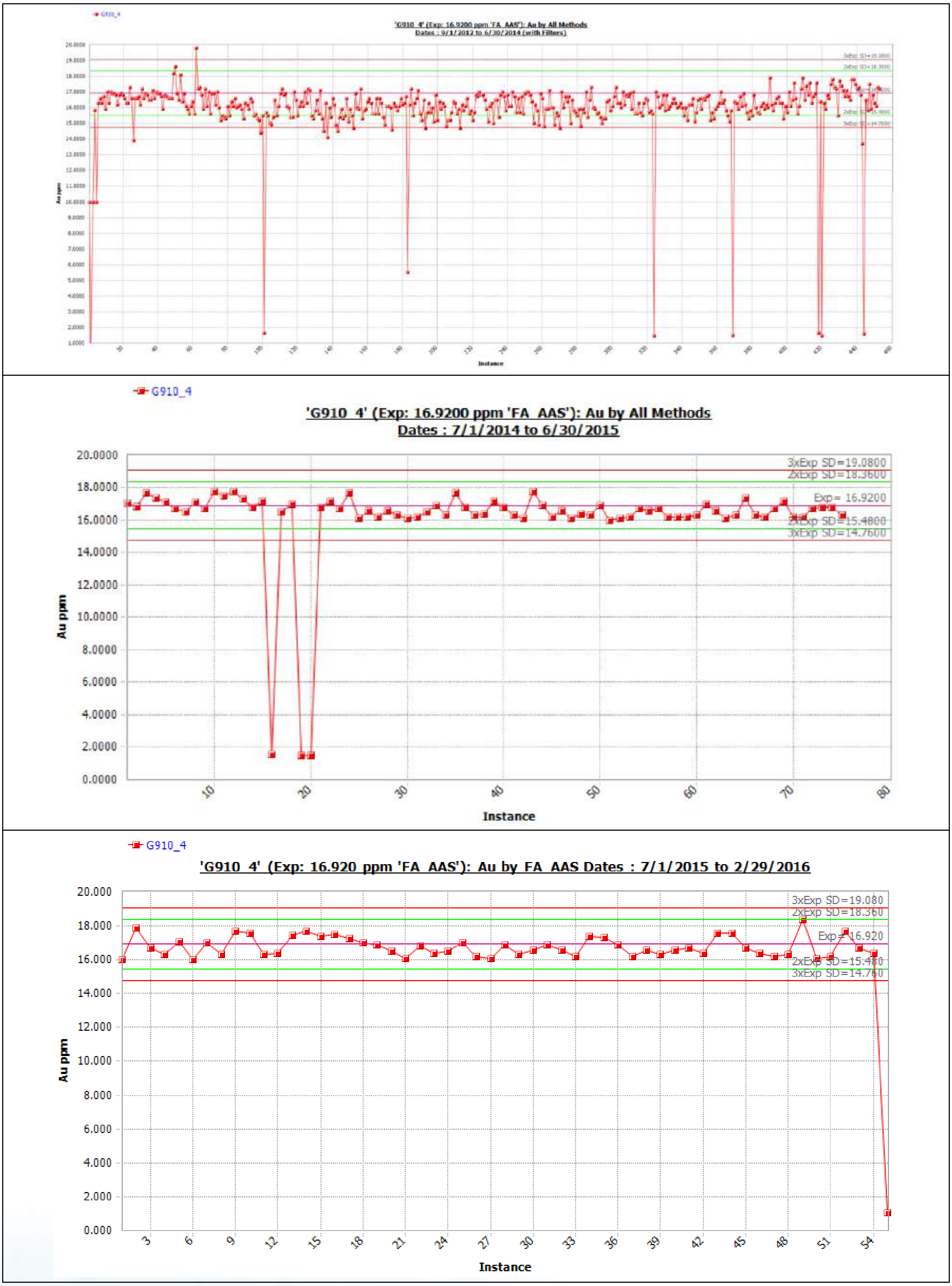
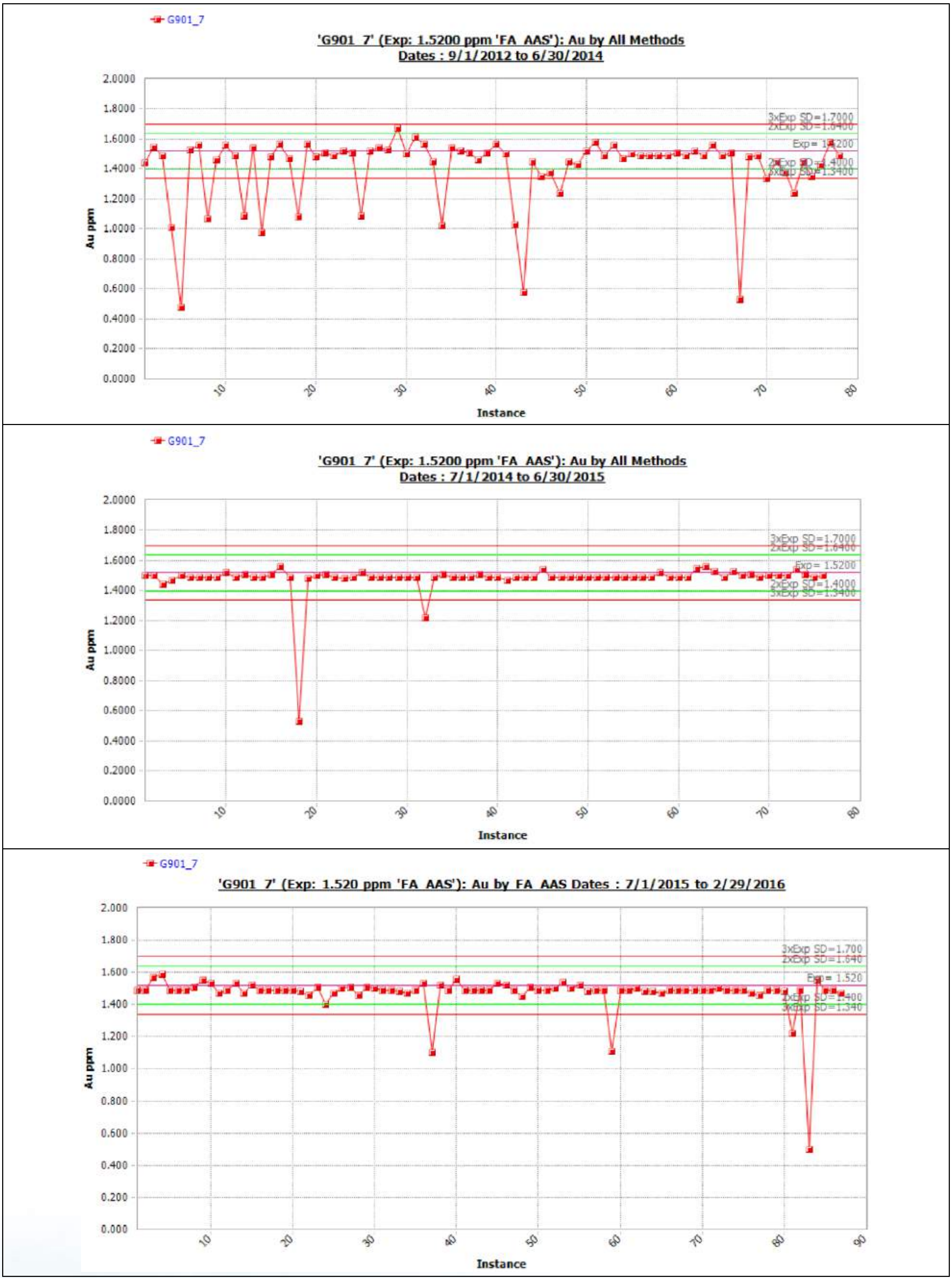


Figure 11.3 Control charts for CRM G901_7 (1.52 g/t Au) for 2012 to 2014 (top), 2014 to 2015 (middle) and 2015 to 2016 (bottom)



Blanks

Blank samples were inserted in the sample sequence at a rate of approximately 1:10 to assess contamination between samples during sample preparation and assaying. The blank material was sourced locally from a granite quarry located in the Cameron Highlands and is not certified.

Snowden (2016) reported that majority of the blanks used until February 2016 returned a value close to the detection limit of 0.01 g/t Au (Figure 11.4). The same trends continued through to the end of 2016. In Snowden's opinion, the blanks generally report within an acceptable range and show that contamination of samples between 2012 and 2016 was negligible.

Field duplicates

Monument submitted field duplicate samples at a ratio of approximately 1:20 samples to assess the precision of the field sampling and laboratory preparation. Field duplicate samples were collected using a riffle splitter and were submitted into the sample stream to the primary assay laboratory.

Scatterplots reported in Monument's internal QAQC reports show that the precision (i.e. repeatability), whilst not ideal is reasonable and quantile-quantile (Q-Q) plots show no bias between the duplicate pairs (Figure 11.5). Whilst reasonable, the less than ideal precision may be due to a combination of poor splitting and subsampling practices in the field and/or laboratory, along with the presence of some potentially coarse gold. At the fourth quarter of FY2016, slightly over 7% of internal laboratory coarse duplicates returned poor results, indicating a reasonably good level of precision was achieved for the sample preparation practices at the laboratory.

Monument indicated that the same trends for field duplicates continued through to the end of 2016.

Figure 11.4 Control charts for uncertified blanks for 2012 to 2014 (top), 2014 to 2015 (middle) and 2015 to 2016 (bottom)

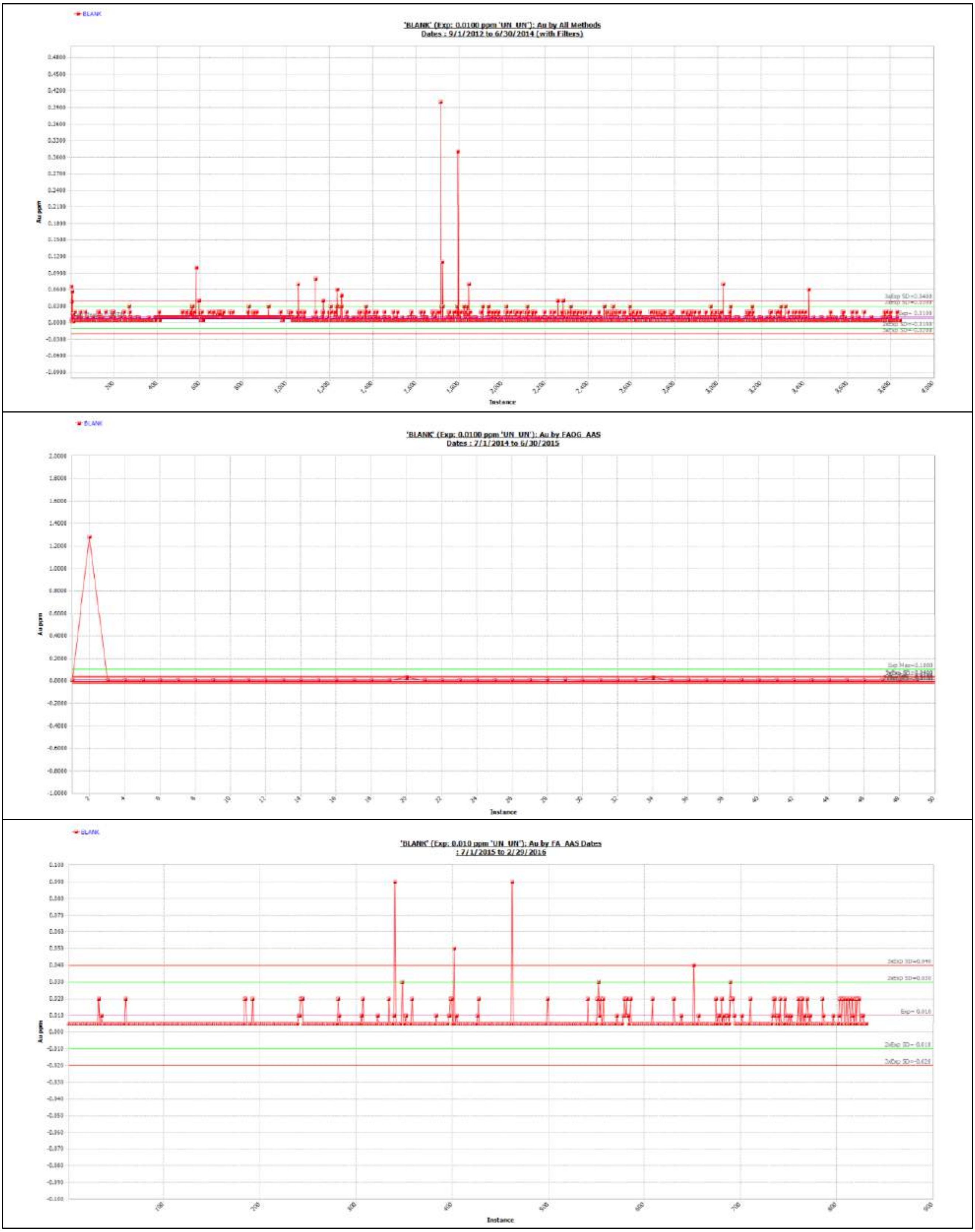
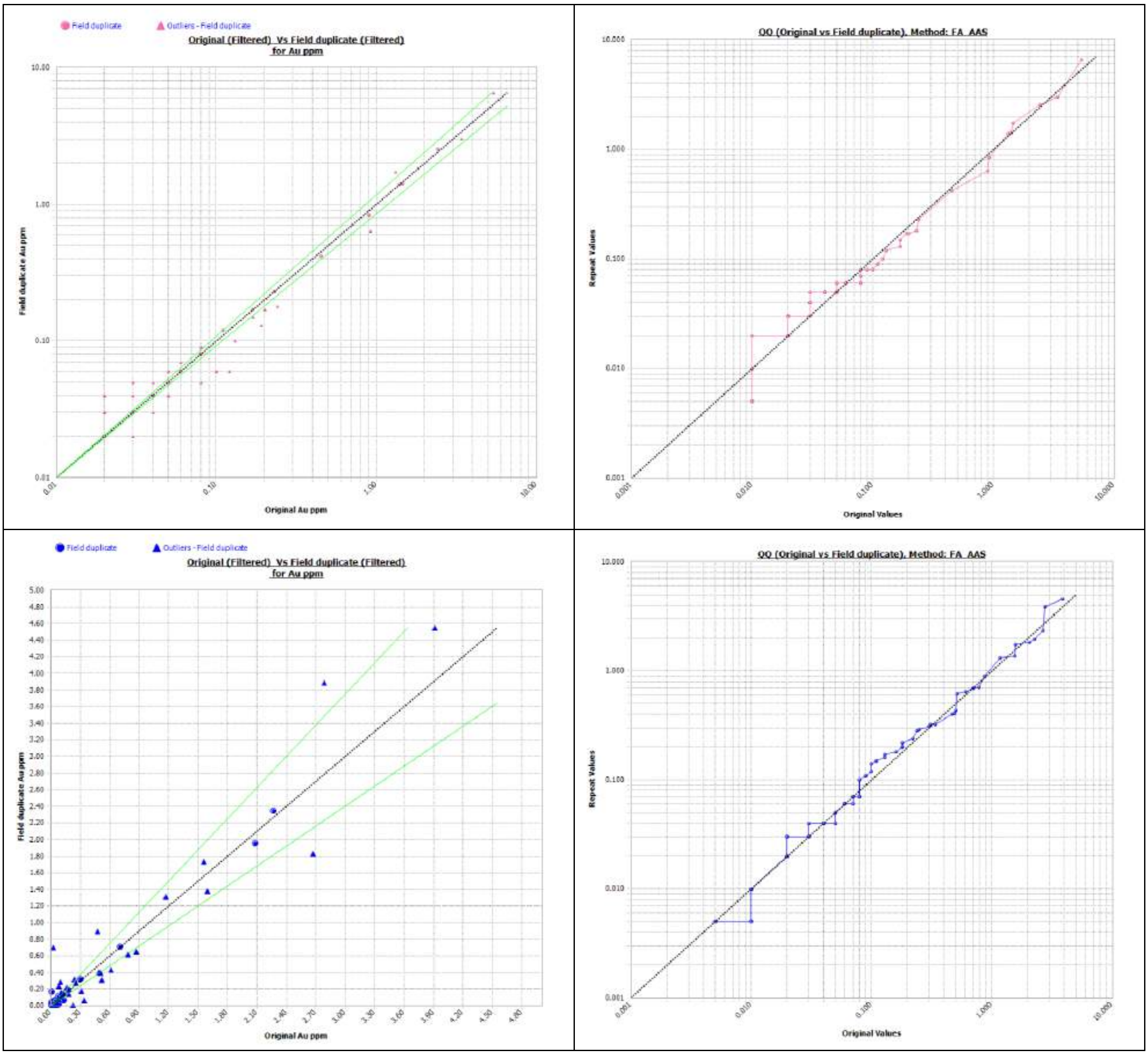


Figure 11.5 Scatterplots (left) and Q-Q plots (right) for field duplicates from 2012 to 2014 (top) and 2015 to 2016 (bottom)



Umpire laboratory checks

To assess the quality of the assay results reported by the primary laboratories, SGS (Mengapur) and PT Intertek (Jakarta), coarse crush duplicates and pulp duplicates from diamond drillholes were dispatched, under instruction by Monument, to the ALS laboratory in Malaga, Western Australia for umpire analysis between March 2014 and March 2015. The ALS laboratory in Malaga is NATA accredited as meeting the requirements of ISO 17025:2005. No duplicate samples were submitted for umpire analysis for the period March 2015 to the end of 2016.

Results from the coarse crush duplicates (Figure 11.6) submitted in 2014 and 2015 show relatively poor precision between laboratories, particularly at lower grades, with a slight bias towards ALS (i.e. ALS reporting slightly higher grades). The relatively poor precision may be due to poor subsampling practices in the various laboratories, along with the splitting methodology employed in the primary laboratory to generate the coarse crush duplicate.

The results from the umpire pulp duplicates (Figure 11.7) show no evidence for bias with reasonable, although not ideal, precision.

Figure 11.6 Scatterplot and Q-Q plot for umpire coarse crush duplicates

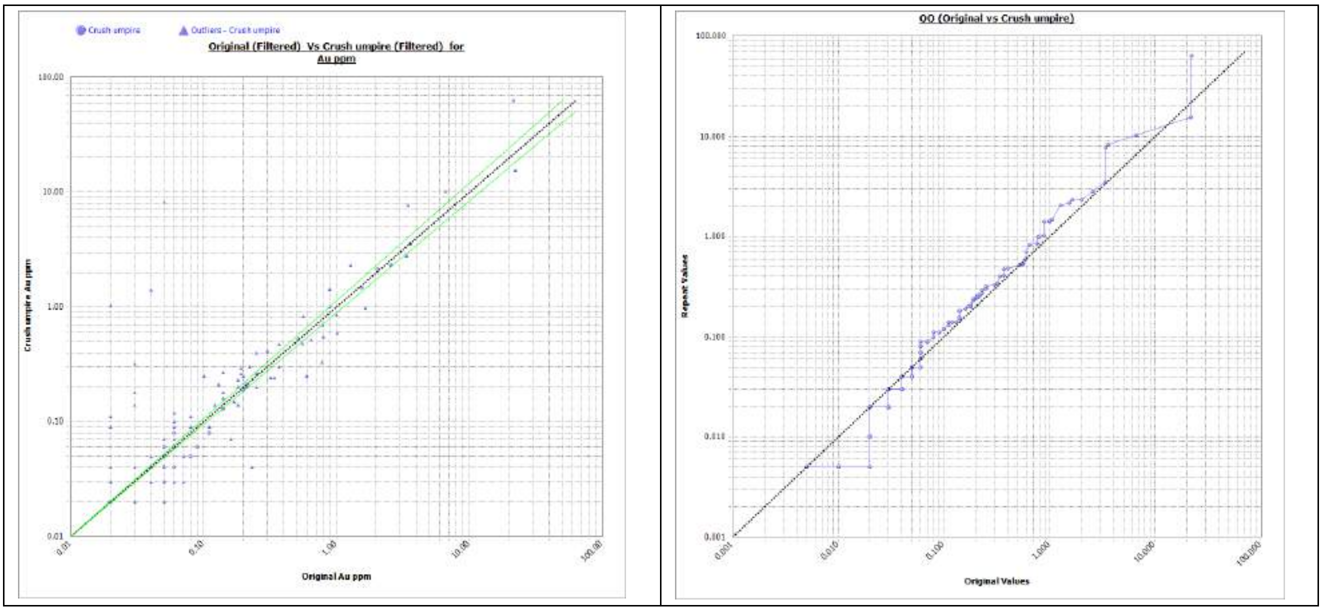
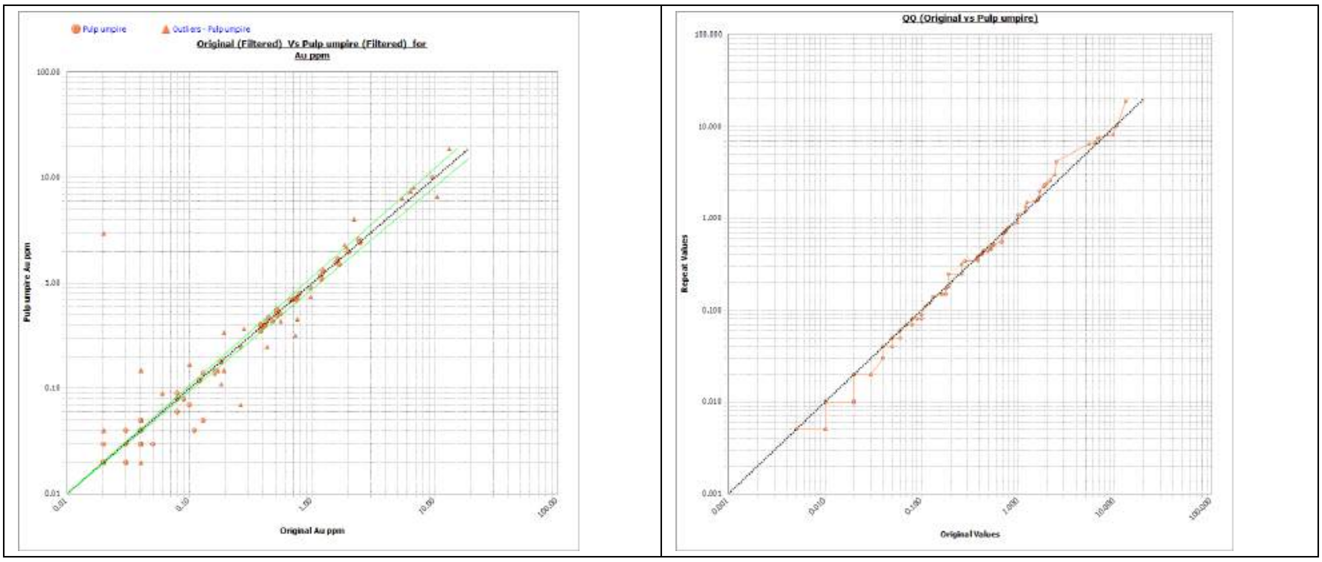


Figure 11.7 Scatterplot and Q-Q plot for umpire pulp duplicates



11.5 Author's opinion on the adequacy of sample preparation, security and analytical procedures

The assaying and sampling of the Damar/Avocet drilling and trenches at Buffalo Reef is not optimal. No independent QAQC was included in sample batches to assess the precision and accuracy of the assaying. Additionally, the assaying methodology is a mixture of fire assaying and titration techniques. Given the refractory nature of the gold mineralisation, the titration assaying has likely resulted in some gold grades being under-reported; however, the assaying method is not recorded in the database and the number of affected samples is not known. Flindell (2003) indicates that all RC and core samples were re-assayed by Damar using fire assay and only trench samples may still have gold grades determined by titration.

Similarly, at Selinsing, drilling conducted before 2007 did not include any independent QAQC in the sample batches; however, the majority of this early data targets the upper portions of the mineralisation which has been mined out. As such, Snowden believes that including this data in the resource estimation is reasonable and has been accounted for in the Mineral Resource classification.

The RC and diamond drilling completed by Monument after 2007 includes independent QAQC samples with the sample batches, the results of which show reasonable precision and analytical accuracy have been achieved. Comparisons between the Damar/Avocet and Monument drilling at Buffalo Reef (Snowden, 2011) do not show any material differences or bias is present. In 2013, SGS Mengapur implemented a number of different routine laboratory checks to assess accuracy and repeatability at all levels, including sample preparation and analyses.

In the author's opinion, the available drillhole and trench data for the Selinsing and Buffalo Reef deposits, is reasonable for use in resource estimation and the procedures for sample preparation, security and QAQC protocols are appropriate to ensure the quality of the assay data. However, given the less than ideal precision shown by the duplicates (field and pulp), Snowden recommended in 2016 that Monument review the field and laboratory sampling and subsampling practices to assess for potential improvements in the procedures (e.g. the tiered riffle splitter used for RC sampling) and to ensure that only best practice methods are employed. Monument indicated that the same trends for field duplicates continued through to the end of 2016 and as such, Snowden's recommendation is still valid.

12 DATA VERIFICATION

12.1 Assay data verification

Assay data validation has been completed through the umpire and field duplicate sampling programs. Monument indicated that for historical assay data compiled from various sources, including the data used by Practical Mining in the 2012 NI 43-101 report, a minimum of 10% of assay values were cross-checked internally by Monument with assay certificates from AssayCorp, Intertek and SGS (Port Klang). No discrepancies were found by Monument with the data. Assays that have been verified have been flagged in the database.

Additionally, a random selection of 10 assay certificates, sourced directly from the SGS laboratory in Mengapur, was checked by Snowden in 2016 against the data within the database. Overall, the assay certificates compared well with the database, however, some discrepancies were identified for three of the certificates and rectified, where appropriate, as summarised in Table 12.1.

Table 12.1 Assay certificate data verification issues

Assay certificate	Issues identified	Comments
MSMC002_14560-14575.pdf	• Drillhole in database for these samples is SELRC0461 • Au grades totally different in database • No As grades in database	The assays reported on this certificate actually correspond to surface samples (not drillhole samples) in the database (in a separate table) with the same sample IDs as SELRC0461. SELRC0461 was drilled in 1997 and assayed by AssayCorp. PDF scan of the original assay certificate from AssayCorp was provided by Monument and shows no discrepancies.
MSM_C_14503-14523_14503-14523.pdf	• Drillhole in database for these samples is SELRC0460 • Au grades totally different in database • No As grades in database	The assays reported on this certificate actually correspond to surface samples (not drillhole samples) in the database (in a separate table) with the same sample IDs as SELRC0460. SELRC0460 was drilled in 1997 and assayed by AssayCorp. PDF scan of the original assay certificate from AssayCorp was provided by Monument and shows no discrepancies.
MSMDD035_25523-25569.pdf	• Drillhole in database for these samples is MSMDD035 • Au and Sb grades ok • No As grades in database	Two fields in the master database for arsenic values (called "As" and "Ars"). All arsenic values were in the database, however, due to incorrect ranking, were not in the export provided to Snowden. This was corrected by Monument and a revised database export was provided.

As only minimal additional drilling has been conducted since February 2016 (the cut-off date for assays to be included in the 2016 Mineral Resource estimate), no additional data verification has been conducted by Snowden.

12.2 Qualified Person's opinion on the adequacy of the data for the purposes used in the technical report

Snowden believes the assay data within the database is robust, although some minor discrepancies were identified in the assay certificate checks in 2016. Snowden understands that validating the database is an ongoing process at Monument. In Snowden's opinion, the drillhole database for Selinsing and Buffalo Reef is suitable for use to generate Mineral Resource estimates.

Snowden has not conducted any independent sampling or assaying to verify the gold tenor of the samples. Given the results of the assay certificate checks and QAQC results, along with the mining production history, Snowden does not believe that independent sampling is required at this stage.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Mineralogy and testwork summary

The main mineralogical and metallurgical investigations on Selinsing and Buffalo Reef ores are listed in date order in Table 13.1. The reference numbers given in this text correspond to the reference numbers in Table 13.1.

Table 13.1 Reference texts for mineralogical and metallurgical investigations on Selinsing and Buffalo Reef

Ref. no.	Year	Title	Author/Organisation	Report date
1	2009	The Biogeochemistry of Gold	G. Southam *et al* Elements, Vol.5, pp303-307	Oct 2009
2	2011	Microscopic & Microanalytic Investigation of Auriferous Metallurgical Samples. Buffalo Reef South Zone, Malaysia (for Inspectorate, Metallurgical Division)	Lehne & Associates – Applied Mineralogy	11 May 2011
3	2011	Report JZMIN 11051: Deportment Study of Gold and Characterization of Carbonaceous Matter in Head and Process Products	Joe Zhou Mineralogy Ltd	20 Nov 2011
4	2011	Geology, Geochemistry and Metallogenesis of Selected Sediment-hosted Gold Deposits in the Central Gold Belt, Peninsular Malaysia (M.Sc. Thesis)	Charles Makoundi, CODES, University of Tasmania	Dec 2011
5	2012	4 Polished thin sections of drill chips: DH7-1, DH7-2, DH7-3, and DH8-HG3	Vancouver Petrographics Ltd	27 Jun 2012
6	2012	Metallurgical Study on the Buffalo Reef Gold Project	Inspectorate Exploration & Mining Services	24 Sep 2012
7	2013	Selinsing Deep Preliminary Process Response - Project 1206609	Inspectorate Exploration & Mining Services	19 Feb 2013
8	2013	Selinsing Phase IV Pre-Feasibility Study	Lycopodium	Feb 2013
9	2013	Geology, geochemistry and metallogenesis of the Selinsing gold deposit, Central Malaysia	Charles Makoundi *et al*, Gondwana Research, 2013	20 Aug 2013
10	2013	Albion Process Plant Engineering Study - Selinsing Gold Project	M Hoorn Xstrata Technology	21 Oct 2013
11	2014	Selinsing Deep Flotation Program for Selinsing Gold Mine Manager Sdn Bhd	D Ocampo SGMM R&D	Feb 2014
12	2014	Flotation Optimization and Variability Testing on Buffalo Reef Composites	S Haznisan SGMM R&D	7 May 2014
13	2015	Albion Process Test work Study on Selinsing Buffalo Reef Refractory Gold	Daniel Mallah, Glencore Technology	8 Apr 2015
14	2016	Study at the North Buffalo Reef Gold Mine - Its Mineralogy and Geochemistry	F Syahmi, Honours Thesis	12 Apr 2016
15	2016	Flotation Pilot Plant for Buffalo Reef Composite Sample	D Ocampo SGMM R&D	27 Jul 2016
16	2016	Mineralogy Report 16/473: Quantitative XRD Analyses on Four Samples (from Intec Pilot Plant Batch 2)	SGS	28 Jul 2016
17	2016	Stage 2 Report-Pilot Plant: Intec Process Pre-Treatment of Buffalo Reef Concentrate - Campaign 2 – Addendum	DCS Technical Pty Ltd	Aug 2016
18	2016	Quantitative Automated Mineralogical Analysis conducted on Three samples from Selinsing Gold Mine Project, Malaysia for Able Return Sdn Bhd: A17352 Mineralogy Report MIN 2622	ALS Metallurgy	Sep 2016

Ref. no.	Year	Title	Author/Organisation	Report date
19	2016	Preliminary Study Report on Acid Leach - Scoping Test	A Taib SGMM	19 Sep 2016
20	2016	Technical Report RCIKMET00707 - Comminution Test work	PT. Geoservices	4 Oct 2016
21	2016	Progress Report #1 on Acid Leach Fine Tuning Testwork Program - Ferric Nitrate Concentration Optimisation	D Ocampo SGMM R&D	14 Oct 2016
22	2016	SEM Microanalysis Report	Dr Syed Fuad, University Sains Malaysia, Penang	Dec 2016
23	2016	Review of Intec Leach Process and Subsequent Metallurgical Test Work Programme: OMC Report No. 7683-03 Rev A	OMC	Dec 2016
24	2017	Metallurgical Test work conducted upon Selinsing Flotation Concentrate and Tailings Samples for Able Return Sdn Bhd - Report A17352	ALS Metallurgy	Jan 2017
25	2017	Flotation Optimization Bench Scale Program for SGMM Sdn Bhd	D Ocampo SGMM R&D	Feb 2017
26	2017	Project MS 1731 Selinsing	J Tan Met-Solve Laboratories	9 Mar 2017
27	2017	BIOX Batch Amenability Test Work Report for Selinsing Gold Mine	Outotec	Mar 2017
28	2017	Ore Mineralogy Microscopy at USM	M Wort SGMM R&D	5 Apr 2017
29	2017	Representativeness of Flotation Concentrate for BIOX Testing	M Wort SGMM R&D	12 May 2017
30	2017	SGSP – Preliminary Report on Rougher-Scavenger Flotation on Selinsing Ore types	M Wort SGMM R&D	7 Jun 2017
30a	2018	SGSP - Preliminary Report on Rougher – Scavenger Flotation on Buffalo Reef Ore types at the Selinsing Mine (Ver 3)	M Wort SGMM R&D	16 Feb 2018
31	2017	SGSP – Preliminary Report on Cleaner Flotation on Buffalo Reef Ore types	M Wort SGMM R&D	14 Jun 2017
32	2017	SGSP – Testing of Auxiliary Collectors for Gold Recovery Improvement	M Wort SGMM R&D	9 Sep 2017
33	2017	Ore Mineralogy Microscopy at USM, 10/10/17.	M Wort SGMM R&D	2 Nov 2017
34	2018	SGSP – Report on Gangue Depression Tests on Buffalo Reef Ore types	M Wort SGMM R&D	5 Feb 2018
35	2018	SGSP – GDT & Other Flotation Tests on Transition Ore Variants	M Wort SGMM R&D	19 May 2018
36	2017	Outotec – Selinsing Gold Mine BAT report for Selinsing III Concentrate	Outotec	13 Dec 2017
37	2018	Outotec – Selinsing Gold Mine BAT report for Selinsing Variability Samples	Outotec	7 Jun 2018
38	2016	Selinsing Gold Mine and Buffalo Reef Project - Malaysia: NI 43-101 Technical Report - Project Number AU5200	Snowden	14 Dec 2016
39	2018	SGSP site flotation testwork program review	OMC	Aug 2018
40	2018	SGSP – Optimisation tests on BRC4_FR and their extension to other LOM pit ores	M Wort SGMM R&D	Oct 2018
41	2018	BRC4 Fresh flotation testwork program	Bureau Veritas	Ongoing (Dec 2018)
42	2018	SGSP Locked Cycle Flotation Test Results	M Wort SGMM R&D	Received Nov 2018
43	2018	Memo regarding BRC4 representativity	R Stangler	May 2018

The geology, geochemistry and metallogenesis of the Selinsing gold deposit has been described in detail by Charles Makoundi for his M.Sc. thesis (4), and by Makoundi *et al* of the CODES Group at the University of Tasmania (9).

In this study, the metallurgical characteristics of the Buffalo Reef deposits which lie to the immediate north of the Selinsing Gold Mine open-cut pits, along strike in the same shear zone and within the same host rocks, are considered to be closely similar to those in the Selinsing Gold Mine. This consideration is based on the documented observation (43) that most of the ore minerals present are the same, the host rocks for the mineralisation are the same, and the gold grades and pattern of elemental distribution are closely comparable. There is however a noted difference in the flotation response of Selinsing and Buffalo Reef ore types. The similarity of observable deposit features is strong evidence for the consanguineous origin of at least the early part of their metallisation history. Evidence of a late stage of mineralisation carrying Sb ions is present in the Buffalo Reef ores but has not been recorded for the Selinsing deposit.

Due to the deformational history of the host rocks along the Raub-Bentong suture zone, the presence of native gold and sulphide mineralisation is not confined to a tightly restricted zone, but rather is diffused so that highest grades occur close to the main shear axis with progressively weaker mineralisation persisting as localised spurs in and around shear and fracture extensions. So, the difference between ore and waste is simply the grade, as measured by exploration drilling and grade control drilling within an operating pit, measured against the cut-off grade designated by the mining schedule.

The two main auriferous sulphide minerals are arsenian pyrite (FeS_2) and arsenopyrite (FeAsS). Stibnite (Sb_2S_3) is locally present at Buffalo Reef and is often intergrown with native gold. The minor sulphide phases in the ore are pyrrhotite ($Fe_{1-x}S$), chalcopyrite ($CuFeS_2$), sphalerite (ZnS) and galena (PbS). The presence of berthierite ($FeSb_2S_4$) in Buffalo Reef ores has also been reported by two authors (2, 5), and berthierite is often intergrown with stibnite. Trace tetrahedrite $(Cu, Fe, Zn, Ag)_{12}Sb_4S_{13}$ has also been reported as a late stage infilling to brecciated pyrite (5).

Gangue minerals are mainly silicates including quartz (SiO_2); muscovite mica ($KAl_2\{AlSi_6O_{10}\}\{F, OH\}_2$) and its fine-grained variety, sericite; with chlorite (an iron, aluminium, magnesium hydrated silicate); illite (a layered aluminosilicate clay mineral); calcite ($CaCO_3$); dolomite $CaMg(CO_3)_2$; ankerite $(Ca,Mg,Mn,Fe)CO_3$; and the oxide phases rutile (TiO_2) and hematite (Fe_2O_3). The grain size of the gangue silicates in the host rock sediments is extremely fine, and they are the source of most of the minus 10 µm gangue that contaminates the sulphide flotation concentrates by means of entrainment. This implies that water recovery to flotation concentrate needs to be minimised in order to minimise gangue recovery.

A problem component of the gangue is the presence of preg-robbing graphite as well as graphite-smeared particles of quartz. Many of the host rocks are graphitic, the graphite probably representing remains of marine life co-deposited with the original sediments.

A consultant mineralogical study by Joe Zhou (3) issued in 2011 confirmed the identity of the main sulphide phases present (pyrite, arsenopyrite and stibnite) and also the presence of two types of graphitic carbon, both of which were preg-robbing. Furthermore, this graphitic carbon contained gold absorbed as aurocyanide during leaching testwork, and gold that was metallic gold on the graphite. Microbiological testwork at the University of Western Ontario (1) has shown that absorption of gold on to organic carbon in gold ores is possible due to the action of bacteria present in ground waters.

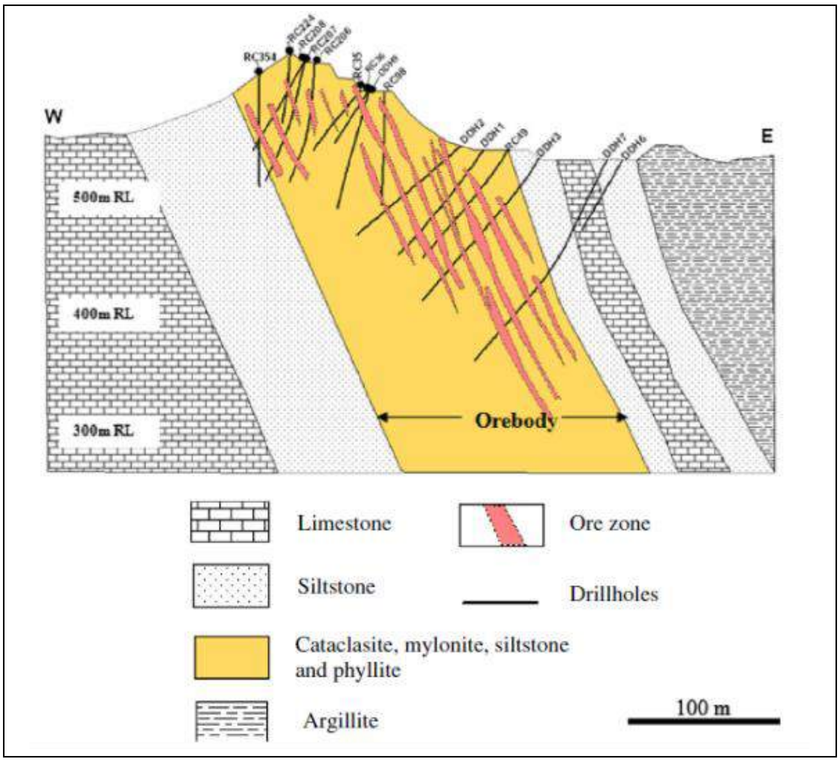
Five separate generations of pyrite have been identified by Makoundi *et al* (9) in the Selinsing ore, commencing with deposition in deep sea carbonaceous muds of tiny 2 µm framboids (Pyrite 1) which became diagenetically enriched in Au, Ag and As together with V, Mo, Se, Te, Ni and Zn. Much of the pyrite is known to be arsenian, and during late diagenesis and the early stages of metamorphism, the generation 5 pyrite released gold which was then able to precipitate as native gold in suitable shear zones and saddle reefs.

The host rocks are essentially argillites, siltstones and phyllites with local zones of mylonite and cataclasite. Mineralisation is closely associated with veining by quartz and/or dolomite, in veinlets and veins 1–200 mm thick. Native gold occurs within the quartz veins (Figure 13.1) along with sulphides. The sulphides are also often within phyllites immediately adjacent to the vein walls.

Visible native gold is not abundant, but can occasionally be seen in blasted ore, and is mostly captured along with liberated sulphides in the Knelson centrifugal concentrator at the existing CIL plant.

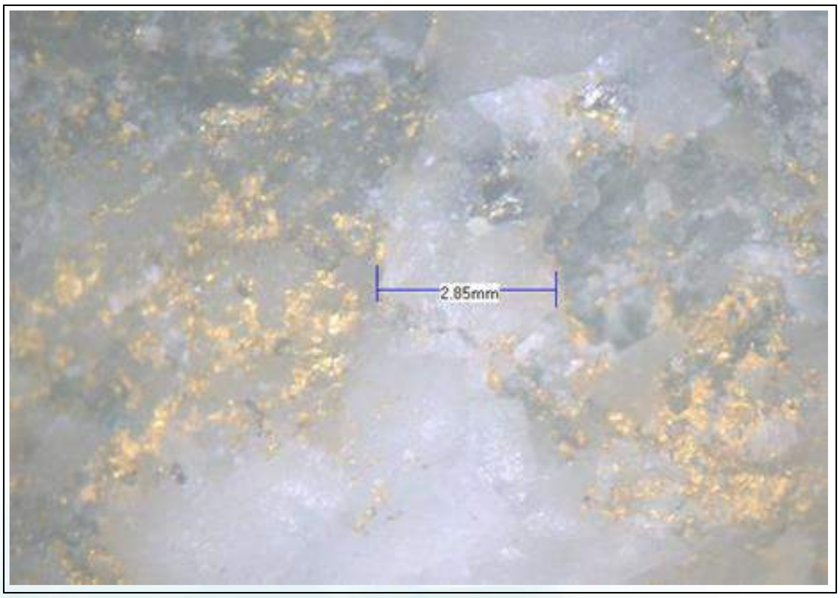
A schematic cross-section of the Selinsing ore deposit is shown below in Figure 13.1.

Figure 13.1 Schematic cross-section of the Selinsing orebody, from Makoundi *et al.* (8)



The hangingwall limestone shown on the eastern side of Figure 13.1 is very dark grey due to its graphitic carbon content. It may be clearly seen from the south end of Pit 4 at Selinsing. Testing of the hanging wall limestone from Selinsing Pit 5, conducted at the site R&D Laboratory and also by Outotec in South Africa has confirmed that the acid neutralising power of this limestone is close to that of AR grade limestone. Accordingly, this limestone will be mined, crushed and ground on site for use as a pH control reagent for the BIOX® operation and subsequent digested slurry neutralisation. The possible gold preg-robbing effect of contained graphitic carbon within this hanging wall limestone on BIOX® residue CIL needs to be established.

Figure 13.2 Native gold in quartz from a location between Selinsing Pit 4 and Pit 5



Dr Syed Fuad (22) of University Sains Malaysia in Penang state completed an SEM examination of flotation pilot plant concentrates from Batch 1 and Batch 2 (representing Buffalo Reef ore types), together with examination of Falcon centrifugal gravity concentrates. The common presence of pyrite, arsenopyrite and stibnite was confirmed, and gangue minerals including albite (plagioclase feldspar), orthoclase (potassium feldspar) and zircon were also noted.

Subsequently ore microscopy was conducted by M. Wort (28) using a reflected light microscope, on products including Acacia Feed from the Selinsing CIL plant, Batch 2 concentrate from the Flotation Pilot Plant and a low-grade Falcon concentrate. In the Acacia Feed the presence of native gold, pyrite and arsenopyrite was confirmed, along with a prismatic form of probable arsenopyrite. The relevant polished section was resubmitted to USM for examination by SEM but could not be completed due to machine breakdown.

A subsequent brief examination by Townend Mineralogy Laboratory in Perth on a laboratory flotation concentrate confirmed mineralogical work by Makoundi, which showed the pyrite to be an arsenian form, carrying arsenic in the crystal lattice.

13.1.1 Sampling program

The testwork investigations conducted both internally and externally by research organisations to assess the performance of relevant processing routes, either alone or in combination, are given in Table 13.1 above, along with their respective completion dates and report titles.

Specific investigations overview

In 2010, the Inspectorate commercial mineral processing laboratory in Canada was commissioned to conduct a testwork program on fresh sulphide ore samples from the north and the south of the Buffalo Reef resource, to investigate relevant competing unit separation and concentration processes. A composite of ores from the northern and southern zones was also tested. The program assessed the possible concentrate production and gold recoveries obtainable by flotation, bioleaching, centrifugal gravity concentration, and direct cyanide leaching. The report on this work (6) was issued in September 2012 and concluded that the best gold recoveries obtained were shown by sulphide flotation followed by bioleaching of the flotation concentrate.

A further testwork program, on Selinsing Deep drill core ore samples was reported by Inspectorate (7) in February 2013 for a two-pronged investigation. The first involved gravity concentration combined with intense cyanide leaching and conventional CIL leaching. The second involved flotation, bioleaching of the flotation concentrate, and CIL on the bioleached concentrate and also on the flotation tailing. A poor response to direct gravity concentration and CIL leaching was obtained and the best gold recoveries, all above 90% for the six submitted samples, were obtained by the flotation plus bioleach route.

A flotation program on Selinsing Deep ore samples was conducted by D Ocampo (11) in February 2014, and the tests achieved high rougher recoveries of both gold and sulphur with mass-pulls ranging from 27.5% to 67.9%. Combined roughing and cleaning achieved mass-pulls of 10–15% in three of the nine tests conducted.

In May 2014, flotation work on four Buffalo Reef Central (BRC) composites was reported by S Haznisan (12) who found that best bench-scale results came from splitting both roughing and cleaning into six stages, possibly due to benefits from staged collector addition.

In July 2014, extensive bench-scale flotation testwork investigations conducted in the Selinsing Gold Mine (SGMM) R&D Laboratory on Buffalo Reef ore types were completed by D Ocampo. Due to unforeseen circumstances, reporting of results from these tests was not completed until early 2017. (25)

In April 2016, a thesis study by Faruq Syahmi (14) on the geology of the Buffalo Reef North deposit became available, as noted earlier above.

From March to May 2016, the earlier bench-scale flotation testwork conducted in 2014 was followed by Buffalo Reef pilot plant flotation trials (15), in which a total of more than 10 t of drill core samples were made available for testing in a specially constructed flotation pilot plant. The amounts of the individual ore types were supplied in the same ratio as their then known contribution to the total Buffalo Reef resource (i.e. 40% BRS, 40% BRC-QA, 15% Felda and 5% BRN).

The flotation pilot plant investigations were augmented by a detailed comminution study conducted by PT Geoservices (20), to determine Bond Work Indices etc for Buffalo Reef Ores.

At the conclusion of the flotation pilot plant runs which achieved an 82.65% recovery of gold, the total amount of concentrate available was 922 kg, obtained with a concentrate mass-pull of 12.36%. This concentrate was then used for further metallurgical testing, and graded 8.8 g/t Au, 2.30 g/t Ag, 7.36% Fe, 5.86% Tot S, 1.01% TOC, 448 ppm Cu, 24.9% Si and 0.94% Sb, with an specific gravity of 2.98.

The logical next step was to select a technology for releasing the gold from the refractory sulphide concentrate, which in addition to containing free gold and auriferous sulphides also contained preg-robbing graphite and graphite-smeared particles of quartz and other gangue.

Due to concerns about the high capital costs of currently available bacterial leaching process plants, a licence was obtained to use the Intec Process owned by DCS Technical. The next processing investigations therefore included two pilot plant leaching campaigns to test the Intec Process, which uses a halide leaching medium. Regrettably, these Intec processing investigations (17) were unsuccessful and failed to provide adequate gold recovery from the leached solids.

Further intensive investigations in Australia to explain the failure of the Intec process were conducted by ALS Metallurgy under the direction of OMC. The many investigations included cyanide leaching and diagnostic leaching tests on Intec residues, removal of preg-robbing graphite by flotation, and x-ray diffraction (XRD) (16) and QEMSCAN (18) and investigations of the mineralogy of leach feeds and residues. OMC (23) then reviewed the results of the original pilot plant testing campaigns of the Intec Process testing and also the subsequent trouble shooting investigations by ALS Metallurgy (18, 24).

An unequivocal explanation for the failure of the Intec Process to achieve complete digestion of the pyrite could not be found, nor for the failure of cyanidation to extract the gold which had been released, but with hindsight it seems possible that the failure to extract gold was due to its passivation by Sb ions released from digested stibnite.

In early 2017, a parcel of crushed ore from the BBX box-cut at Buffalo Reef South was sent to Met-Solve in Canada for testing of gravity recoverable gold (26).

Although the gold head grade was quite high (5.5 g/t Au), the testing achieved only 59.3% gold recovery at 16% mass-pull at a grind size P_{80} of 78 µm. Early passes obtained 50% gold recovery into a 7.4% mass-pull. A repeat gravity test to provide concentrate for cyanide leaching achieved 66.5% gold recovery into 14.1% mass but only 14.1% of the gold in the concentrate could be recovered by cyanide leaching, i.e. 9.38% of the gold in the sample feed. Recoveries of sulphides in the Falcon concentrates cannot be accurately assessed, because no sulphur assays were taken on the gravity concentration test products. However, a stereo binocular examination of size fractions of the scavenger tail by M. Wort showed that recovery of sulphides into the bulk gravity concentrate was essentially complete.

Following elimination of the Intec Process as a viable option for recovery of gold from flotation concentrates, bioleaching now logically became the best process contender. Using a parcel of the 2016 flotation pilot plant concentrate, and in recognition of the earlier 2012 findings by Inspectorate, a BIOX® bacterial leaching study was now undertaken in South Africa by Outotec in early 2017. The BIOX® process currently has the best commercial track record of the available bioleaching processes. This BIOX® program (BIOX-I) was successful and achieved a peak gold release of 90.3% (27). However, on account of the comparatively low head grade (5.86% Tot S), additions of sulphuric acid had to be added to maintain the required low pH.

As part of the diligent evaluation of competing technologies, pilot plant flotation concentrate was also used for a program of bench-scale low-pH leaching using ferric nitrate and sulphuric acid. Leached slurries were neutralised with lime and then subjected to CIL leaching to recover gold. The ferric nitrate bench scale leaching gave higher gold recoveries than were achieved by the Intec Process, but will still need pilot scale confirmation to accurately assess the effects of liquor recycling. Two reports on this work were prepared (19, 21).

As at October 2017, a total quantity of about 507 kg of the 2016 pilot plant flotation concentrate remains, and this material has been provisionally allocated to due diligence testing by the planned ferric nitrate leaching pilot plant campaign, still to be conducted although refurbishment of the leaching pilot plant is still to be effected (Figure 13.3).

Figure 13.3 Acid leaching pilot plant as employed for testing the Intec Process

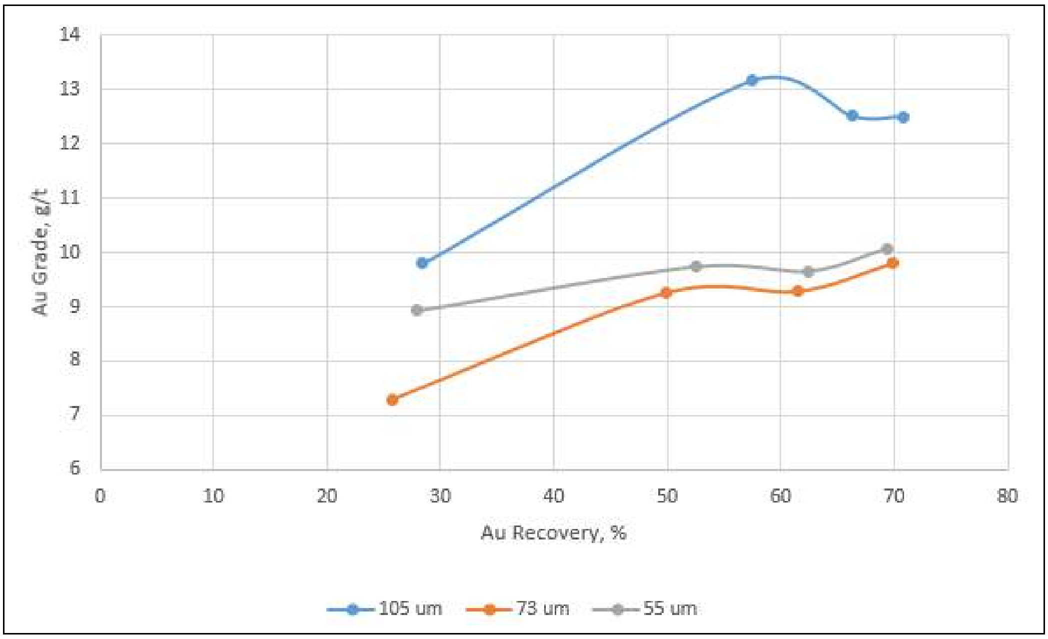
It was realised that to establish the auditable relevance of the pilot flotation work, that the representativeness of the samples used for the pilot flotation program should be documented. Accordingly, in mid-May 2017 a technical memorandum (29) was compiled and issued with input from both the Exploration Department and the R&D Department, documenting the source of the drill cores used to make up each ore type composite, their position in relation to the LOM pit shells as modelled in the NI 43-101 Technical Report prepared by Snowden and issued in December 2016 (38), the sample weights taken, and the sample preparation procedures employed. This memorandum confirmed the representativeness of the samples used for the pilot flotation campaign.

In April 2017, a program of flotation engineering confirmation tests commenced, to gather flotation data to be used for design of a flotation plant to produce concentrates higher in sulphur grade than that which had been achieved in the 2016 pilot plant production run, with a target concentrate sulphur grade of 15–20% S. The higher sulphur content ensures that the process will be acid generating and will not require supplemental additions of sulphuric acid to maintain the required pH range between pH 1.0 and pH 1.5. A further objective was to achieve a standardised mass-pull to final concentrate not exceeding 10% in order to reduce the required capacity and construction cost of the BIOX® bacterial digestion tanks.

While the design of this new program included referral to the previous investigations, which in the rougher-scavenger stages at least had very large mass-pulls, it was hoped where possible to achieve a 10% mass-pull by roughing only, by achieving a more selective flotation of the sulphide particles. Such selective sulphide flotation should be achievable if the fine-grained gangue silicates can be more effectively depressed. In this sense, the 2017 program was looking for a breakthrough improvement in selectivity. Adding a 1 kg/t dose of soda ash to the mill feed to disperse gangue slimes was a part of all preliminary tests, and subsequent testing with a combination of gangue dispersants and depressants resulted in improvements to gold and sulphide recoveries for some ore types, but overall did not achieve the desired result of reducing mass-pull.

In this initial 2017 flotation testwork, six different geometallurgical ore types were tested in a standardised simple testwork program, to identify the most suitable sulphide flotation collectors, best feed size distribution, and best flotation pH range for rougher-scavenger flotation. Results showed that, weighted across all ore types, the best grade/recovery results resulted from a 10' grind (P_{80} 105 µm on average, Figure 13.4), best flotation pH was 5.5–6.0, and best collector was PAX (30). Sulphide recoveries greater than 90% were routinely achieved, but gold recoveries were below 90% for some ore types. Rougher-scavenger combined mass-pull varied between 5% and 40% depending on the ore type and flotation conditions.

Figure 13.4 Effect of grind size on grade/recovery response – weighted by ore type (30)



Best conditions from the rougher-scavenger tests (30) were then applied in a further programme to identify the best auxiliary gold collector. PAX was retained as the base case collector and additions of Aero 3418A, Maxgold MX900 and Aero 208A were assessed. Improvements in rougher-scavenger recovery of gold were achieved for individual ore types, but no single auxiliary collector was found to be best for all ore types (32).

Cleaning tests were initially conducted using PAX alone (31) to serve as a base case for subsequent cleaning optimisation tests.

Detailed results of flotation testing from testwork initiated in 2017 for engineering confirmation are summarised in Section 13.3.1.

13.2 Process selection

13.2.1 Fundamentals

The known mineralogy of the Selinsing and Buffalo Reef ore types immediately suggests the candidate unit processes to be evaluated for their performance, and since 2011 Monument has been proactively investigating and assessing them.

- Liberated sulphides can be floated selectively using xanthate and dithiophosphate collectors, as well as by other collector chemicals such as dithiocarbamates, so sulphide flotation is an important process to be tested
- Liberated gold and electrum are routinely recoverable by centrifugal gravity equipment such as Knelson and Falcon Concentrators, with the Falcon reputed to make better recoveries of gold particles finer than 10 microns
- Liberated fine gold can be recovered by flotation using xanthate and certain other collectors, although maximum recovery of free gold via gravity separation should be the primary focus
- Bioleaching of refractory gold ores to release gold encapsulated within sulphides, or present in solid solution within the sulphide mineral lattice, is a proven commercial technology.

Auriferous sulphide concentrates can be treated by roasting plus calcine cyanidation, by direct smelting, by high temperature pressure oxidation (POX) as at Porgera, by atmospheric pressure leaching at 100°C or below, or by bacterial digestion. Of these processes, the last two carry the lowest environmental pollution risks and permitting problems.

There are several ambient-pressure leaching processes including the Albion Process, the Activox Process as developed by Aqueous Metallurgy, and the Leach ox Process developed by Maelgwyn Mineral Services in the UK. The Albion Process involves ultrafine grinding of flotation concentrate using Glencore's IsaMill technology, followed by oxidative leaching at atmospheric pressure, and is commercially proven.

With regard to the Albion Process, initially a Class 5 (AACE) Scoping Study (10) was commissioned in October 2013 from Xstrata Technology by Mike Kitney for a conceptual Albion Process plant to treat 72,000 t/a of Selinsing flotation concentrate grading 27 g/t Au and 24% sulphide sulphur. However, when actually tested on a bulk concentrate from a BRC flotation program (13), it inexplicably failed to give satisfactory gold recoveries.

The official Intec Process literature makes claims about its efficacy in treating a very wide range of sulphide ores, including those containing stibnite, and the Intec Process had the promise of comparatively cheap capital and operating costs. It had been successfully employed to retreat the tailings from the Aberfoyle mine in Tasmania for recovery of Pb, Zn and Ag. However, and as discussed above, when tested at considerable length on Buffalo Reef pilot plant flotation concentrate, the Intec Process failed to deliver acceptable gold recoveries (23).

13.2.2 Final process selection

The process route selected for this FS, for treatment at the rate of 1 Mt/a is:

- Reuse of the existing grinding and classification circuit from the current CIL plant to produce sulphide flotation feed at a P_{80} sizing of 75–105 microns.
- Reuse of the existing gravity concentration and intensive cyanide leach circuit to maximise pre-flotation recovery of liberated free gold.
- Grinding circuit product will be fed to a bulk sulphide flotation circuit consisting of roughing, scavenging and cleaning duties, where gold-bearing sulphide particles will be separated from gangue and concentrated to form BIOX® feed. Flotation will be conducted under slightly basic pH conditions using a PAX collector, Cu2+ activation and a sulphidization reagent, the combination of which has been demonstrated to give the best flotation response across the LOM ore types.

- Flotation tailings will report directly to final tailings.
- A sulphide flotation concentrate grading 6–8% S at a 10% mass-pull will form feed to a BIOX® bacterial leaching plant, based on the latest recommendation by Outotec. The sulphide slurry will be controlled to a pulp density of 20% solids.
- Fully digested acid BIOX® slurry will then be treated in three consecutive counter-current washing thickeners, and after adjustment to pH 10 and 40% solids will proceed to the new CIL gold recovery plant.
- Acid washings from the digested slurry will be brought back to pH=7 in two stages using ground limestone to pH=5, and hydrated lime from pH=5 to pH=7.

The selected process route is similar to that indicated by the Inspectorate testwork completed in September 2012 (6).

13.3 Engineering confirmation testwork

13.3.1 Flotation

Key results of progressive flotation testing programs (initiated in 2017 and ongoing) are included in this section, with detailed testwork information contained in referenced reports placed in the Appendices.

While preliminary programs failed to meet target recoveries for gold, there has been a steady improvement in performance achieved to the point in October 2018 where satisfactory gold recovery has been achieved in rougher flotation for LOM fresh sulphide ore types.

Preliminary test programs

Preliminary site-based flotation test programs to establish and optimise conditions and define the expected flotation response consisted of the following phases:

- Rougher tests to compare and select base line grind size, pH and collector type (30)
- Preliminary open circuit cleaner tests (31)
- Rougher and open circuit cleaner tests using auxiliary collectors to maximise native gold recovery (32)
- Rougher and open circuit cleaner tests using various gangue depressants with the goal of reducing gangue recovery to concentrate (34)
- Gangue depression and other rougher tests on transitional ore variants (35).

Key points of note during these preliminary programs are:

- Flotation under slightly acidic conditions showed promising early results and this condition was carried though all early test programs.
- Ore type designations were modified mid-way through the early programs to reflect LOM pit designations rather than individual deposits. New and old designations are summarised in Table 13.2.

Table 13.2 Correlation of new LOM pit names with deposit names

LOM pit names	Equivalent previous deposit name
SEL 1 and 2	Unchanged
BRC 2	BRS + FELDA
BRC 3	BRC + FELDA
BRC 4	BRC

The test results achieved in these programs (30 to 35) were reviewed and summarised by OMC in August 2018 (39). Discussion and recommendations to ensure data is usable for recovery estimates and engineering confirmation focused on:

- Staged collector addition and stage time modification to 1, 2, 4, 8, 10 minutes (from 5, 5, 5, 5, 5 minutes used in previous work) in order to minimise gangue recovery and infill early kinetic rate data.
- Standardised scrape rates and regulated air addition.
- High Au and S head grades in some ore types when compared to LOM average values, as well as high variability of the head grades between individual tests. This made assigning achieved test recoveries to individual ore types difficult and made comparison of tests on the same ore type to determine optimum flotation conditions difficult. Thus, ensuring that test head grades were representative of the ore type being tested was important.
- Ensuring that repeatability of test results can be shown.
- Investigating the response possible using sulphidisation and basic pH conditions to enhance Au recovery.

Following these recommendations and with the arrival of a sulphidising agent (Na_2S) at the SGSP laboratory, a further series of rougher and cleaner tests on LOM ore types (fresh variants) using sulphidisation and modified stage times were conducted (40, 42).

This program resulted in a set of flotation results that bettered those obtained in previous preliminary tests, were for the most part representative of the LOM ore types and demonstrated repeatability. In addition, the reagent scheme had been simplified and standardised from previous work. Key results from this program are summarised in Table 13.3 and Table 13.4.

Table 13.3 Summary of rougher test results showing repeatability (40)

ID		Calculated head grade		Combined rougher concentrate				
Ore type	Test no.	Au (g/t)	S (%)	Au grade (g/t)	S grade (%)	Au recovery (%)	S recovery (%)	Mass recovery (%)
FELDA	OMC3	2.22	1.12	9.14	4.66	94.44	95.65	22.91
FELDA	OMC3R	2.02	1.04	9.29	4.56	96.21	91.52	20.96
SEL1 FR	OMC1	0.80	0.49	3.48	2.04	94.44	95.65	19.51
SEL1 FR	OMC1R	1.01	0.72	5.41	3.21	96.21	91.52	17.89
SEL2 FR	OMC7	1.05	1.10	4.62	5.38	84.56	80.88	20.07
SEL2 FR	OMC7R	1.16	1.08	5.11	5.42	96.19	79.81	19.68
BRC2 FR	OMC4R	3.85	0.63	13.62	2.16	98.29	95.73	27.81
BRC2 FR	OMC4 R2	3.62	3.17	12.88	9.10	96.74	78.06	27.16
BRC3 FR	OMC10	1.59	1.12	6.59	4.51	92.85	90.52	22.47
BRC3 FR	OMC10R	1.62	1.04	6.47	4.40	91.66	96.89	22.92
BRC4 FR	OMC17	2.53	1.83	9.13	6.81	94.86	97.87	26.35
BRC4 FR	OMC17R	2.35	1.32	9.50	5.16	96.67	93.51	23.97

Table 13.4 Summary of open circuit cleaner test results (40)

ID		Calculated head grade		Combined cleaner concentrate				
Ore type	Test no.	Au (g/t)	S (%)	Au grade (g/t)	S grade (%)	Au recovery (%)	S recovery (%)	Mass recovery (%)
FELDA	OMC3CL	2.45	1.10	27.13	12.45	90.73	93.12	8.19
SEL1 FR	OMC1CL2	0.97	0.60	9.66	5.91	93.71	92.92	9.44
SEL2 FR	OMC7CL2	1.52	1.09	12.16	9.09	94.08	97.78	11.77
BRC2 FR	OMC4CL	3.82	2.36	24.42	9.30	87.68	95.92	13.72
BRC3 FR	OMC10CL	1.63	1.06	16.27	10.38	91.84	89.88	9.21
BRC4 FR	OMC17CL	2.12	1.53	21.31	15.5	91.87	92.5	9.15

Reagent dosage rates for each of these tests was similar:

- 400 g/t PAX collector
- 100 g/t $CuSO_4$ activator
- 300 g/t Na_2S sulphidising agent
- 2–4 kg/t Na_2CO_3 pH modifier and gangue dispersant achieving an average pH of 9.5
- MIBC frother.

BRC2_FR rougher tests showed some anomalies with regard to calculated head grades and recoveries. Elevated Au head grade of the cleaner test on this material (OMC4CL) suggests that further work needs to be carried out using representative sample head grades.

Work to define expected results from transitional ore variants and using locked cycle cleaner tests for all ore types, using a similar reagent scheme is ongoing (42).

The results of these tests are considered to be sufficient for LOM recovery estimates and basic engineering design, pending confirmatory tests at a third party commercial laboratory.

A dedicated program is being conducted on BRC4_FR material at Bureau Veritas, Perth, overseen by OMC (Section 0.0.0).

Testwork supervised by OMC at the BV Laboratory, Perth

In mid-August 2018, testwork at the Bureau Veritas Laboratory in Perth commenced under OMC supervision on a 200 kg sample of the BRC4-FR ore type (39, 41). This material type and quantity was the best available to begin testwork at the outset of the program. It is believed to be most representative of the BRC fresh ores (43) and their flotation response on average.

The approved program included:

- Rougher, cleaner, and locked-cycle flotation tests using feeds prepared to P_{80} = 75 µm.
- Combined gravity-leach-flotation testing
- Tests to de-slime flotation feed by hydrocyclone to remove silicate slimes.
- Conclusions.

The key focus of the program was:

- To closely replicate head grades for Au and S for the predicted head grade for BRC4-FR ore type in the LOM schedule, and by extension the combined BRC fresh ore types (43)
- To reduce variation in test results with regard to calculated head grade and the applied flotation methodology, and so allow reliable conclusions about the effects of different flotation conditions
- To cooperate with the site R&D team to identify and prove a reagent scheme that would maximise gold recovery and minimise the mass-pull to concentrate
- To demonstrate repeatability of key test results
- Flotation recovery and mass-pull estimates
- Flotation conditions, reagent scheme and consumption rates
- Kinetic/engineering confirmation data for circuit design.

Results from the program to date include:

- Confirmation that the composited BRC4_FR head assay was 1.82 g/t Au, 1.12% total sulphur, 3.5% Fe, 0.50% As and 0.19% organic carbon. These assays are close to the values shown in the LOM mining schedule (1.73 g/t Au, 0.90% S). Individual flotation test calculated head assays also varied little around this assayed head.

- QEMSCAN and XRD analysis confirmed that the mineralogy of the crystalline phases was dominated by quartz, dolomite, mica and plagioclase, with trace quantities of pyrite, arsenopyrite, stibnite and chlorite. Only three grains of free native gold were detected, all -3 µm in size.
- Diagnostic leaching on the BRC4-FR feed sample indicated the presence of only 13.5% of leachable free gold, with the rest either locked in other minerals or in solid solution in sulphides. More than 30% of the gold was locked in silicates.
- Recovery of gold by gravity concentration averaged only 2.3%, and this result together with results from the diagnostic leach confirm the refractory nature of this ore.
- De-sliming tests by hydrocyclone produced a cyclone overflow product with D_{90} of 10.2 µm and this product contained 11.6% of feed gold and 6.1% of feed sulphur in a high mass split of 31.2 wt. % in a low-grade product unsuitable for further treatment. This suggests that de-sliming to reduce concentrate mass-pull will not be a suitable process for treating this ore due to unacceptable gold loss. This result is similar to de-sliming results obtained on site using an up-flow elutriation cone.
- Flotation tests established an effective reagent regime using a combination of copper sulphate activator plus a sulphidiser (NaHS or Na_2S) using PAX collector at a pH of 9.0–9.5. The use of a sulphidiser to deliver improved gold recovery at an alkaline pH has been confirmed for this ore, and the results are supported by flotation testing at the site R&D Laboratory on the other Buffalo Reef ore types which confirm the improvement in gold recovery which can be achieved by use of sodium sulphide at alkaline pH.
- The key interim finding from the program, which is ongoing, for the BRC4-FR ore type was that a repeatable 92.5% gold recovery and 93% sulphur recovery into a 12.5% mass-pull has been achieved by rougher flotation only into a concentrate with 8.5% S, suitable for the BIOX® process.
- Testing to achieve further recovery improvement and reduction in reagent consumption rates are on hold.
- A locked cycle cleaner test with higher rougher mass recovery is pending to complete the Bureau Veritas program.

13.3.2 BIOX® engineering confirmation testwork

Size distribution requirements of flotation concentrate for BIOX® feed

The BIOX® stipulation for grind size is 99% passing 150 µm and 80% passing 75 µm, which is currently being achieved by the existing milling circuit on oxide ores.

Based on the batch grinding conditions employed at the time, flotation concentrates supplied to Outotec for BIOX® testing had the following sizings:

- Selinsing I was 100% passing 75 µm and 97.5% passing 38 µm
- Selinsing II was 100% passing 106 µm and 88% passing 38 µm
- Selinsing III concentrate was found to have sizings of 99% passing 150 µm and 90% passing 45 µm
- SEL1 CL and SEL2 Fresh 99% <150 µm and 80% <38 µm
- BRC2 Transition 99% <150 µm and 80% <15 µm.

The Selinsing III investigation used concentrate produced from BRC2-Fresh ore and was used by Outotec to design the BIOX® plant.

BIOX® batch amenability testwork

A series of batch amenability tests (BATs) were conducted under the supervision of Outotec at the SGS laboratories in Johannesburg, South Africa, during 2017 to establish the suitability of the BIOX® process for subsequent gold recovery from the refractory Selinsing sulphide ores.

The first suite of tests was conducted during February 2017 using low sulphide (Selinsing I) concentrate (27), the second (Selinsing II) was conducted from May to July 2017 using high antimony concentrate, and the third was conducted during October 2017 using a lower antimony (Selinsing III) concentrate (36).

The chemical analyses of the three Selinsing flotation concentrates are provided below in Table 13.5.

Table 13.5 Chemical analyses of BIOX® BAT testwork

Species/Ratio	Unit	Selinsing I	Selinsing II	Selinsing III
Au	g/t	8.8	29.9	39.8
Ag	g/t	2.3	7.2	1.0
Cu	ppm	448	-	-
Si	%	24.9	15.4	22.0
Fe	%	7.4	8.2	12.5
S (total)	%	5.9	14.2	10.5
S (sulphide)	%	4.1	14.1	9.5
S (elemental)	%	<0.3	<0.3	<0.3
S (sulphate)	%		<0.3	<0.4
Hg	ppm	1.4	0.3	1.0
Sb	ppm	9,430	121,000	5,620
As	%	1.7	3.1	4.3
C (total)	%	2.1	-	0.5
C (organic)	%	1.0	0.4	0.1
CO_3^{2-}	%	4.0	3.9	1.0
SG	kg/L	3.0	3.4	3.2
Ratios				
Fe/As molar		5.8	3.5	3.9
Au/S^{2-}		2.1	2.7	4.2
S^{2-}/CO_3^{2-}		1.0	2.8	9.5

Selinsing I BIOX® testwork

Concentrate for the Selinsing I testwork was produced from the Intec sample, a weighted blend of Buffalo Reef flotation concentrates, combining all known ore sources at the mine site. For the adaptation of Selinsing I, a microbial stock culture adapted to a pyrite/arsenopyrite substrate was used as the inoculum source. The adaptation and inoculum build-up stages provided the active microbial culture to commence the batch amenability tests.

Seven batch amenability tests were conducted using the Selinsing I concentrate (27). Each test used a liquid/solid ratio of 3:1, 0K nutrient (iron free), sulphuric acid to pH 1.4, 25% solids concentration and 10% inoculum addition. The individual tests were allowed to run for 2, 3, 5, 8, 10 and 12 days. During each test sulphuric acid or lime was added to maintain pH 1.2–1.4. Ferric and ferrous ion concentration, pH, redox potential and O_2 levels were recorded.

Upon the completion of each test the slurries were filtered, washed and dried. The residues were submitted for chemical analysis and subjected to cyanide leach tests. The cyanide leach tests were performed with a conditioning stage at lime at pH 11.5 followed by additions of activated carbon and sodium cyanide. Leach tests were allowed to run for 24 hours with lime added to maintain minimum pH 10.5. The solids residue, carbon and solution were all analysed to determine gold dissolution.

An average 99.4% sulphide oxidation was achieved when exposed to a BIOX® batch pre-treatment period of 12 days. A 90% gold recovery was achieved from subsequent cyanidation of the BIOX® pre-treatment solids, compared to 13% gold recovery from direct cyanidation of the untreated flotation concentrate. Batch neutralisation of the acidic BIOX® solution yielded an environmentally stable ferric arsenate waste product with an average soluble arsenic concentration of less than 0.05 mg/L, well below the limit of 5 mg/L set by the USA EPA guidelines for tailings disposal.

Selinsing II BIOX® testwork

The concentrate for Selinsing II was derived from the ore type known as BRC QS (i.e. Buffalo Reef Central – Quartz Stibnite), a subspecies of the QA or Quartz Ankerite that dominates BRC mineralisation.

Despite the high antimony concentration in the Selinsing II concentrate the adaptation and inoculum build-up stages progressed well. The high solution redox values, low ferrous ion values and high lime requirement indicate that microbial activity was high. However, the ferric ion in solution was much lower than expected. It was found that the antimony in solution was co-precipitating with ferric ion and forming a compound called Chapmanite, an iron-antimony silicate.

Testing of the BIOX® residue after 11 days indicated 35% sulphide oxidation. Samples of the original concentrate and BIOX® residue after 11 days were subjected to cyanidation testing. The results indicated 34% gold dissolution for the unoxidized concentrate and 39% for the 18-day sample. The prevalence of antimony was believed to have passivated the gold surface and was the main reason for the poor gold recovery.

The Selinsing II testwork program was subsequently abandoned due to the high antimony concentration in the feed. The antimony grade of the concentrate feed to BIOX® was subsequently capped at 4% to ensure no subsequent deleterious effect on the BIOX® circuit. Based on a 10% mass-pull to flotation concentrate, a maximum 0.42% antimony in mill feed was specified, which will be achieved by a combination of selective mining and ore blending.

Selinsing III BIOX® testwork

The concentrate for Selinsing III testwork comprised BRC2 (fresh) or Buffalo Reef Central Pit 2, the orebody which will dominate the first two years of production and which is considered representative of most of the Buffalo Reef ore, but this has not been established.

A sample of Selinsing III concentrate was stabilised overnight at 10% solids and sulphuric acid pH 1.4 before adding active microbial culture drawn from the Selinsing II program. The slurry was maintained between pH 1.2 and 1.6 through the addition of lime and sulphuric acid. Measurements were made of ferrous and ferric ion concentration and solution potential. Once microbial activity started increasing, further concentrate was added to reach 20% solids concentration. When the ferrous ion concentration had decreased to 0.2 g/L inoculum was drawn from this stage to inoculate the Selinsing III BATs.

Seven batch amenability tests were conducted (29) and allowed to run for 3, 6, 9, 13, 15 and 24 days. After each test the residues were filtered and washed. Dried residues and product solutions were analysed to determine the extent of sulphide oxidation and residues were subjected to 24-hour cyanide leach tests.

The analytical results showed that sulphide oxidation had reached 90% after nine days, 98% after 15 days and 99% at test completion after 24 days. Cyanide leaching tests showed 21.4% dissolution on the un-oxidised concentrate and 90.5% recovery from the 99% oxidized, 24-day residue. A repeat test conducted for 24 days also achieved 99% sulphide oxidation with 92% gold recovery after a longer pH adjustment period.

Batch neutralisation of the acidic BIOX® solution was carried out using: two-stage neutralisations with limestone to pH 5 and lime to pH 7; single-stage neutralisation with lime only to pH 5 and slowly raised to pH 7. The resultant products were tested for compliance with the US EPA standards for arsenic. The neutralised effluents contained arsenic concentration of <0.40 mg/L thus satisfying the EPA requirements of a maximum 0.50 mg/L. However, TCLP tests on the solids produced extracts containing 5.5 mg/L and 6.8 mg/L respectively, exceeding the EPA limit of 5 mg/L.

BIOX® variability testwork

Samples of cleaner flotation concentrate from SEL1 (Pit IV), SEL2 (Pit V and VI) and BR C2 (transition) were sent for BIOX® testwork at the SGS Laboratories in Johannesburg, under the supervision of Outotec. SEL1, SEL2 and BR C2 (transition) represent the ore sources to be mined along with BR C2 (fresh) during the first two years of production.

The chemical analyses of the three Selinsing variability flotation concentrates are provided below in Table 13.6.

Table 13.6 Chemical analyses of flotation variability concentrates submitted for BIOX® testing

Species/Ratio	Unit	SEL1	SEL2	BR C2 (transition)
Au	g/t	71.9	12.6	12.8
Ag	g/t	11.3	2.4	0.8
S (total)	%	9.3	9.9	4.5
S (sulphide)	%	8.9	9.7	3.8
Sb	ppm	41	101	10,111
As	%	5.2	5.0	2.5
C (total)	%	0.5	1.8	0.3
C (organic)	%	0.1	0.3	0.2
CO_3^{2-}	%	0.9	6.0	0.4
SG	kg/L	3.3	3.2	2.9
Fe/As molar		2.9	2.7	3.2

The laboratory batch amenability testwork (36) (BAT) followed the previous procedures adopted for Selinsing I, II and III. The adaptation phase employed active Selinsing III inoculum with a composite of the three variability samples added in three stages. Following the adaptation stage, the inoculum build-up stage again involved a blend of the three samples stabilised overnight with added sulphuric acid to pH 1.4 at 20% solids with 0K nutrient (no iron substrate). The slurry was inoculated with the culture from the adaptation stage and maintained at 38–42°C and pH 1.2–1.6. Once the ferrous ion concentration had reached <0.2 g/L the inoculum was drawn from this stage for the batch amenability tests.

Six BATs were conducted, with each concentrate sample tested in duplicate. The BATs were initiated and monitored under the same conditions as for previous tests. Upon completion of each test the slurry was filtered and washed. The filter cake was dried, subsampled for assay and sent for cyanide leach testing. The bioleach liquors were sent for analysis and neutralisation tests. The cyanide tests were conducted over 24 hours after conditioning at lime pH 11. Activated carbon and cyanide was added and pH maintained at minimum 10.5 during the 24-hour leach. At the end of the leaching period the solids were filtered and washed, and assays conducted on the carbon, solids and solution to determine the gold recovery.

Neutralisation tests were conducted on the bioleach solution to determine the efficacy of Selinsing site limestone compared with analytical grade (AR) limestone.

Variability testwork results

Both the adaptation and inoculum build-up stages performed well using the combined concentrate from SEL1, SEL2 and BRC2 (transition). The individual BATs also progressed very well with good oxidation levels reported for all three concentrates.

The SEL1 BAT ran for 16 days and achieved 99.1% sulphide oxidation; SEL2 was stopped after 18 days with 99.0% sulphide oxidation; the BRC2 BAT ran for 11 days, achieving 98.3% sulphide oxidation.

Sulphuric acid consumptions were high during these tests, reflecting the higher proportion of arsenopyrite versus pyrite in each of the three concentrates as well as the high carbonate content of SEL2.

Leachability tests on the unoxidized flotation concentrates gave gold recoveries of 96.8%, 51.0% and 56.7% for SEL1, SEL2 and BRC2 (transition) respectively. Leachability tests on the bioleach residues produced gold recoveries of 99.5%, 94.9% and 94.6% under standard bottle roll conditions. Gold dissolution of 97.4% was achieved on a combined sample comprising equal proportions of the three concentrates.

Neutralisation tests using Selinsing site and AR grade limestone showed that either could be used to produce stable neutralisation precipitates that satisfied the US EPA guidelines for arsenic; the arsenic content of the effluent stream was also below the EPA limits.

Conclusions

The BAT testwork programs showed that the Buffalo Reef and Selinsing flotation concentrates were amenable to the BIOX® process, achieving high levels of sulphide oxidation, typically 98–99%.

Leachability tests on the BIOX® residues produced good results: initial tests on BIOX® residues from Selinsing I and Selinsing III gave around 90% gold recovery; variability testwork has indicated that higher recoveries ranging from 94% to 99% may be possible.

The presence of a high proportion of antimony in the flotation concentrate was shown to be deleterious and would need to be controlled through selective mining and ore blending.

A treatment period of up to 12 days was used for the Selinsing I batch amenability tests and 24 days were used for the Selinsing III batch tests; the variability tests used between 11 and 18 days. The treatment period for continuous operation to achieve the same extent of sulphide mineral oxidation as the batch tests will be lower due to the constant supply of fresh substrate hosting larger and more active microbe populations. Faster microbial assisted leach kinetics are achieved during continuous operation.

No continuous pilot testwork was conducted on the Selinsing concentrate samples to derive the kinetic parameters so the respective final extent and rate constants utilised in the Selinsing design have been sourced from within Outotec's database on a concentrate which had similar chemical characteristics, and which had undergone a continuous pilot trial.

13.3.3 LOM recovery estimate

Based on the test results achieved to date and variable levels of uncertainty, Table 13.7 summarises a realistic estimate of the expected gold recovery for the Selinsing Sulphide Project, and Table 13.8 shows the references, test results used and any adjustments applied to arrive at the estimate. Supporting flotation results are provided in Table 13.9. The following considerations have been applied to ensure the estimates are realistic:

- Flotation recovery estimates:
  - For fresh ores – all ore types have fairly conclusive tests and tests results have been assumed as achievable. The most detailed and conclusive testwork for the BRC FR ore types has been conducted on BRC4 FR material. As per (43), the achieved recoveries for BRC4 FR have been applied to BRC2 FR and BRC3 FR, with a small negative adjustment on BRC2 FR due to uncertainty achieving the higher recovery based on test results to date. The overall BRC FR recovery has been calculated by averaging the recovery for BRC2, BRC3 and BRC4.
  - For transitional ores – recoveries have been consistently lower for these ore types. BRC2 and 4 TR have conclusive test results using sulphidisation and locked cycle cleaning

tests, and these have been assumed achievable. BRC3 TR has had a rougher test only conducted and had a very high mass-pull (37%), this result has been de-rated by 5%. Other transitional variants have assumed the fresh locked cycle cleaner test result minus a further 3% adjustment. The overall BRC TR recovery has been calculated by averaging the recovery for BRC2, BRC3 and BRC4.

- BIOX® recovery estimates:
  - BIOX® testwork (phases I and III) has conclusively shown that variously 90.0%, 90.5% and 92.0% Au recovery is achievable for the BR fresh ore types. An average of these results has been applied (90.8%).
  - Selinsing concentrates yielded 95–99% BIOX® Au recovery in variability testwork, and the average recovery (97.2%) has been assumed.
  - BRC2 TR concentrates yielded 94.6% BIOX® Au recovery and this has been assumed as achievable.
  - Concentrate derived from other transitional ore types was not specifically tested and are assumed to give a similar BIOX® Au recovery to BRC2 TR (94.6%).

The overall project is expected to yield 85.0% recovery of contained gold from the combined gravity separation, flotation and BIOX® processes.

Table 13.7 LOM Au recovery estimates

Ore type	Gravity/Flotation Au recovery (%)	BIOX® Au recovery (%)	Total Au recovery (%)	Weighting by Au content (%)	LOM Au recovery (% contribution)
SEL FR	93.5	97.2	90.9	19.5	17.7
SEL TR	90.5	94.6	85.6	1.0	0.9
BRC2 FR	93.5	90.8	85.0		
BRC2 TR	78.2	94.6	74.0		
BRC3 FR	94.5	90.8	85.9		
BRC3 TR	85.1	94.6	80.5		
BRC4 FR	94.5	90.8	85.9		
BRC4 TR	83.1	94.6	78.6		
BRC FR *			85.6	60.9	52.1
BRC TR *			77.7	16.9	13.1
BRN FR	78.4	90.8	71.2	0.3	0.2
BRN TR	75.4	94.6	71.3	1.4	1.0
Total LOM Au recovery estimate (%)					**85.0**

** Note – average of BRC2, BRC3 and BRC4 – used for LOM recovery estimate*

Table 13.8 Flotation test references and comments

Ore type	Test no. (Ref)	Test Au recovery (%)	Adjustment for estimate (%)	Comment
SEL FR	Average of SEL1_FR LCR and SEL2_FR LCR (42)	92.5	+1.0	Locked cycle cleaner test using sulphidization. Low head grade and mass-pull, positive adjustment.
SEL TR	-	-	-3.0 (from fresh)	No tests.
BRC2 FR	-	-	0.0	Assumed same as BRC4_FR.
BRC2 TR	BRC2_TR OMC14 LC (42)	78.2	0.0	Locked cycle cleaner test using sulphidization.
BRC3 FR	-	-	0.0	Assumed same as BRC4_FR.
BRC3 TR	BRC3_TR OMC2 (42)	90.1	-5.0	Rougher test only, high mass-pull, negative adjustment.
BRC4 FR	OMC17 LC, RPT1, RPT2 and RPT3 (42)	95.5	-1.0	Repeated locked cycle cleaner test using sulphidization. High mass-pulls (13–14%), 1% negative adjustment.

BRC4 TR	BRC4_TR OMC1 LC (42)	83.1	0.0	Locked cycle cleaner test using sulphidization.
BRN FR	BRN_FR CL1 (31)	78.4	0.0	Pre-sulphidization result. Should be able to achieve better with sulphidization.
BRN TR	-	-	-3.0 (from fresh)	No tests.

Table 13.9 Flotation tests – supporting results

Test no.	Calculated head grade (Au, g/t)	Rougher concentrate			Cleaner concentrate		
		Grade	Recovery		Grade	Recovery	
		Au (g/t)	Mass (%)	Au (%)	Au (g/t)	Mass (%)	Au (%)
SEL1_FR LCR	0.91	-	-	-	10.36	8.31	94.72
SEL2_FR LCR	1.18	-	-	-	9.95	10.74	90.29
BRC4_FR OMC17 LC	1.76	-	-	-	16.76	9.06	94.22
BRC4_FR OMC17 LC RPT1	1.63	-	-	-	10.67	14.70	96.13
BRC4_FR OMC17 LC RPT2	1.79	-	-	-	11.59	14.71	95.22
BRC4_FR OMC17 LC RPT3	1.64	-	-	-	11.62	13.65	96.62
BRC4_TR OMC1 LC	2.22	-	-	-	14.64	12.58	83.10
BRN_FR CL1	2.31	8.85	21.69	83.09	18.24	9.94	78.41
BRC2_TR OMC14 LC	1.93	-	-	-	18.59	8.13	78.21
BRC3_TR OMC2	2.17	5.20	37.64	90.07	-	-	-

Recommendations

Further flotation testwork is recommended to better define recovery estimates should include:

- Repeat and optimise on site testing of the BRC2 fresh ore type at representative head grades using sulphidization reagents.
- Carry out complete locked-cycle cleaner tests for all ore types on site (underway), summarise and report results in a single document.
- Carry out locked-cycle cleaner tests on the BRC4 FR ore type at Bureau Veritas (underway).
- Undertake third party laboratory confirmation of the results achieved at the site laboratory for LOM ore types other than BRC4 FR. Rather than testing individual ore types it may be more efficient to test composites that represent different phases of the mine plan.

Antimony distribution at Buffalo Reef

This section documents the occurrence and distribution of antimony (Sb) in the Buffalo Reef model areas (BRSCF and BRN) in relation to the LOM designed pits. Antimony (Sb) has only been estimated in the Buffalo Reef South Central Felda (BRSCF) and Buffalo Reef North (BRN) block models. Antimony was not estimated at Selinsing due to the limited Sb assay data.

Antimony has a refractory effect on the BIOX® Process and also on the recovery of gold by cyanidation. Ideally the Sb concentration of flotation feed should not exceed 4,200 ppm, however, the BRC–QS bulk sample of QS (quartz-stibnite) mineralisation type at Buffalo Reef has a Sb head grade of 15,200 ppm.

Antimony grades for fresh sulphide blocks in the mineralised zones, above 0.7 g/t Au for the BRSCF area are shown in Figure 13.5 and Figure 13.6. It is apparent that there is a high proportion of blocks with very low Sb grade (<100 ppm). The locations of higher Sb (>1,000 ppm) are reasonably well defined and associated with interpreted QS mineralisation.

Antimony is generally low (<100 ppm) in the Buffalo Reef LOM pits. Higher Sb grades are predominantly associated with the QS mineralisation type, which is more pronounced in the BRC4 LOM pit. The antimony distribution is very erratic, although some continuous higher-grade zones are observed (<100 ppm) in BRC2, BRC3 and BRC4 LOM pits. Monument believes that the more continuous high-Sb zones can be defined during grade-control with close spaced drilling and mined selectively.

The average Sb grade within the BRSCF LOM pits is 2,100 ppm, which is approximately half of the metallurgical threshold of 4,200 ppm. A long section showing the Sb within the LOM pit shells is shown in Figure 13.7, which suggests that the Sb distribution is somewhat erratic both vertically and horizontally.

Figure 13.5 Sb estimated grades within BRSCF Modelled Mineralised Sulphide

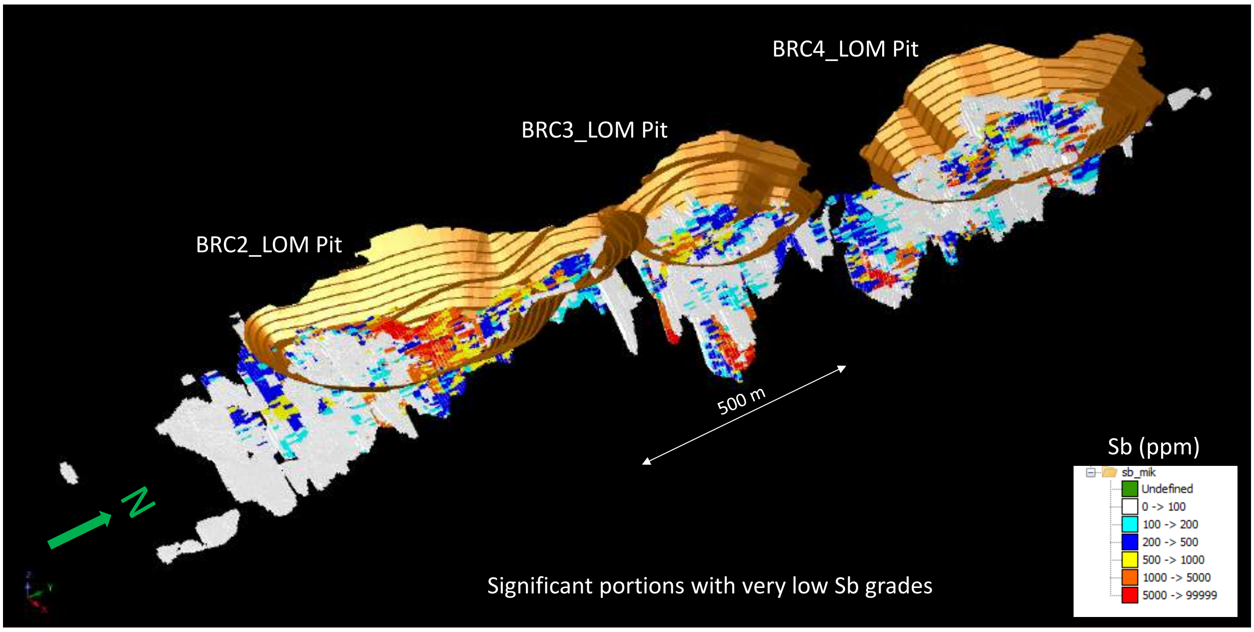


Figure 13.6 Sb estimated grades within BRSCF Modelled Mineralised Sulphide >= 0.70 g/t Au within the LOM pits

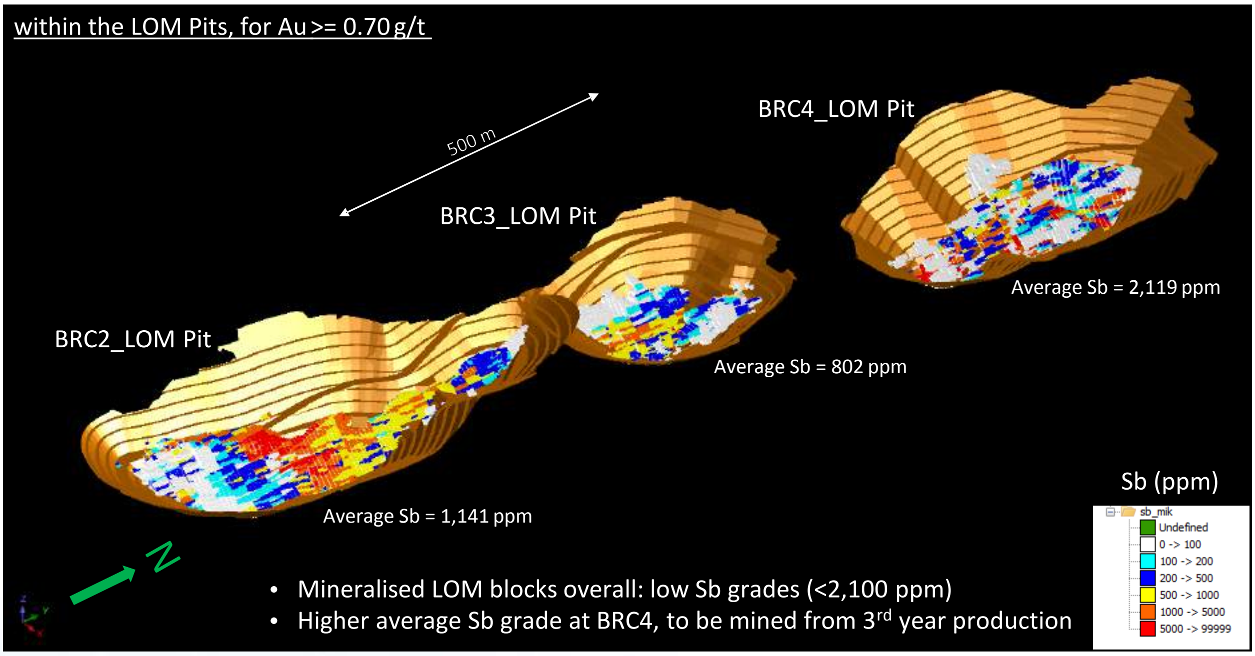


Figure 13.7 Long section (looking west): BRSCF Modelled Mineralised Sulphide >= 0.70 g/t Au within the LOM pits

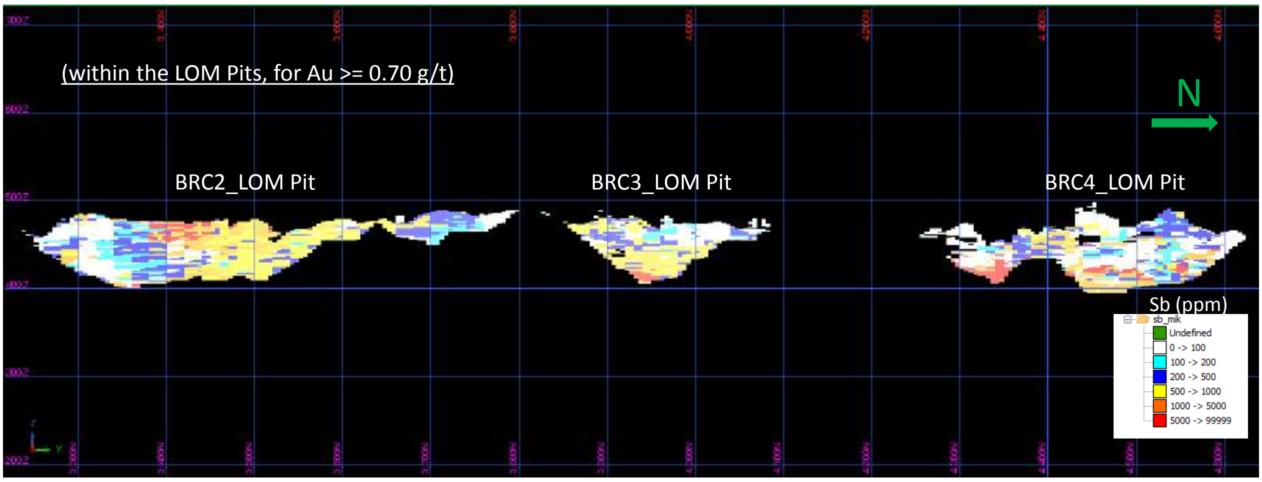


Figure 13.8 shows the Sb distribution in the fresh sulphide mineralised blocks for the BRN model area, in relation to the LOM pit shells, which shows that the majority of BRN has a very low Sb grade, below the 4,200 ppm threshold.

Figure 13.9 shows the Sb distribution in the fresh sulphide mineralised blocks for the BRN model, within the LOM pit shells only and >=0.7 g/t Au. No fresh mineralisation has been modelled for the BRN1 LOM pit. The average Sb grade for the BRN2 LOM pit is approximately 140 ppm.

Figure 13.8 Sb estimated grades within BRN Modelled Mineralised Sulphide

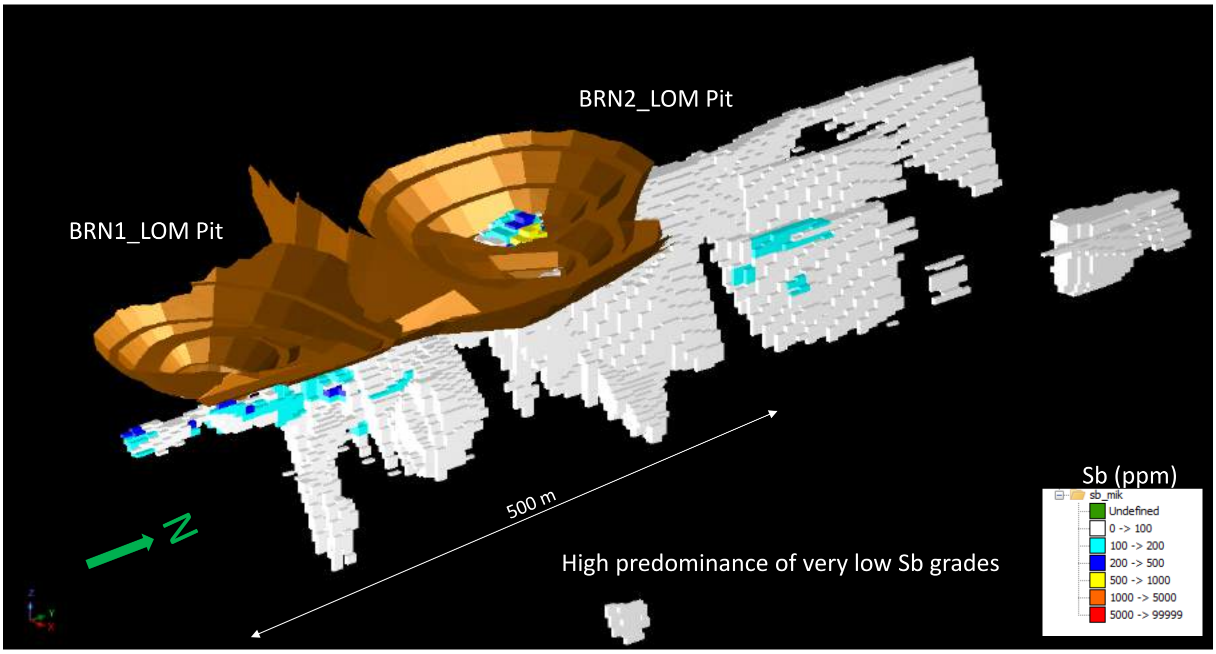


Figure 13.9 Sb estimated grades within BRN Modelled Mineralised Sulphide >= 0.70 g/t Au within the LOM pits

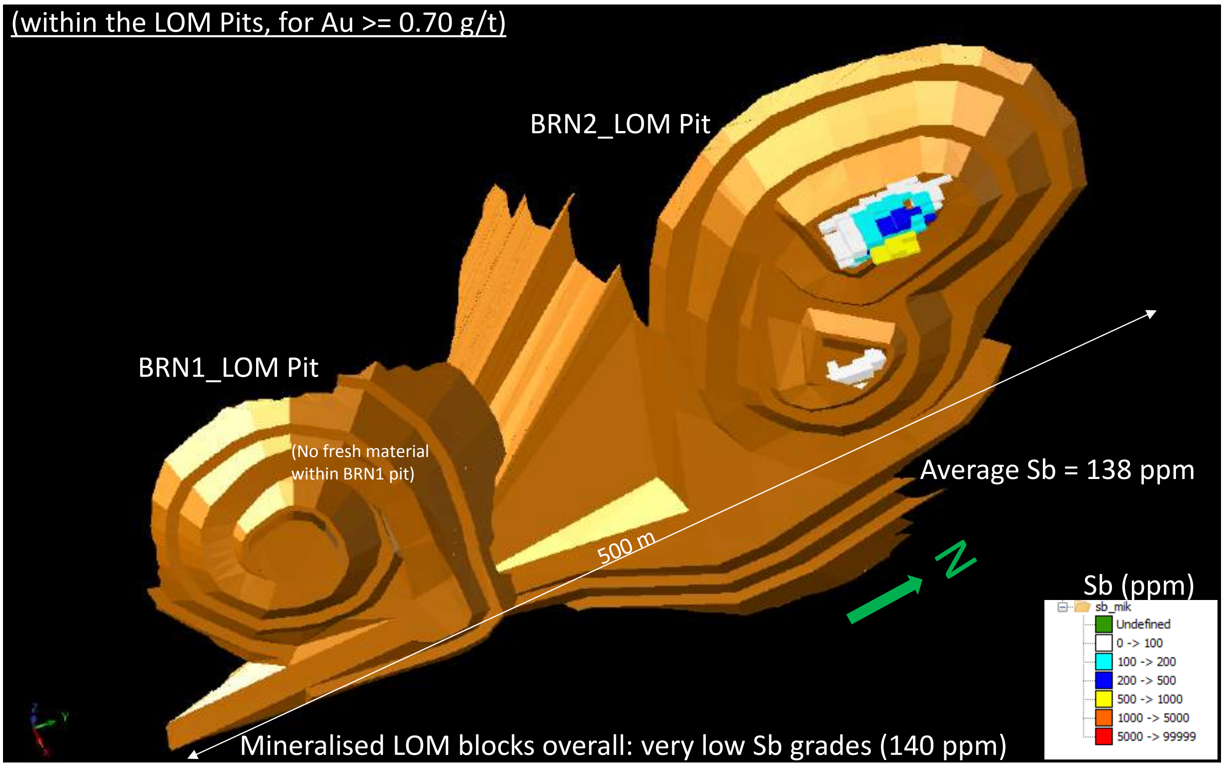


Figure 13.10 and Figure 13.11 show the locations of the QS samples in relation to the LOM pit shells. The samples are mostly within and beneath the BRC4 pit and to the south, between the BRC4 and BRC3 LOM pits. Stibnite-bearing ore was tested by flotation and the flotation concentrate was used for the second BIOX® metallurgical testwork program. The BRC-QS sample can be considered representative of the sulphide mineralisation occurring within or beneath the pit, in a broad sense (bulk sample scale). The composited bulk sample has approximately half of the composing intervals within the BRC4 pit (bottom of the pit) and half beneath that, within the same geological environment.

The metallurgical drilling that supplied core intersections selected for the bulk sample used to generate the flotation concentrate, was conducted prior to the design of the LOM BR pit shells. At that time (first half of 2016) the intention was to get samples representative of the sulphide resource at the Buffalo Reef/Felda deposits. A portion of the resulting flotation concentrate was later sent for BIOX® testwork.

A total of 179 sample intervals were used to generate the sulphide BRC-QS composited sample, 215.85 m in total, 99.50 m (46%) of which are within the BRC4 LOM designed pit, and the remaining 116.35 m (54%) are beneath and to the south of this LOM designed pit.

The QS samples do not always coincide with the pure QS mineralisation style, i.e. consistent high grades of gold associated with stibnite quartz veins occurring in depth at the BRC and Felda deposits. These QS samples can appear at shallower depths mixed with quartz-ankerite (QA) samples.

Given that approximately half of the QS tested samples occur within one of the LOM pits (BRC4), mostly near the bottom of this pit, with the other half beneath this pit, the composited sample can be considered indicative of later LOM pit stages, comprising the gradual transition of predominantly QA mineralisation (shallower) into predominant QS type (deeper).

Figure 13.10 BRC-QS sample location (in red), in relation to the BRC4 LOM pit

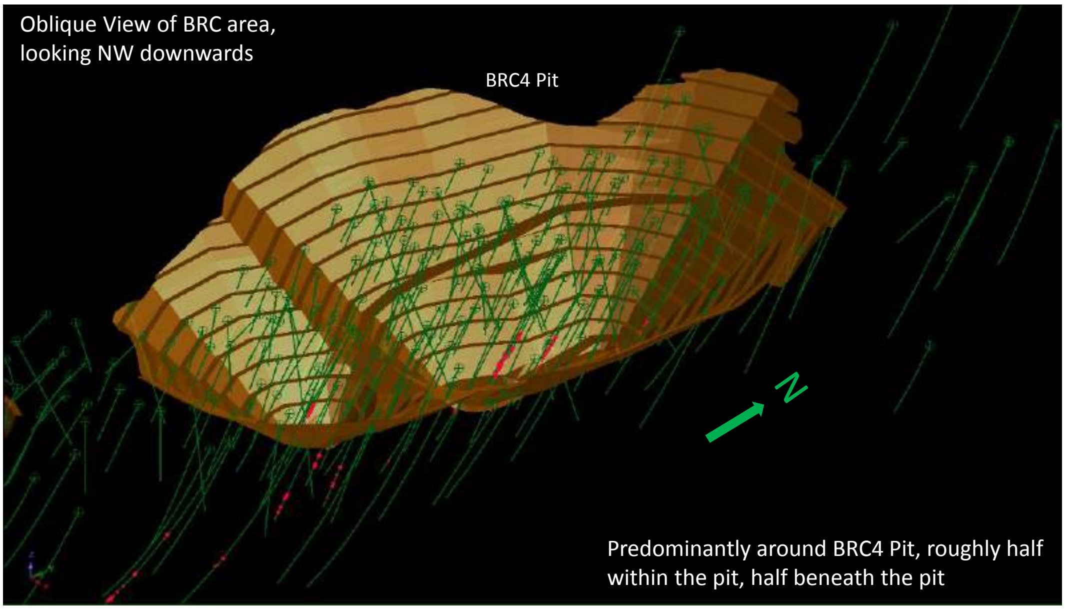
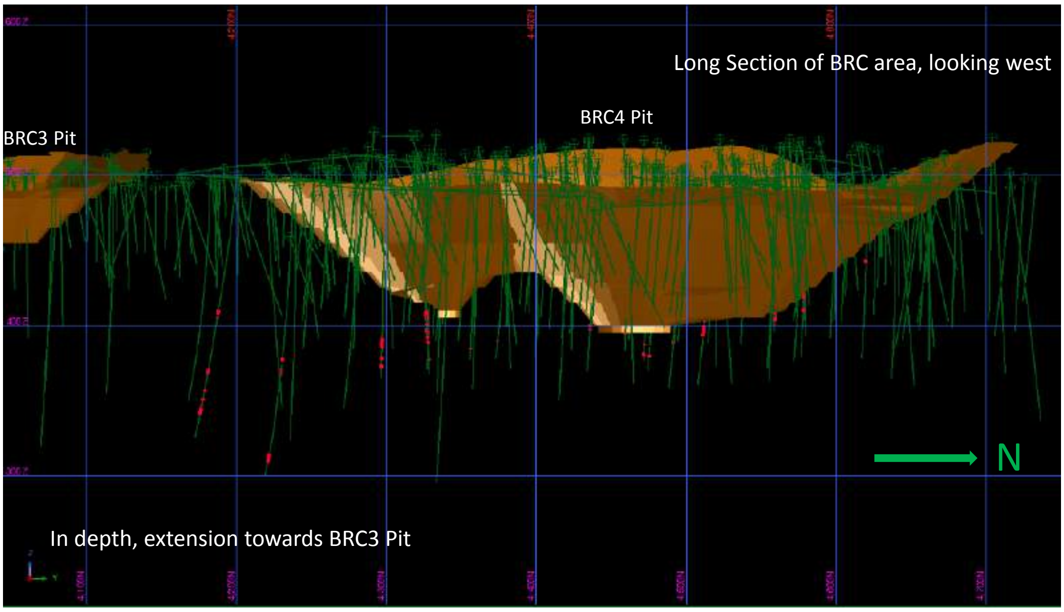


Figure 13.11 BRC-QS sample location (in red), in relation to the BRC3 and BRC4 LOM pits



The Sb statistics for 1.5 m composite samples used as input data for the block grade estimation at BRSCF and BRN are shown in Section 14.4.2. Overall, Sb grades tend to be quite low, especially at BRN. However, the data shows a mixed distribution with a low-grade population centred on approximately 40 ppm Sb and a high-grade population centred on approximately 2,500 ppm Sb (especially for BRSCF).

Within the modelled mineralisation, Au and Sb show a weak to moderate correlation (Section14.5.2). The scatterplots in log scale show that high Sb grades are associated with high Au grades; however, high Au grades are not always associated with high Sb, likely as a result of two phases of mineralisation – the more common QA, and more localised (between BRC3 and BRC4) QS. The two mineralisation types commonly occur in proximity to one another; however, it is anticipated that grade control will allow the two mineralisation types to be mined selectively.

As discussed in the metallurgy section, Stibnite (Sb_2S_3) is locally present at Buffalo Reef, as opposed to the main auriferous sulphides, pyrite (FeS_2) and arsenopyrite (FeAsS), which are spread throughout the entire Buffalo Reef and Selinsing deposits. Stibnite often contains intergrowths of native gold.

A mineralogical study compiled in 2011 by an external consultant (Joe Zhou, 2011) confirmed the presence of stibnite as one of the sulphide phases present together with pyrite and arsenopyrite. Stibnite was also observed in polished sections of flotation concentrates, and in hand specimens collected at BRN deposit, examined using a stereobinocular microscope (Wort, April 2017).

The mineral berthierite ($FeSb_2S_4$) in Buffalo Reef has also been reported by two authors (Lehne & Associates, 2011, and Vancouver Petrographics Ltd, 2012), and is often intergrown with stibnite. In addition, trace tetrahedrite $(Cu,Fe)_{12}Sb_4S_{13}$ has been reported as well as a late stage infilling of brecciated pyrite (Vancouver Petrographics Ltd, 2012).

Whilst antimony grades have not been estimated in the Selinsing resource model, core samples from diamond holes within the Selinsing 1 and Selinsing 2 LOM pits, which were assessed for BIOX® variability testwork, have been assayed for Sb, which, whilst limited, provides an indication of the expected Sb grades in the fresh sulphide mineralisation at the Selinsing deposit.

The SEL1_FRESH composite bulk sample showed an average grade of 294 ppm Sb and median grade of 51 ppm Sb, ranging from 8 ppm Sb to 3,315 ppm Sb and only one value of >70 ppm Sb. The SEL2_FRESH composite bulk sample showed an average grade of 108 ppm Sb and median of 73 ppm, ranging from 31 ppm to 835 ppm and only one value >180 ppm Sb. However, these antimony grades are only considered indicative and further sampling and assaying is required to allow adequate modelling of the Sb grades in the resource model and to improve the confidence in the Sb distribution assumptions.

13.4 BIOX® variability testwork – sample details

Core and coarse reject samples were collected for BIOX® process variability testwork, with the aim of covering each LOM pit as defined in the 2016 Selinsing PFS (Snowden, 2016).

Transition and fresh samples in the modelled mineralised zones were identified and further verified for availability, with a nominal lower cut-off grade of 0.7 g/t Au (based on the cut-off grade used for the 2016 Mineral Resource; Snowden, 2016). The selected samples were distributed throughout each proposed LOM pit, with preference given to consistent, continuously mineralised intersections, including some internal waste.

Where samples were not available within the proposed LOM pit (e.g. SEL2_Fresh), sample intervals located just below the pit floor in the same mineralisation environment were selected.

The selected composited sample groups are:

- SEL1_Fresh
- SEL2_Fresh
- BRC2_Transition
- BRC2_Fresh
- BRC3_Transition
- BRC3_Fresh
- BRC4_Transition
- BRC4_Fresh
- BRN2_Transition.

These nine sample groups comprise a total of 245 individual samples from 53 drillholes, with a total weight of 915 kg (Table 13.10).

Table 13.10 List of composite sample groups used for BIOX® testwork

Composite sample group	No. of individual samples	No. of source drillholes	Total weight (kg)
SEL1_Fresh	18	3	76
SEL2_Fresh	34	9	158
BRC2_Fresh	33	7	205
BRC2_Transition	20	4	73
BRC3_Fresh	26	4	105
BRC3_Transition	18	9	42
BRC4_Fresh	37	5	129
BRC4_Transition	40	7	93
BRN2_Transition	19	5	34
Total	**245**	**53**	**915**

For the SEL1 LOM pit, all transition material has been mined. For the SEL2 LOM pit, only fresh samples from below the pit were available.

The locations of the samples in relation to the proposed LOM pit shells are shown in Figure 13.12 to Figure 13.14, with the estimated gold distribution in the block model.

Figure 13.12 Selected samples for SEL1 and SEL2 Fresh

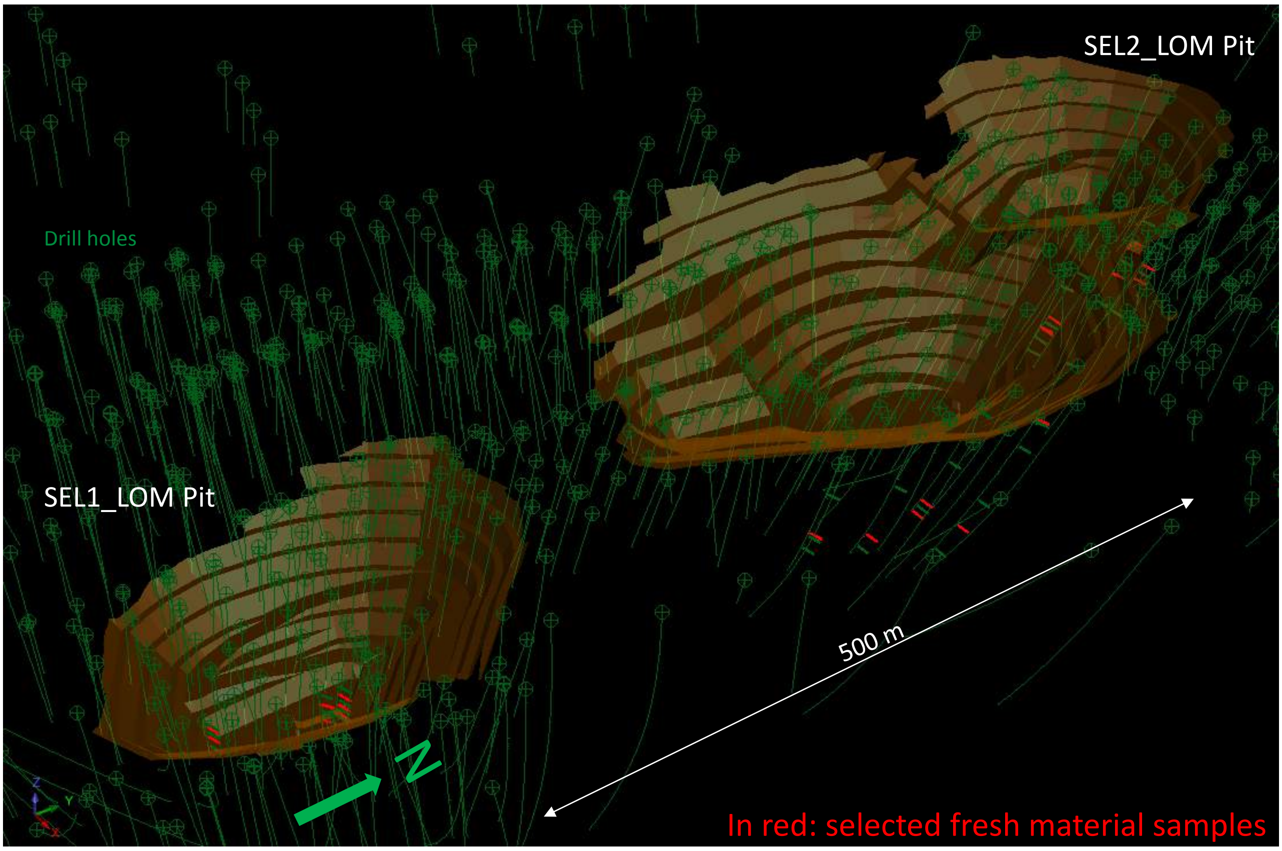


Figure 13.13 SEL1 and SEL2 LOM pits and mineralised Fresh blocks

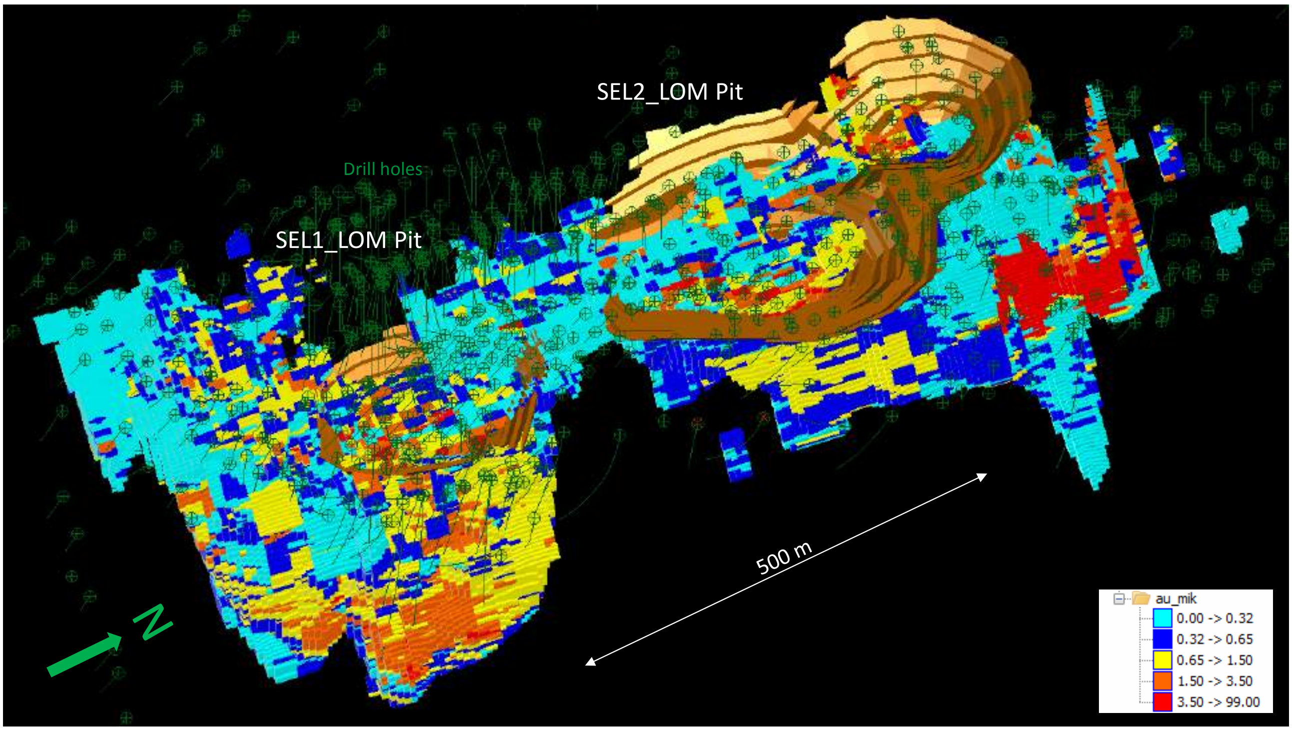
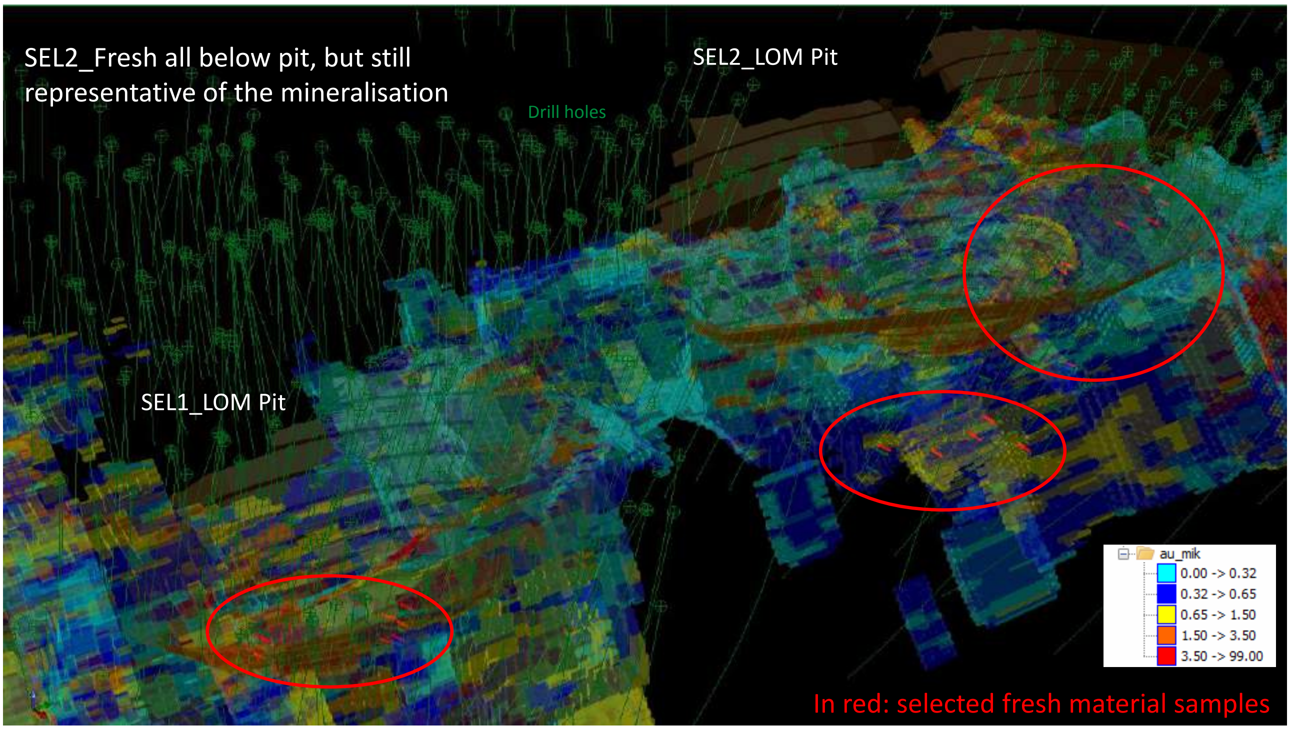


Figure 13.14 Selected samples in relation to SEL LOM pits and mineralised Fresh blocks



The location of fresh and transition samples for BRC2/BRC3 and BRC4 in relation to the LOM pit shells are shown in Figure 13.15 and Figure 13.16 respectively. For sample BRC3_Transition, a portion of the sample was collected from drillholes below the proposed pit but is still considered representative of the mineralisation in this area.

The estimated gold distribution in the BRSCF block model and the sample locations are shown in Figure 13.17 to Figure 13.19.

Figure 13.15 Selected samples for BRC2 and BRC3 Fresh

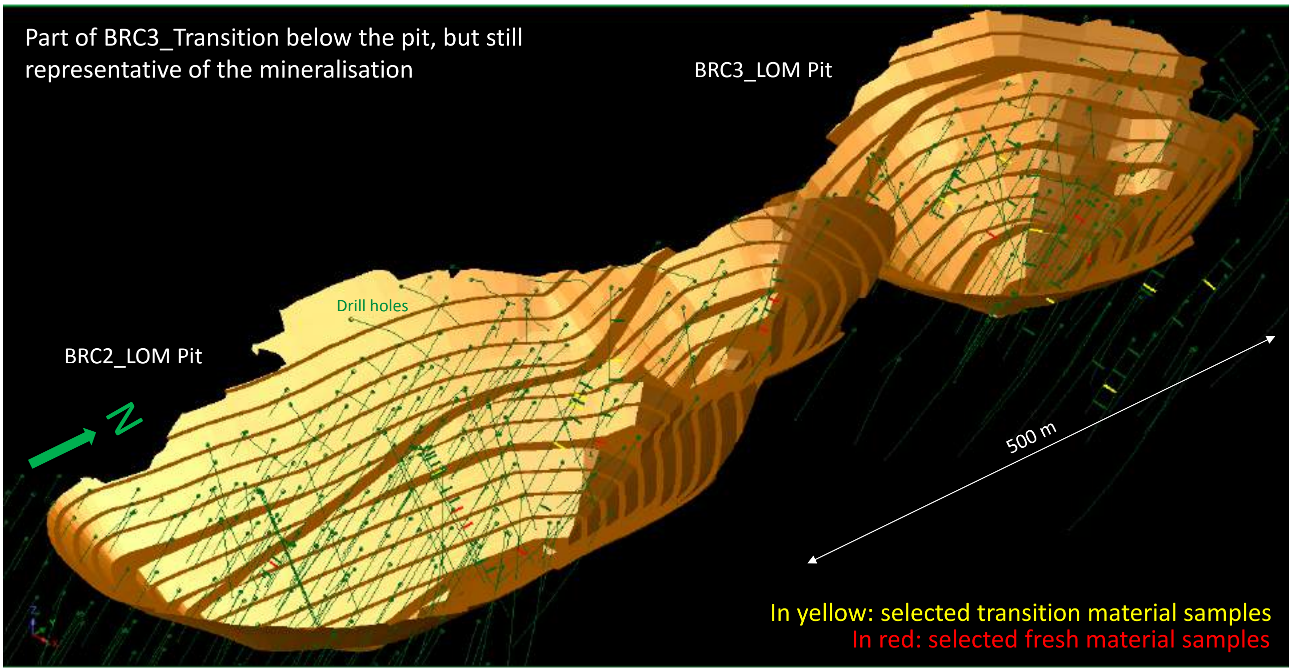


Figure 13.16 Selected samples for BRC4 Fresh

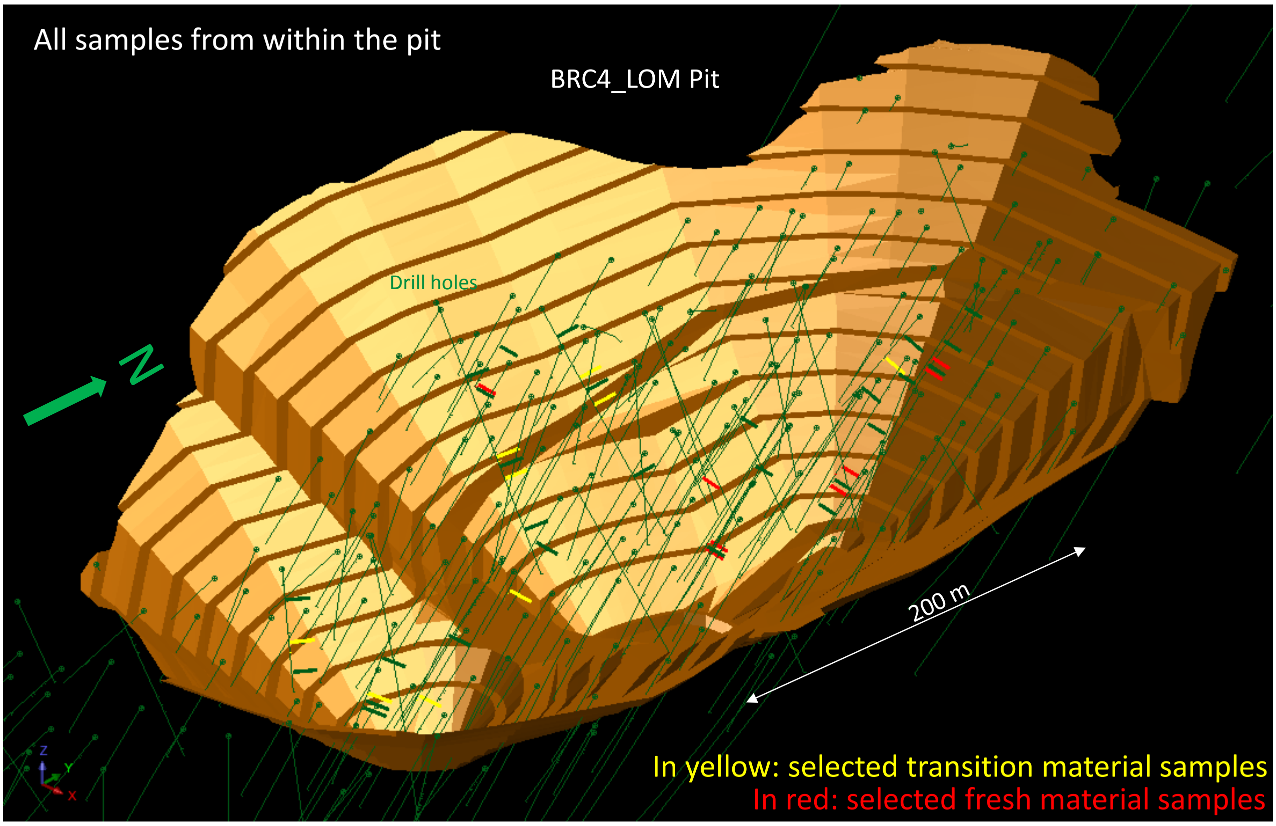


Figure 13.17 BRC2, BRC3 and BRC4 LOM pits and mineralised Fresh blocks

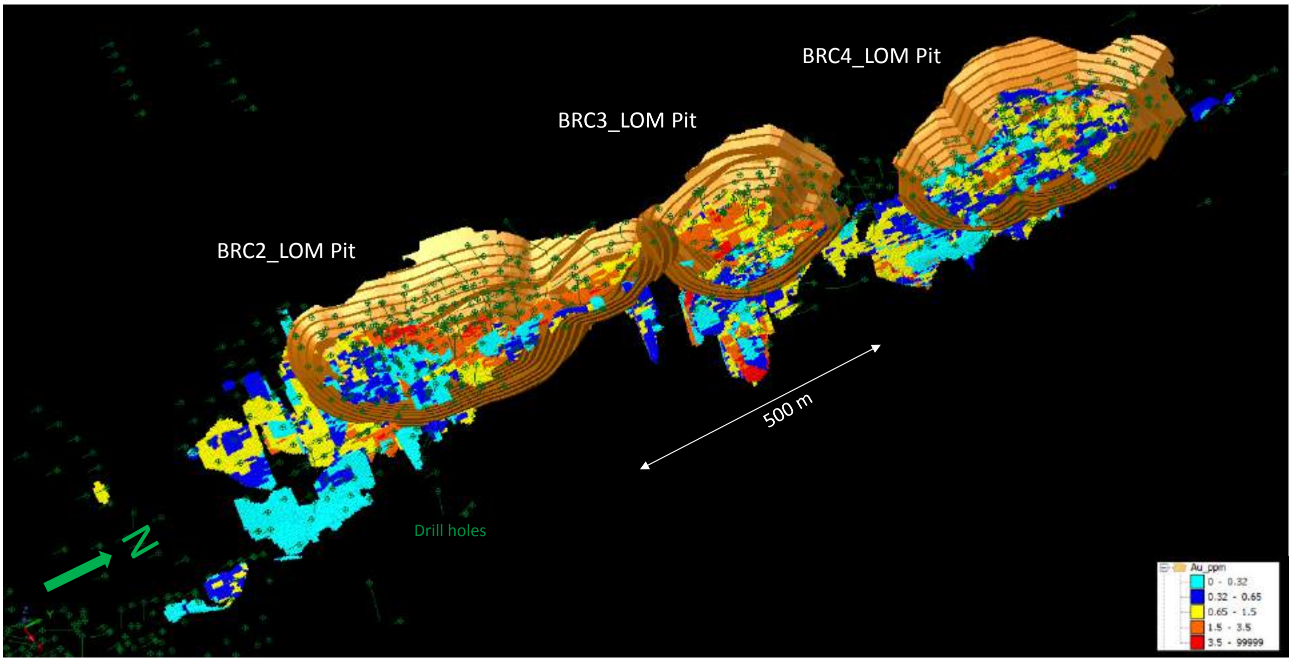


Figure 13.18 Selected samples in relation to BRC2 and BRC3 LOM pits and mineralised Fresh blocks (Fresh samples circled in white)

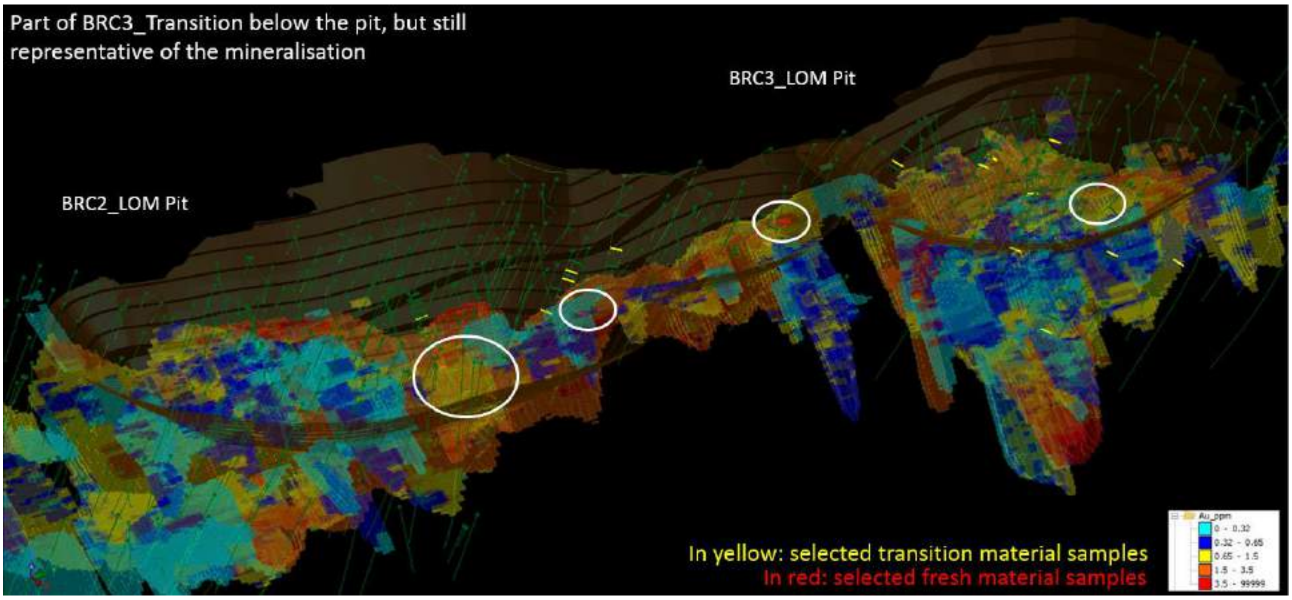
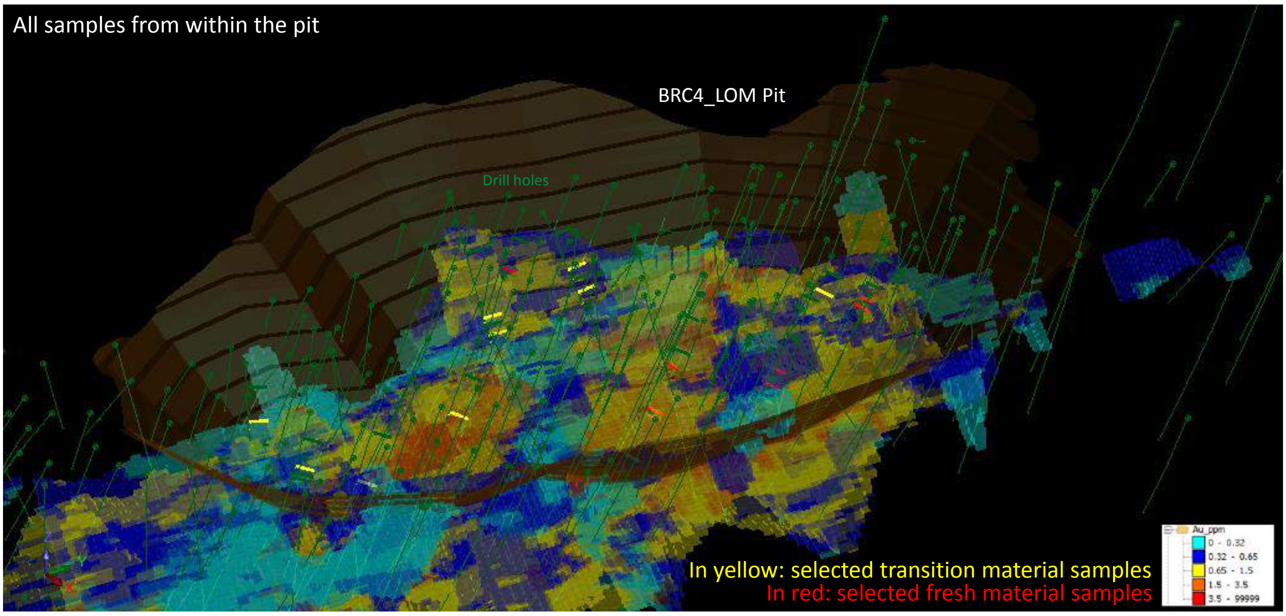


Figure 13.19 Selected samples in relation to BRC4 LOM pits and mineralised Fresh blocks



No fresh material occurs within the BRN1 and BRN2 LOM pits and no transition samples are available at BRN1. Part of sample BRN2_Transition comprises samples from below the proposed pits, but still considered representative of the mineralisation in this area.

The location of the samples in relation to the LOM pit shells, with the estimated gold distribution in the resource block model, are shown in Figure 13.20 and Figure 13.21.

Figure 13.20 BRN1 and BRN2 LOM pits and mineralised Transition blocks

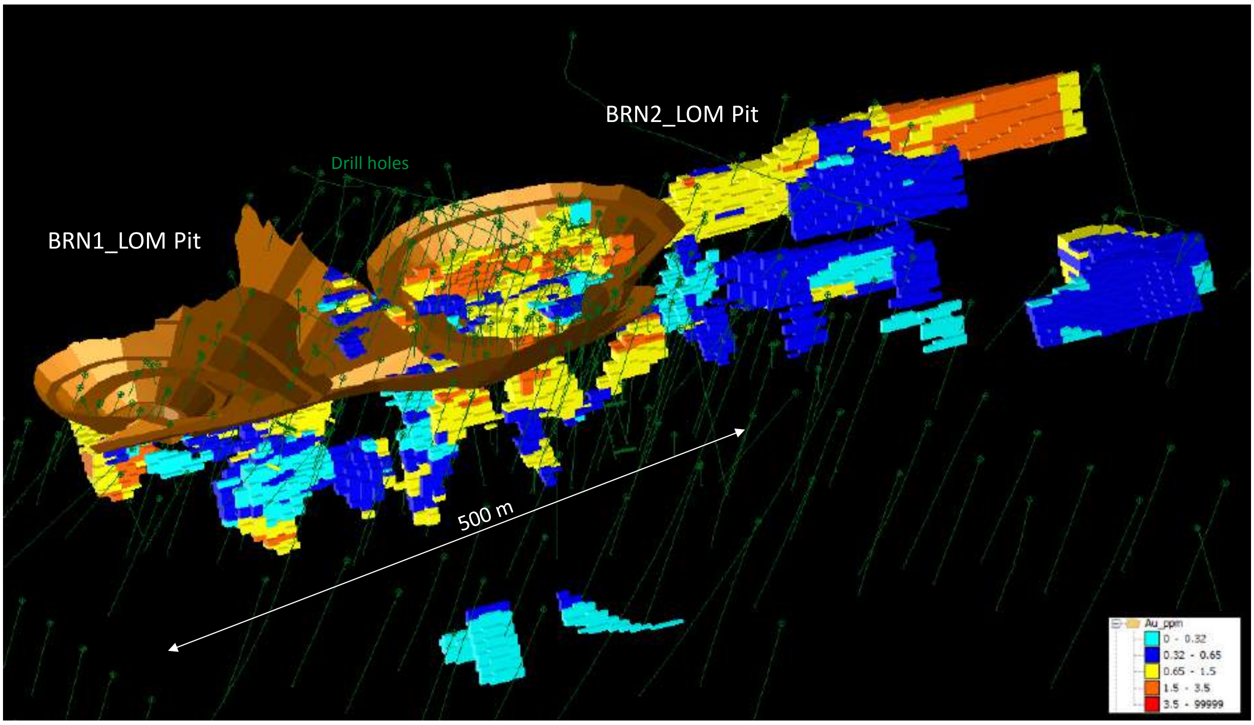
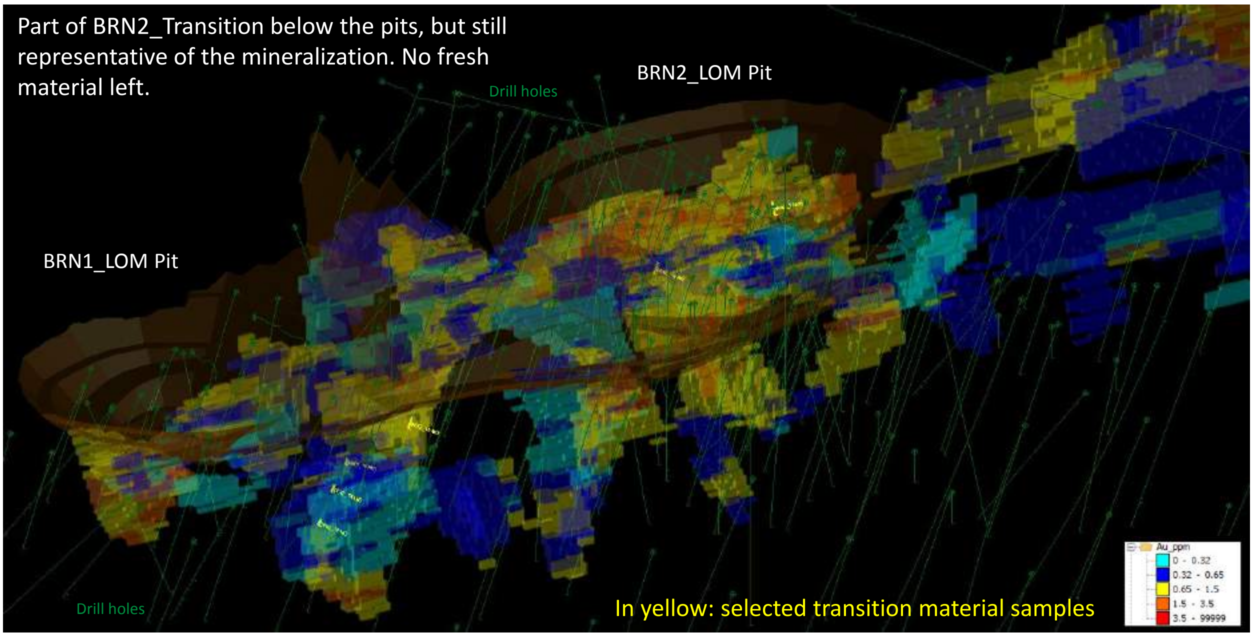


Figure 13.21 Selected samples in relation to BRN1 and BRN2 LOM pits and mineralised Transition



The samples selected for use in the BIOX® variability testwork are considered as representative of the overall mineralisation in each proposed LOM pit. Some samples comprise material taken from drillholes below the proposed pits; however, Monument indicated that these samples come from the same mineralisation environment as within the pit shells.

14 MINERAL RESOURCE ESTIMATE

14.1 Introduction

Mineral Resource estimates for the Selinsing and Buffalo Reef deposits were primarily prepared by Mr J. Graindorge, Principal Consultant, a full-time employee of Snowden. Mr Graindorge is a Qualified Person as defined in NI 43-101. Snowden is independent of Monument. The 2016 resource models have been depleted for mining to end of March 2018 and are constrained within a revised pit shell. Additionally, Monument added sulphur grade estimates to the model, which were not included in the original 2016 models.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The methodology description included in the Snowden 2016 NI 43-101 Technical Report is largely used in this section, updated and amended as required.

The changes to the 2016 resource model were:

- The depletion for mining
- The addition of a sulphur grade estimate
- The use of a larger resource pit shell with some change in the parameters
- The change in the cut-off for transition and sulphide, from 0.7 g/t Au to 0.5 g/t Au.

14.2 Known issues that materially affect Mineral Resources

Snowden is unaware of any issues that materially affect the Mineral Resources in a detrimental sense. These conclusions are based on the following:

- Monument has the relevant mining permits in good standing; the mining approvals in the Felda Block 7 area at Buffalo Reef which were outstanding in 2016, are now fully approved by the relevant authorities, and the right to mine has been secured from the landowner.
- Monument has represented that there are no outstanding legal issues; no legal action, and injunctions pending against the Project.
- Monument has represented that the mineral and surface rights have secure title.
- There is no known marketing, political or taxation issues.
- Monument has represented that the Project has strong local community support.
- Monument has successfully mined the Selinsing deposit for a number of years and has successfully treated oxide and transitional ore. Additionally, oxide ore has been successfully treated from the Buffalo Reef deposits.
- Metallurgical testwork has shown that recovery of gold from the refractory sulphide ore is possible using either a previously proposed bioleach process or the currently proposed BIOX® process. Successful commercial scale treatment of the sulphide ore will depend on successful implementation of the BIOX® process at the Selinsing processing plant.
- There are no known infrastructure issues.

14.3 Previous estimates

Snowden (2016) reported a Mineral Resource for Selinsing and Buffalo Reef, as of June 2016, estimating a total of 9,906 kt of Measured + Indicated Resources grading at 1.52 g/t Au for a total of 485 koz Au, plus 4,373 kt of Inferred Resource grading at 1.98 g/t Au for a total of 279 koz Au.

The 2016 block model forms the basis of the current Selinsing and Buffalo Reef Mineral Resource estimate. Changes to the block model and Mineral Resource reporting include:

- Depletion for mining to March 2018

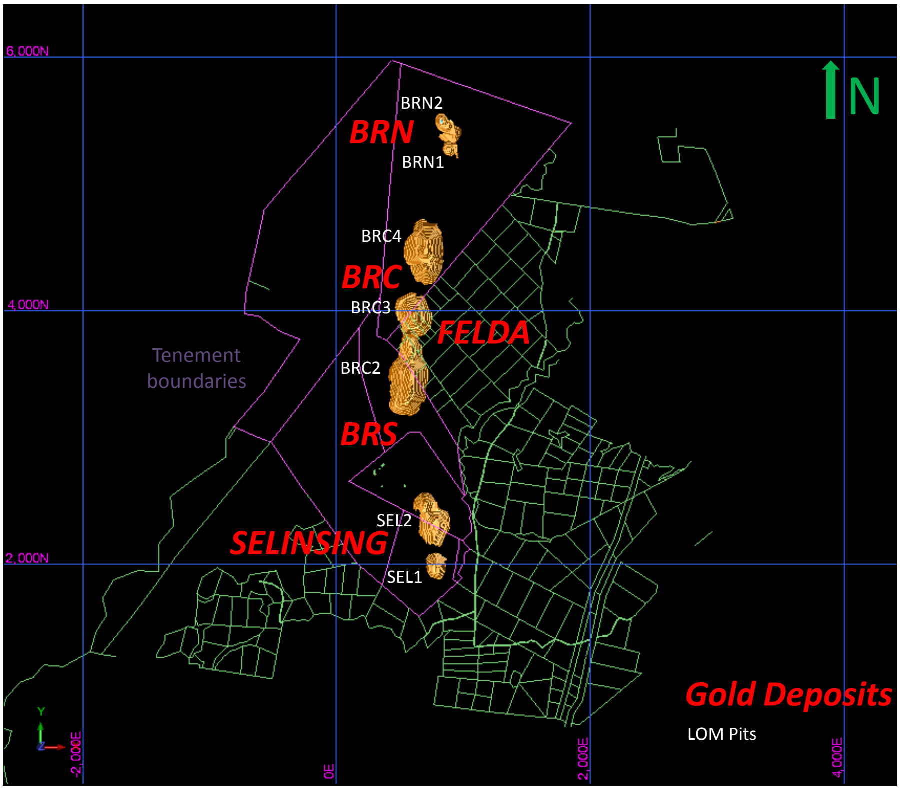
- Updated the pit shell used to constrain the reported Mineral Resource
- Addition of sulphur grade estimates for Selinsing and Buffalo Reef
- Changed the cut-off grade for reporting of the transition and sulphide mineralisation from 0.7 g/t Au to 0.5 g/t Au.

A block estimation was completed in house in 2017 for element sulphur (S%), as an additional field for the Selinsing and Buffalo Reef North resource models completed for the NI 43-101 technical report issued in December 2016 (Snowden, 2016).

Sulphur grade is an important metallurgical consideration for the BIOX® gold metallurgical process in the Sulphide Project, in particular for fresh mineralised material within the seven designed pit shells comprising the Selinsing LOM production schedule.

A plan view of the seven LOM pits in relation to the tenements and gold deposits is presented in Figure 14.1

Figure 14.1 Location of Selinsing Operation gold deposits and designed LOM pits



14.4 Selinsing Mineral Resource estimate

The following section has been taken from Snowden (2016) and updated to reflect the changes to the block model and reported Mineral Resource.

14.4.1 Geological interpretation

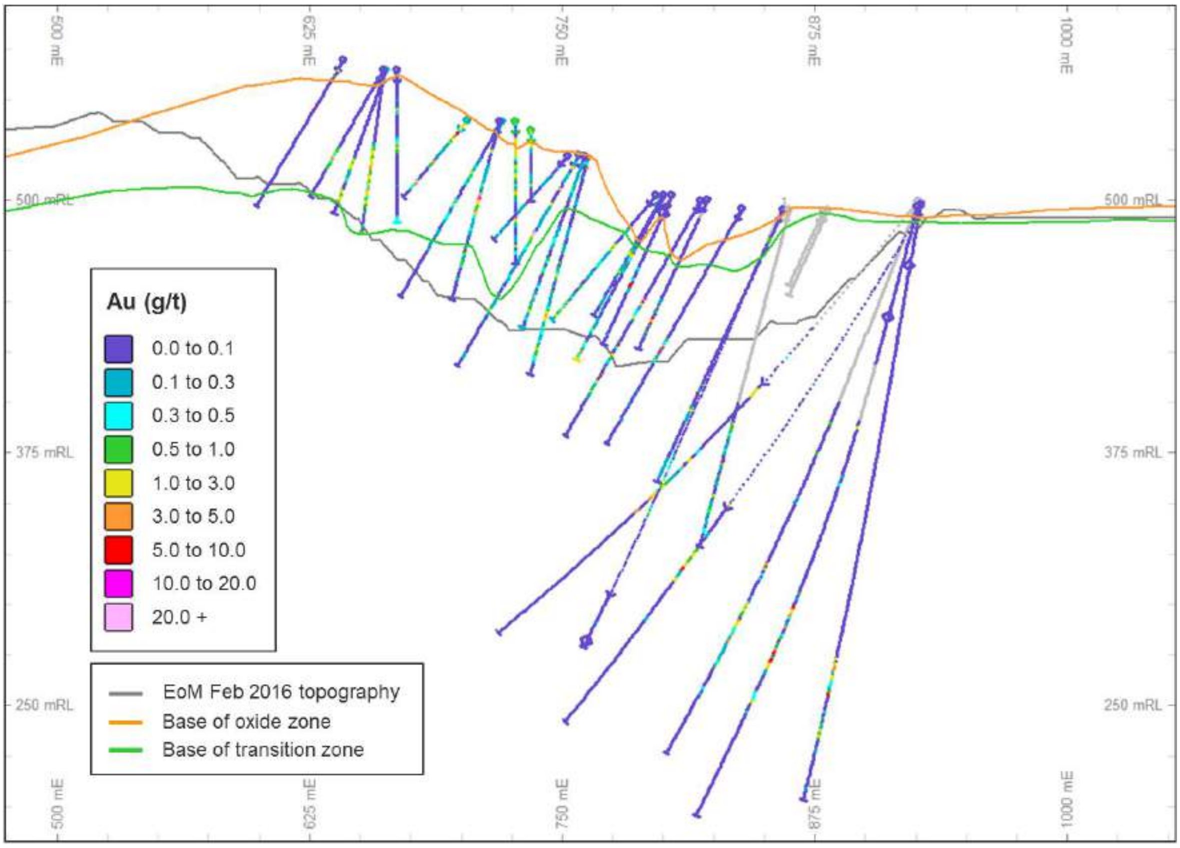
Oxidation

Monument interpreted surfaces for the base of complete oxidation (BOCO) and top of fresh rock (TOFR) based on the logging of the oxidation state of the drillholes. Monument noted that the logging of historical holes is incomplete in some cases and validation with actual mining shows that the historical logging tends to underestimate the oxidation (i.e. slightly biased towards fresh). As such, holes drilled prior to July 2015 were re-logged to ensure consistency between the logging of the older drilling and the more recent drilling.

The interpretation preferentially used diamond drillholes, with the RC holes given a secondary priority. Logged vein intensity was also used to assist with the interpretation of the oxidation surfaces as the weathering tends to be influenced by the presence of mineralised quartz veins and the associated alteration.

The surfaces provided by Monument were expanded by Snowden to cover the entire model area. An example west-east cross section showing the oxidation surfaces is shown in Figure 14.2.

Figure 14.2 Example west-east section (1920 mN) showing interpretation of oxidation surfaces



Source: Snowden

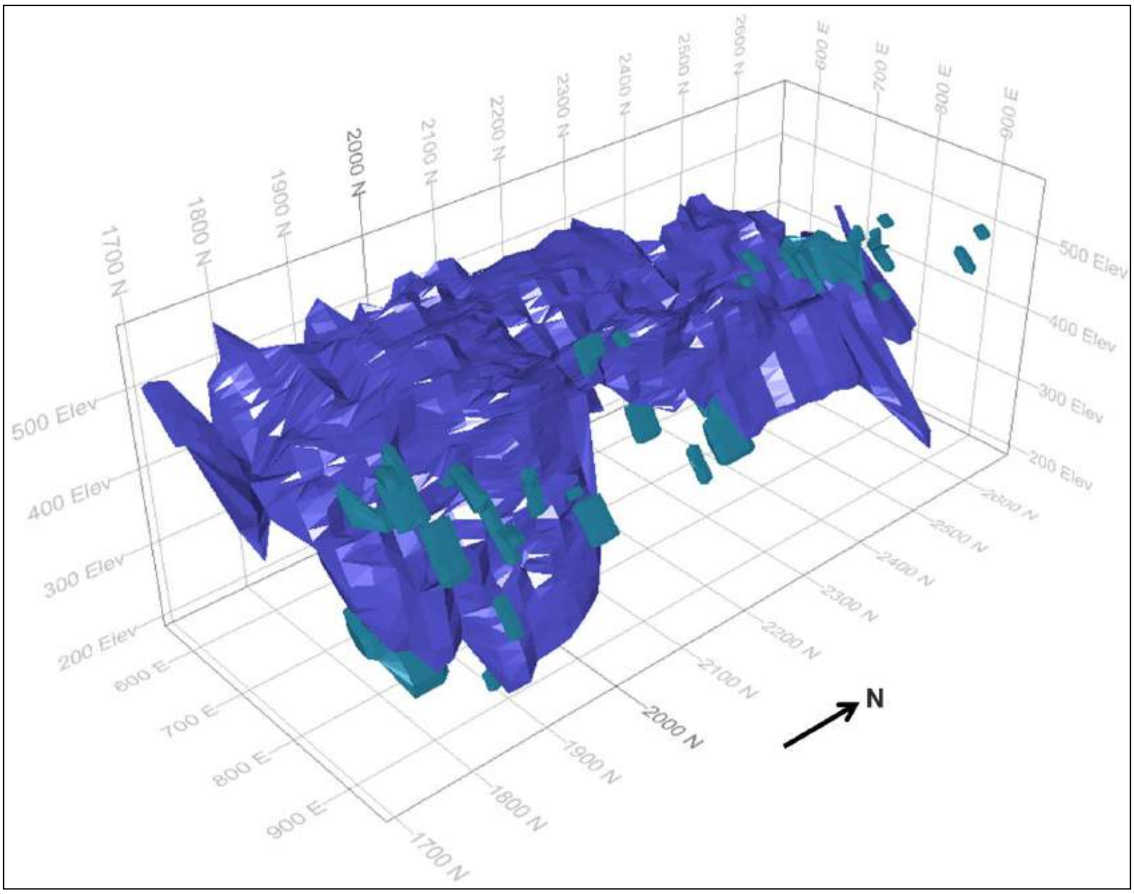
Mineralisation

The sectional interpretation of the Selinsing gold mineralisation was completed by Monument based on east-west sections with a section spacing of 20 m. Logging information from RC and diamond drillholes, including vein intensity, alteration and structural information, along with the gold grades, were utilised by Monument to interpret the gold mineralisation. A nominal threshold of 0.15 g/t Au has been used to guide the interpretation of the gold mineralisation. Monument noted that elevated gold grades are typically associated with zones of higher intensity quartz veining; however, zones of veining also occur in unmineralised rocks. The zones of quartz veins are normally associated with higher hydrothermal alteration with sulphide minerals including pyrite, chalcopyrite, arsenopyrite and stibnite.

The interpreted mineralisation at Selinsing comprises a single main mineralised zone which is typically 30–50 m thick (locally up to 80 m thick) in the southern half and narrows in the northern half to 10–20 m thick, along with numerous minor mineralised structures. Compared to Buffalo Reef, where the mineralisation is more continuous and tabular, the Selinsing mineralisation, whilst broadly continuous at the deposit scale, is locally discontinuous and very mixed, especially within the southern half of the Selinsing area.

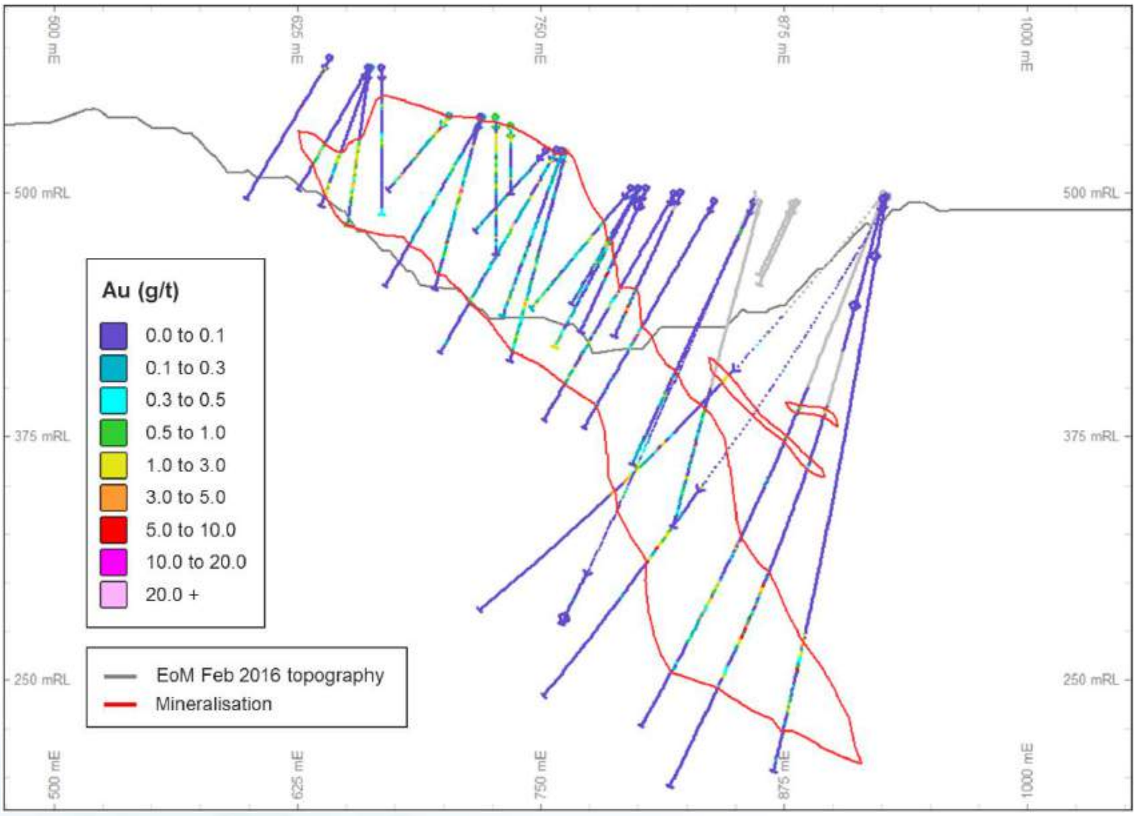
An oblique view of the Selinsing mineralised zones is shown in Figure 14.3 and an example cross section is shown in Figure 14.4.

Figure 14.3 Oblique view (looking northwest) of the Selinsing mineralisation interpretation



Source: Snowden

Figure 14.4 Example west-east section (1920 mN) showing mineralisation interpretation

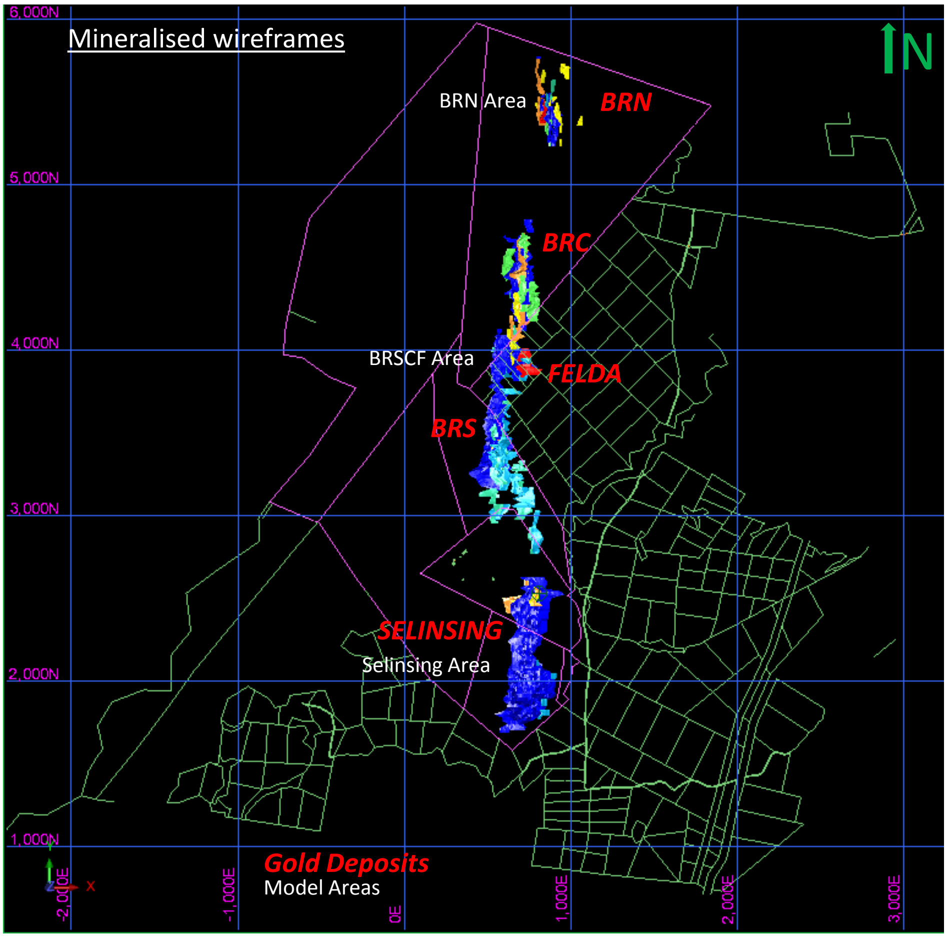


Source: Snowden

The interpreted wireframes for mineralisation (in Selinsing and Buffalo Reef areas) are shown in the Figure 14.5.

A minimum of three sample intervals (generally 3 m for RC holes and similar for diamond holes) was used for the interpreted mineralised intersections. External dilution was only included for isolated intervals with continuity from neighbouring intervals, either down-dip in the same section or along strike in adjacent sections. Extrapolation of the interpretation down-dip and along strike was limited to approximately half the drillhole spacing.

Figure 14.5 Plan view of the mineralised wireframes interpreted



14.4.2 Drillhole data analysis

For the Selinsing model completed by Snowden in 2016, the drillhole dataset was limited to south of 2900 mN. Table 14.1 summarises the drilling data subset used for the Selinsing resource model. A very small amount of air-core (AC) drilling was included; however, samples from the AC drilling were only used for the estimates in the waste blocks. Only RC and diamond drilling data were used for the grade estimates in the mineralised domains (Figure 14.6).

The cut-off date for the data used in the Selinsing model was for drilling assays received up to 24 February 2016. An additional nine diamond core holes, for 648 m of drilling, completed in 2016 were not included in the 2016 Selinsing resource model (Snowden 2016). This additional data was included by Monument in the dataset used for the estimation of sulphur; however, the additional data is not considered to be material with respect to the gold grade estimates, which remain unchanged. Table 14.1 and Table 14.2 show a breakdown of the metres by drilling type.

Table 14.1 Drilling data used for Selinsing model (south of 2900 mN), Snowden 2016

Drilling method	Total length (m)	Proportion
Reverse circulation (RC)	42,211.5	55.5%
Diamond drilling (DD)	28,297.2	37.2%
RC with diamond tail	4,095.2	5.4%
Air-core (AC) *	1,412.7	1.9%
Total	**76,016.6**	**100.0%**

** AC includes some minor auger drillholes and banka holes; the majority of which target the old tailings and were excluded from the resource estimation*

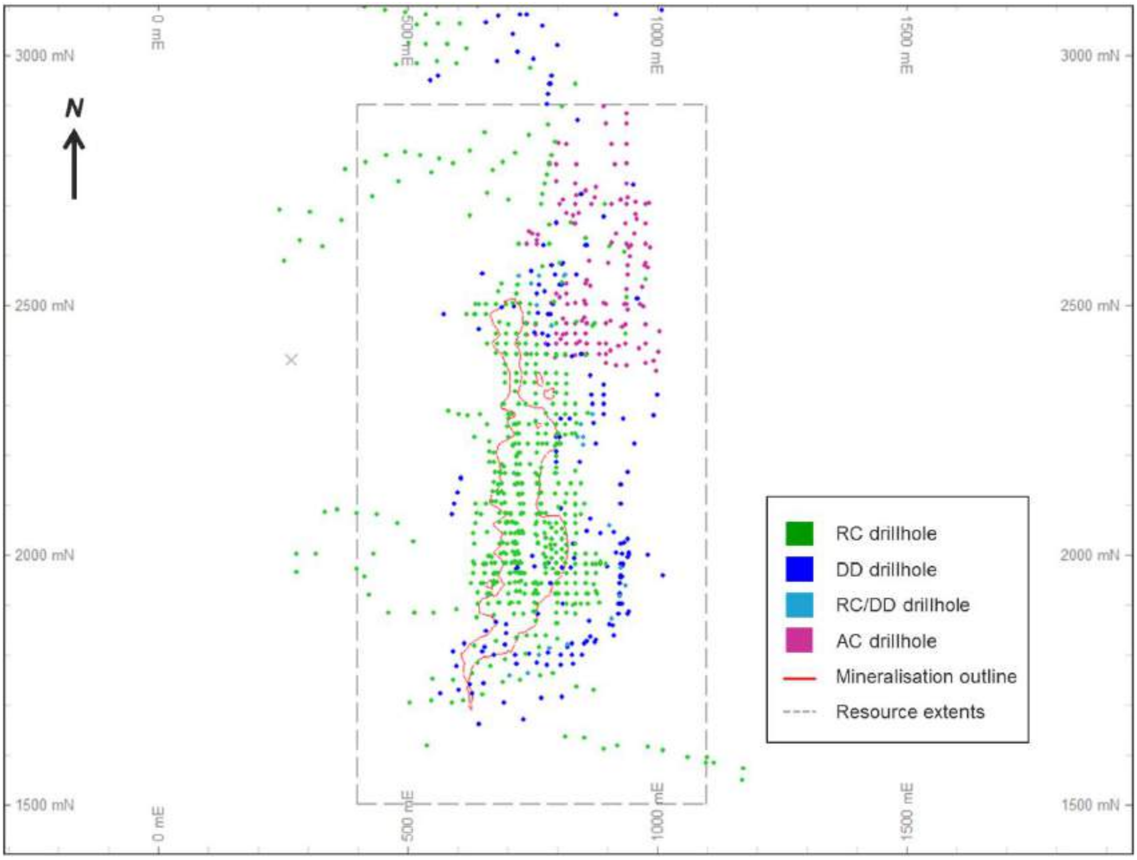
Table 14.2 Summary of Selinsing drilling after end February 2016*

Hole type	No. of holes	Length (m)
Diamond (DD)	9	648
Total	**9**	**648**

** For assays received after 24 February 2016*

The drilling at Selinsing is based on a section spacing of approximately 20 m with drilling typically completed at 20 m intervals on section (i.e. a drill spacing of 20 mN x 20 mE). In the central portion of Selinsing, the drill spacing can be locally down to 10 m. The drill spacing in the deeper portions of the mineralisation, as well as the northern and southern extremities, is typically wider spaced, averaging 40 mN x 40 mE to 60 mN x 40 mE.

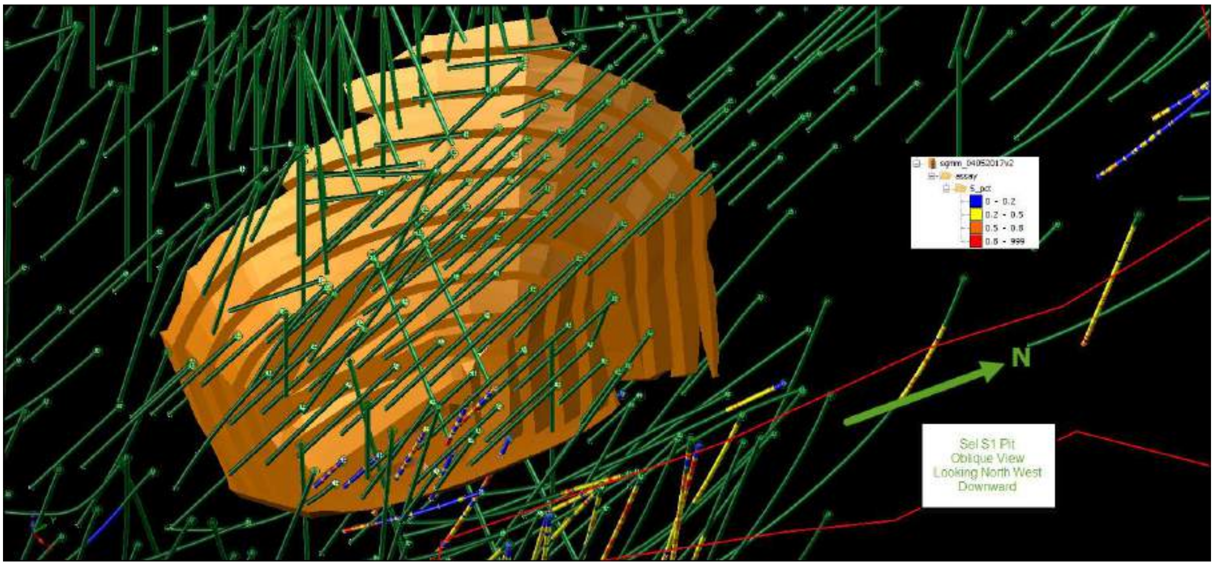
Figure 14.6 Collar location plan for Selinsing resource model area, Snowden 2016



Sulphur data coverage

The available sulphur assay data, as validated and incorporated in the database, come entirely from Monument drilling campaigns undertaken since 2014. Sulphur has not been assayed consistently for the Selinsing deposit, resulting in an irregular sample assay coverage (Figure 14.7).

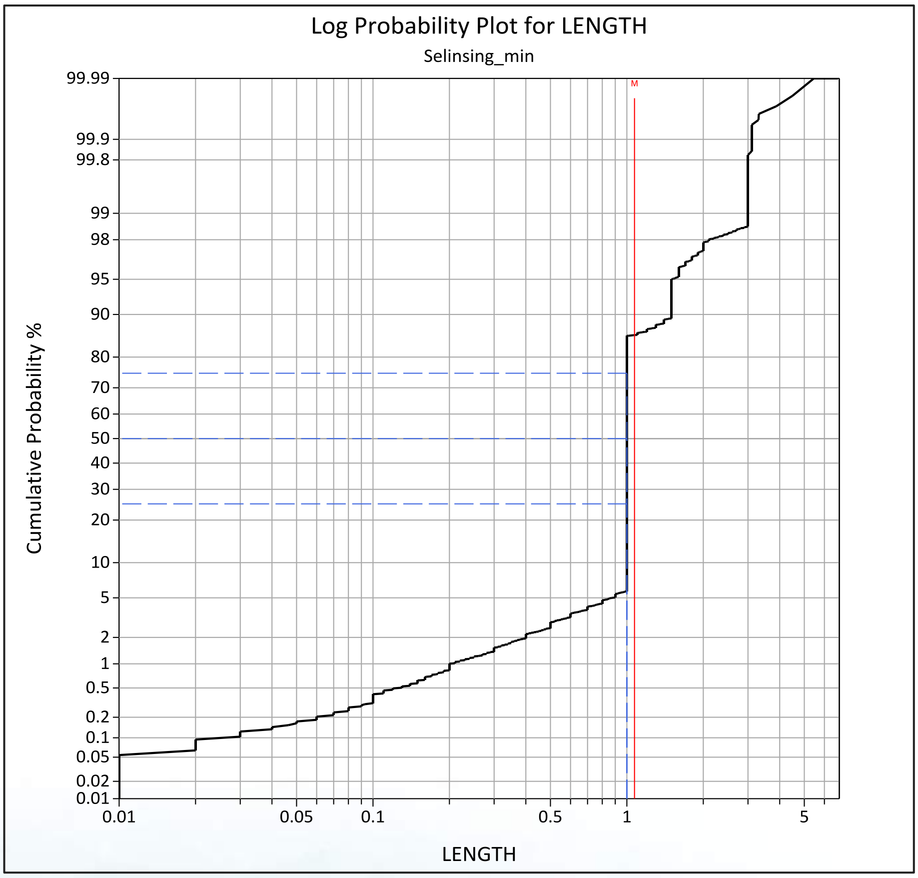
Figure 14.7 Oblique view of Selinsing S1 LOM pit showing drilling coverage and limited sulphur assays (green drillhole trace: not assayed for sulphur)



Sample compositing

The drillhole data was composited downhole prior to running the estimation process using a 1.5 m compositing interval for Au and 3 m for S, to minimise any bias due to sample length. A log probability plot of the raw drillhole sample lengths within the Selinsing mineralised domains is presented in Figure 14.8. The compositing interval of 1.5 m (and 3 m for S) was chosen, rather than 1 m, to avoid excessive sample splitting as a significant proportion of the samples have sample lengths of 1.5 m.

Figure 14.8 Log probability plot of sample lengths within Selinsing mineralisation



Source: Snowden

The compositing was run within the attribute fields to ensure that no composite intervals crossed any lithological or grade boundaries. To allow for uneven sample lengths within each of the domains, the composite process was run using the variable sample length method. This adjusts the composite intervals, where necessary, to ensure all samples are included in the composite file (i.e. no residuals) while keeping the composite interval as close to the desired interval as possible.

The compositing process was checked by:

- Comparing the lists of attribute field values in the raw and composite files; these should match
- Comparing the sample length statistics in the raw and composite files; the two total length values should match, and the mean composite interval should be 1.5 m.

No discrepancies were identified during the compositing process.

Statistical analysis

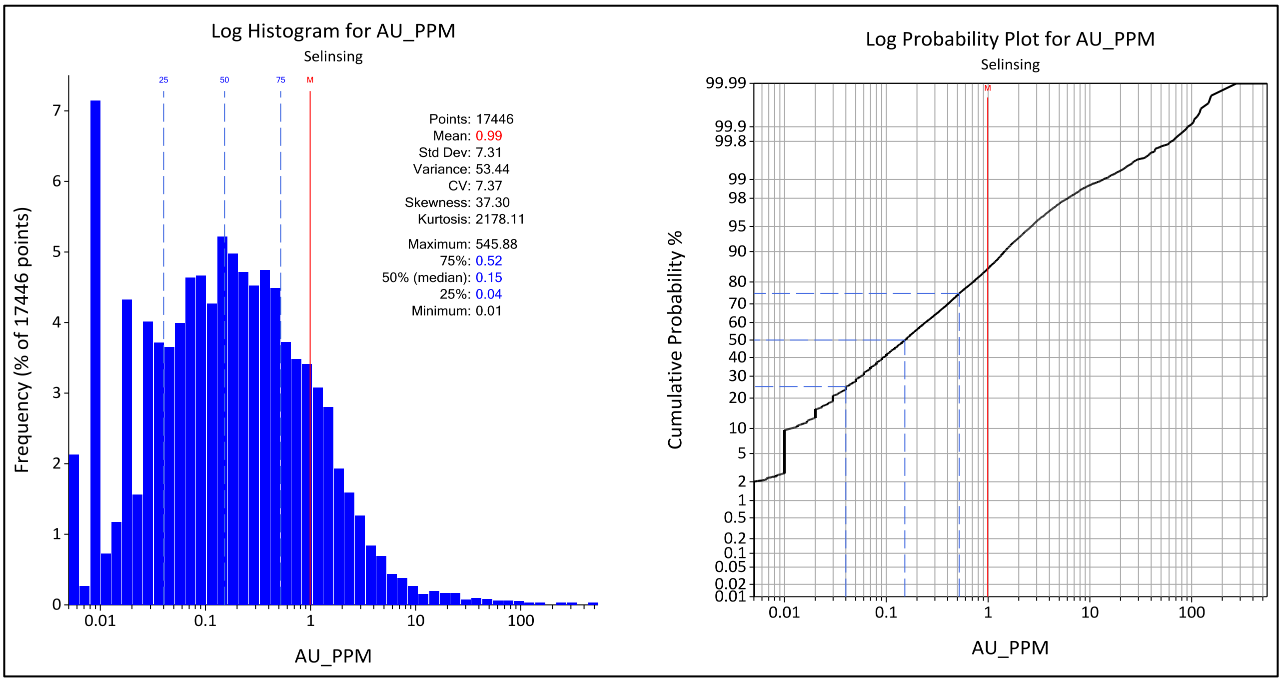
Statistical analysis was carried out on the composited dataset for gold grades.

Summary statistics of composites for gold for the combined mineralised domain (i.e. MINZONE=3000) are presented in Table 14.3 and log histograms and log probability plots are presented in Figure 14.9. The statistics show that the Selinsing mineralisation has a strongly positively skewed gold grade distribution with a very high coefficient of variation ("CV" = ratio of the standard deviation to the mean).

Table 14.3 Summary gold statistics for combined Selinsing mineralisation composites

Statistic		MINZONE 3000 Au (g/t)
Samples		17,446
Minimum		0.01
Maximum		545.88
Mean		0.99
Standard deviation		7.31
CV		7.37
Percentiles:	10%	0.01
	20%	0.03
	30%	0.06
	40%	0.09
	50%	0.15
	60%	0.24
	70%	0.40
	80%	0.72
	90%	1.45
	95%	2.66
	97.5%	4.81
	99%	13.56

Figure 14.9 Log histogram and log probability plot for gold for combined Selinsing mineralised domains



Source: Snowden

Due to the extremely skewed nature of the gold grades (CV>>2) in both the mineralised domain and the waste domain, along with evidence of mixed populations, Snowden elected to use multiple indicator kriging (MIK) to estimate the block gold grades.

Summary statistics of composites for sulphur within the fresh mineralised domain are shown in Table 14.4.

Table 14.4 Summary sulphur statistics for Selinsing mineralised composites

Statistic		MINZONE 3000 S (%) Fresh
Samples		185
Minimum		0.001
Maximum		1.876
Mean		0.443
Standard deviation		0.336
CV		0.292
Percentiles:	10%	0.084
	20%	0.135
	30%	0.187
	40%	0.260
	50%	0.336
	60%	0.415
	70%	0.537
	80%	0.691
	90%	1.058
	95%	1.258
	97.5%	1.344

The correlation between sulphur and gold is low to very low within the oxide zone, increasing with depth for the transition zone and low to moderate (correlation coefficient = 0.17) in the fresh, sulphide zone. However, given the low number and poor coverage of sulphur assays, the correlations are noted to be somewhat irregular and unclear.

Top-cuts

No top-cuts were applied to the gold assay data for the Selinsing gold estimate due to the application of MIK. No top-cuts were applied for sulphur estimation due to the low CV.

14.4.3 Variography

Indicator variograms for gold were generated for the combined mineralised domain (MINZONE 3000). The indicator variogram models from the mineralised domain were applied to the waste domain, with the threshold grades adjusted as per Table 14.5. For the waste domain, only eight thresholds were used, rather than 12 used for the mineralised domain, as several of the percentiles have the same grade value due to a significant proportion of samples being close to the detection limit.

Table 14.5 Indicator variogram thresholds and variogram model mapping

Threshold percentile	Mineralised domain MINZONE=3000		Waste domain MINZONE=0	
	Au (g/t)	Variogram reference no.	Au (g/t)	Variogram reference no.
10%	0.01	1	-	-
20%	0.03	2	0.005	2
30%	0.06	3	-	-
40%	0.09	4	-	-
50%	0.15	5	0.01	5
60%	0.24	6	-	-
70%	0.40	7	0.013	7
80%	0.72	8	0.02	8
90%	1.45	9	0.04	9
95%	2.66	10	0.07	10
97.5%	4.81	11	0.10	11
99%	13.56	12	0.15	12

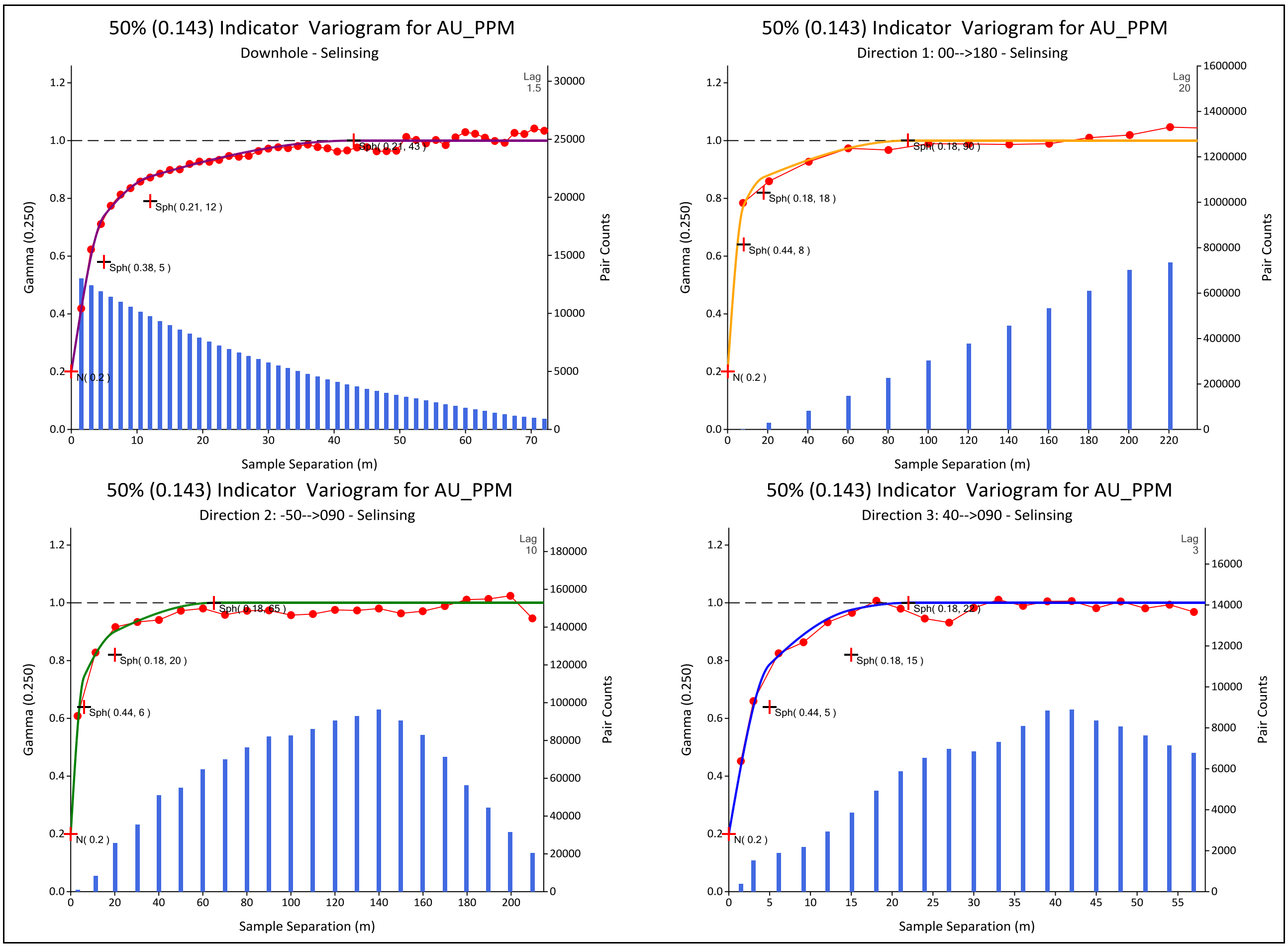
Variograms for the mineralised domain were modelled based on the following general approach:

- Due to the steepening of the dip at depth, the drillhole composites were filtered (using coordinate fields) to provide an area of relatively even dip and strike
- All variograms were standardised to a sill of one
- The nugget effect was modelled from the true downhole variogram
- Variograms were modelled using three nested, spherical structures
- The variograms were evaluated using indicator variograms for a total of 12 grade thresholds
- Variograms were only modelled for the mineralised domain.

The maximum and intermediate directions of continuity (directions 1 and 2 respectively) were aligned with the overall strike and down dip directions respectively. The minor direction of continuity (direction 3) was aligned in the true thickness direction. The variogram directions did not change for different thresholds (i.e. no rotating anisotropy).

The indicator variogram models for the mineralised domain are summarised in Table 14.7 and the median indicator (i.e. 50th percentile) variogram model is shown in Figure 14.10.

Figure 14.10 Median (50%) indicator variogram model for mineralised domain



Source: Snowden

Upper and lower tail modelling

The upper and lower tails of the gold grade distributions, above the 99th percentile and below the 10th percentile, were modelled using a hyperbolic and power model respectively. Grades were modelled up to the maximum composite grade for the domains, as per Table 14.4. The upper and lower tail model parameters are summarised in Table 14.6 for each domain.

Table 14.6 Distribution tail modelling – Selinsing

Domain (MINZONE)	Tail	Model type	Model parameter
0	Lower	Power	1.00
0	Upper	Hyperbolic	1.20
3000	Lower	Power	0.80
3000	Upper	Hyperbolic	1.15

Table 14.7 Indicator variogram models for gold for the Selinsing mineralised domain (MINZONE 3000)

Threshold	Variogram ref. no.	Directions			Nugget	1st spherical variogram structure				2nd spherical variogram structure				3rd spherical variogram structure			
		Dir1	Dir2	Dir3		Sill	Range Dir1	Range Dir2	Range Dir3	Sill	Range Dir1	Range Dir2	Range Dir3	Sill	Range Dir1	Range Dir2	Range Dir3
10%	1	00→180	-50→090	40→090	0.16	0.44	10	6	5	0.20	150	50	25	0.20	170	140	50
20%	2	00→180	-50→090	40→090	0.16	0.44	10	6	5	0.20	150	50	25	0.20	170	140	50
30%	3	00→180	-50→090	40→090	0.18	0.44	10	6	5	0.20	50	25	20	0.18	140	125	50
40%	4	00→180	-50→090	40→090	0.18	0.44	10	6	5	0.20	36	20	15	0.18	138	95	30
50%	5	00→180	-50→090	40→090	0.20	0.44	8	6	5	0.18	18	20	15	0.18	90	65	22
60%	6	00→180	-50→090	40→090	0.23	0.44	8	6	5	0.13	18	17	14	0.20	75	55	16
70%	7	00→180	-50→090	40→090	0.30	0.37	8	5	5	0.13	18	13	10	0.20	55	30	12
80%	8	00→180	-50→090	40→090	0.30	0.37	8	5	5	0.13	18	9	8	0.20	40	17	9
90%	9	00→180	-50→090	40→090	0.33	0.34	8	5	4	0.13	18	9	5	0.20	20	13	6
95%	10	00→180	-50→090	40→090	0.35	0.32	8	5	2	0.13	15	9	3	0.20	17	13	4
97.5%	11	00→180	-50→090	40→090	0.35	0.32	5	5	1	0.13	9	9	2	0.20	13	13	3
99%	12	00→180	-50→090	40→090	0.35	0.32	5	5	1	0.13	9	9	2	0.20	11	11	3

14.4.4 Block model and grade estimation

Kriging neighbourhood analysis

A kriging neighbourhood analysis (KNA) was performed by Snowden (2016) using Snowden Supervisor software to optimise and validate various kriging parameters, based on the median (50th percentile) indicator variogram for gold. Snowden used the results of the KNA to verify the choice of parent block size, number of informing samples and the search ellipse radii. Based on the KNA results, along with consideration of the geometry of the mineralisation and the current open-pit bench height at Selinsing of 2.5 m, the following parameters were selected for the Selinsing model:

- Parent block size of 10 mE x 20 mN x 2.5 mRL
- A minimum of eight samples and maximum of 25 samples for the initial search pass
- Search ellipse radii of 7.5 m in the thickness direction by 30 m along strike by 15 m down dip for the initial search pass.

Volume model construction

The block model extents for the Selinsing deposit, along with parent and sub-cell sizes are listed in Table 14.8. The Selinsing block model was originally constructed by Snowden (2016) using Datamine Studio 3 software. Subsequently, Monument converted the Selinsing model from Datamine to Surpac format. The converted Surpac model was validated by Monument to ensure that there was no impact due to the conversion.

Table 14.8 Selinsing block model settings (local mine grid)

Model setting	Value
Origin – X	400
Origin – Y	1,500
Origin – Z	150
Maximum – X	1,100
Maximum – Y	2,900
Maximum – Z	600
Parent cell size – X	10
Parent cell size – Y	20
Parent cell size – Z	2.5
Minimum cell size – X	2.5
Minimum cell size – Y	5
Minimum cell size – Z	1.25
Rotation	None

Block model coding

The block model was coded based on wireframe surfaces and/or solids of the gold mineralisation and oxidation zones, along with the June 2016 end of month topographical surface and a solid defining the extent of old tailings material.

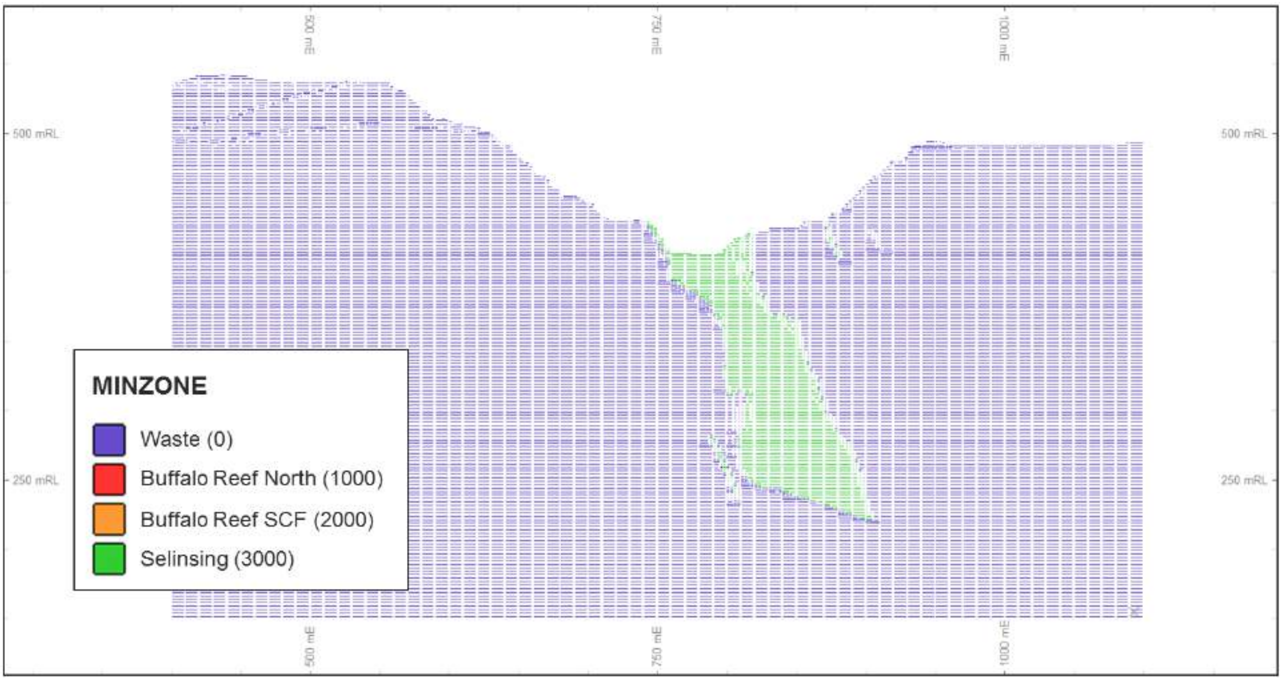
Mineralisation

The gold mineralisation was coded using a field called MINZONE, with individual lodes coded using a field called LODE. The LODE numbers are based on the wireframe numbers. Field codes are defined in Table 14.9 and an example west-east cross section showing the MINZONE coding is shown in Figure 14.11.

Table 14.9 MINZONE and LODE field coding

Field	Value	Description
MINZONE	0	Waste
	3000	Mineralisation (Selinsing)
LODE	0	Waste
	301	Main mineralised zone
	302 to 306, 399	Minor mineralised zones
	999	Old tailings material

Figure 14.11 Example west-east section (1950 mN) showing block model MINZONE coding



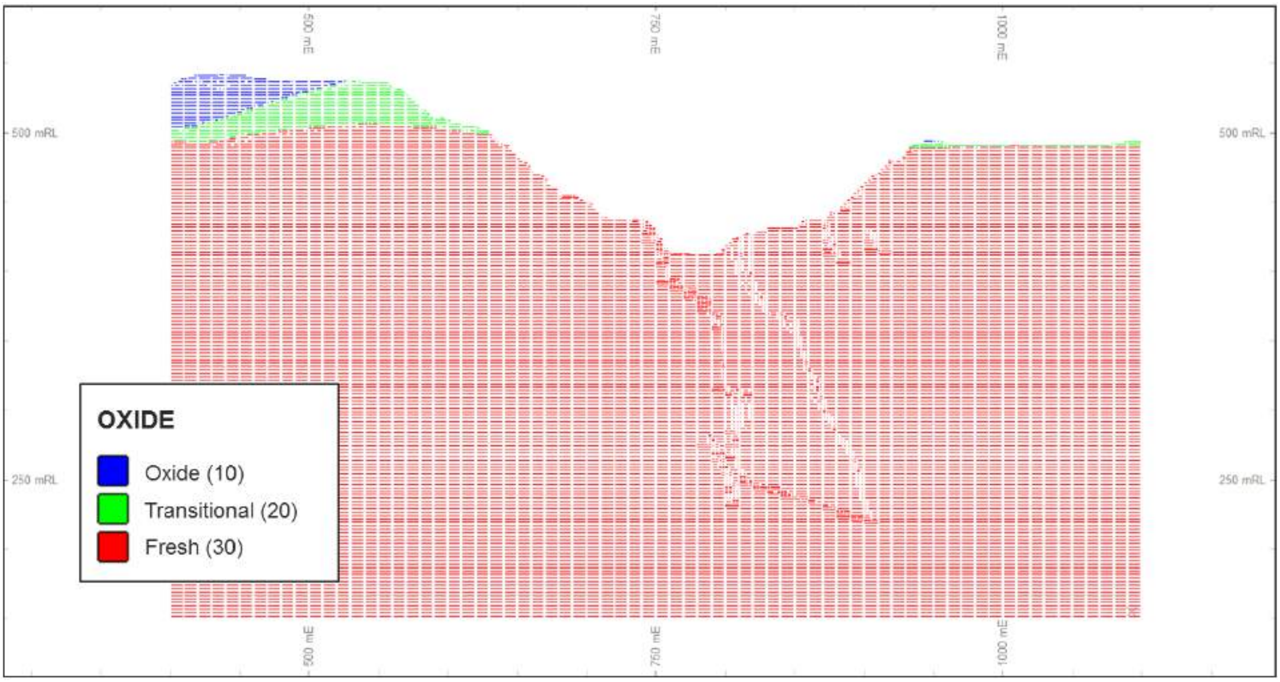
Oxidation

The oxidation zones were coded using a field called OXIDE. Field codes are defined in Table 14.10 and an example west-east cross section showing the OXIDE coding is shown in Figure 14.12.

Table 14.10 OXIDE field coding

Field	Value	Description
OXIDE	10	Oxide zone
	20	Transitional zone
	30	Fresh (sulphide) zone

Figure 14.12 Example west-east section (1950 mN) showing block model OXIDE coding



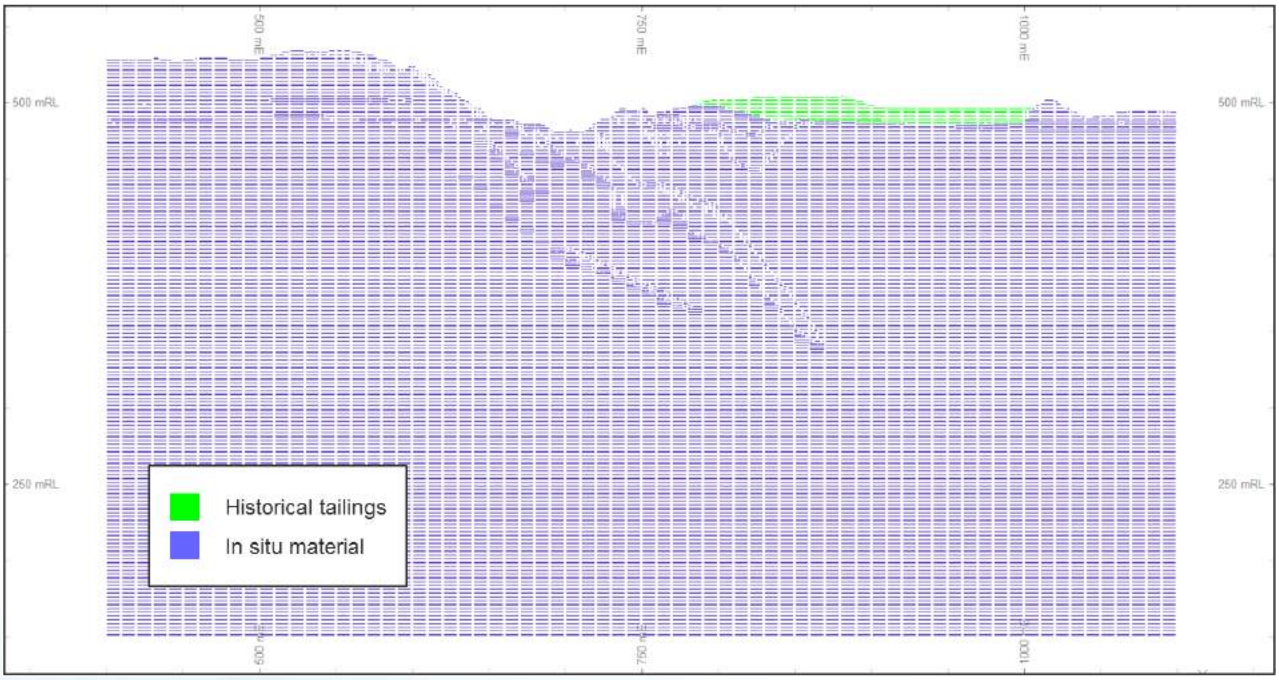
Tailings

The extent of the Old Tailings material was coded using a field called TAILS, based on a wireframe solid provided by Monument. Field codes are defined in Table 14.11 and an example west-east cross section showing the TAILS coding is shown in Figure 14.13.

Table 14.11 TAILS field coding

Field	Value	Description
TAILS	0	In situ (i.e. outside the tailings wireframe solid)
	1	Old Tailings material

Figure 14.13 Example west-east section (2400 mN) showing block model TAILS coding



Grade estimation methodology – Au (g/t)

Due to the strongly skewed nature of the gold grades (CV>>2) in both the mineralised domain and the waste domain, Snowden elected to use MIK to estimate the block gold grades. The MIK estimate was compiled using a total of 12 grade thresholds, based on the population deciles (10%, 20%,…, 80%, 90%) with additional thresholds at the 95%, 97.5% and 99% included to model the higher grade portion of the distributions.

Datamine Studio 3 software was used to estimate the gold grade of the MINZONE domains using MIK using the gold grade field in the drillhole file. The results were written to a field called AU_MIK. The MINZONE field was used to constrain the MIK gold grade estimation with hard boundaries between all domains. All estimates were parent cell estimates. To ensure that the grade estimates were not unduly influenced by samples from within the current pit, the input sample data was restricted to 10 m above the February 2016 pit floor.

Due to the variable dip of the Selinsing mineralisation, dynamic anisotropy was used to locally adjust the orientation of the search ellipse and variogram models. The mineralisation wireframes were used to create a point file where each point relates to a triangle centroid and contains the true dip and true dip direction of the wireframe triangle. All points related to the edges of the wireframes were manually removed to avoid anomalies in these areas. This point file was then used to estimate the local true dip and dip direction into the block model for each block. The estimates of true dip and dip direction were subsequently used to locally adjust the variogram and search orientations during the grade estimation.

The POSTIK process in the GSLIB suite of software was used for post-processing the MIK output from Datamine to enable order relation corrections to be applied and to allow the skewed tails of the gold grade populations to be modelled and used as part of the estimation process. The upper tail was modelled with a hyperbolic function between the last indicator cut-off and the maximum composite grade, while the lower tail was modelled with a power function below the first indicator cut-off (Table 14.6). The final MIK product was an e-type estimate (i.e. average grade of the block based on the MIK probability estimates) which was subsequently imported back into Datamine. No change of support was applied to the MIK estimates as only an e-type estimate was produced.

Where no gold estimate could be made due to sparse data, a default value was applied as per Table 14.12. The median value was used for the default gold grade due to the skewed nature of the grade distributions.

Table 14.12 Default gold grade values for un-estimated blocks

MINZONE	Default AU_MIK value (g/t Au)
0	0.02
3000	0.15

Search neighbourhood parameters – Au (g/t)

A three-pass search strategy was utilised for all grade estimates with the same search neighbourhood parameters applied to all domains. The search radii for the first search pass (30 m x 15 m x 7.5 m) corresponds to the range of continuity interpreted around the 80th percentile, while the second search pass (double the range of the first pass) corresponds to the range around the 60th percentile. Details of the estimation search parameters are presented in Table 14.13. The number of samples per drillhole was limited to three to ensure that a reasonable number of drillholes were used to estimate each block, with at least three drillholes within the search neighbourhood required for the first and second search passes.

Table 14.13 Selinsing model search neighbourhood parameters

Parameter	Value
Search ellipse rotation angles (Datamine format)	
Z axis	90
X axis	50
Z axis	0
Search radii (along strike x down dip x thickness)	
Search pass 1	30 m x 15 m x 7.5 m
Search pass 2	60 m x 30 m x 15 m
Search pass 3	120 m x 60 m x 30 m
Number of samples (minimum – maximum)	
Search pass 1	8 – 25
Search pass 2	8 – 25
Search pass 3	2 – 25
Maximum number of samples per drillhole	**3**

Grade estimation methodology – S (%)

Due to the poor coverage of sulphur assay data, variograms showed very poor quality and the decision was made to estimate the sulphur grade using inverse distance squared (ID^2).

Monument used the variograms for Au, modelled by Snowden in 2016, as a reference for the search ellipse orientation and ranges, for both fresh and transition domains. Based on this, the following parameters were selected for the sulphur estimation:

- A minimum of two samples and maximum of 24 samples for the initial search pass.
- Search ellipse radii of 285 m in the thickness direction by 1,000 m along strike by 666 m down dip for the initial search pass. The very large search ellipse ranges were required due to the very poor coverage of sulphur assays.

Sulphur for the fresh sulphide zone was estimated using a hard boundary with only sulphide samples coded within the mineralisation envelopes used. The search ellipse anisotropy was based on the parameters used for the Au estimation, with the maximum direction oriented along the north-south strike, the intermediate direction oriented down dip (dipping to the east), and the minor direction across the thickness. For the transition zone a soft boundary was used, with all samples (mineralised and unmineralised) contributing to the estimation. The search strategy for the transition zone used the same parameters as the sulphide zone. The oxide zone also used a soft boundary, with a sub-horizontal search orientation.

Model validation – Au (g/t)

The block grade estimates were validated using:

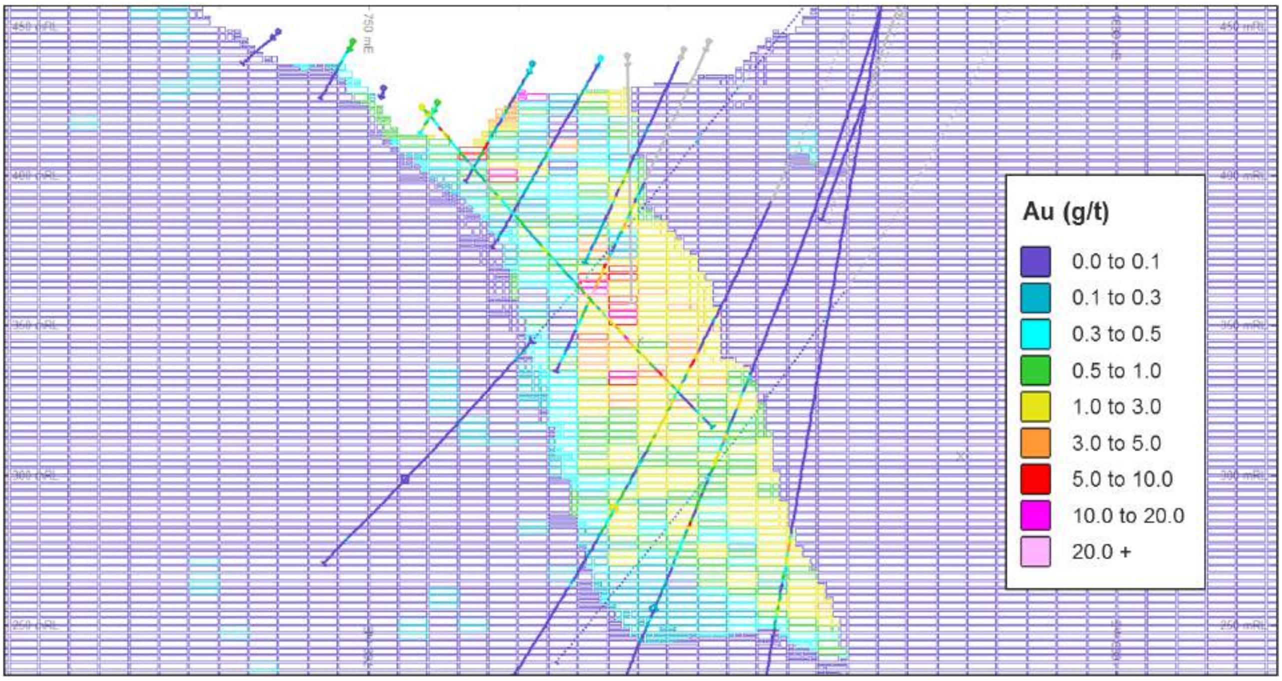
- A visual comparison of block grade estimates and the input drillhole data
- A global comparison of the average composite (naïve and de-clustered) and estimated block grades
- Moving window averages comparing the mean block grades to the composites.

The conclusions from the model validation work are as follows:

- Visual comparison of the model grades and the corresponding drillhole grades shows a good correlation (Figure 14.14)
- A comparison of the global drillhole mean grade with the mean grade of the block model estimate (for each domain) shows that the block model mean grade is within 15% of the drillhole mean which is a reasonable outcome (Table 14.14)

- With the exception of poorly sampled regions, the grade trend plots show a good correlation between the patterns in the block model grades compared with the drillhole grades (Figure 14.15).

Figure 14.14 Example west-east section (1990 mN; ±10 m) showing block grade estimates against the input drillhole composites



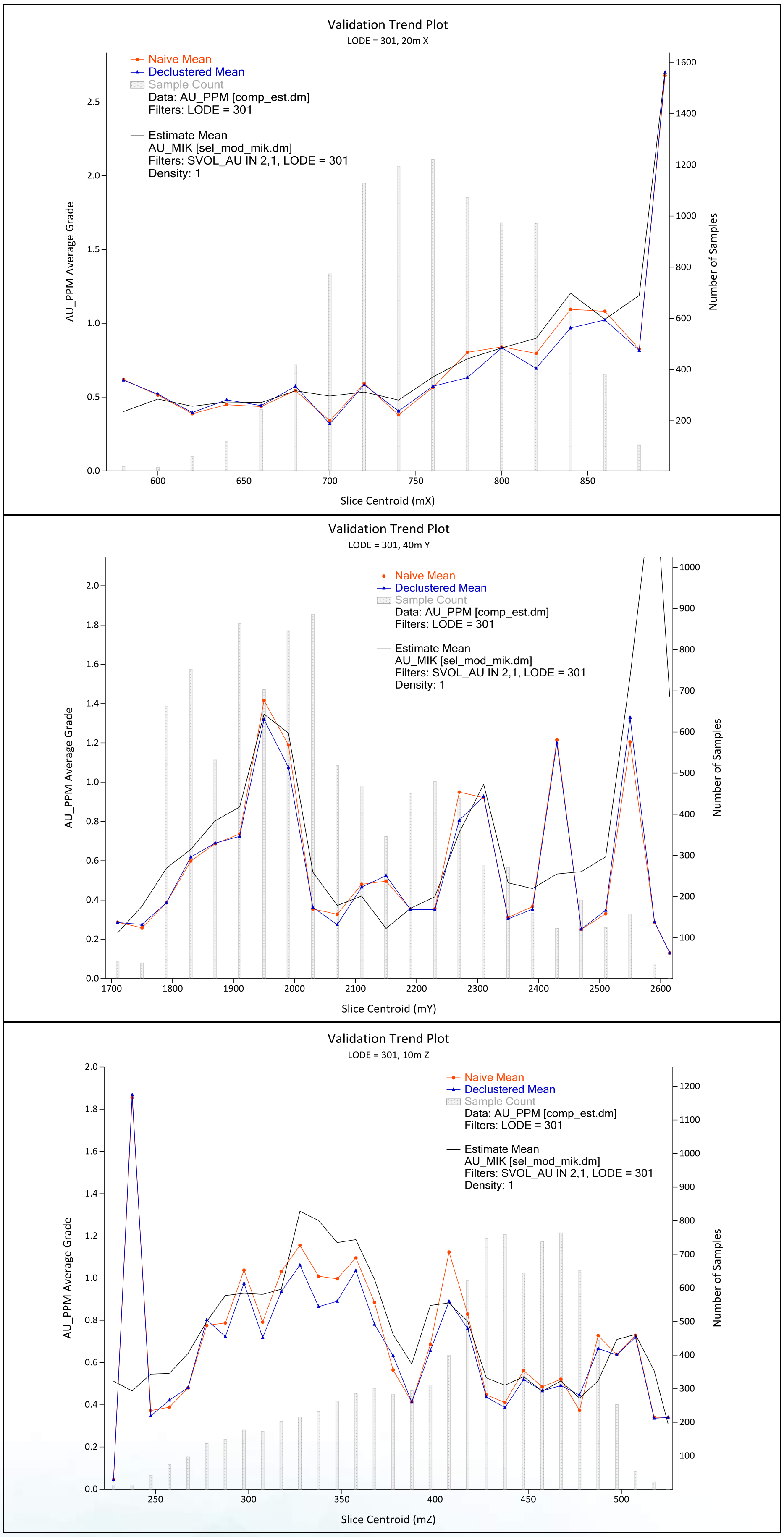
Source: Snowden

Table 14.14 Selinsing model validation summary statistics – gold

Statistic	Input samples		Block model estimate Au (g/t)
	Naïve Au (g/t)	De-clustered Au (g/t)	
Number	9,377	9,377	
Mean	**0.66**	**0.63**	**0.73**
Variance	9.64	7.95	1.26
Maximum	122.36	122.36	22.18
75%	0.46	0.46	0.80
50%	0.12	0.13	0.39
25%	0.03	0.03	0.19
Minimum	0.005	0.005	0.01

Notes: Sample data restricted to 10 m above February 2016 pit floor; model restricted to blocks estimated in the first and second search passes only; validation restricted to main mineralisation only (LODE=301)

Figure 14.15 Validation trend plots in the Y (top) and Z (bottom) directions – gold



Source: Snowden

Model validation – S (%)

For the sulphur block grade estimates, validation checks were undertaken both visually and statistically. Given the irregular and very low coverage of the sulphur assay data, the block estimates are considered to be reasonable overall, although highly smoothed.

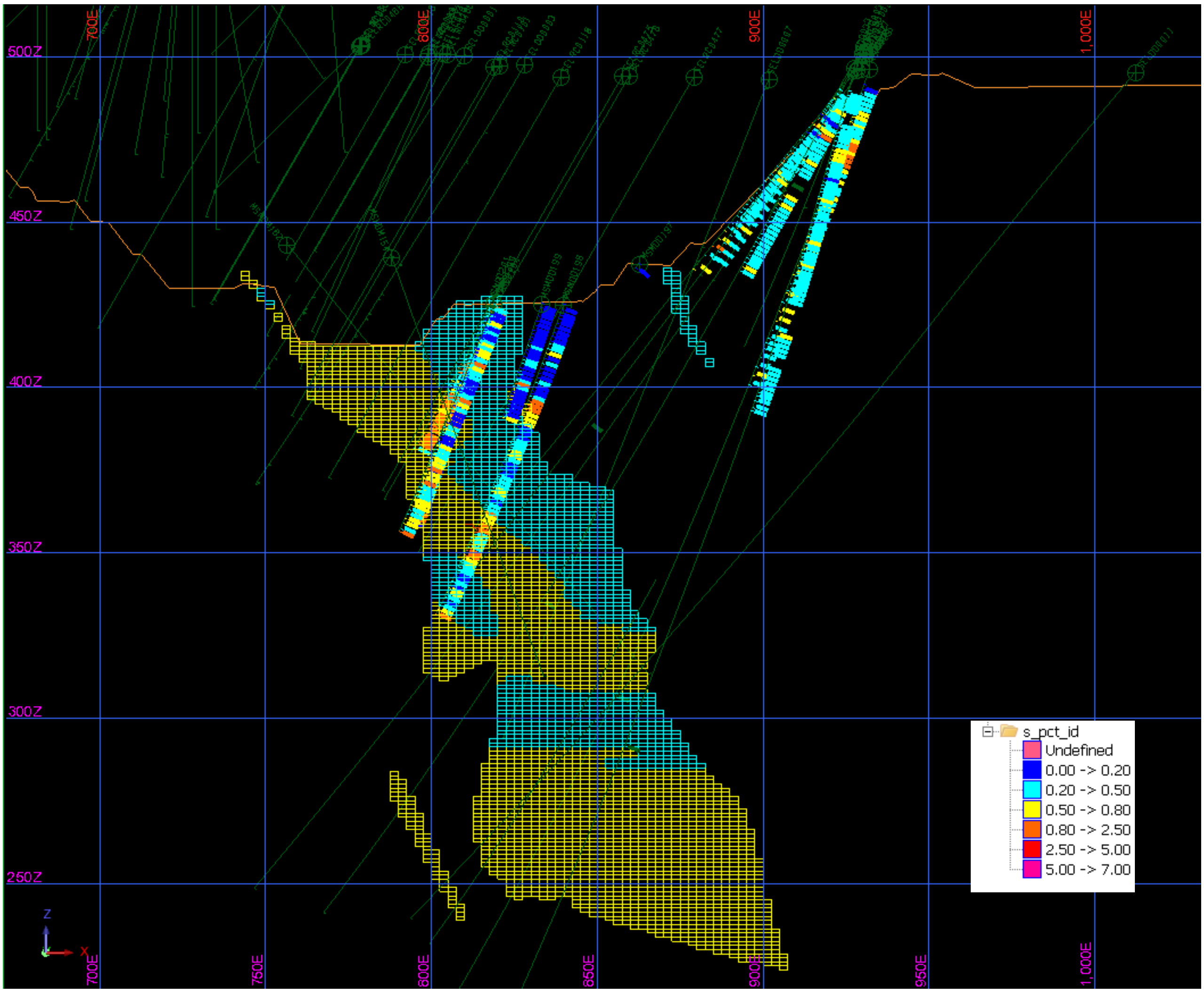
A global statistical comparison shows that the block grade estimates are within approximately 15% of the input sample data (Table 14.15) for the fresh zone for Selinsing.

Table 14.15 Selinsing model validation summary statistics – sulphur (fresh zone only)

Deposit	Input samples		Block model estimate S (%)
	Naïve S (%)	De-clustered S (%)	
Selinsing (fresh)	0.44	0.55	0.46

A cross-section showing the block sulphur grade estimates is shown in Figure 14.16.

Figure 14.16 Cross section Y=1960m +- 20m looking north showing Selinsing block S% grade estimates for sulphide blocks



14.4.5 Bulk density

Monument supplied Snowden with a spreadsheet containing 2,324 bulk density measurements of hand specimens collected from the Selinsing open pit. Snowden reviewed the data and identified significant errors, including incorrect bulk density calculations and erroneous weights (e.g. wax-coated dry sample weights lower than the uncoated dry weight). Monument subsequently provided a revised bulk density dataset, comprising 602 bulk density measurements, which Snowden believes are reasonable.

The samples average approximately 1–2 kg in weight, ranging from 0.5 kg up to 4.5 kg. The location of each sample was recorded by Monument based on the local mine grid.

The bulk density measurements were taken by Monument using the Archimedes Principle with wax-coating used to account for the porosity.

The procedure used by Monument to calculate the bulk density for a sample is as follows:

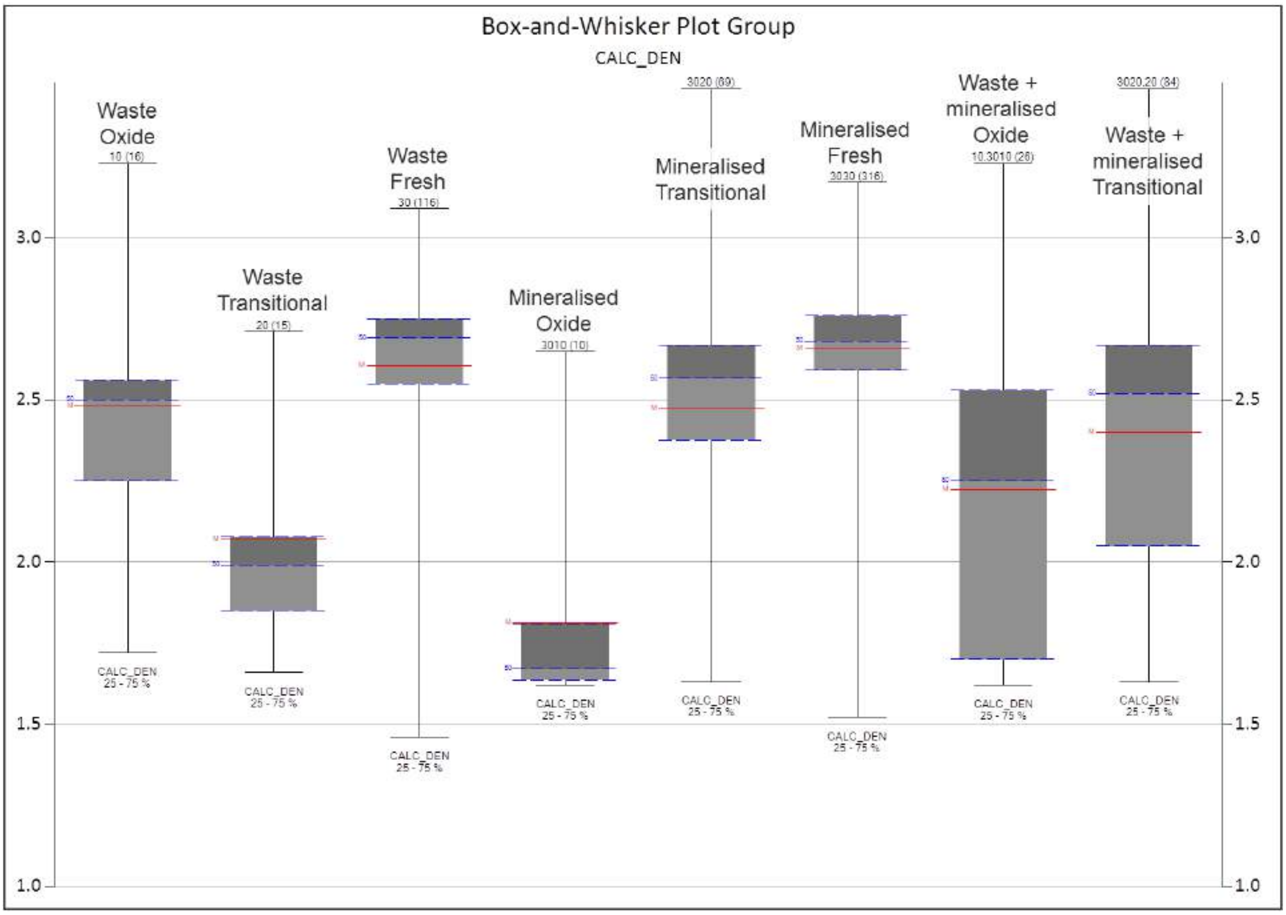
- The sample is dried in an oven at 105°C for 48 hours.
- Dry weight is determined.
- Sample is coated in wax and the dry weight with the wax is determined.
- The wax-coated sample is placed into a basket and submerged in water to determine the weight of the wax-coated sample suspended in water. Care is taken to ensure the sample is fully submerged and does not touch the bottom.

The equation used to calculate the bulk density is as follows:

$$bulk\ density = \frac{M_{dry}}{\left(M_{dry\ with\ wax} - M_{water\ with\ wax}\right) - \left(\frac{M_{dry\ with\ wax} - M_{dry}}{density_{wax}}\right)}$$

Snowden imported the density data into Datamine and intersected/coded the samples with the Selinsing oxidation surfaces and mineralisation wireframes. The data was then analysed to derive default values for each combination of oxidation state and mineralisation domain. Snowden noted 60 samples at depth within the fresh zone with anomalously low bulk density values, in the order of 1.5–1.7 t/m^3; the low bulk density values for these samples were considered to be likely caused by errors in the determination process and were removed from the dataset for the statistical analysis. A box-and-whisker plot is presented in Figure 14.17.

Figure 14.17 Box-and-whisker plot of bulk density samples from Selinsing open pit



Source: Snowden

Based on the analysis, default bulk density values were assigned to the model blocks based on the OXIDE and MINZONE coding, as per Table 14.16. Due to the low number of oxide samples, the waste and mineralised domains were combined to assess the bulk density of the oxide zone. Similarly, the mineralised samples were combined with the waste samples for the transitional zone to derive a value for the transitional waste material due to the low number of samples in the waste domain. A default bulk density of 1.44 t/m^3 for the Old Tailings material was used, as per discussion in Section 14.7.

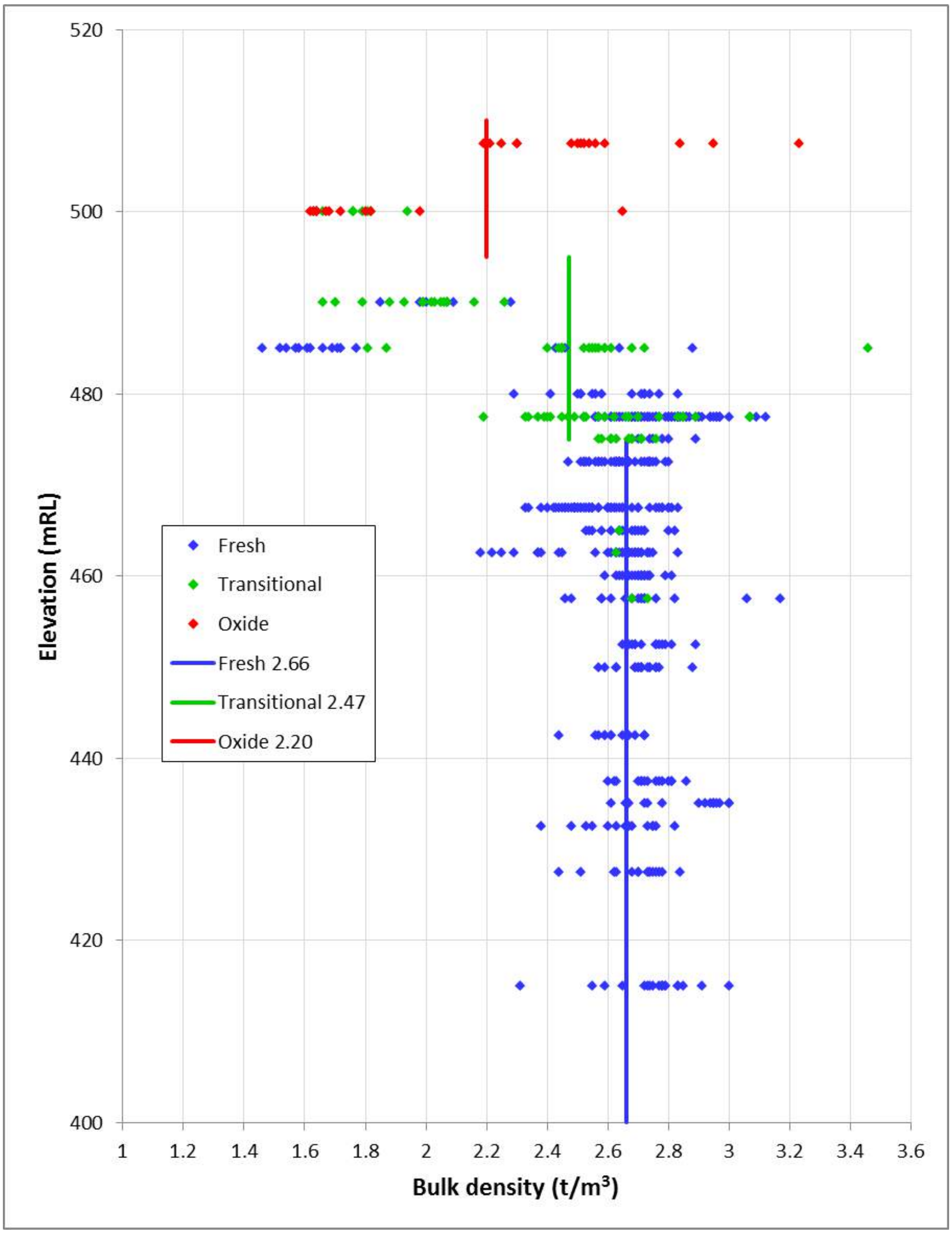
A plot of the bulk density values of all samples against the depth, with the assigned bulk density values for the mineralisation shown by solid lines, is presented in Figure 14.18 (the depth of the oxide and transitional boundaries in Figure 14.18 are indicative only). Snowden believes that the values assigned to the model are reasonable for the Selinsing mineralisation.

Given the errors identified by Snowden during the analysis of the bulk density data, Snowden recommends that Monument conduct an audit of the Selinsing site bulk density sample collection and determination processes to ensure the accuracy of the data being collected.

Table 14.16 Default bulk density values applied to Selinsing block model

Mineralisation state MINZONE	Oxidation state OXIDE	Bulk density (t/m^3)
Waste (0)	Oxide (10)	2.20
	Transitional (20)	2.40
	Fresh (30)	2.61
Mineralised (3000)	Oxide (10)	2.20
	Transitional (20)	2.47
	Fresh (30)	2.66
Old Tailings	-	1.44

Figure 14.18 Bulk density depth profile for Selinsing



Source: Snowden

14.4.6 Mineral Resource classification

The Selinsing Resource estimate was classified by Snowden as a combination of Indicated and Inferred Resources in accordance with CIM guidelines. The classification was developed based on an assessment of the following criteria:

- Nature and quality of the drilling and sampling methods
- Drilling density
- Confidence in the understanding of the underlying geological and grade continuity
- Analysis of the QAQC data
- A review of the drillhole database and Monument's sampling and logging protocols
- Confidence in the estimate of the mineralised volume
- The results of the model validation

- Production history and reconciliation.

The resource classification scheme adopted by Snowden for the Selinsing pit Mineral Resource estimate is outlined as follows:

- Where the drilling density was approximately 40 m along strike x 20 m down dip (or less), mineralisation within the main mineralised lode (i.e. LODE = 301) was classified as an Indicated Resource.
- Where the drilling density was greater than 40 m x 20 m, the mineralisation was classified as an Inferred Resource.
- All minor lodes were classified as Inferred Resources due to the limited geological continuity within these domains.
- Only unmined mineralisation has been considered by Snowden in this scheme. Old Tailings are discussed in Section 14.7, and Stockpiles Mineral Resources are discussed in Section 14.6.

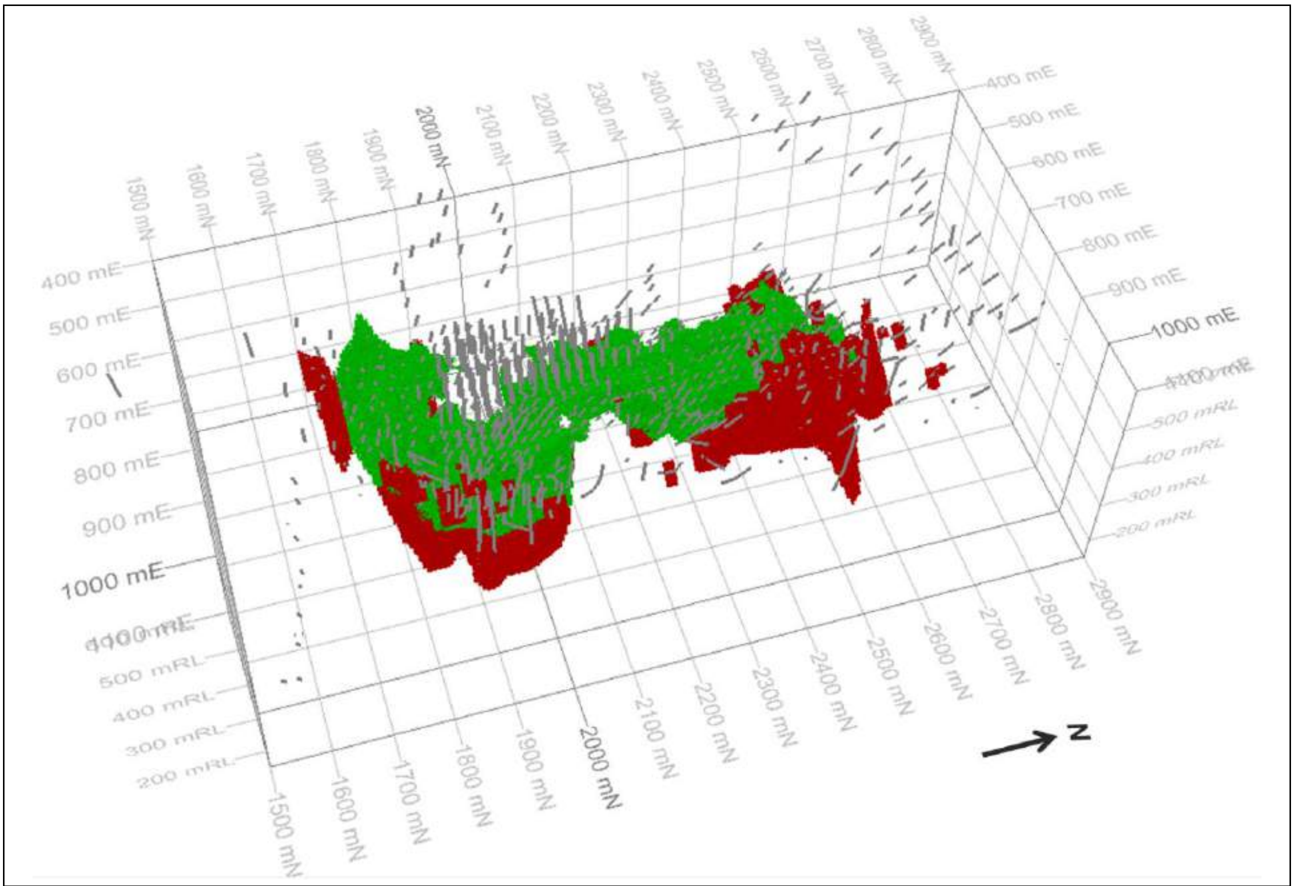
For Selinsing, sulphur was estimated in addition to gold, as this element is considered by Monument to be important for the processing. Snowden notes that comparatively very few samples contain assays for this element and as such, estimates of this variable should be considered to be of low confidence. As such, the resource classification applies to the gold grade estimates only. Due to the low confidence, the sulphur grade estimates are considered to be indicative only.

The classification was recorded in the model using a field called RESCAT as described in Table 14.17. The resulting classification is shown in Figure 14.19.

Table 14.17 Resource classification model field codes

RESCAT	Description
0	Not classified (waste)
2	Indicated
3	Inferred

Figure 14.19 Orthogonal view showing Selinsing resource classification (green = Indicated; red = Inferred)



Source: Snowden

14.4.7 Mineral Resource reporting

Mining selectivity assumptions for Mineral Resource

Given the broad nature of the interpretation of the mineralisation, which incorporates a broad mineralised envelope with internal waste, Snowden cautions that the Mineral Resource estimate and associated grade-tonnage curve assumes a relatively coarse level of selectivity. If Monument is able to achieve a higher level of selectivity during mining via adequate grade control, Snowden anticipates that a lower tonnage at higher grade may be realised than is predicted by the Mineral Resource estimate.

Cut-off grade

The Mineral Resource for Selinsing has been reported above a 0.3 g/t Au cut-off grade for oxide material and above a 0.5 g/t Au cut-off grade for transitional and sulphide material. The cut-off grades are based on cost and metal price parameters detailed in Section 16 (Mining Methods), along with the assumptions discussed below, and practical considerations with respect to the mining grade control.

Moisture

All Mineral Resources have been reported on a dry tonnage basis.

Depletion for mining

The Selinsing Mineral Resource has been depleted for all open pit mining to end of March 2018.

Mineral Resource statement

The Mineral Resource estimate for the Selinsing deposit is provided in Table 14.18. The Mineral Resource is limited to a pit shell generated by Monument based on a long-term potential gold price of US$2,400/oz. This pit shell was used by Monument to define the likely limits of potential open pit mining. The mining and cost parameters and assumptions used by Monument to generate the resource pit shell are listed in Table 14.19.

Table 14.18 Selinsing Mineral Resource statement, inclusive of Mineral Reserves, depleted for mining to end of March 2018

Classification	Oxidation	Cut-off (g/t Au)	Tonnes (kt)	Au (g/t)	S (%)	Ounces (koz Au)
Indicated	Oxide	0.3	64	0.62	0.03	1
	Transitional	0.5	100	1.16	0.38	4
	Fresh	0.5	5,007	1.51	0.47	243
Indicated – total			**5,171**	**1.49**	**0.46**	**248**
Inferred	Oxide	0.3	8	0.98	0.03	0.3
	Transitional	0.5	3	1.14	0.17	0.1
	Fresh	0.5	1,680	2.02	0.52	109
Inferred – total			**1,691**	**2.02**	**0.51**	**110**

Notes: Small discrepancies may occur due to rounding. The classification applies to the Au grades only; S grades are considered indicative only. Mineral Resources are inclusive of Mineral Reserves.

Table 14.19 Resource pit optimisation parameters and assumptions

Input parameters	Unit	Selinsing	BRSCF	BRN
Mining cost	US$/t	1.89	1.66	1.71
Ore loss	%	5	5	5
Mining dilution	%	5	5	5
Processing cost – oxide	US$/t	8.00	8.00	8.00
Processing cost – sulphide	US$/t	10	10	10
Processing recovery – oxide	%	80	80	80
Processing recovery – sulphide	%	90	90	90
Overall slope	°	50	50	50
Selling price	US$/troy oz	2,400	2,400	2,400
Cut-off grade – oxide	g/t Au	0.3	0.3	0.3
Cut-off grade – sulphide	g/t Au	0.5	0.5	0.5
Revenue factor		1	1	1

Note: The foregoing information was used to generate an optimisation pit shell based on a reasonable prospect for extraction of Mineral Resources. No economic analysis has been conducted on the Mineral Resources.

Comparison to 2016 Mineral Resource

An increase of 48 koz (24%) for Indicated Resources and 45 koz (69%) for Inferred Resources can be observed for the Selinsing deposit, compared to the 2016 Mineral Resource (Snowden, 2016).

The increase is primarily within the sulphide portion of the mineralisation and is due to a larger constraining pit shell along with a recalculation of the cut-off grades used for reporting the resource. However, due to mining depletion, the oxide resource has decreased compared to the 2016 Mineral Resource. A breakdown of the differences is shown in Table 14.20 for Selinsing.

Table 14.20 **Differences between the current and previous (Snowden, 2016) Selinsing Mineral Resource estimates**

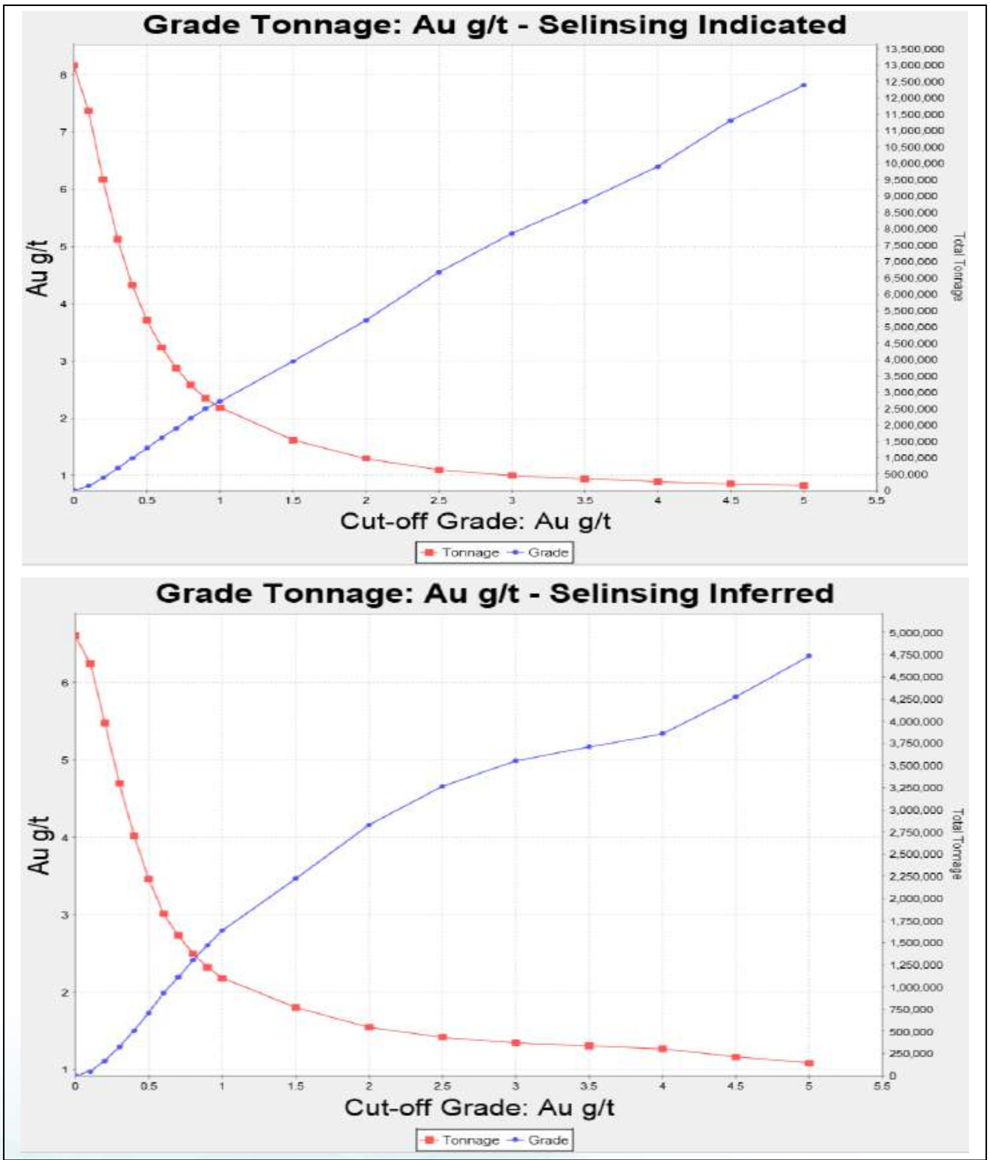
Classification	Oxidation	Cut-off (g/t Au)	Tonnes (kt)	Ounces (koz Au)
Indicated	Oxide	0.3	-26	-1
	Transitional	0.5	10	0
	Fresh	0.5	1,967	50
Indicated – total			**1,951**	**48**
Inferred	Oxide	0.3	-2	0
	Transitional	0.5	0	0
	Fresh	0.5	1,140	44
Inferred – total			**1,138**	**45**

Notes: Small discrepancies may occur due to rounding. Positive values indicate an increase in the resource.

Grade-tonnage curves

Grade-tonnage curves for the Selinsing Mineral Resource estimates, limited to the US$2,400/oz resource pit shell, are presented in Figure 14.20.

Figure 14.20 **Grade-tonnage curves for Selinsing, Indicated Resource (top) and Inferred (bottom) for sulphide resources**



14.5 Buffalo Reef Mineral Resource estimate

14.5.1 Geological interpretation

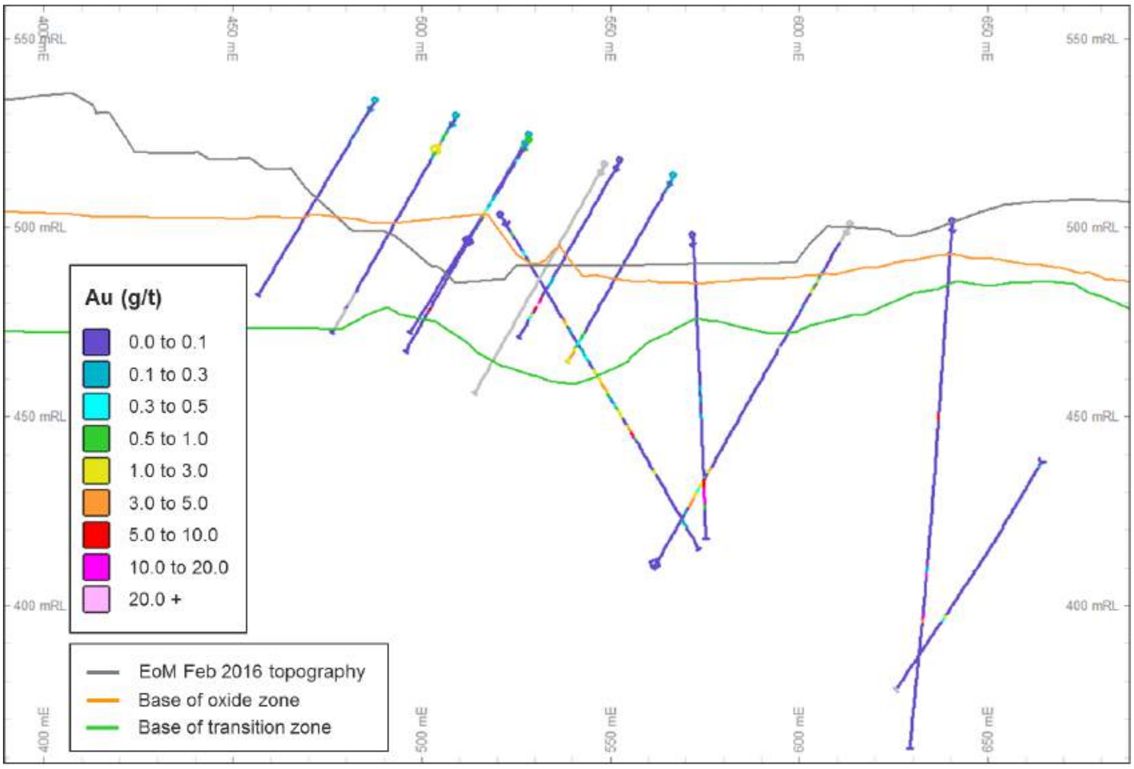
Oxidation

Monument interpreted surfaces for the BOCO and TOFR based on the logging of the oxidation state of the drillholes. Monument noted that the logging of historical holes is incomplete in some cases and validation with actual mining shows that the historical logging tends to underestimate the oxidation (i.e. slightly biased towards fresh). As such, holes drilled prior to July 2015 were re-logged to ensure consistency between the logging of the older drilling and the more recent drilling.

The interpretation preferentially used diamond drillholes, with the RC holes given a secondary priority. Logged vein intensity was also used to assist with the interpretation of the oxidation surfaces as the weathering tends to be influenced by the presence of mineralised quartz veins and the associated alteration.

The surfaces provided by Monument were expanded by Snowden to cover the entire model area. An example west-east cross section showing the oxidation surfaces is shown in Figure 14.21.

Figure 14.21 Example west-east section (3440 mN) showing interpretation of oxidation surfaces



Source: Snowden

Mineralisation

The sectional interpretation of the Buffalo Reef gold mineralisation was completed by Monument based on east-west sections with a section spacing of 20 m. Logging information from RC and diamond drillholes (along with a few surface trenches), including vein intensity, alteration and structural information, along with the gold and antimony grades, were utilised by Monument to interpret the gold mineralisation. A nominal threshold of 0.15 g/t Au was used to guide the interpretation of the gold mineralisation. Monument noted that elevated gold grades are typically associated with zones of higher intensity quartz veining; however, zones of veining also occur in unmineralised rocks. The zones of quartz veins are normally associated with higher hydrothermal alteration with sulphide minerals including pyrite, chalcopyrite, arsenopyrite and stibnite.

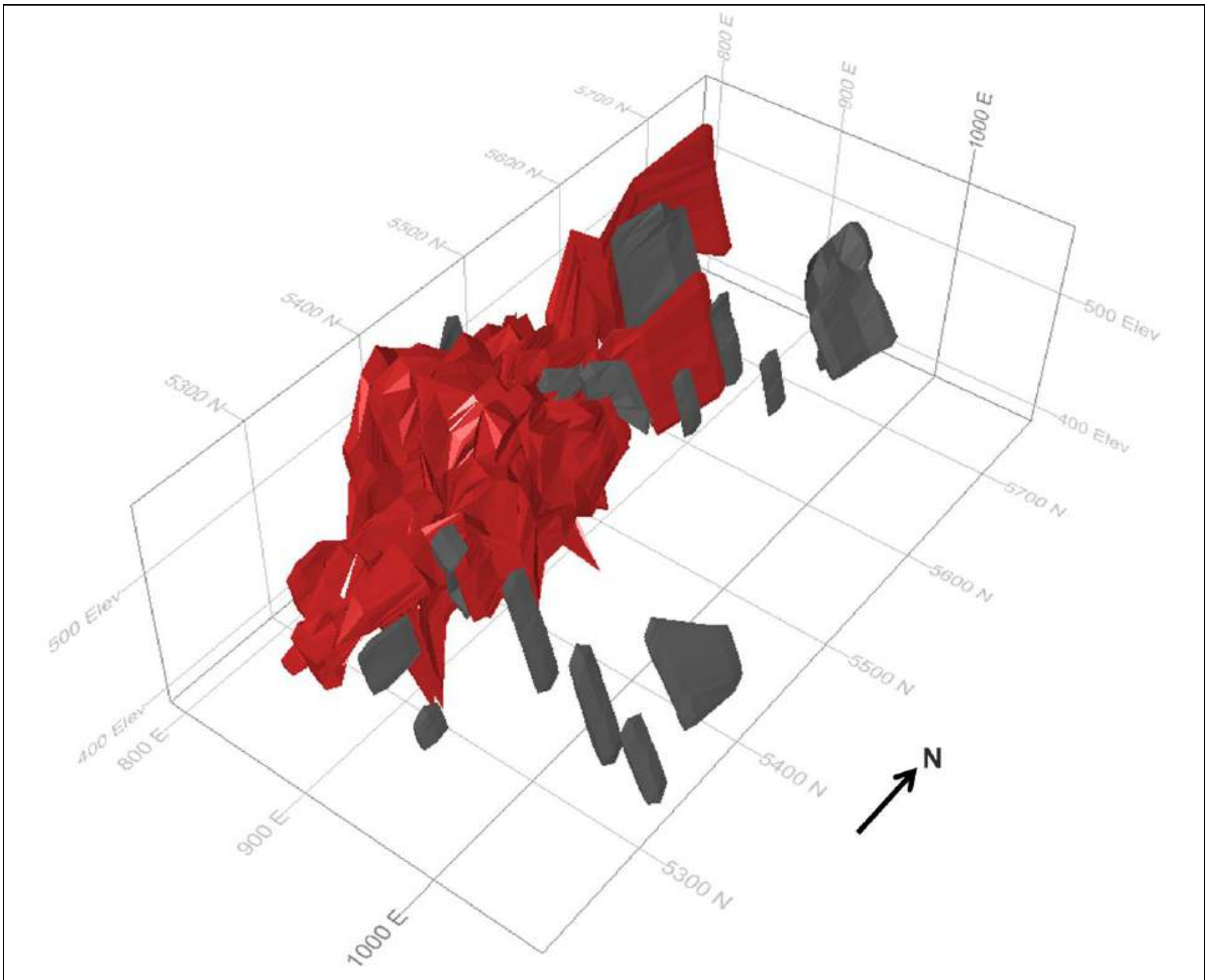
The interpreted mineralisation at Buffalo Reef South-Central-Felda (BRSCF) comprises a main tabular mineralised zone which is typically 10–20 m thick, along with numerous minor sub-parallel mineralised structures. At Buffalo Reef North (BRN), the mineralisation is less continuous and narrower when compared to BRSCF. The interpreted mineralisation at BRN comprises a number of sub-parallel zones, with several sub-parallel minor mineralised structures.

Oblique views of the Buffalo Reef mineralised zones are shown in Figure 14.22 and Figure 14.23 respectively for BRN and BRSCF. Example cross sections for BRN and Buffalo Reef South (BRS) are shown in Figure 14.24 and Figure 14.25 respectively.

The interpreted mineralisation wireframes showing the connection between the Selinsing and Buffalo Reef areas is shown in the Figure 14.26.

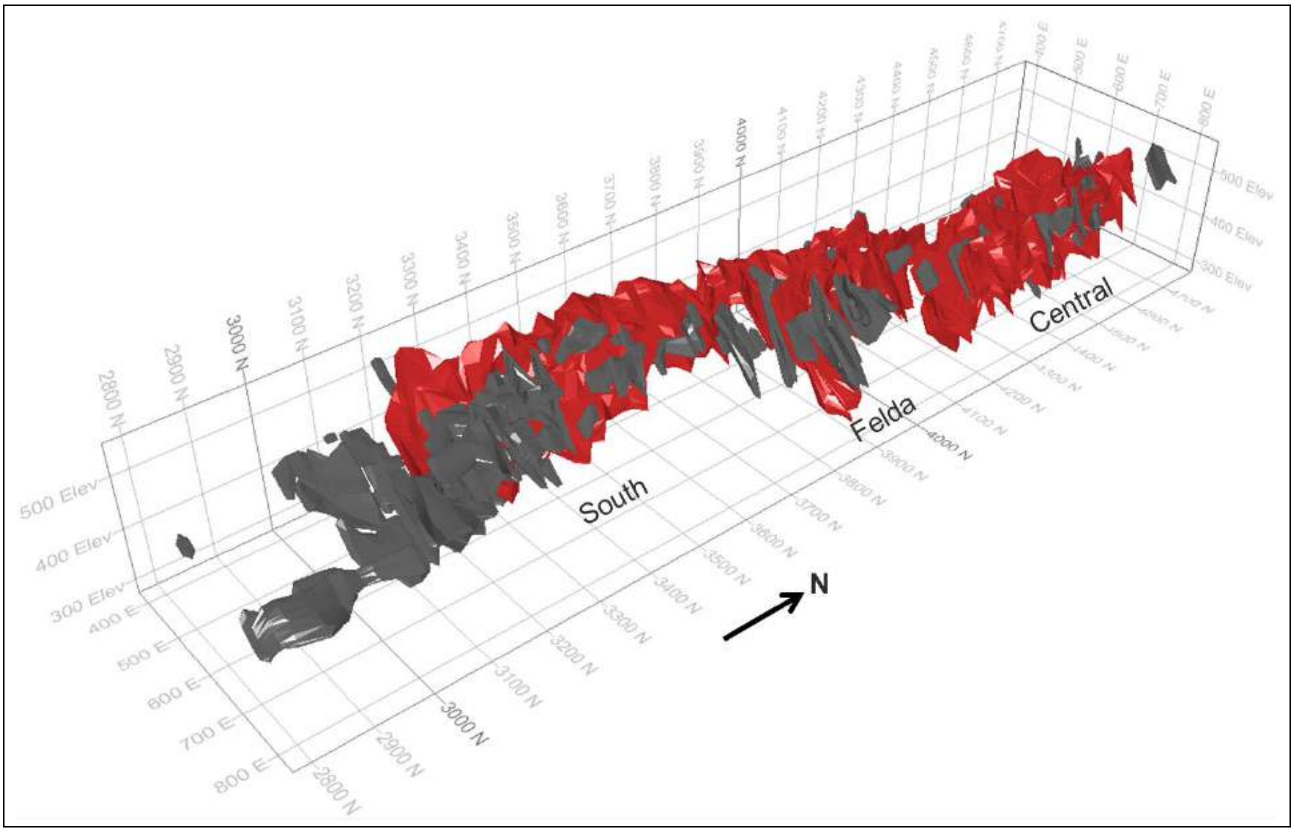
A minimum of three sample intervals (generally 3 m for RC holes and similar for diamond holes) was used for the interpreted mineralised intersections. External dilution was only included for isolated intervals with continuity from neighbouring intervals, either down dip in the same section or along strike in adjacent sections. Internal waste lenses were interpreted for Buffalo Reef where the internal waste displayed continuity along strike and/or down dip. Extrapolation of the interpretation down dip and along strike was limited to approximately half the drillhole spacing.

Figure 14.22 Oblique view (looking northwest) of the BRN mineralisation interpretation



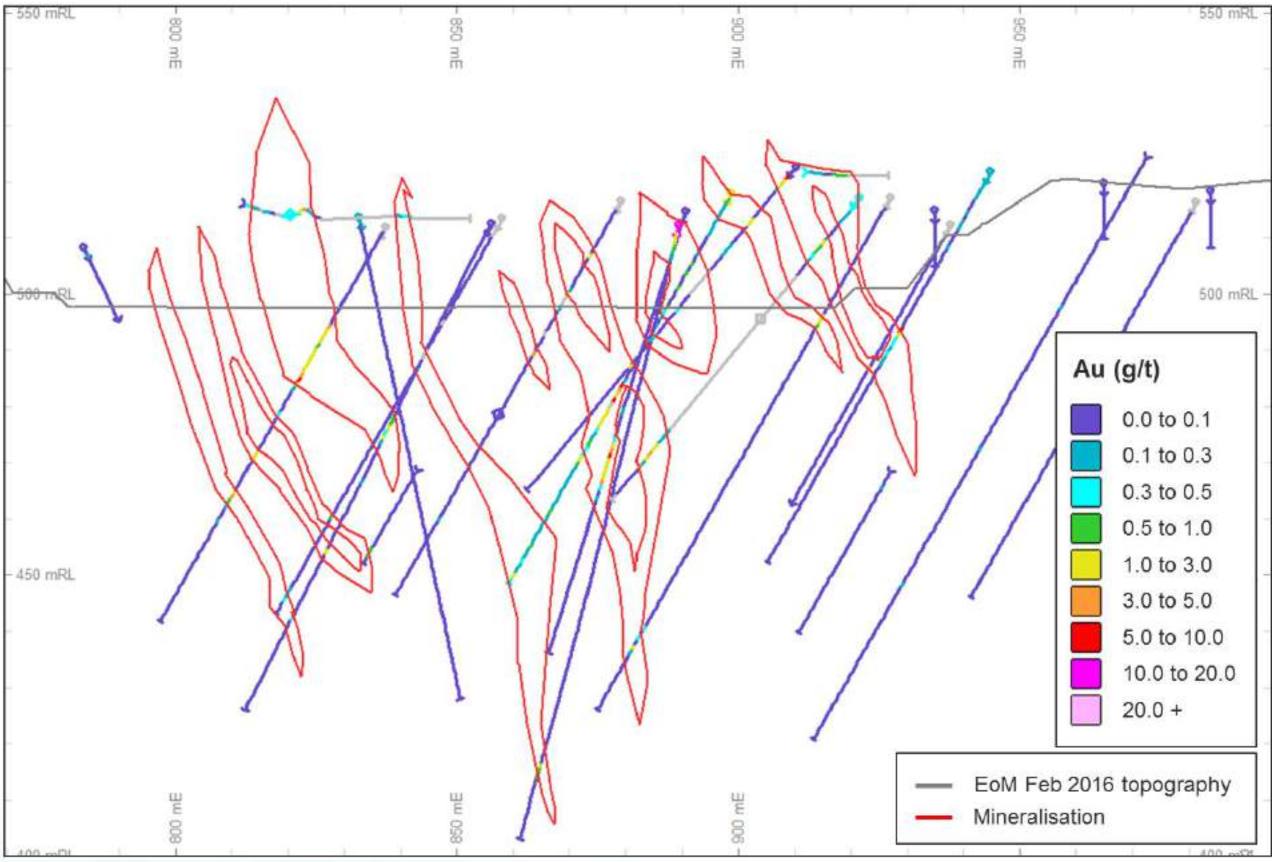
Source: Snowden

Figure 14.23 Oblique view (looking northwest) of the BRSCF mineralisation interpretation



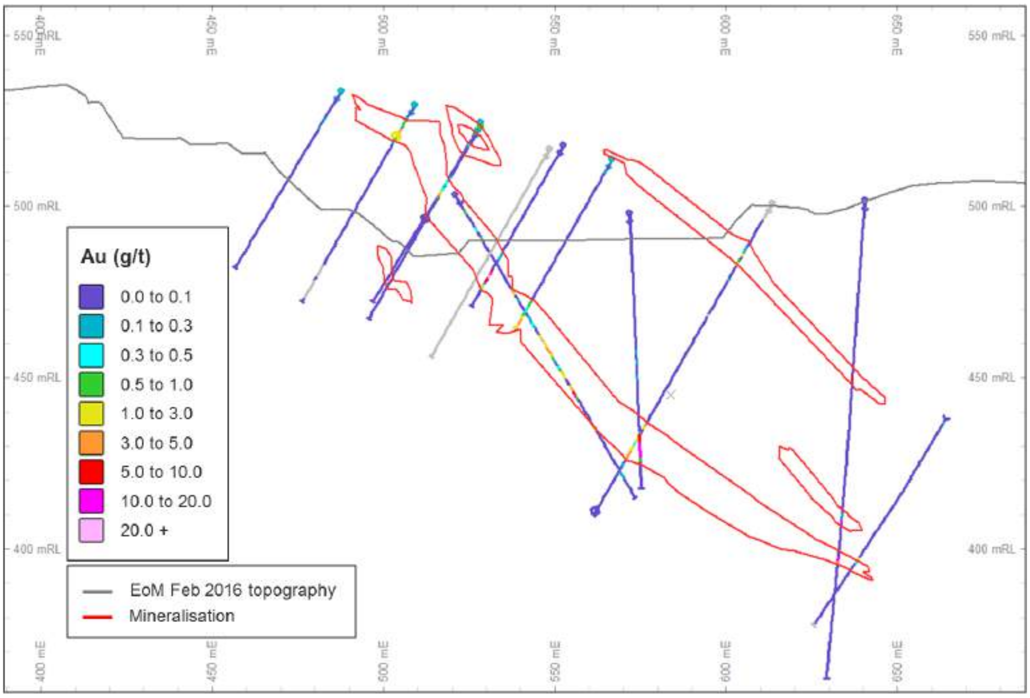
Source: Snowden

Figure 14.24 Example west-east section (5460 mN, BRN) showing mineralisation interpretation



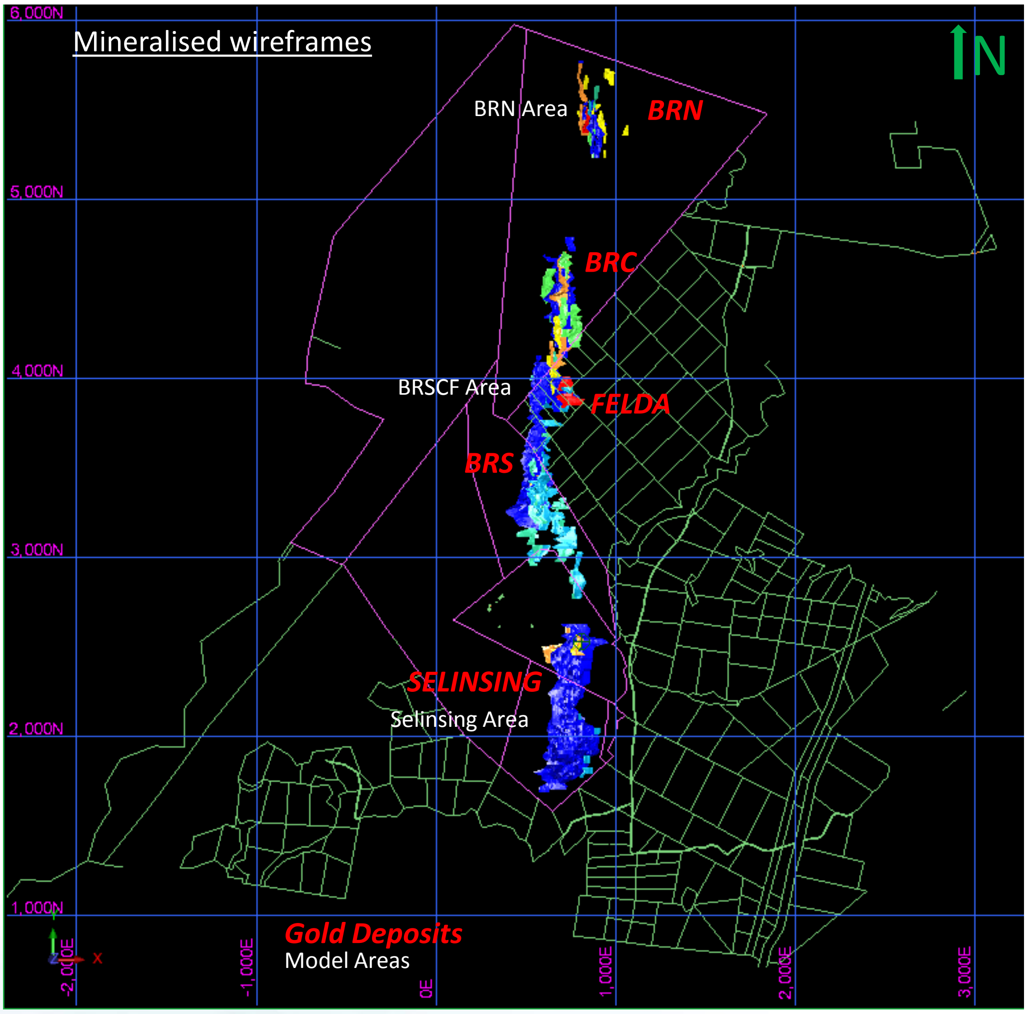
Source: Snowden

Figure 14.25 Example west-east section (3440 mN, BRS) showing mineralisation interpretation



Source: Snowden

Figure 14.26 Plan view of the mineralised wireframes interpreted



14.5.2 Drillhole data analysis

For the Buffalo Reef model, the drillhole dataset was limited to north of 2650 mN (Figure 14.27). Table 14.21 summarises the drilling data subset used for the Buffalo Reef resource model. A very small amount of AC drilling was included; however, samples from the AC drilling were only used for the estimates in the waste blocks. Only RC and diamond data, along with a small amount of trench data was used for the grade estimates in the mineralised domains. The trench data was assessed by Snowden in 2011 (Snowden, 2011) and found to be reasonable for use in resource estimation. Snowden understands that no additional trench data within the block model boundaries has been completed since this study was done and as such the data was included in the resource estimation. Moreover, in the southern portion of the deposit, the majority of the area covered by the trench data has been mined out.

The cut-off date for the data used in the Buffalo Reef model was for drilling assays received up to 24 February 2016. An additional 59 drillholes (RC and diamond core), for 5,694 m of drilling, completed in 2016 were not included in the 2016 Buffalo Reef resource model (Snowden, 2016). This additional data was included by Monument in the dataset used for the estimation of sulphur; however, the additional data is not considered material with respect to the gold grade estimates, which remain unchanged. Table 14.21 and Table 14.22 show a breakdown of the metres by drilling type.

Table 14.21 Drilling data used for Buffalo Reef model (north of 2650 mN)

Drilling method	Total length (m)	Proportion
Reverse circulation (RC)	34,191.5	43.8%
Diamond drilling (DD)	41,420.5	53.1%
RC with diamond tail	600.5	0.8%
Trench	1,345.8	1.7%
Air-core (AC) *	461.5	0.6%
Total	**78,019.8**	

** AC includes some minor auger drillholes, banka and grade control holes; DD includes diamond metallurgical holes*

Table 14.22 Summary of Buffalo Reef and Selinsing drilling after end February 2016*

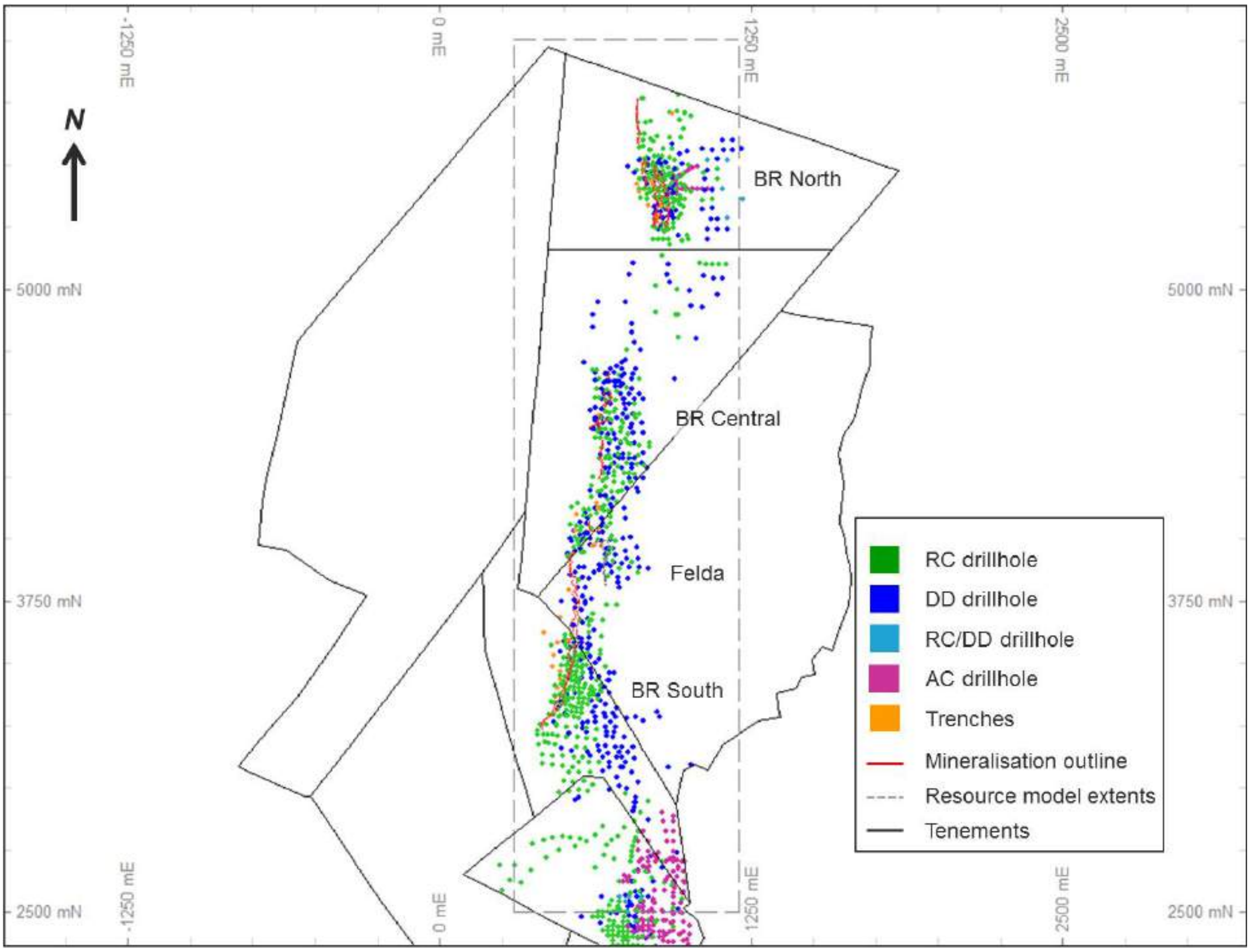
Hole type	No. of holes	Length (m)
Diamond (DD)	13	1,612
DD (metallurgical)	16	1,452
Reverse circulation (RC)	10	681
RC + DD (metallurgical)**	20	1,949
Total	**59**	**5,694**

** For assays received after 24 February 2016*

***Metallurgical DD drilling pre-collared by RC drilling*

The drilling at Buffalo Reef is based on a section spacing of approximately 20 m with drilling typically completed at 20 m intervals on section (i.e. a drill spacing of 20 mN x 20 mE). At BRN, the drill spacing can be locally down to 15 m along strike. The drill spacing in the deeper portions of the mineralisation, as well as the extremities of each area, is typically wider spaced, averaging 40 mN x 40 mE to 60 mN x 40 mE.

Figure 14.27 Collar location plan for Buffalo Reef resource model area



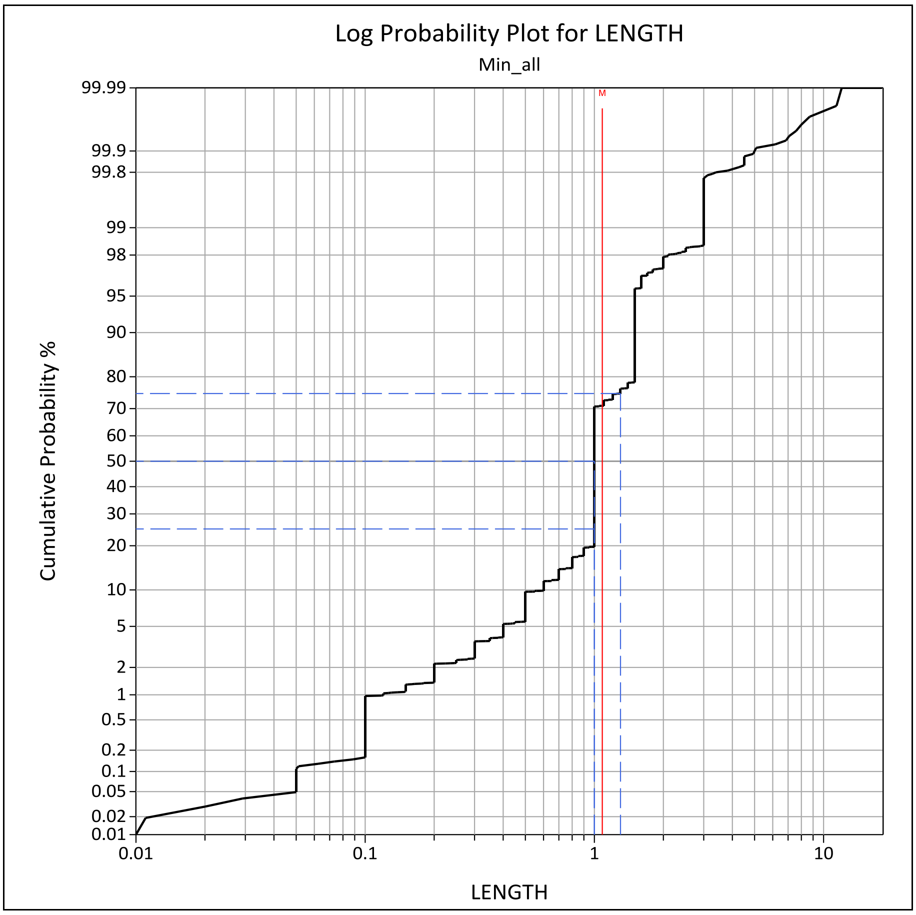
Sulphur data coverage

The available sulphur assay data, as validated and incorporated in the database, come entirely from Monument drilling campaigns undertaken since 2014. Similar to Selinsing, sulphur has not been assayed consistently for the Buffalo Reef deposit, resulting in an irregular sample assay coverage, especially in the BRN area.

Sample compositing

The drillhole data was composited downhole prior to running the estimation process using a 1.5 m compositing interval for Au, As and Sb, and a 3 m interval for S, to minimise any bias due to sample length. A log probability plot of the raw drillhole sample lengths within the Buffalo Reef mineralised domains is presented in Figure 14.28. The compositing interval of 1.5 m (and 3 m for S) was chosen, rather than 1 m, to avoid excessive sample splitting as a significant proportion of the samples have sample lengths of 1.5 m.

Figure 14.28 Log probability plot of sample lengths within Buffalo Reef mineralisation



The compositing was run within the attribute fields to ensure that no composite intervals crossed any lithological or grade boundaries. To allow for uneven sample lengths within each of the domains, the composite process was run using the variable sample length method. This adjusts the sample intervals, where necessary, to ensure all samples are included in the composite file (i.e. no residuals) while keeping the composite interval as close to the desired interval as possible.

The compositing process was checked by:

- Comparing the lists of attribute field values in the raw and composite files; these should match
- Comparing the sample length statistics in the raw and composite files; the two total length values should match, and the mean composite interval should be 1.5 m.

No discrepancies were identified during the compositing process.

Statistical analysis

Statistical analysis was carried out on the composited dataset for Au, As, Sb and S grades.

Gold

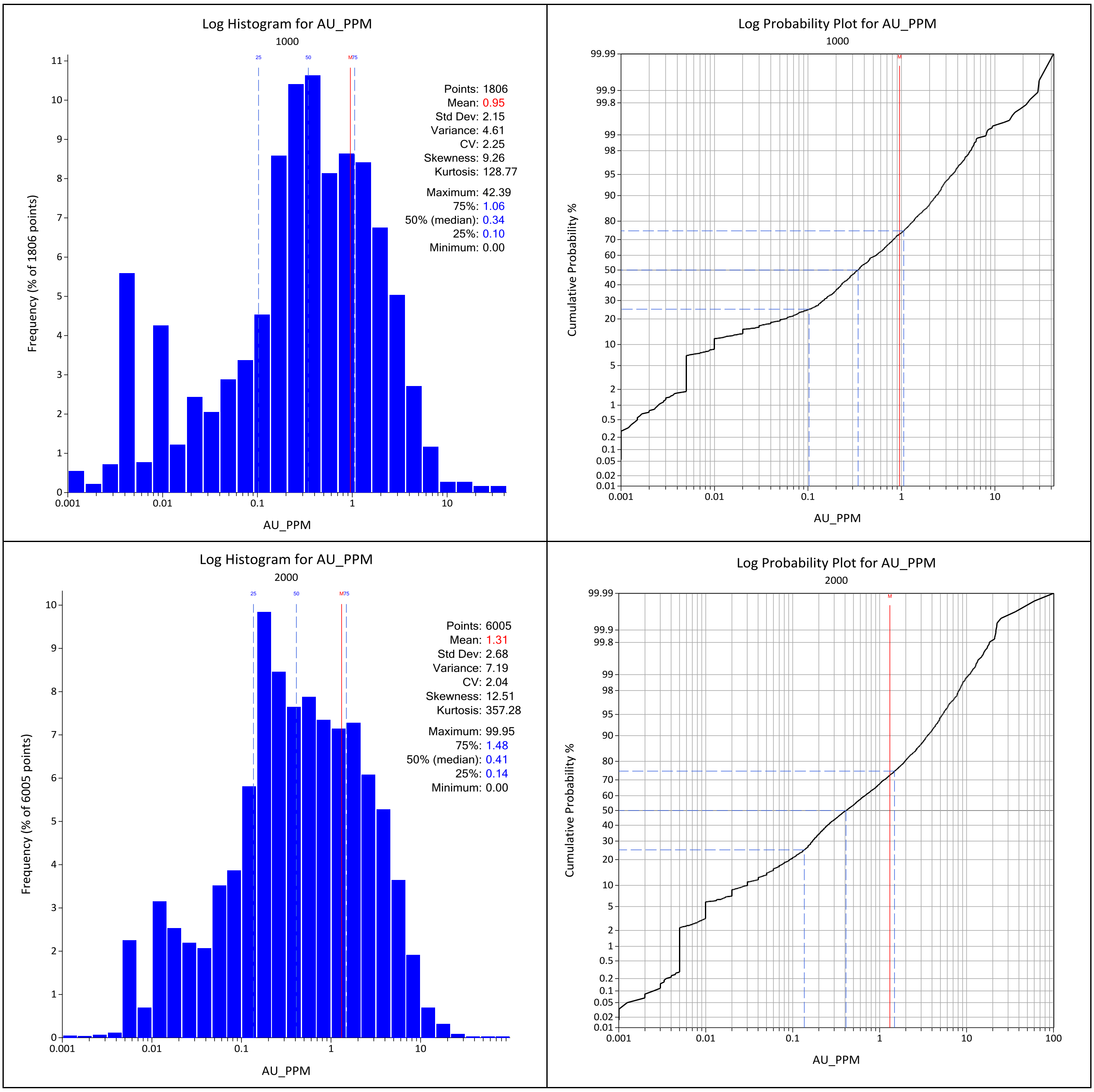
Summary statistics of composites for gold for Buffalo Reef North (BRN, MINZONE=1000) and Buffalo Reef South-Central-Felda (BRSCF, MINZONE=2000) are presented in Table 14.23 and log histograms and log probability plots are presented in Figure 14.29. The statistics show that the Buffalo Reef mineralisation has a positively skewed gold grade distribution with a high CV.

Due to the skewed nature of the gold grades (CV>1.5) in both the mineralised domain and the waste domain, Snowden elected to use ordinary kriging (OK) to estimate the block gold grades, with top-cuts applied to control the influence of extreme outliers.

Table 14.23 Summary gold statistics for BRN and BRSCF mineralised composites

Statistic		BRN MINZONE 1000 Au (g/t)	BRSCF MINZONE 2000 Au (g/t)
Samples		1,806	6,005
Minimum		0.005	0.005
Maximum		42.39	99.95
Mean		0.95	1.31
Standard deviation		2.15	2.68
CV		2.25	2.04
Percentiles:	10%	0.01	0.03
	20%	0.06	0.09
	30%	0.15	0.17
	40%	0.23	0.25
	50%	0.34	0.41
	60%	0.54	0.68
	70%	0.82	1.12
	80%	1.34	1.95
	90%	2.37	3.56
	95%	3.50	5.31
	97.5%	4.99	7.71
	99%	8.07	10.79

Figure 14.29 Log histogram and log probability plot for gold for BRN (top) and BRSCF (bottom) mineralised combined domains



Source: Snowden

Arsenic

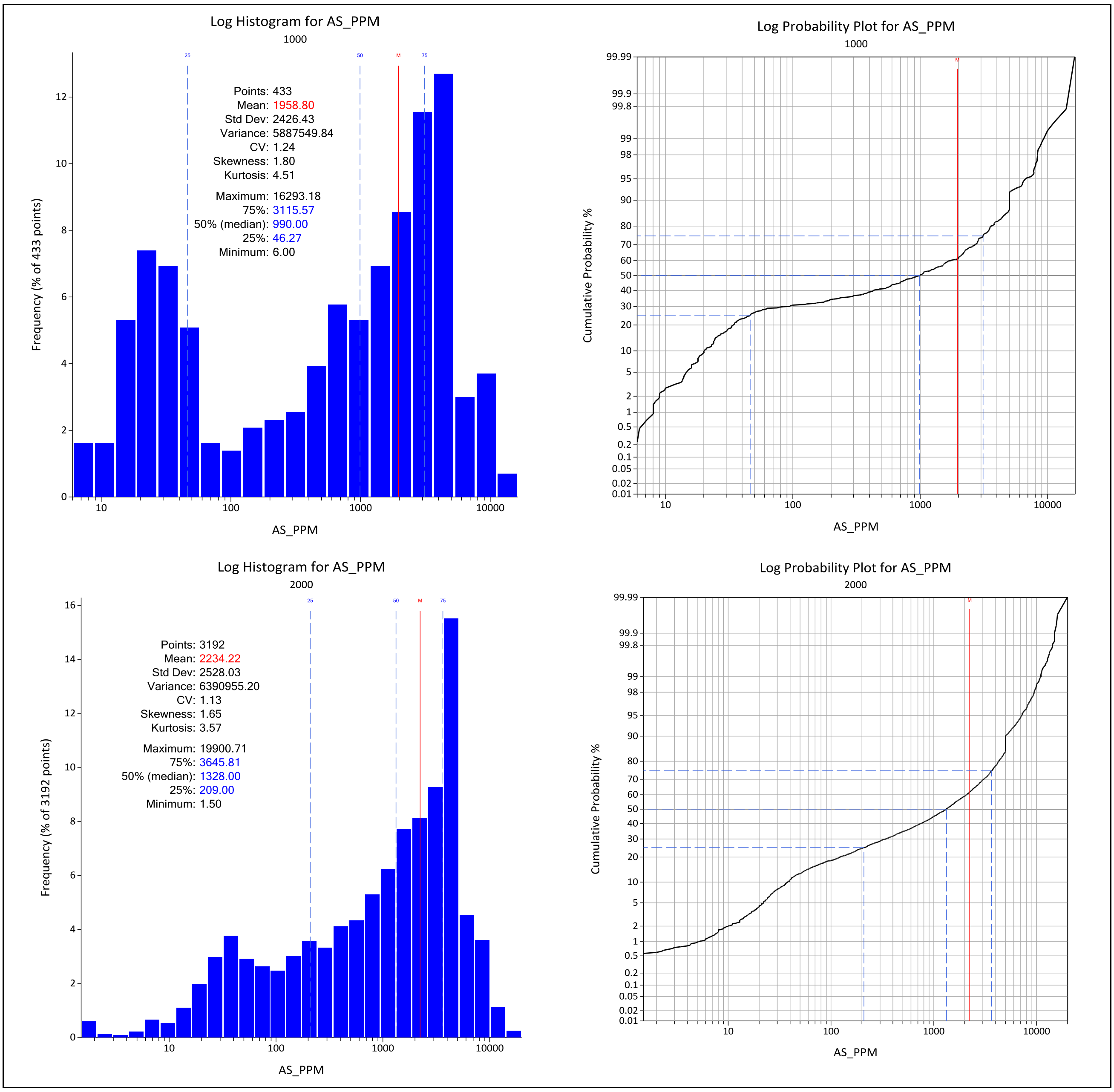
Summary statistics of composites for arsenic for Buffalo Reef North (BRN, MINZONE=1000) and Buffalo Reef South-Central-Felda (BRSCF, MINZONE=2000) are presented in Table 14.24 and log histograms and log probability plots are presented in Figure 14.30. The statistics show that the Buffalo Reef mineralisation has a positively skewed arsenic grade distribution with a moderate CV. Additionally, the statistics show mixed populations, with a low grade population centred on approximately 35 ppm As and a high grade population centred on approximately 3,500 ppm As, which Snowden believes may relate to two phases of mineralisation at Buffalo Reef – one associated with pyrite (low-As) and one associated with arsenopyrite (high-As); however this has not been clearly identified by Monument in the drilling.

Due to the moderate CV (CV<1.5) in both the mineralised domains and the waste domain, Snowden elected to use OK to estimate the block arsenic grades. No top-cuts were required in the mineralised domains due to the moderate CV and absence of outliers.

Table 14.24 Summary arsenic statistics for BRN and BRSCF mineralised composites

Statistic		BRN MINZONE 1000 As (ppm)	BRSCF MINZONE 2000 As (ppm)
Samples		433	3,192
Minimum		6	2
Maximum		16,293	19,901
Mean		1,959	2,234
Standard deviation		2,426	2,528
CV		1.24	1.13
Percentiles:	10%	20	38
	20%	34	122
	30%	94	341
	40%	430	740
	50%	990	1,328
	60%	1,739	2,075
	70%	2,712	3,060
	80%	3,609	4,274
	90%	5,000	5,000
	95%	6,735	7,146
	97.5%	8,300	8,976
	99%	9,318	11,000

Figure 14.30 Log histogram and log probability plot for arsenic for BRN (top) and BRSCF (bottom) mineralised combined domains



Source: Snowden

Antimony

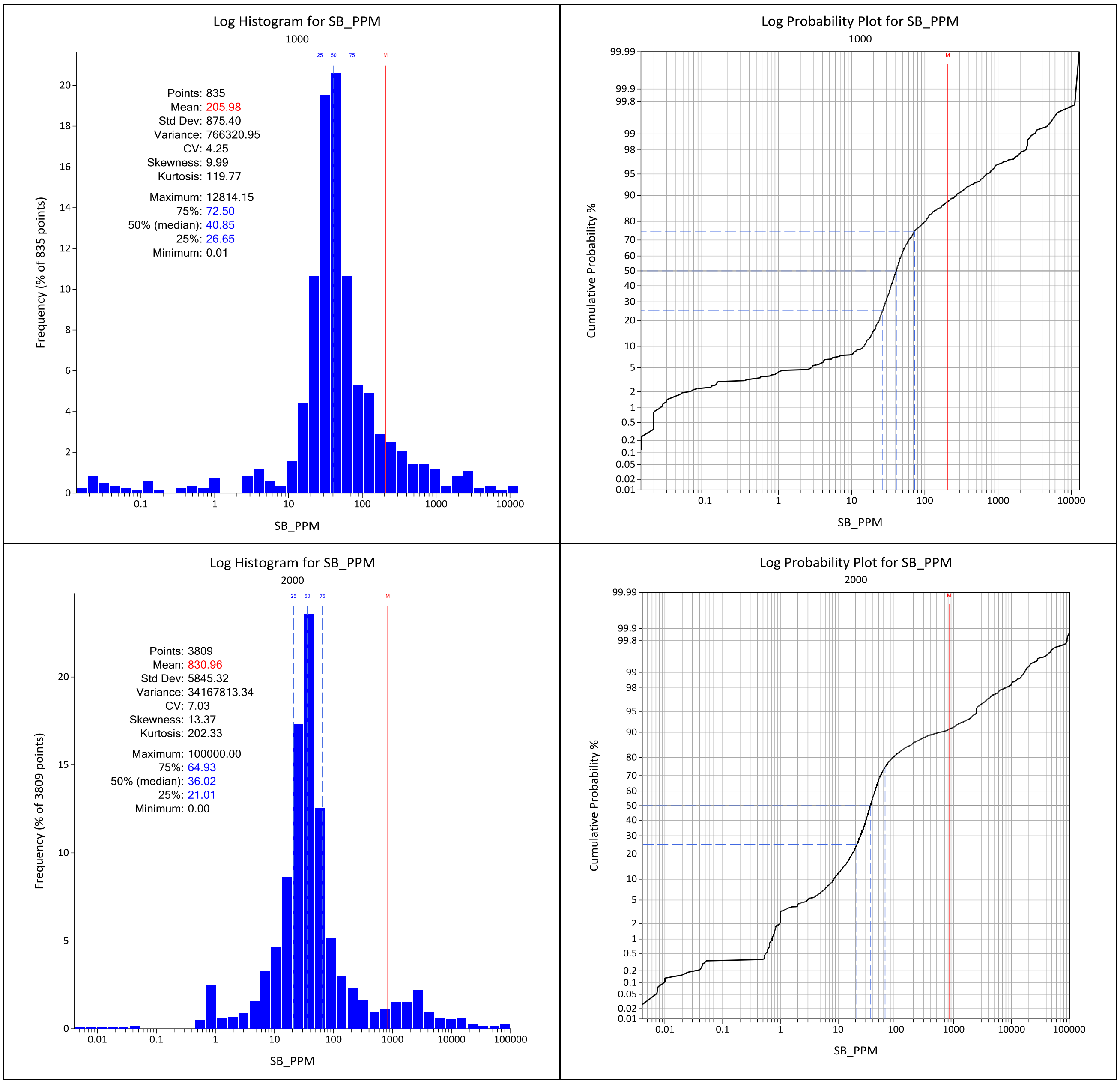
Summary statistics of composites for antimony for Buffalo Reef North (BRN, MINZONE=1000) and Buffalo Reef South-Central-Felda (BRSCF, MINZONE=2000) are presented in Table 14.25 and log histograms and log probability plots are presented in Figure 14.31. The statistics show that the Buffalo Reef mineralisation has a strongly positively skewed antimony grade distribution with a very high CV. Additionally, the statistics show mixed populations, with a low-grade population centred on approximately 40 ppm Sb and a high-grade population centred on approximately 2,500 ppm Sb (especially for BRSCF). The high antimony grades relate to samples with significant stibnite (Sb_2S_3).

Due to the very high CV (CV>>2) in both the mineralised domains and the waste domain, Snowden elected to use MIK to estimate the block antimony grades.

Table 14.25 Summary antimony statistics for BRN and BRSCF mineralised composites

Statistic		BRN MINZONE 1000 Sb (ppm)	BRSCF MINZONE 2000 Sb (ppm)
Samples		835	3,809
Minimum		0.01	0.01
Maximum		12,814	100,000
Mean		206	831
Standard deviation		875	5,845
CV		4.25	7.03
Percentiles:	10%	15	8
	20%	24	17
	30%	29	24
	40%	34	30
	50%	41	36
	60%	48	44
	70%	61	54
	80%	102	89
	90%	271	569
	95%	703	2,500
	97.5%	1,895	5,491
	99%	3,107	16,702

Figure 14.31 Log histogram and log probability plot for antimony for BRN (top) and BRSCF (bottom) mineralised combined domains



Source: Snowden

Sulphur

Summary statistics of composites for sulphur within the fresh mineralised domains are shown in Table 14.26.

Table 14.26 Summary sulphur statistics for BRSCF and BRN mineralised composites

Statistic		BRSCF MINZONE 2000 S (%) Fresh	BRN MINZONE 1000 S (%) Fresh
Samples		1,262	85
Minimum		0.005	0.054
Maximum		4.854	2.478
Mean		0.797	0.619
Standard deviation		0.663	0.450
CV		0.833	0.400
Percentiles:	10%	0.137	0.104
	20%	0.227	0.150
	30%	0.334	0.209
	40%	0.470	0.306
	50%	0.614	0.450
	60%	0.805	0.539
	70%	1.026	0.768
	80%	1.282	1.129
	90%	1.678	1.465
	95%	2.007	1.682
	97.5%	2.432	2.328

Multivariate statistics

Correlation matrices for BRN and BRSCF are presented in Table 14.27 and Table 14.28 respectively. The tables show similar relationships between gold, arsenic and antimony for the two areas, although the correlations are slightly stronger at BRSCF. The matrices show a weak to moderate correlation between gold and arsenic and between gold and antimony, which relates to the association of gold with arsenopyrite and stibnite. The correlations are shown in scatterplots (log-scale) in Figure 14.32.

The scatterplots show that typically, high-Sb is associated with high-Au; however, high-Au is not always associated with high-Sb, which is likely caused by two phases of mineralisation.

The correlation between sulphur and gold is low to very low in the oxide zone, increasing with depth for the transition zone and moderate in the fresh, sulphide material. The correlation (Figure 14.23) is clearer at the Buffalo Reef deposits, especially BRC/Felda where there are more sulphur assays compared to Selinsing, although the coverage of sulphur is still considered poor.

Table 14.27 Correlation matrix – BRN

Variable	Au	As	Sb	S
Au	1.00	0.37	0.20	
As	0.37	1.00	0.08	
Sb	0.20	0.08	1.00	
S				1.00

Table 14.28 Correlation matrix – BRSCF

Variable	Au	As	Sb	S
Au	1.00	0.40	0.25	0.62
As	0.40	1.00	0.18	0.69
Sb	0.25	0.18	1.00	0.49
S	0.62	0.69	0.49	1.00

Figure 14.32 Scatterplots between Au-As and Au-Sb, for BRN (top) and BRSCF (bottom)

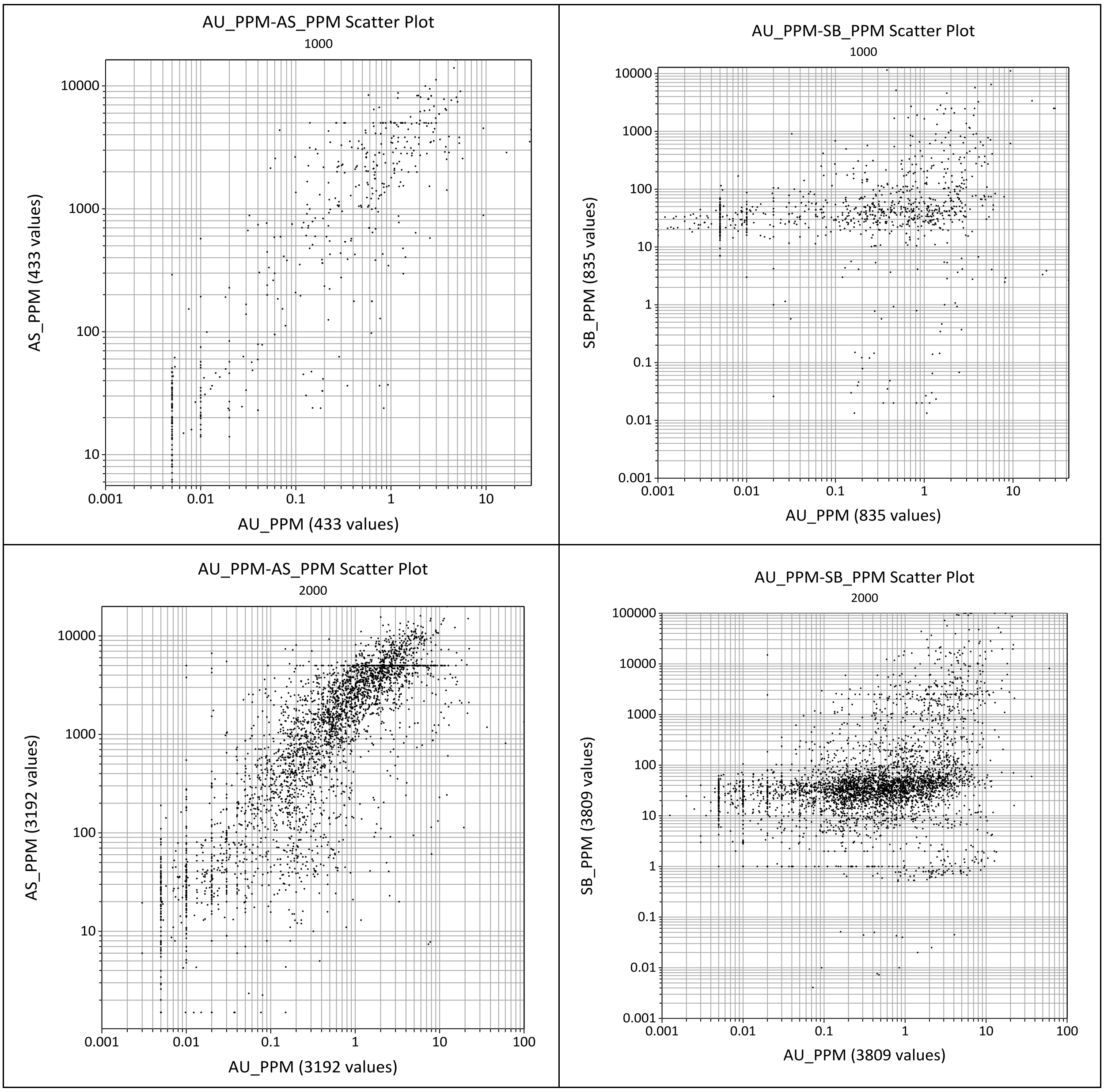
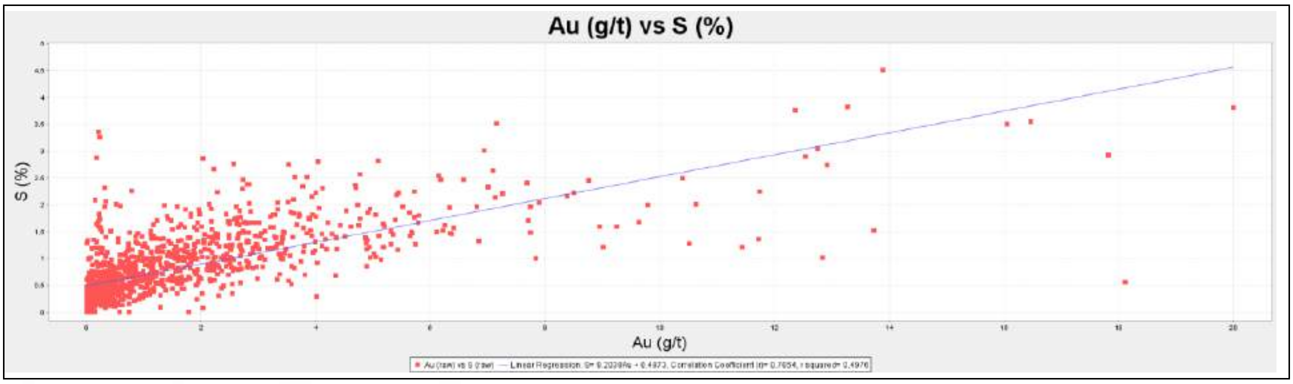


Figure 14.33 Scatterplot for Au (g/t) vs S (%) composites for BRCSF area, fresh material (Au g/t cut at maximum 20 g/t)



Top-cuts

Top-cuts were applied to gold and arsenic to minimise the impact of extreme grades on the local block grade estimates. The top-cuts were selected based on an assessment of probability plots and histograms, along with the impact of the top-cut on the mean grade and CV. No top-cuts were applied to the antimony assay data for the Buffalo Reef estimate due to the application of MIK.

For the mineralised domains, top-cuts were only applied to the gold grades. Due to the low CV, no top-cuts were applied to the arsenic or sulphur grades within the mineralised domains.

The top-cuts are summarised in Table 14.29.

Table 14.29 Buffalo Reef top-cut values

MINZONE	Variable	Top-cut value	No. of samples	Raw data		Top-cut data		Number cut	Percent cut
				Mean	CV	Mean	CV		
0	Au	0.5	40,117	0.02	4.40	0.02	1.78	86	0.2%
0	As	500	14,245	77	4.59	52	1.86	359	2.5%
1000	Au	8	1,806	0.95	2.25	0.87	1.54	19	1.0%
2000	Au	20	6,005	1.31	2.04	1.29	1.71	11	0.2%

14.5.3 Variography

Gold and arsenic

Variograms for gold and arsenic for the mineralised domains were modelled based on the following general approach:

- Variograms were modelled for individual lodes where enough data was available to enable a reasonable variogram to be modelled.
- For BRN, a variogram was generated from all combined lodes due to the limited continuity.
- For lodes with insufficient data to generate a reliable variogram, the variogram model from the nearest lode was applied.
- Variograms were only modelled for the mineralised domains; for the waste domain, the variogram from the main BRSCF lode was applied.
- All variograms were standardised to a sill of one.
- The nugget effect was modelled from the true downhole variogram.
- Variograms were modelled using one, two or three nested, spherical structures.
- For BRSCF, the variograms were evaluated using a normal scores transform due to the strong positive skewness of the grade distributions. For BRN, due to the limited grade continuity, a median indicator variogram was modelled.

The maximum and intermediate directions of continuity (directions 1 and 2 respectively) were aligned with the overall strike and down dip directions respectively. The minor direction of continuity (direction 3) was aligned in the true thickness direction.

The variogram models for gold and arsenic are summarised in Table 14.32 and the gold variogram for the main BRSCF lode (201) is provided in Figure 14.34. The median indicator gold variogram for the combined BRN domain is provided in Figure 14.35.

Antimony

Indicator variograms for antimony were generated for the combined mineralised domain for BRSCF (MINZONE 2000). Indicator variograms for BRN were attempted but not deemed to be reliable and as such, the variogram models from BRSCF were applied to the BRN antimony estimate, with the threshold grades adjusted as per Table 14.30. Additionally, the indicator variogram models from the BRSCF mineralised domain were applied to the waste domain.

Table 14.30 Indicator variogram thresholds and variogram model mapping

Threshold percentile	Mineralised (BRSCF) MINZONE=2000		Mineralised (BRN) MINZONE=1000		Waste MINZONE=0	
	Sb (ppm)	Variogram reference no.	Sb (ppm)	Variogram reference no.	Sb (ppm)	Variogram reference no.
10%	8	1	15	1	1	1
20%	17	2	24	2	5	2
30%	24	3	29	3	9	3
40%	30	4	34	4	13	4
50%	36	5	41	5	17	5
60%	44	6	48	6	22	6
70%	54	7	61	7	27	7
80%	89	8	102	8	33	8
90%	570	9	270	9	43	9
95%	2,500	10	700	10	55	10
97.5%	5,500	11	1,900	11	79	11
99%	16,700	12	3,100	12	150	12

Variograms for the combined BRSCF mineralised domain were modelled based on the following general approach:

- All variograms were standardised to a sill of one
- The nugget effect was modelled from the true downhole variogram
- Variograms were modelled using three nested, spherical structures
- The variograms were evaluated using indicator variograms for a total of 12 grade thresholds.

The maximum and intermediate directions of continuity (directions 1 and 2 respectively) were aligned with the overall strike and down-dip directions respectively. The minor direction of continuity (direction 3) was aligned in the true thickness direction. The variogram directions did not change for different thresholds (i.e. no rotating anisotropy).

The indicator variogram models for antimony for the BRSCF mineralised domain are summarised in Table 14.33.

Upper and lower tail modelling

The upper and lower tails of the antimony grade distributions, above the 99th percentile and below the 10th percentile, were modelled using a hyperbolic and power model respectively. Grades were modelled up to the maximum composite grade for the domains, as per Table 14.25. The upper and lower tail model parameters are summarised in Table 14.31 for each domain.

Table 14.31 Distribution tail modelling – Buffalo Reef

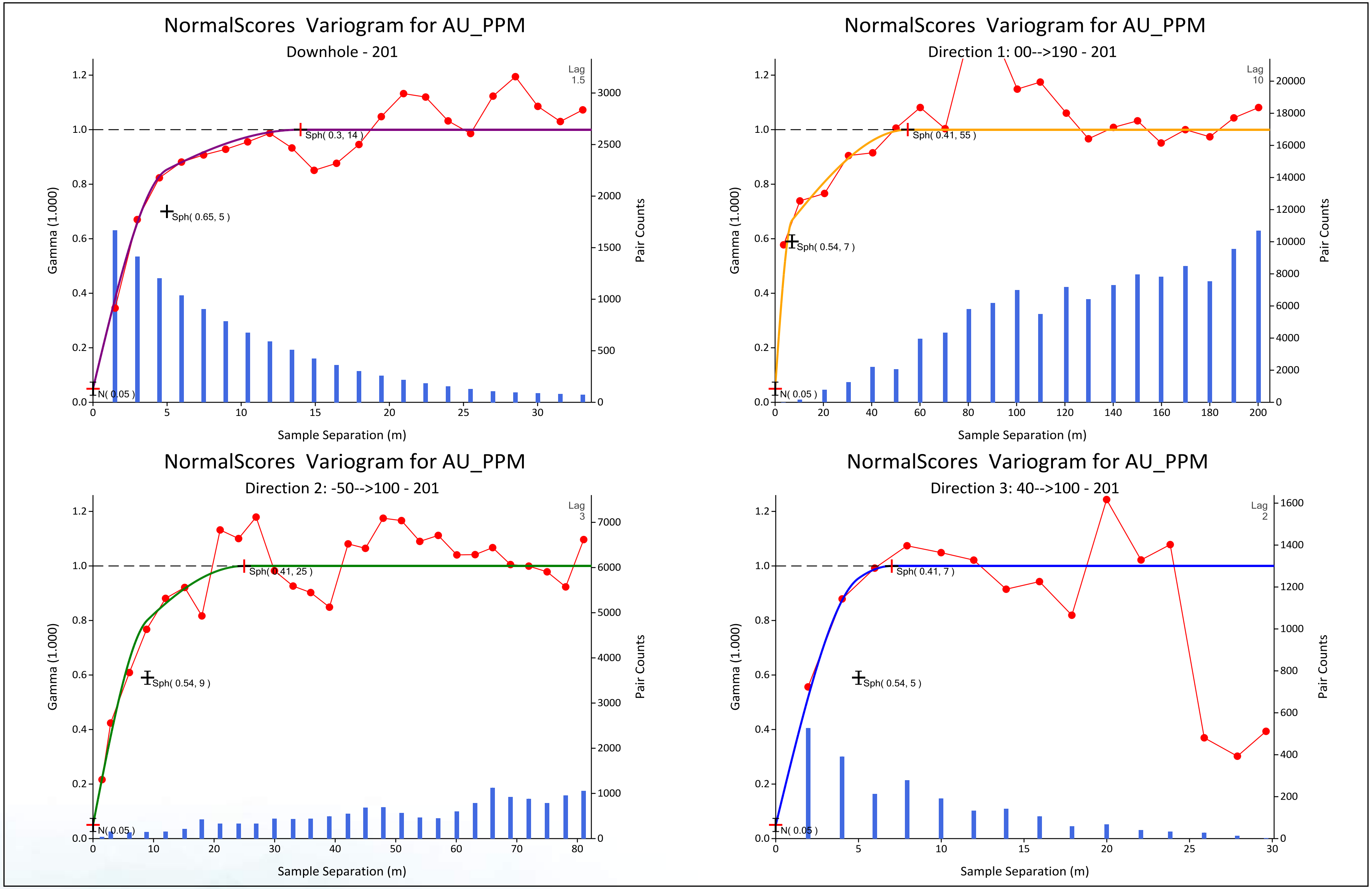
Domain (MINZONE)	Tail	Model type	Model parameter
0	Lower	Power	0.60
	Upper	Hyperbolic	1.10
1000	Lower	Power	0.40
	Upper	Hyperbolic	1.20
2000	Lower	Power	0.80
	Upper	Hyperbolic	1.50

Table 14.32 Gold and arsenic variogram models for the BRSCF and BRN mineralised domains

Variable	LODE	Directions			Nugget	1st spherical variogram structure				2nd spherical variogram structure				3rd spherical variogram structure			
		Dir1	Dir2	Dir3		Sill	Range Dir1	Range Dir2	Range Dir3	Sill	Range Dir1	Range Dir2	Range Dir3	Sill	Range Dir1	Range Dir2	Range Dir3
Au	BRN	00→175	-70→085	20→085	0.15	0.63	8	8	4	0.22	55	20	6	-	-	-	-
Au	201	00→190	-50→100	40→100	0.12	0.65	7	9	5	0.23	55	25	7	-	-	-	-
Au	210	00→180	-50→090	40→090	0.10	0.90	50	50	5	-	-	-	-	-	-	-	-
Au	212	00→180	-55→090	35→090	0.33	0.67	40	12	12	-	-	-	-	-	-	-	-
Au	213	00→175	-50→085	40→085	0.19	0.81	55	35	6	-	-	-	-	-	-	-	-
Au	214	00→175	-55→085	35→085	0.10	0.90	40	25	5	-	-	-	-	-	-	-	-
Au	220	00→180	-55→090	35→090	0.11	0.54	8	5	4	0.35	85	25	17	-	-	-	-
Au	221	00→180	-50→090	40→090	0.10	0.62	15	5	2	0.28	70	20	9	-	-	-	-
As	BRN	00→175	-70→085	20→085	0.06	0.47	5	6	6	0.31	12	12	8	0.16	60	20	10
As	201	00→190	-50→100	40→100	0.07	0.35	5	5	4	0.59	45	45	11	-	-	-	-
As	212	00→180	-55→090	35→090	0.21	0.31	20	80	9	0.48	125	105	12	-	-	-	-
As	213	00→175	-50→085	40→085	0.1	0.9	60	25	7	-	-	-	-	-	-	-	-
As	214	00→175	-55→085	35→085	0.09	0.91	40	25	7	-	-	-	-	-	-	-	-
As	220	00→180	-55→090	35→090	0.22	0.49	9	8	8	0.29	60	13	13	-	-	-	-
As	221	00→180	-50→090	40→090	0.18	0.26	60	4	9	0.56	70	40	11	-	-	-	-

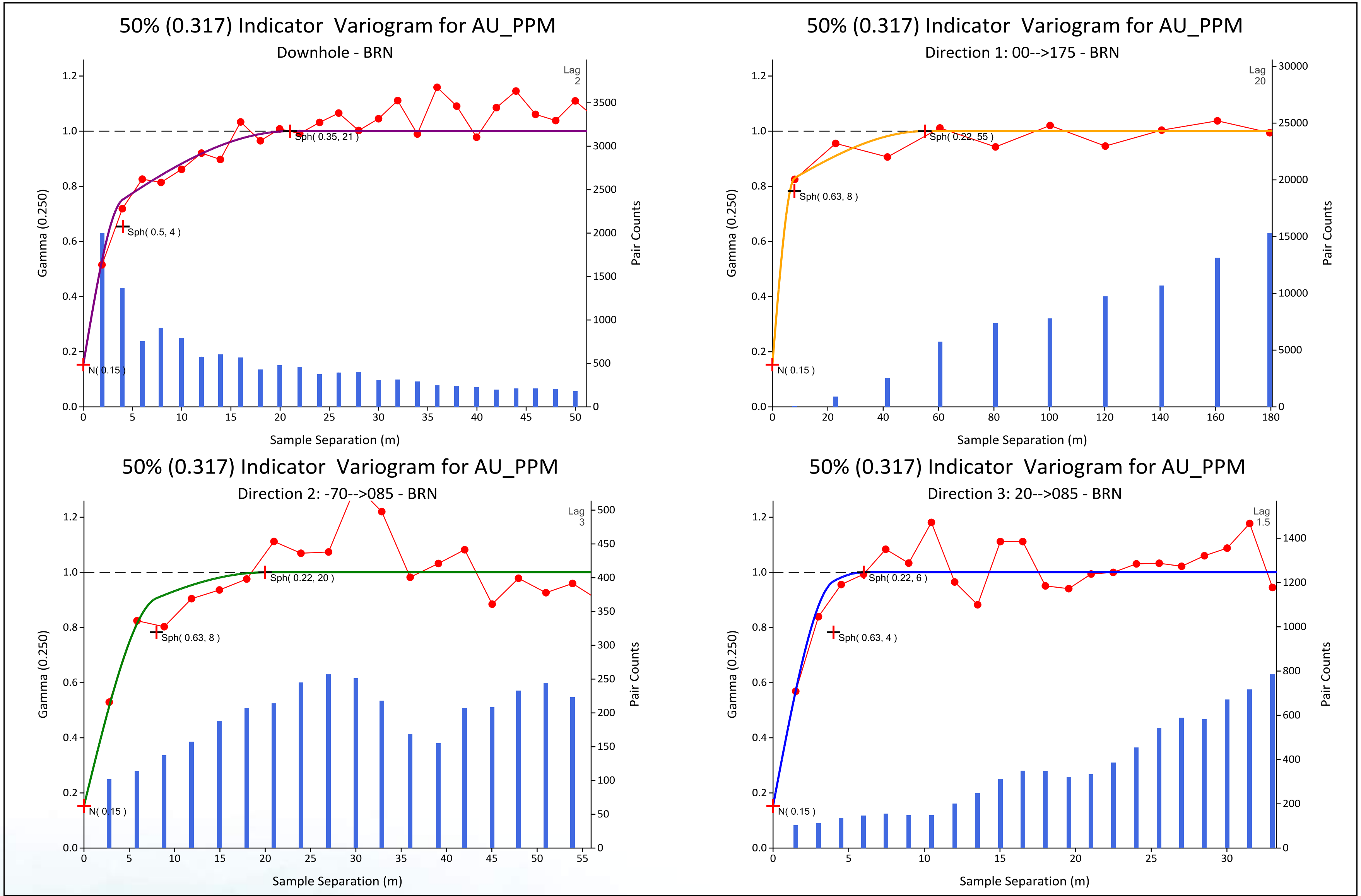
Note: Lode numbers 201 to 221 are from BRSCF; BRN Au variogram is median indicator, all others are back-transformed normal scores variograms

Figure 14.34 Normal scores variogram model for Au for main BRSCF mineralised domain



Source: Snowden

Figure 14.35 Median indicator variogram model for Au for combined BRN mineralised domains



Source: Snowden

Table 14.33 Antimony indicator variogram models for the BRSCF mineralised domain (MINZONE 2000)

Threshold	Variogram ref. no.	Directions			Nugget	1st spherical variogram structure				2nd spherical variogram structure				3rd spherical variogram structure			
		Dir1	Dir2	Dir3		Sill	Range Dir1	Range Dir2	Range Dir3	Sill	Range Dir1	Range Dir2	Range Dir3	Sill	Range Dir1	Range Dir2	Range Dir3
10%	1	00→190	-50→100	40→100	0.12	0.15	20	45	15	0.26	70	135	67	0.47	650	150	115
20%	2	00→190	-50→100	40→100	0.12	0.15	20	45	15	0.26	70	135	67	0.47	500	150	115
30%	3	00→190	-50→100	40→100	0.15	0.20	20	30	15	0.27	70	110	67	0.38	400	150	100
40%	4	00→190	-50→100	40→100	0.15	0.25	15	10	12	0.29	70	85	60	0.31	360	135	65
50%	5	00→190	-50→100	40→100	0.15	0.25	15	10	6	0.32	35	30	48	0.28	280	100	55
60%	6	00→190	-50→100	40→100	0.15	0.25	10	10	5	0.36	35	20	30	0.24	175	65	55
70%	7	00→190	-50→100	40→100	0.20	0.20	10	10	5	0.38	30	20	15	0.22	90	30	42
80%	8	00→190	-50→100	40→100	0.20	0.25	10	10	5	0.34	20	12	16	0.21	30	15	17
90%	9	00→190	-50→100	40→100	0.24	0.21	10	8	5	0.34	12	10	7	0.21	15	13	10
95%	10	00→190	-50→100	40→100	0.24	0.21	7	3	3	0.34	10	5	4	0.21	13	7	5
97.5%	11	00→190	-50→100	40→100	0.24	0.21	7	3	2	0.34	10	5	3	0.21	12	6	4
99%	12	00→190	-50→100	40→100	0.28	0.17	7	3	2	0.34	9	4	3	0.21	11	5	4

14.5.4 Block model and grade estimation

Kriging neighbourhood analysis

A KNA was performed for the Buffalo Reef deposit using Snowden Supervisor software to optimise and validate various kriging parameters, based on the variogram for gold for the main BRSCF lode. The KNA assesses the impact of the kriging parameters on the kriging efficiency and slope of regression statistics. The main aim of a KNA is to assess the level of conditional bias (i.e. degree of over-smoothing) induced by various kriging parameters such as the parent block size, number of informing samples and search ellipse dimensions.

Snowden used the results of the KNA to verify the choice of parent block size, number of informing samples and the search ellipse radii. Based on the KNA results, along with consideration of the geometry of the mineralisation and the current open-pit bench height at Selinsing of 2.5 m, the following parameters were selected:

- Parent block size of 8 mE x 20 mN x 2.5 mRL. A slightly smaller block size of 8 mE was selected for Buffalo Reef due to the more selective nature of the geological interpretation and to ensure reasonable volume resolution.
- A minimum of 10 samples and maximum of 40 samples for the initial search pass.
- Search ellipse radii of 10 m in the thickness direction by 50 m along strike x 25 m down dip for the initial search pass.

Volume model construction

The block model extents for the Buffalo Reef deposit, along with parent and sub-cell sizes are listed in Table 14.34. The Buffalo Reef block model was originally constructed by Snowden (2016) using Datamine Studio 3 software. Subsequently, Monument converted the Buffalo Reef model from Datamine to Surpac format. The Buffalo Reef model was split into two separate models – BRSCF and BRN. The converted Surpac models were validated by Monument to ensure there was no impact due to the conversion.

Table 14.34 Buffalo Reef block model settings (local mine grid)

Model setting	Value
Origin – X	300
Origin – Y	2,500
Origin – Z	250
Maximum – X	1,204
Maximum – Y	6,000
Maximum – Z	650
Parent cell size – X	8
Parent cell size – Y	20
Parent cell size – Z	2.5
Minimum cell size – X	2
Minimum cell size – Y	5
Minimum cell size – Z	1.25
Rotation	None

Block model coding

The block model was coded based on wireframe surfaces and/or solids of the gold mineralisation and oxidation zones, along with the February 2016 end of month topographical surface and a solid defining the extent of Old Tailings material.

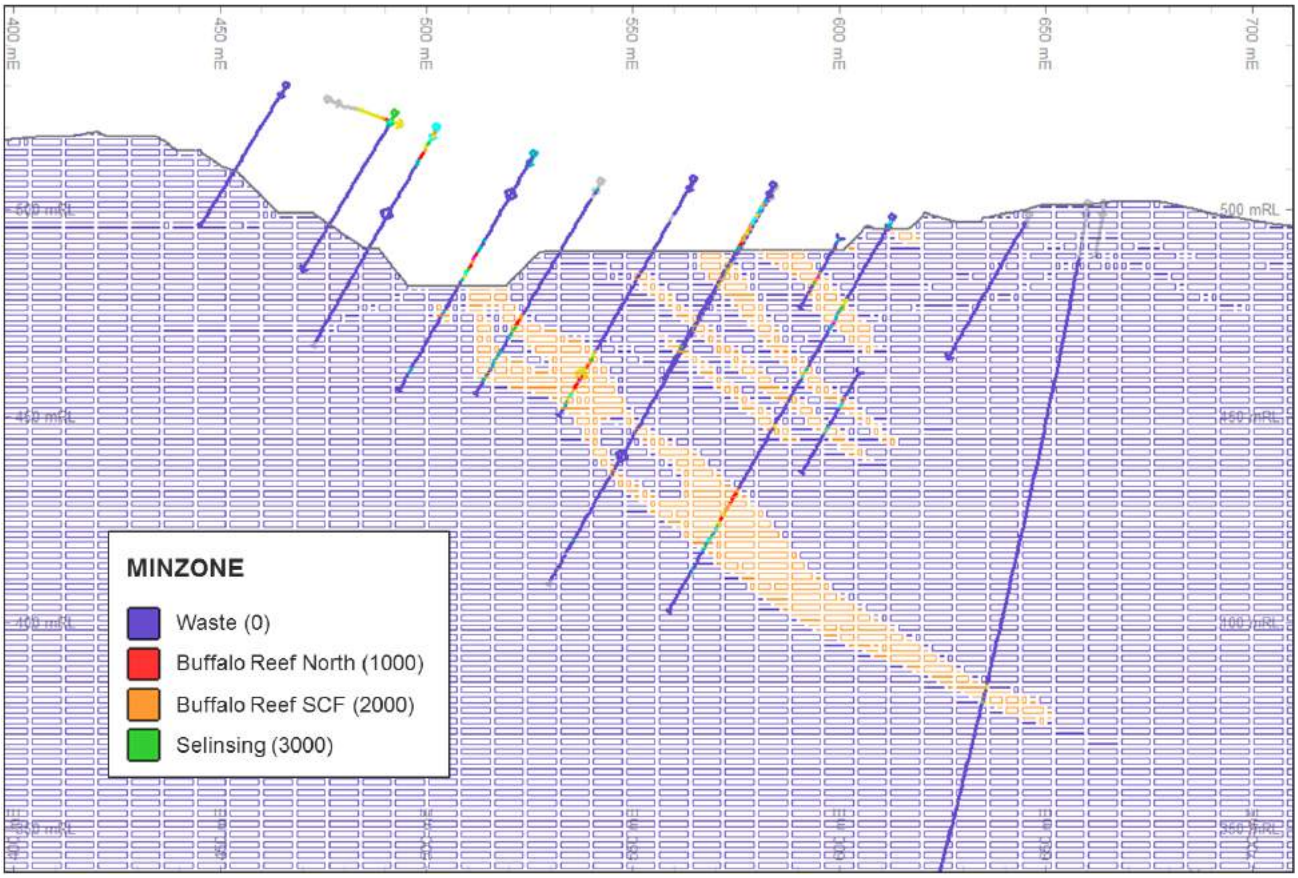
Mineralisation

The gold mineralisation was coded using a field called MINZONE, with individual lodes coded using a field called LODE. The LODE numbers are based on the wireframe numbers. Field codes are defined in Table 14.35 and an example west-east cross section showing the MINZONE coding is shown in Figure 14.36.

Table 14.35 MINZONE and LODE field coding

Field	Value	Description
MINZONE	0	Waste
	1000	Mineralisation (BRN)
	2000	Mineralisation (BRSCF)
LODE	0	Waste
	101 to 199	BRN – mineralised zone
	201, 220	BRSCF – main mineralised zones
	202 to 219, 221 to 299	BRSCF – minor mineralised zones
	999	Old tailings material

Figure 14.36 Example west-east section (3400 mN) showing block model MINZONE coding



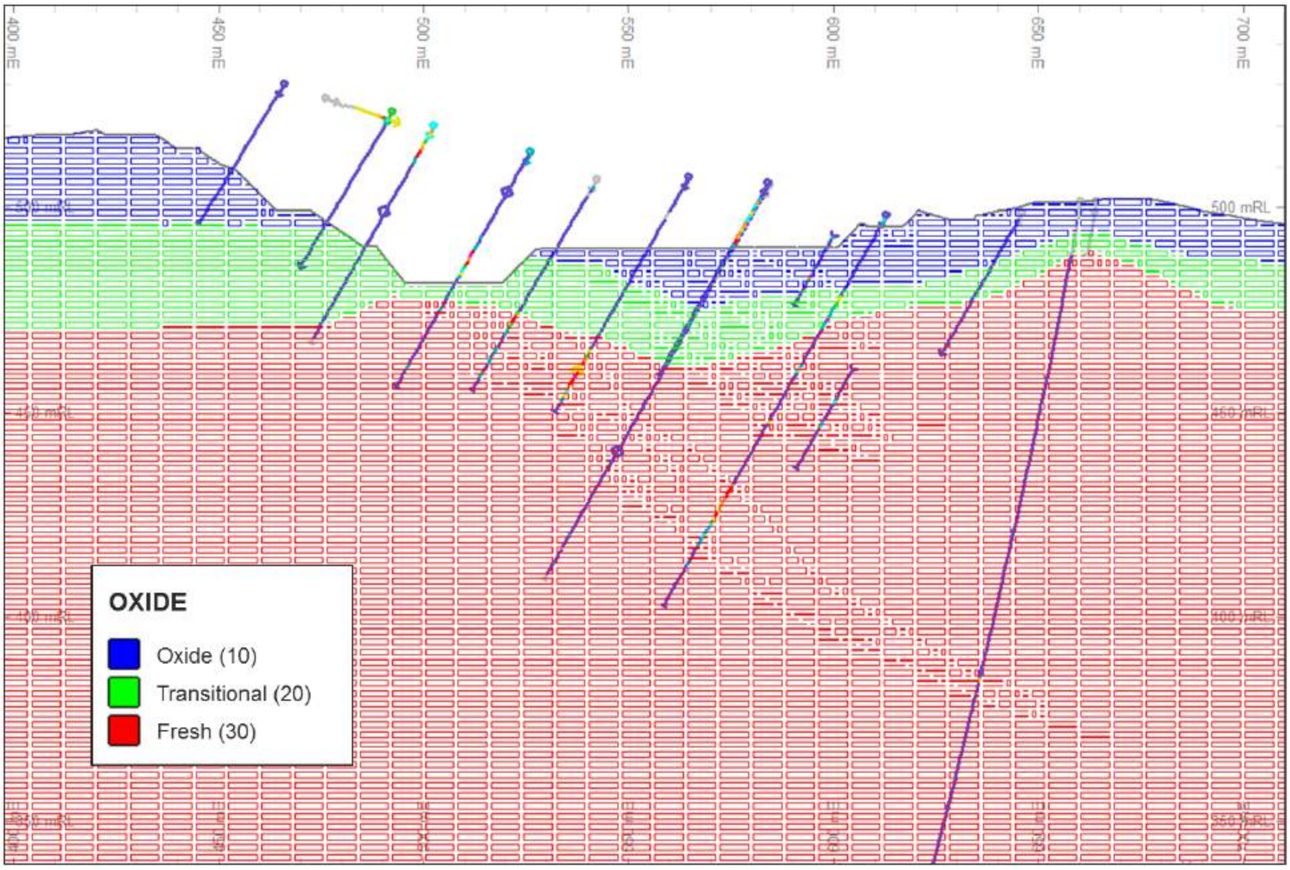
Oxidation

The oxidation zones were coded using a field called OXIDE. Field codes are defined in Table 14.36 and an example west-east cross section showing the OXIDE coding is shown in Figure 14.37.

Table 14.36 OXIDE field coding

Field	Value	Description
OXIDE	10	Oxide zone
	20	Transitional zone
	30	Fresh (sulphide) zone

Figure 14.37 Example west-east section (3400 mN) showing block model OXIDE coding



Tailings

The extent of the Old Tailings material was coded as per Section 14.4.4. The tailings material only impacts a small portion of the south-eastern region of the Buffalo Reef model.

Grade estimation methodology – Au (g/t), Sb (ppm) and As (ppm)

For Buffalo Reef, gold and arsenic grades were estimated using OK, with top-cuts applied to control the influence of extreme grades on the local block estimates. Due to the strongly skewed nature of the antimony grades (CV>>2) in both the mineralised domain and the waste domain, along with mixed populations, Snowden elected to use MIK to estimate the block antimony grades. The antimony MIK estimate was compiled using a total of 12 grade thresholds, based on the population deciles (10%, 20%,…, 80%, 90%) with additional thresholds at the 95%, 97.5% and 99% included to model the higher grade portion of the distributions.

Datamine Studio 3 software was used to estimate the gold and arsenic grades of the individual lodes using OK using the gold and arsenic grade fields in the drillhole file. The results were written to fields called AU and AS respectively. The LODE field was used to constrain the OK estimation with hard boundaries between all domains. All estimates were parent cell estimates. As previously discussed, antimony was estimated using MIK, with the results written to a field called SB_MIK. The MINZONE field was used to constrain the MIK estimation with hard boundaries between all domains. The MINZONE field was used, rather than the LODE field, due to the large number of lodes, some of which have very few samples.

Due to the variable dip of the Buffalo Reef mineralisation, dynamic anisotropy was used to locally adjust the orientation of the search ellipse and variogram models. The mineralisation wireframes were used to create a point file where each point relates to a triangle centroid and contains the true dip and true dip direction of the wireframe triangle. All points related to the edges of the wireframes were manually removed to avoid anomalies in these areas. This point file was then used to estimate the local true dip and dip direction into the block model for each block. The estimates of true dip and dip direction were subsequently used to locally adjust the variogram and search orientations during the OK and MIK grade estimation.

The POSTIK process in the GSLIB suite of software was used for post-processing the MIK output from Datamine to enable order relation corrections to be applied and to allow the skewed tails of the gold grade populations to be modelled and used as part of the estimation process. The upper tail was modelled with a hyperbolic function between the last indicator cut-off and the maximum composite grade, while the lower tail was modelled with a power function below the first indicator cut-off (Table 14.31). The final MIK product for antimony was an e-type estimate (i.e. average grade of the block based on the MIK probability estimates) which was subsequently imported back into Datamine. No change of support was applied to the MIK estimates as only an e-type estimate was produced.

Where no estimate could be made due to sparse data, a default value was applied as per Table 14.37. Either the median or mean values were used for the default grade depending on the nature of the grade distribution.

Table 14.37 Default grade values for un-estimated Buffalo Reef blocks

Field	MINZONE	LODE	Default value
AU	0	0	0.001
AU	1000	103	1.0
AU	1000	199	1.0
AU	2000	207	0.3
AU	2000	218	0.5
AU	2000	220	1.5
AU	2000	228	0.6
AU	2000	229	1.9
AU	2000	230	0.9
AU	2000	231	0.7
AU	2000	299	0.5
SB_MIK	0	0	17
SB_MIK	1000	101 to 199	40
SB_MIK	2000	201 to 299	35
AS	0	0	1
AS	1000	101	1400
AS	1000	102	1700
AS	1000	103	2400
AS	1000	104	2300
AS	1000	106	1500
AS	1000	107	1200
AS	1000	110	300
AS	1000	111	300
AS	1000	199	2100
AS	2000	201	2600
AS	2000	202	100
AS	2000	203	500
AS	2000	205	2200
AS	2000	206	1200
AS	2000	207	900
AS	2000	209	800
AS	2000	210	2000
AS	2000	211	2300
AS	2000	213	2900
AS	2000	214	1900
AS	2000	218	1100
AS	2000	220	2800
AS	2000	225	2300
AS	2000	226	100
AS	2000	227	200
AS	2000	228	800
AS	2000	229	2800
AS	2000	230	1800
AS	2000	231	400
AS	2000	299	1300

Search neighbourhood parameters – Au (g/t), Sb (ppm) and As (ppm)

A three-pass search strategy was utilised for all grade estimates with the same search neighbourhood parameters applied to all domains. The search radii for the gold and arsenic estimates, for the first and second search passes (50 m x 25 m x 10 m), corresponds to the range of continuity for gold interpreted for both BRN and BRSCF. The search radius for antimony was reduced to 5 m in the thickness direction as antimony was estimated using the MINZONE field to constrain the estimation (due to the lower number of assays and requirement for MIK estimation), rather than the LODE field which was used for gold and arsenic.

Details of the estimation search parameters are presented in Table 14.38. The number of samples per drillhole was limited to four to ensure that a reasonable number of drillholes were used to estimate each block, with at least three drillholes within the search neighbourhood required for the first search pass and two drillholes required for the second search passes.

Table 14.38 Buffalo Reef model search neighbourhood parameters

Parameter	Value	
	Gold and arsenic	Antimony
Search ellipse rotation angles (Datamine format):		
Z axis	100	100
X axis	50	50
Z axis	0	0
Search radii (along strike x down dip x thickness):		
Search pass 1	50 m x 25 m x 10 m	50 m x 25 m x 5 m
Search pass 2	50 m x 25 m x 10 m	50 m x 25 m x 5 m
Search pass 3	100 m x 50 m x 20 m	100 m x 50 m x 10 m
Number of samples (minimum – maximum):		
Search pass 1	10 – 40	10 – 40
Search pass 2	5 – 40	5 – 40
Search pass 3	2 – 40	2 – 40
Maximum number of samples per drillhole	**4**	**4**

Grade estimation methodology – S (%)

Due to the poor coverage of sulphur assay data, variograms showed very poor quality and the decision was made to estimate the sulphur grade using inverse distance squared (ID^2).

Monument used the variograms for Au, model by Snowden in 2016, as a reference for the search ellipse orientation and ranges, for both fresh and transition domains. Based on this, the following parameters were selected for the sulphur estimation:

- A minimum of two samples and maximum of 24 samples for the initial search pass.
- BRN: Search ellipse radii of 11 m in the thickness direction by 100 m along strike x 36 m down dip for the initial search pass.
- BRSCF: Search ellipse radii of 20 m in the thickness direction by 100 m along strike x 50 m down dip for the initial search pass.

Sulphur for the fresh sulphide zone was estimated using a hard boundary with only sulphide samples coded within the mineralisation envelopes used. The search ellipse anisotropy was based on the parameters used for the Au estimation, with the maximum direction oriented along the north-south strike, the intermediate direction oriented down dip (dipping to the east), and the minor direction across the thickness. For the transition zone, a soft boundary was used, with all samples (mineralised and unmineralised) contributing to the estimation. The search strategy for the transition zone used the same parameters as the sulphide zone. The oxide zone also used a soft boundary, with a sub-horizontal search orientation.

Model validation – Au (g/t), Sb (ppm) and As (ppm)

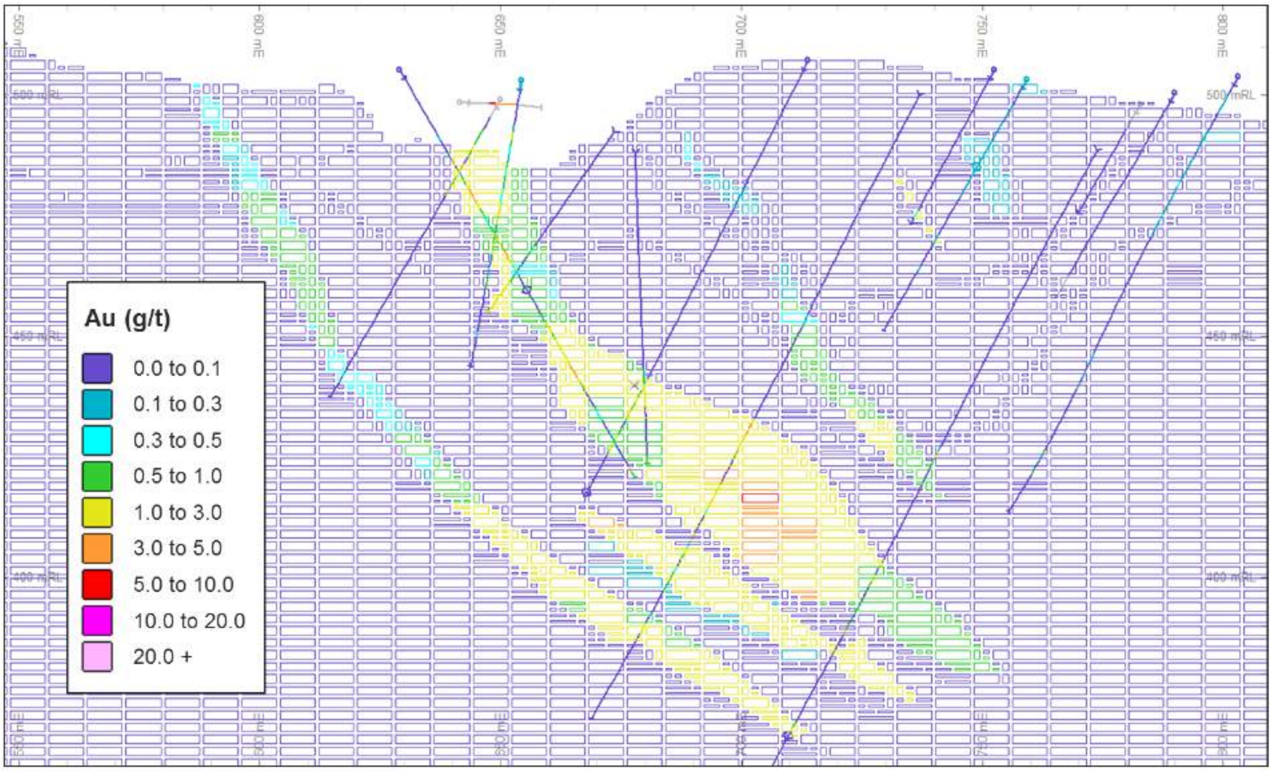
The block grade estimates were validated using:

- A visual comparison of block grade estimates and the input drillhole data
- A global comparison of the average composite (naïve and de-clustered) and estimated block grades
- Moving window averages comparing the mean block grades to the composites.

The conclusions from the model validation work are as follows:

- Visual comparison of the model grades and the corresponding drillhole grades shows a good correlation (Table 14.38).
- A comparison of the global drillhole mean grade with the mean grade of the block model estimate (for each domain and each element) shows that the block model mean gold grade is within 10% of the drillhole mean for Au and within 15% for As, which is a reasonable outcome (Table 14.39 to Table 14.41).
- For antimony, the global comparison suggests that the model overestimates the antimony grade somewhat. Analysis of grade trend plots and visual validation shows a reasonable correlation in well supported areas where the grade distribution is not overly skewed. In Snowden's opinion, due to the extreme skewness of the antimony grades, the quality of the antimony estimate is unlikely to be improved significantly without further constraining the elevated antimony areas by additional appropriate domaining (specific for antimony).
- With the exception of poorly sampled regions, the grade trend plots show a good correlation between the patterns in the block model grades compared with the drillhole grades (Figure 14.39 and Figure 14.40).

Figure 14.38 Example west-east section (4490 mN; ±10 m) showing block grade estimates at BRSCF against the input drillhole composites



Source: Snowden

Table 14.39 Buffalo Reef model validation summary statistics – gold

Statistic	Input composites		Block model estimate Au (g/t)
	Naïve Au (g/t)	De-clustered and top-cut Au (g/t)	
BRN			
Number	1,806	1,806	
Mean	**0.95**	**0.77**	**0.77**
Variance	4.61	1.55	0.26
Maximum	42.39	8.00	4.01
75%	1.06	0.86	1.04
50%	0.34	0.32	0.64
25%	0.10	0.12	0.39
Minimum	0.005	0.005	0.02
BRSCF			
Number	6,005	6,005	
Mean	**1.31**	**1.11**	**1.06**
Variance	7.19	4.06	1.13
Maximum	99.95	20.00	7.54
75%	1.48	1.19	1.73
50%	0.41	0.34	0.95
25%	0.14	0.12	0.39
Minimum	0.005	0.005	0.02

Notes: Model restricted to blocks estimated in the first and second search passes only

Table 14.40 Buffalo Reef model validation summary statistics – arsenic

Statistic	Input composites		Block model estimate As (ppm)
	Naïve As (ppm)	De-clustered As (ppm)	
BRN			
Number	433	433	
Mean	**1,959**	**1,885**	**1,949**
Variance	5,887,550	5,248,269	1,483,171
Maximum	16,293	16,293	7,407
75%	3,116	3,064	2,683
50%	990	980	1,839
25%	46	43	990
Minimum	6	6	19
BRSCF			
Number	3,192	3,192	
Mean	**2,234**	**2,068**	**2,361**
Variance	6,390,955	5,885,452	2,058,101
Maximum	19,901	19,901	10,488
75%	3,646	3,351	3,339
50%	1,328	1,139	2,238
25%	209	179	1,281
Minimum	1.5	1.5	4.9

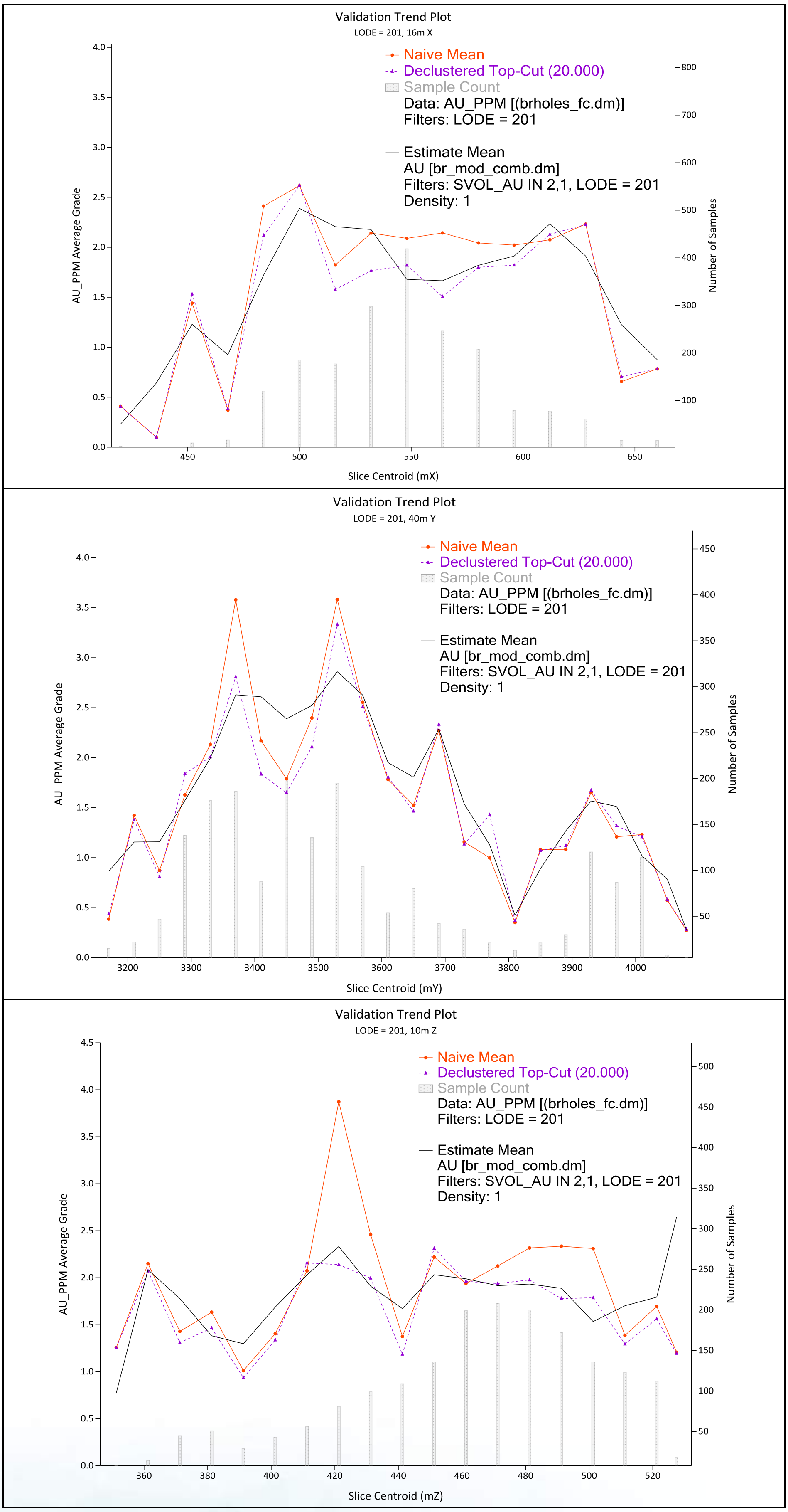
Notes: Model restricted to blocks estimated in the first and second search passes only

Table 14.41 Buffalo Reef model validation summary statistics – antimony

Statistic	Input composites		Block model estimate Sb (ppm)
	Naïve Sb (ppm)	De-clustered Sb (ppm)	
BRN			
Number	835	835	
Mean	**206**	**197**	**162**
Variance	766,321	745,858	91,877
Maximum	12,814	12,814	2,743
75%	73	61	162
50%	41	37	49
25%	27	24	31
Minimum	0.01	0.01	4
BRSCF			
Number	3,809	3,809	
Mean	**831**	**679**	**984**
Variance	34,167,813	25,007,874	5,338,990
Maximum	100,000	100,000	26,375
75%	65	59	624
50%	36	34	97
25%	21	20	32
Minimum	0.01	0.01	4

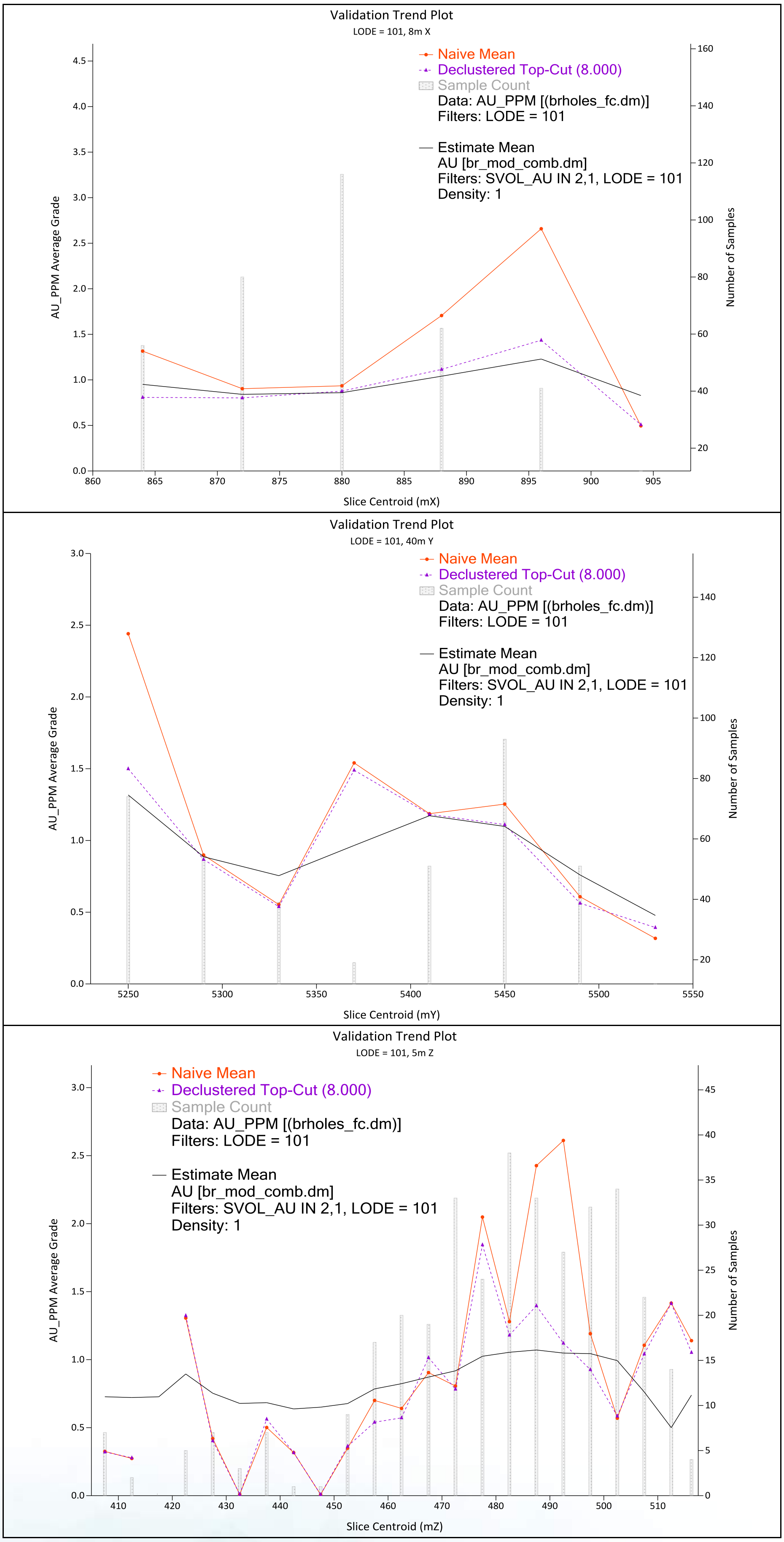
Notes: Model restricted to blocks estimated in the first and second search passes only

Figure 14.39 Validation trend plots – BRSCF main lode (LODE=201), gold



Source: Snowden

Figure 14.40 Validation trend plots – BRN main lode (LODE=101), gold



Source: Snowden

Model validation – S (%)

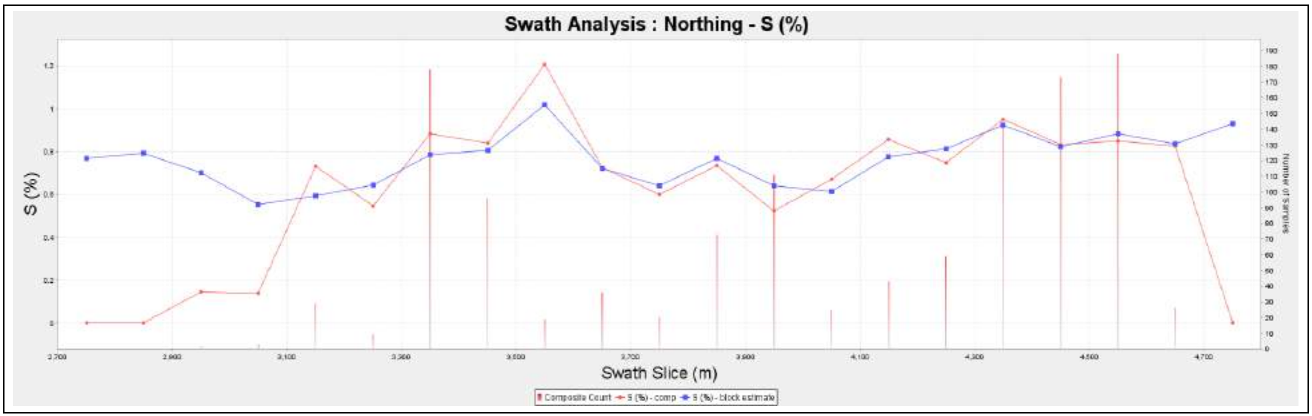
For the sulphur block estimates, validation checks were undertaken both visually and statistically. Given the irregular and generally low coverage of sulphur assay data at Buffalo Reef, the block estimates are considered to be reasonable overall, although somewhat smoothed.

A global statistical comparison shows that the block grade estimates for sulphur are within approximately 5% for BRSCF and within 15% for BRN, where there is significantly less sulphur data, (Table 14.42) for the fresh zone. Figure 14.41 shows a good matching between the sampling and block model grades for BRCSF fresh, except in the poorly sampled regions at the boundaries.

Table 14.42 Buffalo Reef model validation summary statistics – sulphur (fresh zone only)

Deposit	Input samples		Block model estimate S (%)
	Naïve S (%)	De-clustered S (%)	
BRSCF	0.797	0.754	0. 761
BRN	0.619	0.491	0. 568

Figure 14.41 Validation swath-plot for BRSCF fresh in northing direction



Cross-sections showing the block sulphur grade estimates at BRSCF and BRN are shown in Figure 14.42 and Figure 14.43.

Figure 14.42 Cross section Y=4560m +- 10m looking north showing BRSCF block S% grade estimates for sulphide blocks

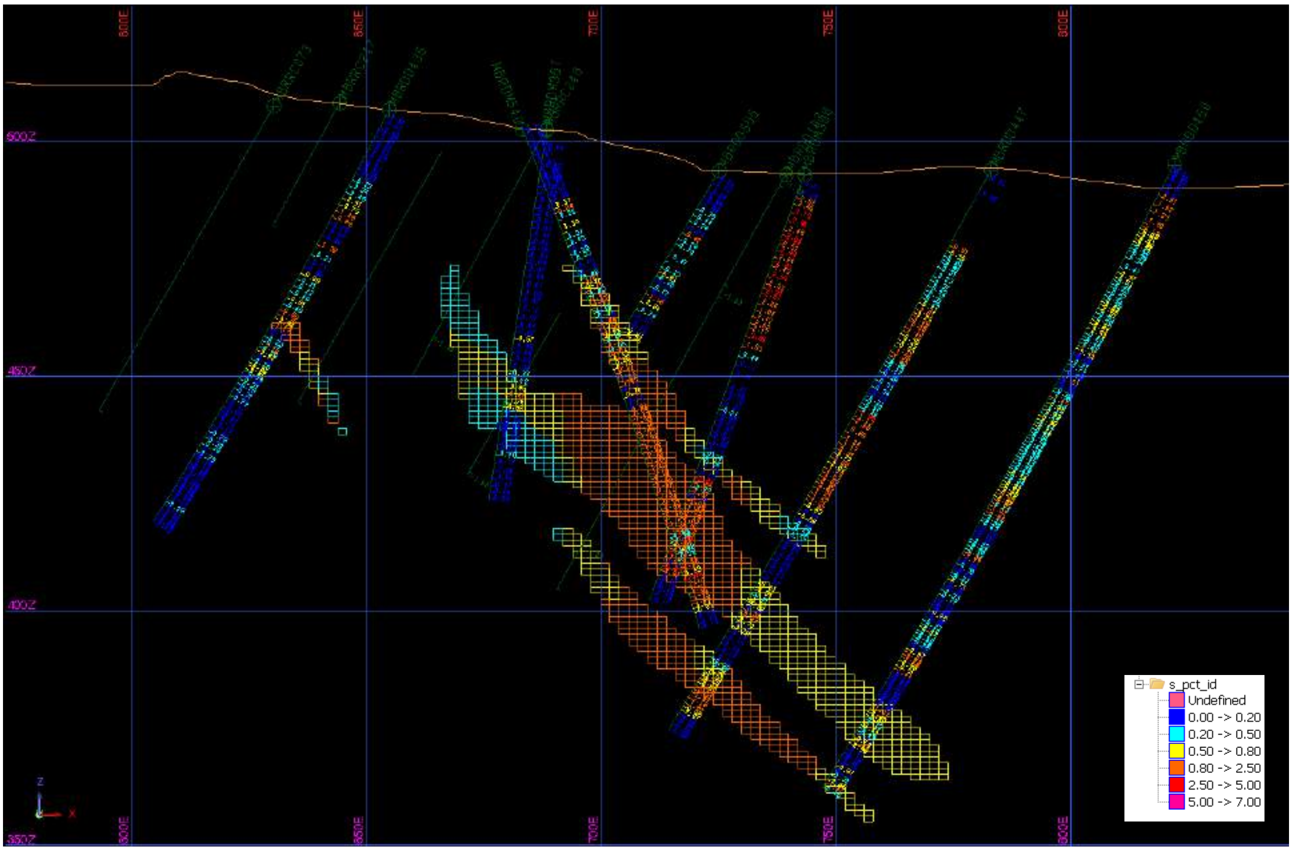
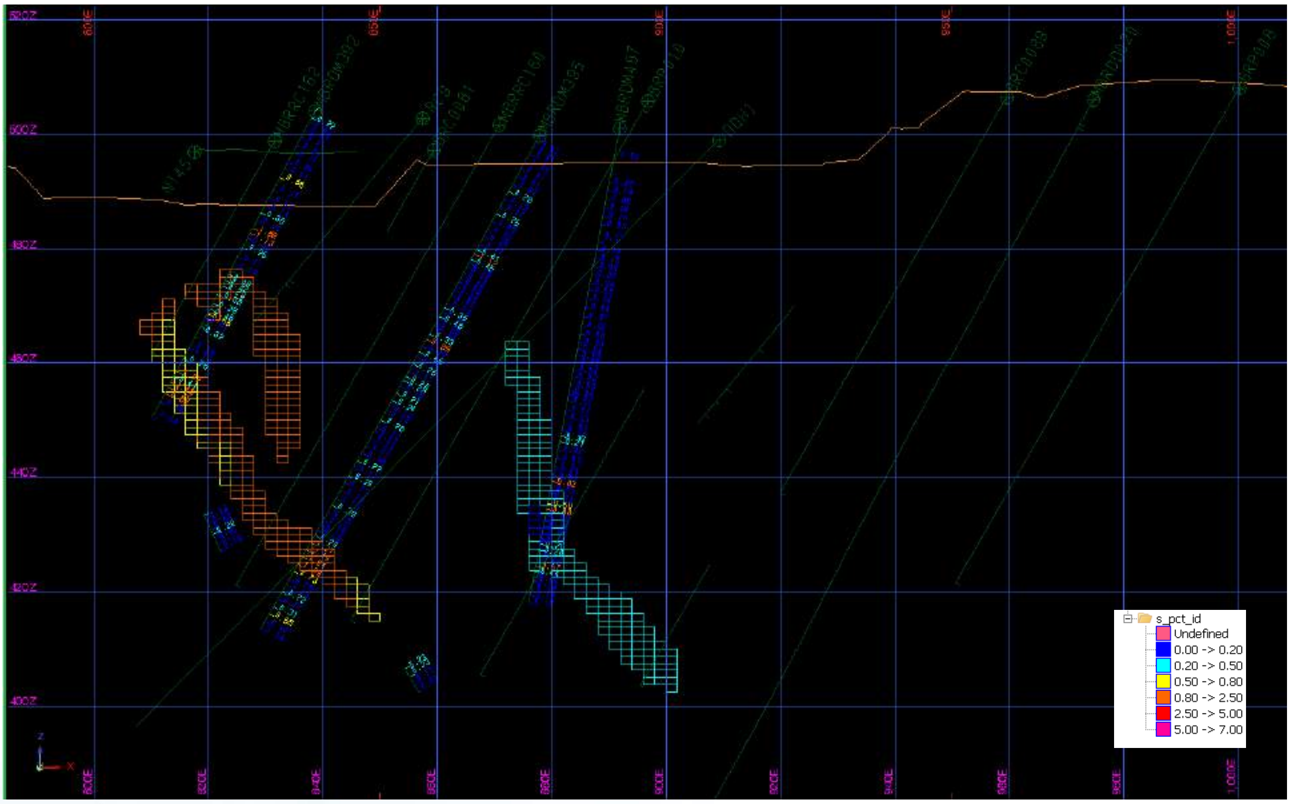


Figure 14.43 Cross section Y=550km +- 10m looking north showing BRN block S% grade estimates for sulphide blocks

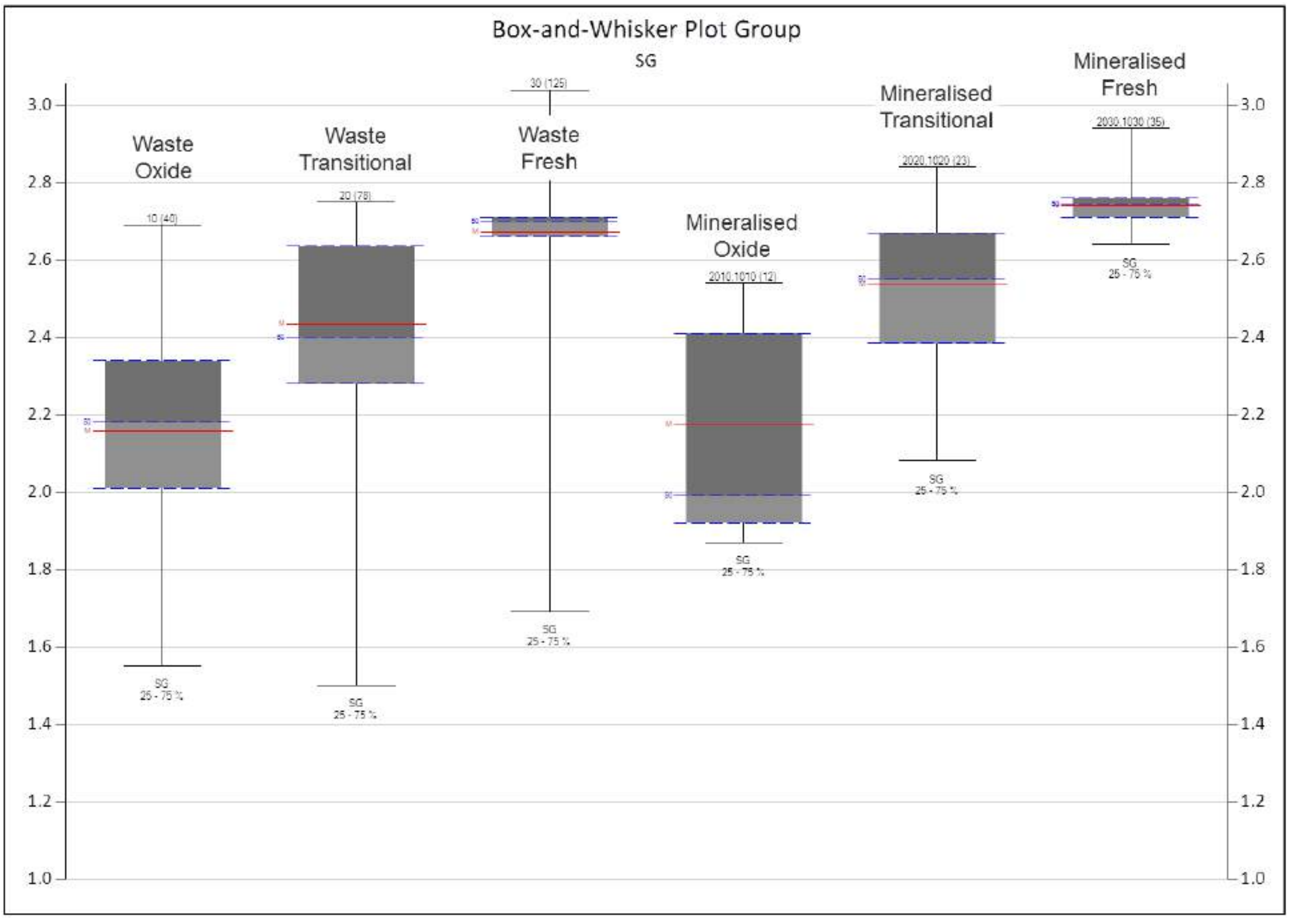


14.5.5 Bulk density analysis

Monument supplied Snowden with a spreadsheet containing 332 bulk density measurements of core from diamond drilling at Buffalo Reef. The bulk density measurements were completed using the same processes as detailed in Section 14.4.5.

Snowden imported the density data into Datamine and intersected/coded the samples with the Buffalo Reef oxidation surfaces and mineralisation wireframes. The data was then analysed to derive default values for each combination of oxidation state and mineralisation domain. A box-and-whisker plot is presented in Figure 14.44. Snowden notes that there is no correlation between the density and the gold, arsenic or antimony content.

Figure 14.44 Box-and-whisker plot of bulk density samples from Buffalo Reef



Source: Snowden

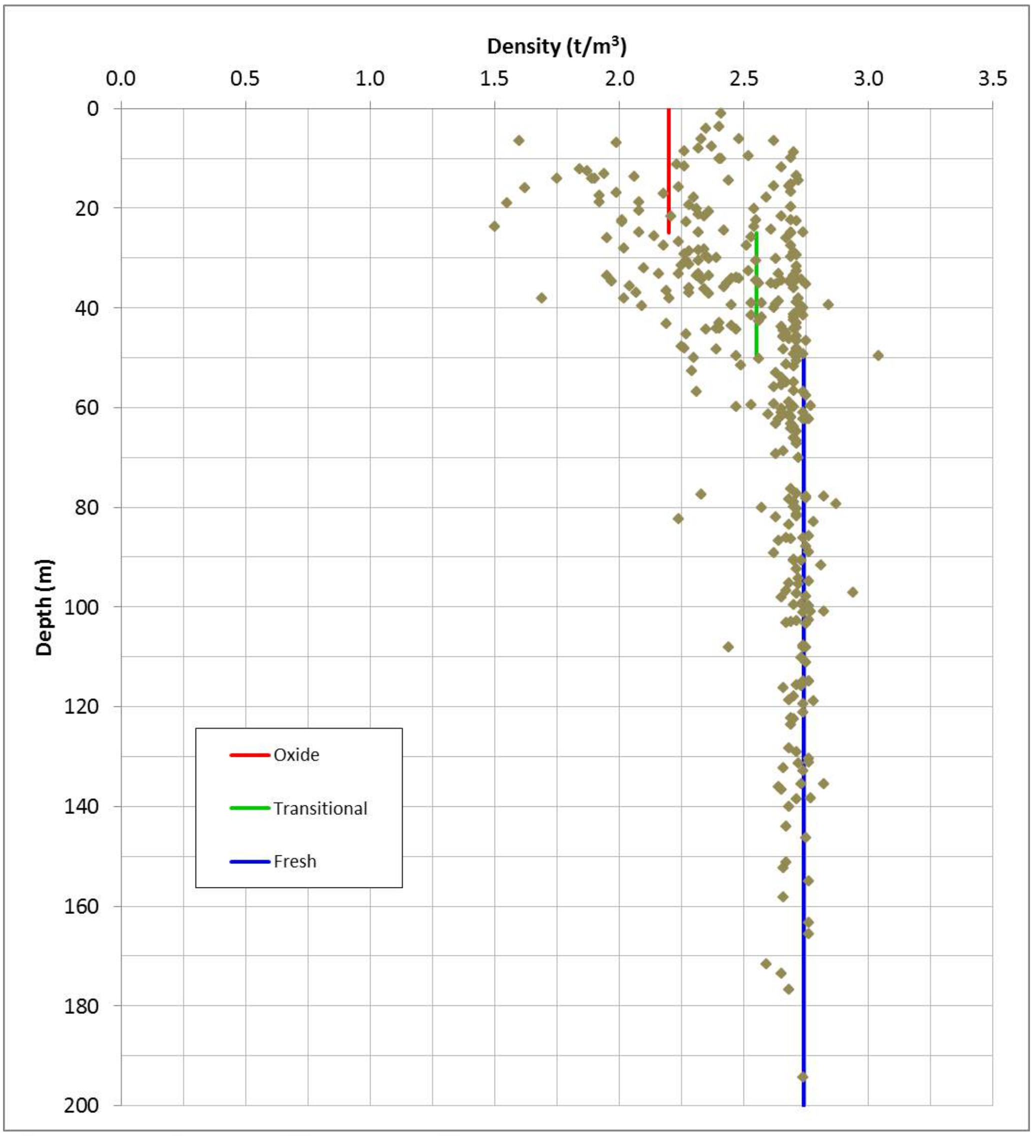
Based on the analysis, default bulk density values were assigned to the model blocks based on the OXIDE and MINZONE coding, as per Table 14.43. Due to the low number of BRN samples, the BRN and BRSCF domains were combined to assess the bulk density of the combined mineralised domains. A default bulk density of 1.44 t/m^3 for the Old Tailings material was used (refer to Section 14.7).

Table 14.43 Default bulk density values applied to Buffalo Reef block model

Mineralisation state MINZONE	Oxidation state OXIDE	Bulk density (t/m^3)
Waste (0)	Oxide (10)	2.16
	Transitional (20)	2.43
	Fresh (30)	2.67
Mineralised (1000, 2000)	Oxide (10)	2.20
	Transitional (20)	2.55
	Fresh (30)	2.74
Old tailings	-	1.44

A plot of the bulk density values of all samples against the depth, with the assigned bulk density values for the mineralisation shown by solid lines, is presented in Figure 14.45 (the depth of the oxide and transitional boundaries in Figure 14.45 are indicative only). Snowden believes the values assigned to the model are reasonable for the Buffalo Reef mineralisation.

Figure 14.45 Bulk density depth profile for Buffalo Reef



Source: Snowden

14.5.6 Mineral Resource classification

The Buffalo Reef Mineral Resource estimate has been classified as a combination of Indicated and Inferred Resources in accordance with CIM guidelines.

The classification was developed based on an assessment of the following criteria:

- Nature and quality of the drilling and sampling methods
- Drilling density
- Confidence in the understanding of the underlying geological and grade continuity
- Analysis of the QAQC data
- A review of the drillhole database and Monument's sampling and logging protocols
- Confidence in the estimate of the mineralised volume
- The results of the model validation
- Production history and reconciliation.

The resource classification scheme adopted by Snowden for the Buffalo Reef Mineral Resource estimate is outlined as follows:

- Where the drilling density was approximately 20 mE x 40 mN (or less), mineralisation within the main BRSCF mineralised lodes (LODE 201, 206–210, 212–214, 220–228 and 230) were classified as an Indicated Resource.
- Where the drilling density was approximately 20 mE x 20 mN down dip (or less), mineralisation within the BRN mineralised lodes (MINZONE = 1000) were classified as an Indicated Resource. The requirement for closer spaced drilling is due to the lower continuity and increased complexity of the BRN mineralisation.
- Where the drilling density was greater than 20 mE x 40 mN at BRSCF or greater than 20 mE x 20 mN at BRN, the mineralisation was classified as an Inferred Resource.
- All minor lodes were classified as Inferred Resources due to the limited geological continuity within these domains.
- Only unmined mineralisation has been considered by Snowden. Old Tailings are discussed in Sections 14.4.4 and 14.5.4.

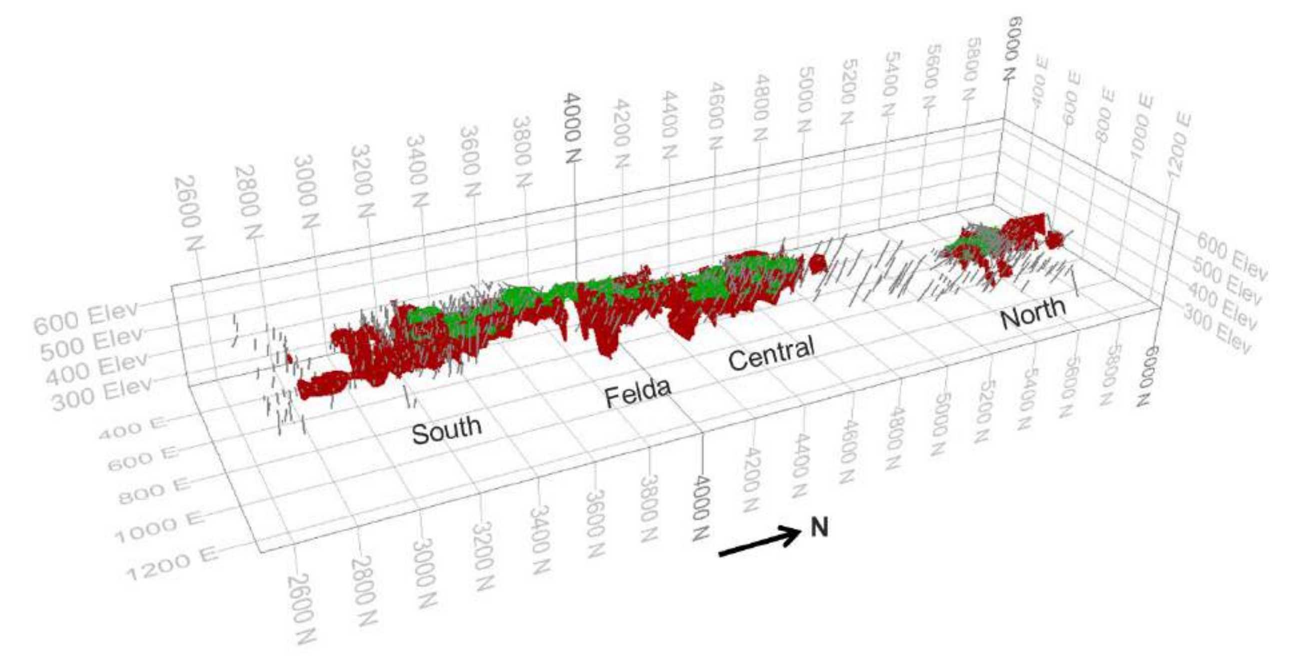
For Buffalo Reef, arsenic, antimony and later in 2017 sulphur were estimated, in addition to gold, as these elements are considered by Monument to be important for the processing. Snowden notes that not all samples contain assays for these elements (in particular for sulphur) and as such, estimates of these variables should be considered to be of low to moderate confidence only. As such, the resource classification applied to the model applies to the gold grade estimates only.

The classification was recorded in the model using a field called RESCAT as described in Table 14.44.

Table 14.44 Resource classification model field codes

RESCAT	Description
0	Not classified (waste)
2	Indicated
3	Inferred

Figure 14.46 Orthogonal view showing Buffalo Reef resource classification (green = Indicated; red = Inferred)



Source: Snowden

14.5.7 Mineral Resource reporting

Cut-off grade

The Mineral Resource for Buffalo Reef deposits has been reported above a 0.3 g/t Au cut-off grade for oxide material and above a 0.5 g/t Au cut-off grade for transitional and sulphide material. The cut-off grades are based on cost and metal price parameters detailed in Section 16 (Mining Methods), along with the assumptions discussed below, and practical considerations with respect to the mining grade control.

Moisture

All Mineral Resources have been reported on a dry tonnage basis.

Depletion for mining

The Buffalo Reef Mineral Resource has been depleted for all open-pit mining to end of March 2018.

Buffalo Reef Mineral Resource statement

The Mineral Resource estimate for the Buffalo Reef deposit is provided in Table 14.45. The Mineral Resource is limited to a pit shell generated by Monument based on a long-term potential gold price of US$2,400/oz. This pit shell was used by Monument to define the likely limits of potential open pit mining. The mining and cost parameters and assumptions used by Monument to generate the resource pit shell are listed in Table 14.46.

Table 14.45 Buffalo Reef Mineral Resource statement, inclusive of Mineral Reserves, depleted for mining to end of March 2018

Classification	Oxidation	Zone	Cut-off (g/t Au)	Tonnes (kt)	Au (g/t)	As (ppm)	Sb (ppm)	S (%)	Ounces (koz)
Indicated	Oxide	BRN	0.3	121	0.97	1,843	185	0.35	4
		BRC	0.3	158	0.81	1,565	141	0.50	4
		Felda	0.3	160	1.11	2,273	232	0.08	6
		BRS	0.3	55	2.06	3,076	837	0.15	4
	Oxide total			**494**	**1.08**	**2,031**	**258**	**0.29**	**18**
	Transitional	BRN	0.5	151	1.16	2,132	196	0.57	6
		BRC	0.5	382	1.06	2,167	96	0.40	13
		Felda	0.5	219	1.48	2,776	291	0.33	10
		BRS	0.5	234	2.57	2,895	3,143	0.59	19
	Transitional total			**986**	**1.53**	**2,470**	**878**	**0.45**	**48**
	Fresh	BRN	0.5	86	1.05	2,122	83	0.71	3
		BRC	0.5	1,106	1.56	3,157	1,796	0.88	55
		Felda	0.5	686	1.66	2,711	878	0.71	37
		BRS	0.5	1,167	2.06	2,732	1,111	0.87	77
	Fresh total			**3,045**	**1.76**	**2,864**	**1,278**	**0.83**	**172**
Indicated Total				**4,525**	**1.63**	**2,687**	**1,080**	**0.69**	**238**
Inferred	Oxide	BRN	0.3	72	0.87	1,646	89	0.10	2
		BRC	0.3	114	1.10	1,459	60	0.20	4
		Felda	0.3	66	1.03	1,424	141	0.31	2
		BRS	0.3	89	1.14	1,296	180	0.05	3
	Oxide total			**341**	**1.05**	**1,453**	**113**	**0.16**	**11**
	Transitional	BRN	0.5	127	1.12	1,854	88	0.72	5
		BRC	0.5	179	1.18	1,891	156	0.28	7
		Felda	0.5	68	1.24	1,731	143	0.46	3
		BRS	0.5	108	1.37	1,708	673	0.49	5
	Transitional total			**482**	**1.22**	**1,818**	**252**	**0.47**	**20**
	Fresh	BRN	0.5	102	1.18	2,761	52	0.71	4
		BRC	0.5	1,851	1.71	2,592	1,708	0.87	102
		Felda	0.5	1,263	1.80	3,104	1,066	0.82	73
		BRS	0.5	668	1.46	2,353	516	0.54	31
	Fresh total			**3,883**	**1.68**	**2,722**	**1,251**	**0.80**	**210**
Inferred Total				**4,706**	**1.59**	**2,537**	**1,066**	**0.72**	**241**

Notes: Small discrepancies may occur due to rounding. The classification applies to the Au grades only; As, Sb and S grades are considered indicative only.

Table 14.46 Resource pit optimisation parameters and assumptions

Input parameters	Unit	Selinsing	BRSCF	BRN
Mining cost	US$/t	1.89	1.66	1.71
Ore loss	%	5	5	5
Mining dilution	%	5	5	5
Processing cost – oxide	US$/t	8.00	8.00	8.00
Processing cost – sulphide	US$/t	10	10	10
Processing recovery – oxide	%	80	80	80
Processing recovery – sulphide	%	90	90	90
Overall slope	°	50	50	50
Selling price	US$/troy oz	2,400	2,400	2,400
Cut-off grade – oxide	g/t Au	0.3	0.3	0.3
Cut-off grade – sulphide	g/t Au	0.5	0.5	0.5
Revenue factor		1	1	1

The foregoing information was used to generate an optimisation pit shell based on a reasonable prospect for extraction of Mineral Resources. No economic analysis has been conducted on the Mineral Resources.

Comparison to 2016 Mineral Resource

For the Buffalo Reef deposit, Indicated Resources have decreased by 4 koz (2%), while Inferred Resources have increased by 31 koz (15%), compared to the 2016 Mineral Resource (Snowden, 2016).

The increase in Inferred Resources is primarily within the sulphide portion of the mineralisation and is due to a larger constraining pit shell along with a recalculation of the cut-off grades used for reporting the resource. However, similar to Selinsing, due to mining depletion, the oxide Indicated Resource has decreased compared to the 2016 Mineral Resource. A breakdown of the differences is shown in Table 14.47 for Buffalo Reef.

Table 14.47 Differences between the current and previous (Snowden 2016) Buffalo Reef Deposit Mineral Resources

Classification	Oxidation	Zone	Cut-off (g/t Au)	Tonnes (kt)	Ounces (koz)
Indicated	Oxide	BRN	0.3	-59	-2
		BRC	0.3	-12	0
		Felda	0.3	-100	-5
		BRS	0.3	-45	-3
	Oxide total			**-216**	**-10**
	Transitional	BRN	0.5	1	0
		BRC	0.5	72	1
		Felda	0.5	29	0
		BRS	0.5	4	0
	Transitional total			**106**	**1**
	Fresh	BRN	0.5	16	1
		BRC	0.5	116	2
		Felda	0.5	66	2
		BRS	0.5	37	0
	Fresh total			**235**	**5**
INDICATED TOTAL				**125**	**-4**
Inferred	Oxide	BRN	0.3	-28	0
		BRC	0.3	-6	0
		Felda	0.3	-4	0
		BRS	0.3	-1	0
	Oxide total			**-39**	**0**
	Transitional	BRN	0.5	37	1
		BRC	0.5	39	1
		Felda	0.5	18	1
		BRS	0.5	18	1
	Transitional total			**112**	**4**
	Fresh	BRN	0.5	72	3
		BRC	0.5	351	13
		Felda	0.5	223	7
		BRS	0.5	138	4
	Fresh total			**783**	**27**
INFERRED TOTAL				**856**	**31**

Notes: Small discrepancies may occur due to rounding. Positive values indicate an increase in the resource.

Grade-tonnage curve

Grade-tonnage curves for the BRSCF and BRN Mineral Resource estimates, limited to the US$2,400/oz resource pit shell, are presented in Figure 14.47 and Figure 14.48 respectively.

Figure 14.47 Grade-tonnage curves for BRSCF, Indicated Resource above and Inferred below for sulphide resources

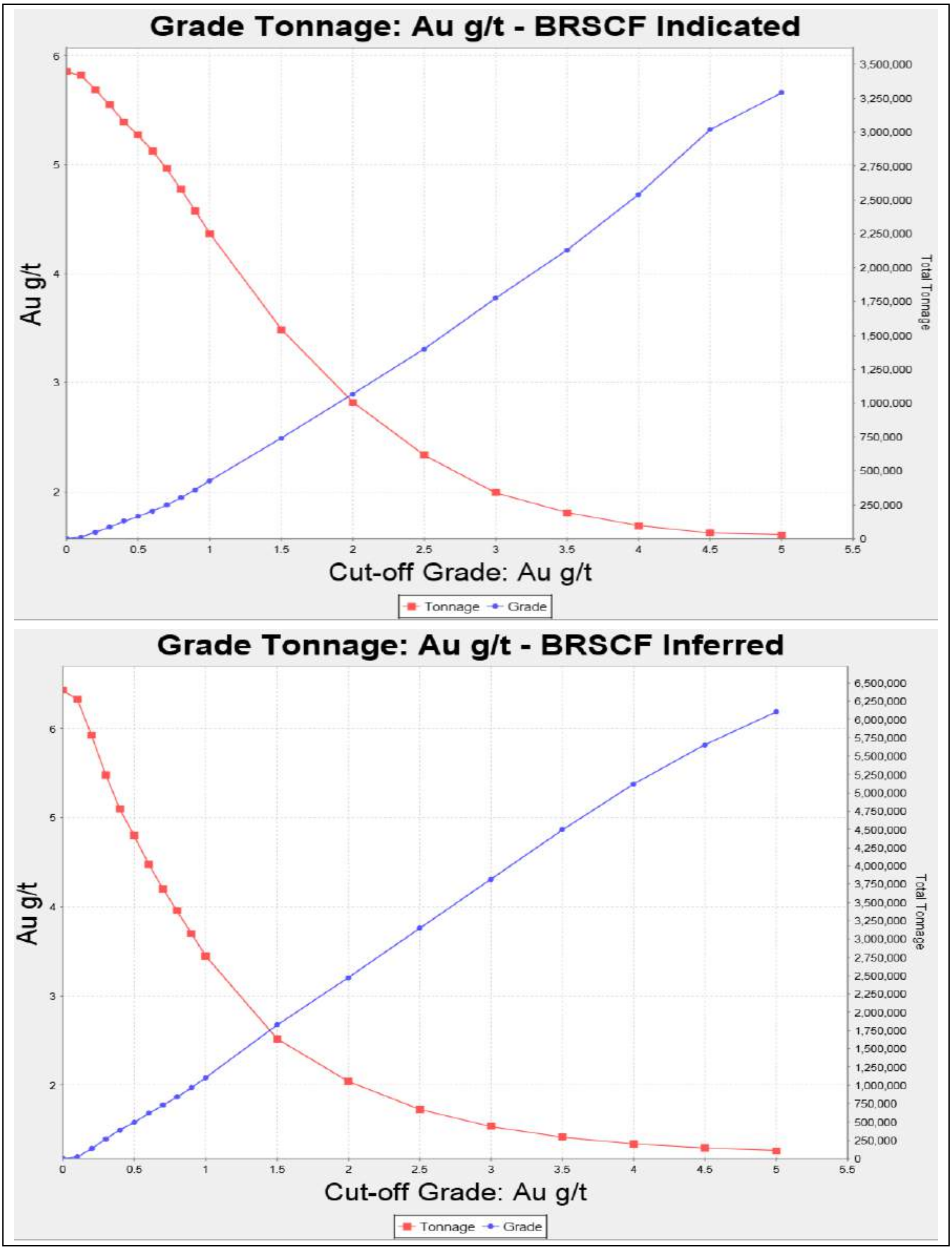
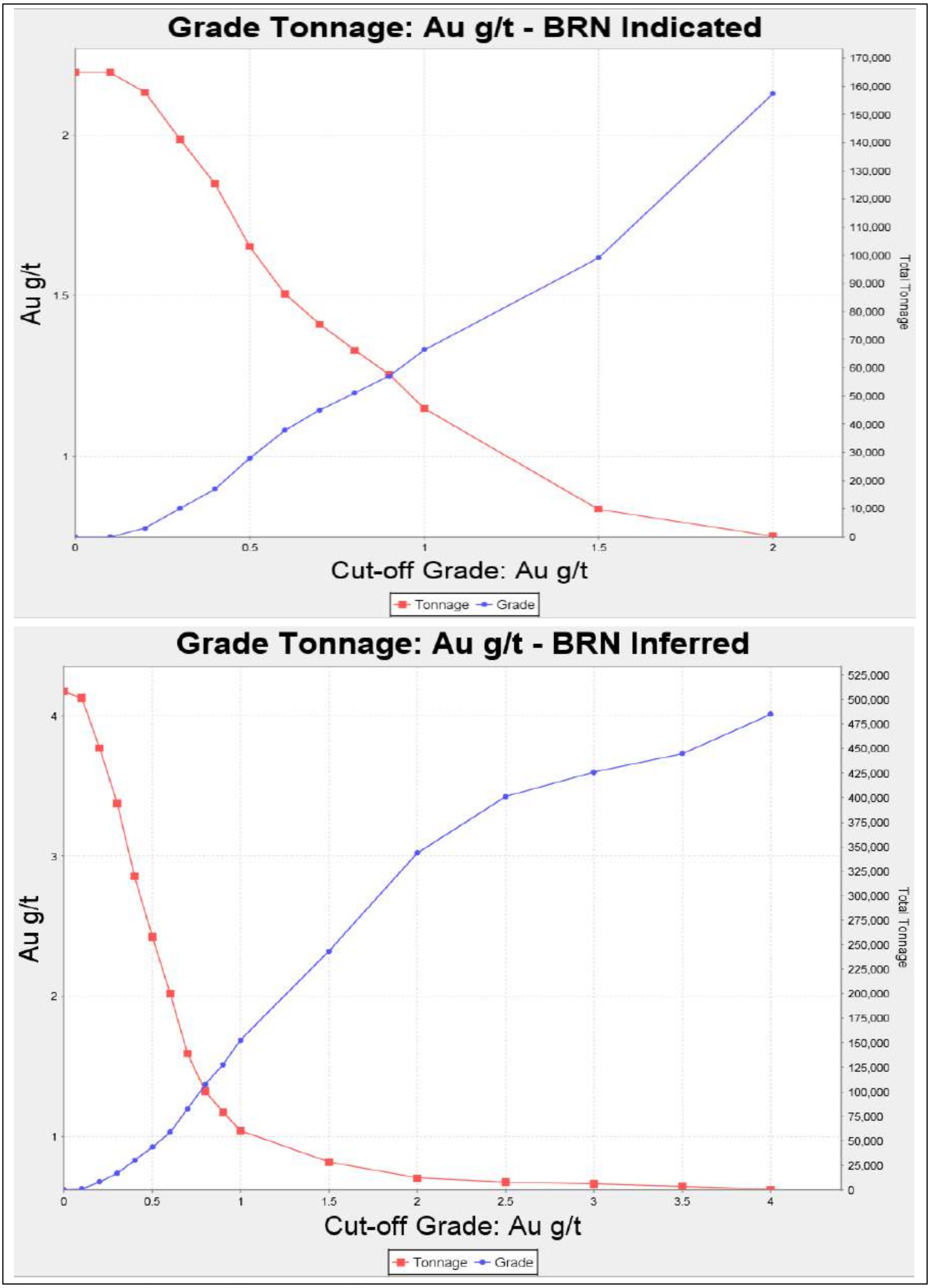


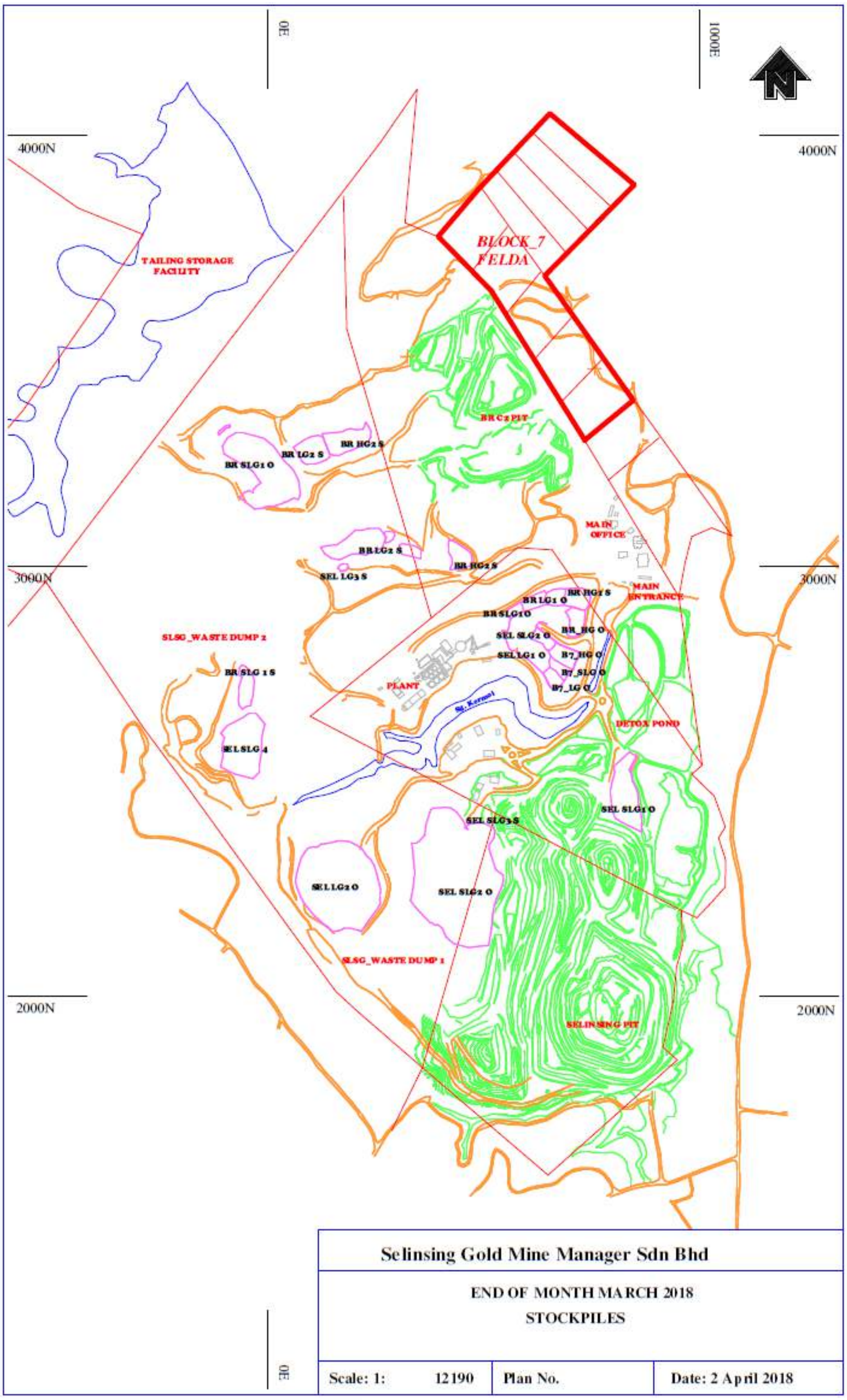
Figure 14.48 Grade-tonnage curves for BRN, Indicated Resource above and Inferred below for sulphide resources



14.6 Stockpile Mineral Resources

Stockpiles at the Selinsing Project include ore mined from the Selinsing and Buffalo Reef pits. The location of the stockpiles as at the end of March 2018 is shown in Figure 14.49.

Figure 14.49 Stockpile location plan, as at end of March 2018.



Source: Monument

14.6.1 Stockpiling strategy

Ore is stockpiled according to the source (Selinsing or Buffalo Reef) and oxidation state (oxide and sulphide ore) along with the gold grade. The grade designation for stockpiles is as follows:

- Super low grade (SLG): 0.30 g/t Au to 0.65 g/t Au
- Low grade (LG): 0.65 g/t Au to 1.50 g/t Au
- High grade (HG): 1.50 g/t Au to 3.50 g/t Au
- Super high grade (SHG): >3.50 g/t Au (none at end of March 2018).

For the sulphide ore, the stockpiles are further subdivided based on the leachability of the ore (designated as “leachable” and “non-leachable”). The leachability designation refers to the current processing plant configuration.

14.6.2 Volume estimate

Stockpile volumes are surveyed by the SGM survey department on a monthly basis. Stockpiles surveyed include the crushed ore stockpile (COS), all ROM stockpiles and other, longer term stockpiles. Volume calculations are completed using Surpac software.

14.6.3 Stockpile bulk density

The bulk density of the stockpiles is based on applying a 25% swell factor to the density. The density determination procedure is described in Section 14.4.5. Approximately 10–30 hand specimens are collected from ore blocks and waste each month for bulk density determination.

The density values are assigned to the stockpiles based primarily on the oxidation state.

14.6.4 Grade estimate

The grade of each stockpile is primarily based on grade control estimates of the source ore blocks during mining. The grade of the stockpiles is then adjusted each month according to the opening balance, material added through mining (from grade control and haulage estimates) and material sent to the crusher.

14.6.5 Mineral Resource classification

The Mineral Resources contained in the stockpiles at the Selinsing Project are classified as Measured Resources in accordance with CIM guidelines. Snowden believes that a Measured classification is appropriate for the stockpile resources based on the following:

- High confidence in the stockpile volumes which are surveyed on a monthly basis
- Stockpile grade estimates are based on grade control of ore blocks during mining
- Reconciliation of tonnes and grade with plant production.

14.6.6 Stockpile Mineral Resource report

Mineral Resources for the stockpiles at the Selinsing Project, as at the end of March 2018, are summarised in Table 14.48.

Table 14.48 Stockpile Mineral Resources as at end of March 2018

Stockpile name	Stockpile ID	lcm	Tonnes (kt)	Au (g/t)	Ounces (koz)
Oxide Stockpiles					
High Grade 1 (Oxide)	SEL HG1 O	-	-	-	-
Low Grade 1 (Oxide)	SEL LG1 O	6,081	12	1.1	0.4
Low Grade 2 (Oxide)	SEL LG2 O	3,189	6	0.73	0.2
Super Low Grade 1 (Oxide)	SEL SLG1 O	2,191	4	0.44	0.1
Super Low Grade 2 (Oxide)	SEL SLG2 O	72,179	991	0.43	13.9
Super Low Grade 4 (Oxide)	SEL SLG 4	36,293	78	0.50	1.3
BR High Grade 1 (Oxide)	BR HG1 O	566	1	3.28	0.1
BR Low Grade 1 (Oxide)	BR LG1 O	928	2	1.02	0.1
BR Super Low Grade 1 (Oxide)	BR SLG1 O	84,030	168	0.55	3.0
B7 High Grade 1 (Oxide)	B7 HG1 O	1,767	3	2.12	0.2
B7 Low Grade 1 (Oxide)	B7 LG1 O	77	0.13	1.02	0.01
B7 Super Low Grade 1 (Oxide)	B7 SLG1 O	449	0.73	0.69	0.01
Oxide – Total		**607,750**	**1,265**	**0.47**	**19.1**
Leachable Sulphide Stockpiles					
High Grade 1 (Leachable Sulphide)	SEL HG1 S	-	-	-	-
Low Grade 1 (Leachable Sulphide)	SEL LG1 S	-	-	-	-
BR High Grade 1 (Leachable Sulphide)	BR HG1 S	879	1.776	1.86	0.1
BR Low Grade 1 (Leachable Sulphide)	BR LG1 S	1,237	2.499	0.97	0.1
High Grade 2 (Leachable Sulphide)	SEL HG2 S	-	-	-	-
Low Grade 3 (Leachable Sulphide)	SEL LG3 S	213	0.46	0.68	0.01
Low Grade 4 (Leachable Sulphide)	SEL LG4 S	-	-	-	-
BR High Grade 2 (Leachable Sulphide)	BR HG2 S	5,186	11.399	2.72	1.0
BR Super Low Grade 1 (Leachable Sulphide)	BR SLG1 S	2,145	4.333	0.55	0.1
B7 High Grade 1 (Leachable Sulphide)	B7 HG1 S	-	-	-	-
B7 Low Grade 1 (Leachable Sulphide)	B7 LG1 S	-	-	-	-
B7 Super Low Grade 1 (Leachable Sulphide)	B7 SLG1 S	-	-	-	-
Leachable Sulphide – Total		**9,660**	**20.5**	**1.92**	**1.3**
Non-Leachable Sulphide Stockpiles		0	0		0
BR Low Grade 2 (Non-Leachable Sulphide)	BR LG2 S	11,570	23.115	1.25	0.9
Super Low Grade 3 (Non-Leachable Sulphide)	SEL SLG3 S	748	1.511	0.6	0.03
B7 High Grade 2 (Non-Leachable Sulphide)	B7 HG2 S	-	-	-	-
B7 Low Grade 2 (Non-Leachable Sulphide)	B7 LG2 S	-	-	-	-
Non-Leachable Sulphide – Total		**12,318**	**24.626**	**1.21**	**0.9**
GRAND TOTAL		**630,000**	**1,312**	**0.51**	**21.3**

Notes:

1. *All stockpiles classified as Measured Resources with 100% conversion to Proven Reserves*
2. *lcm = loose cubic metres; stockpile volume and tonnes are not rounded as based on surveyed volumes*
3. *BR = Buffalo Reef stockpile; SEL = Selinsing stockpile*
4. *SLG = super low grade (0.30 g/t Au to 0.65 g/t Au); LG = low grade (0.65 g/t Au to 1.50 g/t Au); HG = high grade (1.50 g/t Au to 3.50 g/t Au)*

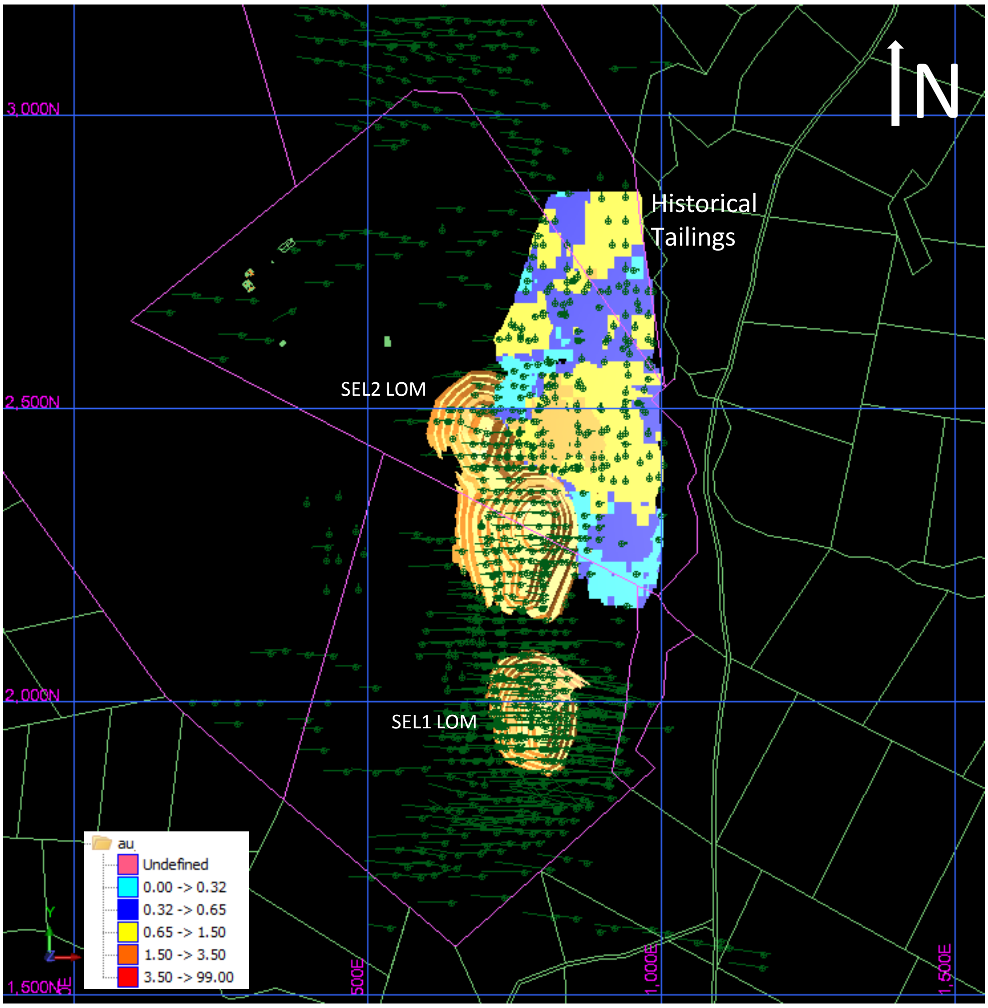
14.7 Old Tailings Mineral Resources

14.7.1 Old Tailings

Od Tailings at the Selinsing Project include the balance of old tailings dams located next to Pit V and Pit VI of the Selinsing deposit pits. Most of the OldTailings material originated from the oxide mining operation by the local operator, Tshu Lian Seng Mining since 1987 until the operation ceased in late 1995. Additionally, a small amount of tailings deposited from underground mining from the early century occur, however the boundary between the underground tailings and tailings from the oxide operation is not well defined. As such, the Old Tailings are modelled as a single volume of "total" tailings.

The current location of the OldTailings is shown in Figure 14.50

Figure 14.50 Old Tailings block model location plan in relation to the tenement boundaries, drilling and LOM pit designed shells (SEL1 and SEL2), as at end of March 2018



14.7.2 Volume estimate

A total of 201 drillholes, totalling 1,503 m and assayed for gold, were used for estimating the grade of blocks coded as Old Tailings (coding discussed in Section 14.4.4 and 14.5.4) in the Selinsing Resource block model. The Buffalo Reef resource model also has a small portion of blocks coded as tailings, but they are entirely overlapped by the blocks covered by Selinsing block model, and as such only the Selinsing Block model was used for estimation.

Drilling covering the old tailings was completed in 1996–1997 using AC, auger and RC drilling in an incomplete grid of 20 m x 20 m spacing. More recent diamond drilling completed by Monument intercepted the old tailings close to the collar. The combined historical and Monument drilling was used previously by Practical Mining in 2013 to define a wireframe representing the volume of the tailings. This surface was checked visually by Monument against the drillhole positions, verifying the grades and logging information, along with comparing to the current topographic surface, previous tailings boundaries and documented extensions of the old tailings. Based on this review, the surface representing the base of theold Tailings is considered reasonable.

14.7.3 Old Tailings bulk density

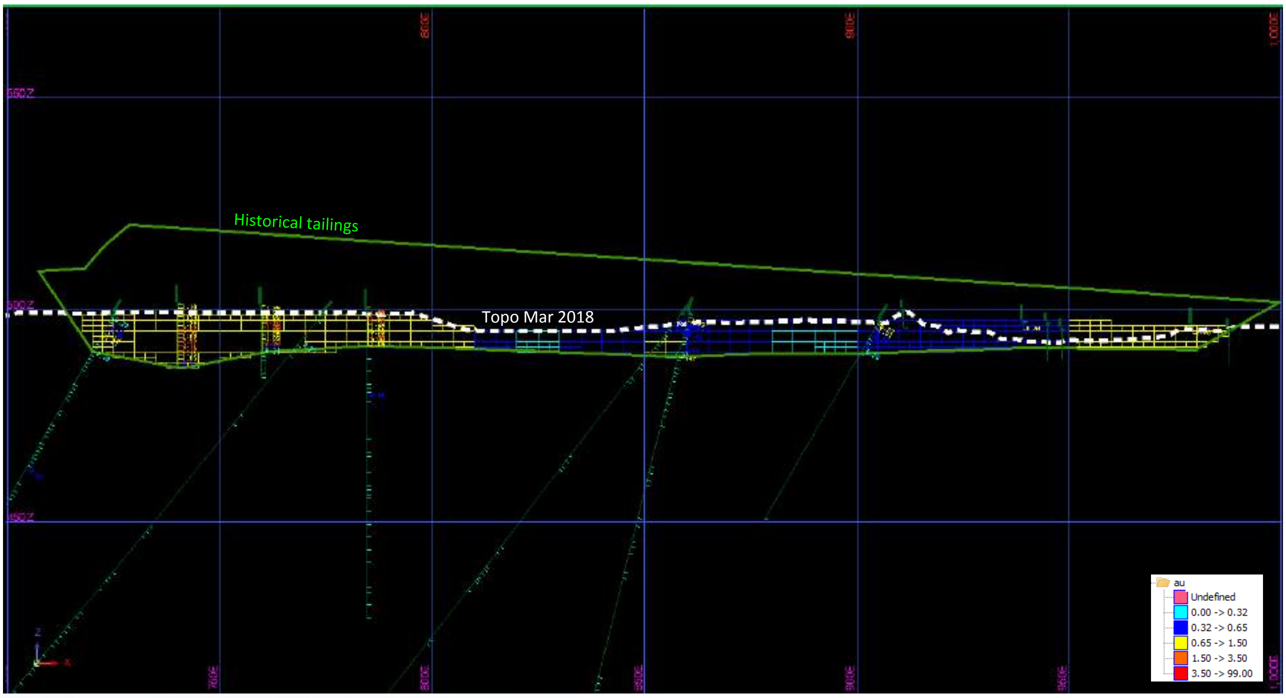
The bulk density of the Old Tailings, based on determinations of bulk samples, is discussed in a historical resource report for the Selinsing project by Johnston (1998). An average dry bulk density of 1.44 t/m^3 was determined and is considered by Monument to be reasonable, and subsequently assigned to the Selinsing block model for material coded as Historical Tailings.

14.7.4 Grade estimate

The gold grade of blocks coded as tailings in the Selinsing block model were estimated using ID^2, from 1.5 m composites of the raw drillhole sample intervals within the tailings wireframe (model field “au_isd_tail”). A top-cut of 5 g/t Au was applied to cap the composite grades, with samples above 3 g/t Au only used if within 20 m of the block centroid. A search ellipse with a radius of up to 100 m in both the northing and easting orientations was used, with a ratio of five times shorter in the vertical direction to reflect the horizontal stratification of the tailings dam. A maximum of 16 samples and a minimum of two samples were used for the interpolation, with a maximum of four samples per hole. A small number of blocks not estimated with these parameters, located in the southeast corner of theOld Tailings area, were estimated using a second search pass with a 300 m radius in the northing and easting orientations, and no limitation on the maximum number of samples per hole.

Estimates were validated visually (e.g. Figure 14.51) and statistically, the results of which show that the block grade estimates are representative of the input drilling data.

Figure 14.51 Cross section Y=2620m +- 10m looking north showing drillholes grades in relation to the block estimates



14.7.5 Mineral Resource classification

The Mineral Resources contained in the Old Tailings at the Selinsing Project are classified as Indicated Resources in accordance with CIM guidelines. Snowden believes an Indicated classification is appropriate for the Old Tailings Resources based on the following:

- Moderate to high confidence in the tailings volumes currently processed, which are surveyed monthly as part of the routine end of month surveying
- Relatively low variability of the grade distribution of the tailings samples
- Most of the tailings grade estimates are based on a drillhole grid spacing of up to 20 m x 20 m
- Reconciliation of tonnes and grade with plant production (discussed in Section 15 – Mineral Reserve Estimate).

14.7.6 Old Tailings Mineral Resource report

Mineral Resources for the Old Tailings balance at the Selinsing Project, as at the end of March 2018, are summarised in Table 14.49. A cut-off of 0.30 g/t Au is used to report the Old Tailings, similar to the deposit oxide resources. A mining recovery factor of 80% was applied for the likely practical limits of reclaim mining, accounting for losses associated with scattered waste material in the tailings, such as material used to construct bunds/walls between the ponds and the dam retaining wall on the west side.

Table 14.49 Selinsing Old Tailings Mineral Resource, inclusive of Mineral Reserves, depleted for reclaiming to end of March 2018

Classification	Cut-off (g/t Au)	Tonnes (kt)	Au (g/t)	Ounces (koz Au)
Indicated total	0.30	975	0.75	24

Notes: Small discrepancies may occur due to rounding.

14.8 Summary of Mineral Resource statement

The Mineral Resources figures for the Selinsing, Buffalo Reef deposits, along with the stockpiles andOld Tailings are summarised in Table 14.50. Only Au grades are reported. Further detail of each respective resource/deposit are provided in the relevant sections of Section 14.

Table 14.50 Combined Selinsing Project summarised Mineral Resource statement, inclusive of Mineral Reserves, depleted for mining or reclaiming to end of March 2018

Resource type	Classification	Oxidation	Cut-off (g/t Au)	Tonnes (kt)	Au (g/t)	Ounces (koz Au)
In situ Resource (Selinsing + Buffalo Reef)	Indicated	Oxide	0.3	558	1.03	19
		Transitional	0.5	1,086	1.49	52
		Fresh	0.5	8,052	1.60	415
	Indicated Total			**9,696**	**1.56**	**486**
	Inferred	Oxide	0.3	349	1.05	12
		Transitional	0.5	485	1.22	19
		Fresh	0.5	5,563	1.79	319
	Inferred Total			**6,397**	**1.70**	**350**
Stockpiles	Measured	Oxide	N/A	1,267	0.47	19
		Fresh		45	1.54	2
	Measured Total			**1,312**	**0.51**	**21**
Old tailings	**Indicated Total**	**Oxide**	0.3	**975**	**0.75**	**24**

Notes: Small discrepancies may occur due to rounding. The classification applies to the Au grades only; other estimated grades are considered indicative only.

15 MINERAL RESERVE ESTIMATE

Mineral Reserve estimates are currently reported for the mining operations at the Selinsing Gold Mine Project, comprising reserves for the Selinsing and Buffalo Reef deposits as well as tailings and stockpiles for Selinsing and Buffalo Reef.

Mineral Reserves, which are inclusive of the identified economic portion of the Mineral Resources described in Section 14, were reviewed by Snowden for the Project as part of the FS. The CIM terms “Mineral Reserve”, “Probable Mineral Reserve” and “Proven Mineral Reserve” have the meanings ascribed to those terms by the Canadian Institute of Mining, Metallurgy and Petroleum, as the CIM Definition Standards on Mineral Resources and Mineral Reserves, as adopted by CIM Council, as amended 2014.

As provided for under the NI 43-101 instrument, Snowden has used an acceptable foreign code as the “Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves” as the JORC 2012 Edition, for the Selinsing Gold Mine Project Mineral Reserve estimates. The CIM definitions 2014 and JORC (2012) use slightly different terminology to describe ore classifications and the terminology is aligned as provided in Table 15.1.

Table 15.1 CIM Definitions 2014 and JORC (2012) terminology

JORC (2012 edition)	CIM Definitions 2014
Ore Reserves	Mineral Reserves
Probable Ore Reserves	Probable Mineral Reserves
Proved Ore Reserves	Proven Mineral Reserves
Competent Person	Qualified Person

There are no material differences between the tonnes and grade estimates as defined using the reserve categories between these codes.

The Reserves use the assumptions, designs and parameters defined predominantly in Section 16 and from other relevant sections of this report, applied as modifying factors.

In accordance with the CIM Definition Standards on Mineral Resources and Mineral Reserves (as adopted and amended), Mineral Reserves are classified as either “Probable” or “Proven” Mineral Reserves and are based on Indicated and Measured Mineral Resources only in conjunction with “estimation of Mineral Resource and Mineral Reserve best practice guidelines” as provided by the CIM. No Mineral Reserves have been estimated using Inferred Mineral Resources.

15.1 Summary

The estimation of Mineral Reserves used the recently completed estimate of Measured and Indicated Mineral Resources for the Project as reported in Section 14 of this report. Monument estimated gold Mineral Resources and Mineral Reserve estimates for Monument’s Selinsing gold deposit, under Qualified Person supervision. Monument updated the mining inventory based on the new Mineral Resource estimates from end of March 2018. The Selinsing Gold Mine Project Mineral Reserve estimates were classified as Probable Mineral Reserves and the Selinsing stockpiles were classified as a combination of Proven and Probable Mineral Reserves in accordance with CIM guidelines. A summary of the pit Mineral Reserves is provided in Table 15.2. The existing stockpile Mineral Reserves are provided in Table 15.3 and the existing re-treatable tailings Mineral Reserves are provided in Table 15.4.

Note that tonnes and ounces have been rounded and this may have resulted in minor discrepancies.

Table 15.2 Selinsing Gold Mine Project Mineral Reserve estimate as at end of March 2018

Classification	Oxidation	Zone	Approximate cut-off (g/t Au)	Tonnes (kt)	Au (g/t)	Ounces (koz)
Probable	Oxide	Selinsing	0.4	51	0.66	1.1
		BRN	0.4	80	1.14	3.0
		BRC	0.4	77	0.90	2.2
		Felda	0.4	233	1.34	10.0
		BRS	0.4	41	0.93	1.2
	Oxide – Total			**483**	**1.13**	**17.5**
	Transitional	Selinsing	0.75	43	1.61	2.2
		BRN	0.75	73	1.31	3.1
		BRC	0.75	171	1.11	6.1
		Felda	0.75	171	1.66	9.1
		BRS	0.75	299	2.23	21.4
	Transitional – Total			**757**	**1.72**	**41.9**
	Fresh	Selinsing	0.75	578	2.28	42.4
		BRN	0.75	13	1.31	0.6
		BRC	0.75	699	1.78	40.0
		Felda	0.75	476	1.79	27.5
		BRS	0.75	913	2.21	64.8
	Fresh – Total			**2,680**	**2.03**	**175.1**
PROBABLE – TOTAL				**3,919**	**1.86**	**234.6**

Reserves for the stockpiles at the Selinsing Project, as at end of March 2018, are summarised in Table 15.3. Note that tonnes and ounces have been rounded and this may have resulted in minor discrepancies.

Table 15.3 Stockpile Mineral Reserves as at end of March 2018

Stockpile name	Stockpile ID	lcm	Tonnes (kt)	Au (g/t)	Ounces (koz)
Oxide Stockpiles					
High Grade 1 (Oxide)	SEL HG1 O	-	-	-	-
Low Grade 1 (Oxide)	SEL LG1 O	6,081	12	1.1	0.4
Low Grade 2 (Oxide)	SEL LG2 O	3,189	6	0.73	0.2
Super Low Grade 1 (Oxide)	SEL SLG1 O	2,191	4	0.44	0.1
Super Low Grade 2 (Oxide)	SEL SLG2 O	72,179	991	0.43	13.9
Super Low Grade 4 (Oxide)	SEL SLG 4	36,293	78	0.50	1.3
BR High Grade 1 (Oxide)	BR HG1 O	566	1	3.28	0.1
BR Low Grade 1 (Oxide)	BR LG1 O	928	2	1.02	0.1
BR Super Low Grade 1 (Oxide)	BR SLG1 O	84,030	168	0.55	3.0
B7 High Grade 1 (Oxide)	B7 HG1 O	1,767	3	2.12	0.2
B7 Low Grade 1 (Oxide)	B7 LG1 O	77	0.13	1.02	0.01
B7 Super Low Grade 1 (Oxide)	B7 SLG1 O	449	0.73	0.69	0.01
Oxide – Total		**607,750**	**1,265**	**0.47**	**19.1**
Leachable Sulphide Stockpiles					
High Grade 1 (Leachable Sulphide)	SEL HG1 S	-	-	-	-
Low Grade 1 (Leachable Sulphide)	SEL LG1 S	-	-	-	-
BR High Grade 1 (Leachable Sulphide)	BR HG1 S	879	1.776	1.86	0.1
BR Low Grade 1 (Leachable Sulphide)	BR LG1 S	1237	2.499	0.97	0.1
High Grade 2 (Leachable Sulphide)	SEL HG2 S	-	-	-	-
Low Grade 3 (Leachable Sulphide)	SEL LG3 S	213	0.46	0.68	0.01
Low Grade 4 (Leachable Sulphide)	SEL LG4 S	-	-	-	-
BR High Grade 2 (Leachable Sulphide)	BR HG2 S	5186	11.399	2.72	1.0
BR Super Low Grade 1 (Leachable Sulphide)	BR SLG1 S	2145	4.333	0.55	0.1
B7 High Grade 1 (Leachable Sulphide)	B7 HG1 S	-	-	-	-
B7 Low Grade 1 (Leachable Sulphide)	B7 LG1 S	-	-	-	-
B7 Super Low Grade 1 (Leachable Sulphide)	B7 SLG1 S	-	-	-	-
Leachable Sulphide – Total		**9,660**	**20.5**	**1.92**	**1.3**
Non-Leachable Sulphide Stockpiles		0	0		0
BR Low Grade 2 (Non-Leachable Sulphide)	BR LG2 S	11570	23.115	1.25	0.9
Super Low Grade 3 (Non-Leachable Sulphide)	SEL SLG3 S	748	1.511	0.6	0.03
B7 High Grade 2 (Non-Leachable Sulphide)	B7 HG2 S	-	-	-	-
B7 Low Grade 2 (Non-Leachable Sulphide)	B7 LG2 S	-	-	-	-
Non-Leachable Sulphide – Total		**12318**	**24.626**	**1.21**	**0.9**
GRAND TOTAL		**630,000**	**1,312**	**0.51**	**21.3**

Note: All stockpiles classified as Proven Mineral Reserves; lcm = loose cubic metres

The reportable Old Tailings Reserve is summarised in Table 15.4.

Table 15.4 Old Tailings Reserve as at end of March 2018

Classification	Volume (m^3)	Tonnes (kt)	Au (g/t)	Ounces (koz)
Probable	**352,958**	**508**	**0.71**	**12**

15.2 Disclosure

Mineral Reserves reported were based on feasibility studies conducted by Monument and reviewed by Frank Blanchfield (Snowden) who is a Qualified Person for this report.

The Mineral Reserves were based on the FS and the Ore Reserves were estimated and reported using the guidelines of the JORC Code (2012 Edition), under the supervision of Mr Frank Blanchfield who is a Qualified Person as defined in NI 43-101, an employee of Snowden. Snowden is independent of Monument. The JORC estimated Ore Reserves are equivalent to similar categories and upheld by the definitions of the CIM. As such, the Mineral Reserves are reported using CIM standards as required by NI 43–101.

In addition, the scientific and technical information in this report that relates to Process Metallurgy is based on information reviewed by Fred Kock, who is an employee of OMC has sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity being undertaken to qualify as a Qualified Person as defined by NI 43-101. Engineering and Process related costs are based on information included in this report reviewed by Mike Kitney, Independent Technical Director of Monument oversaw Metallurgical Design, who has extensive experience in process plant construction and operation and to this area of activity being undertaken to qualify as a Qualified Person as defined by NI 43-101.

15.2.1 Known issues that materially affect Mineral Reserves

Monument is unaware of any issues that materially affect the Mineral Reserves in a detrimental sense.

The Mineral Reserves could be affected by changes in metal price, capital and operating costs, metallurgical performance, infrastructure requirements, permitting or other factors. These factors are discussed in other sections of this report. The major risks to the Mineral Reserves are factors that either effect the costs to exploit resource or the revenues received for the products produced.

Permitting is not expected to be a material risk to the project as there have been no indications to date that there are any social, regulatory or community issues that cannot be managed through best practice operating standards and/or risk management planning and mitigation measures.

There are no perceived infrastructure risks that hinder the estimation of a Mineral Reserve. The infrastructure is either existing or of a relatively standard type to install during construction of the project.

The key modifying factors used to estimate the Mineral Reserve are based on those documented in Snowden (Dec 2016) and Monument employee experience in this type of deposit and style of mineralisation. Table 15.5 summarises the status of material aspects of end of March 2018 Selinsing Mineral Reserve estimate, against the items listed in the JORC Table 1 Section 4 table. Although not required under NI43-101 guidelines, Table 15.5 providing an additional level of review and certaintyof reserve estimation.

Table 15.5 Qualified Person's assessment of Mineral Reserve estimation for Selinsing and Buffalo Reef deposits and Selinsing stockpiles, JORC Code (2012), Table 1, Section 4

Item	Comment
Mineral Resource for conversion to Ore Reserve	Monument Mining Limited (Monument) prepared the updated Selinsing Mineral Resource estimate in March 2018. The Mineral Resource estimate was classified using CIM guidelines and a summary is provided below. No planned dilution was applied to these estimates. The Selinsing Mineral Resources comprise the Selinsing and Buffalo Reef deposits as well as existing stockpiles and old tails and are inclusive of Mineral Reserves. The Selinsing Gold Mine Project Indicated Mineral, Stockpile and Tailings Resources used as a basis for the Mineral Reserves are summarised in Table 1.1 to Table 1.6. The Selinsing Gold Mine Project Indicated Mineral Resources, Measured stockpile and Indicated tailings Mineral Resources are inclusive of Selinsing property and BuffaloReef property Probable Mineral Reserves and Proven stockpile and Probable tailings Mineral Reserves.
Site visits	A site visit to the Selinsing Gold Mine Project site was undertaken by Mr Frank Blanchfield in March 2016. Mr Frank Blanchfield is the Mineral Reserves Qualified Person for the current NI 43-101 Technical Report. Mike Kitney is the Principal Consultant of Metallurgical Design and visited the Selinsing Property on multiple occasions between mid-2008 and mid-2016 inclusive and is QP for engineering design and process capital and operating cost estimation.
Study status	The current NI 43-101 Technical Report is for a feasibility study (FS) to establish the viability of sulphide ore extraction through the extension of the existing oxide plant to incorporate additional sulphide ore extraction. This study includes work as part of a feasibility study, including: • Metallurgical review by Orway Mineral Consultants (OMC). • Upgraded processing plant for sulphide ore treatment using flotation and BIOX®; review by PIE and CES. • Geotechnical review by Peter O'Bryan and Associates (POB) (2018). • Snowden also reviewed mining and geological aspects of this study. Previous studies include a prefeasibility study (PFS) by Snowden Mining Industry Consultants Pty Ltd (Snowden) and another study completed by Practical Mining LLC in 2012 for the extraction of sulphides from the Selinsing and Buffalo Reef deposits. Snowden re-evaluated this work in the PFS using reports from Lycopodium that updated the metallurgy costs and recoveries in 2016.
Cut-off parameters	A nominal cut-off grade of 0.40 g/t Au was applied to oxides and 0.75 g/t Au for sulphides and 0.35 g/t Au for tailings when developing the Mineral Reserve estimate, based on the economic cut-off grade.
Mining factors and assumptions	To identify the Selinsing and Buffalo Reef Mineral Reserve, a process of optimisation using the Deswik Pseudo Flow, staged pit design, production scheduling and mine cost modelling was undertaken by Monument. The mining method is conventional open pit drill and blast, load and haul on a 2.5 m mining flitch with a 10 m high blasting bench, reflective of semi-selective mining. The maximum excavator bucket size of 2.3 m^3 is matched to this selectivity. A stripping ratio of approximately 6 was identified. Overall, dilution assumption used has reduced the recovered ounces by approximately 2% and marginally increased the ore tonnage processed by 2%.

<table>
<tr><th>Item</th><th>Comment</th></tr>
<tr><td>Metallurgical factors and assumptions</td><td>
The Selinsing Gold Mine Project was originally developed on the basis of treating oxide ore via conventional crushing and ball milling followed by gravity recovery of free gold and cyanidation of gravity concentrate. Gravity tails are subjected to conventional carbon-in-leach (CIL). Final gold recovery from carbon strip solution and gravity concentrate leach solution is by electrowinning onto stainless steel cathodes. In 2009, mining operations commenced at Selinsing. Since then, Monument developed an open pit mine and construction of a 1,200 tonnes per day Au treatment plant in three phases.
The 2016 PFS proposed conventional CIL extraction from oxide ores and bio-leach for transition and fresh ores.
Since that time, review as part of this study has indicated treatment of transition and fresh ores by floatation and then bio oxidation BIOX® followed by CIL.
All the oxide unit processes included in the design are standard and common to many current gold operations, including:

Crushing
Grinding and classification
Gravity concentration (Knelson centrifugal concentrator)
Intense leaching (Acacia reactor) of gravity concentrate
CIL with cyanidation and carbon adsorption
Carbon desorption
Electrowinning
Smelting
Tailings disposal and effluent reclaim
Cyanide detoxification.

For the sulphide fresh and transit ore treatment, the following has been done as part of the FS:

Process design criteria
Process design and flow diagrams
Engineering design criteria
Mechanical and electrical equipment lists
Process plant layout
Capital cost estimates.

The metallurgical factors for sulphide were developed by Monument and in-house and independent testwork by Outotec and reviewed by OMC. The oxide metallurgical factors are from site data.
The metallurgical recovery parameters applied are:
<table>
<tr><th>Material treated</th><th>Deposit</th><th>Gold grade (g/t Au)</th><th>Recovery (%)</th></tr>
<tr><td>Old Tailings</td><td>Selinsing
Buffalo Reef</td><td>All</td><td>74</td></tr>
<tr><td>Oxide</td><td>Selinsing
Buffalo Reef</td><td>All</td><td>75</td></tr>
<tr><td rowspan="2">Transition</td><td>Selinsing</td><td>All</td><td>85</td></tr>
<tr><td>Buffalo Reef</td><td>All</td><td>85</td></tr>
<tr><td rowspan="2">Fresh/Sulphides</td><td>Selinsing</td><td>All</td><td>85</td></tr>
<tr><td>Buffalo Reef</td><td>All</td><td>85</td></tr>
</table>
It is the Qualified Person’s opinion that the plant production numbers are accurate and correct. Given superior flotation response and bioleach response from metallurgical testwork conducted for Phase IV of the project, it is reasonable to assume that the results obtained, and design criteria and process flowsheet adapted for Phase IV are reasonable and adequate for a FS level of accuracy.
</td></tr>
<tr><td>Environmental</td><td>
Rock characterisation was completed in Malaysia and potentially acid-forming acid rock drainage items were identified. The waste dumps are recommended to be designed at a final angle of 18° but the final landform designs will require completion prior to mining; however, Monument has verified that there is enough space for these designs. A cost provision has been made for the construction of the final land forms.
Currently, an exploration licence is approved. An MLA (mining application licence) was submitted for approval in October.
The MLA allows provision for tailings dams and waste dumps.
</td></tr>
<tr><td>Infrastructure</td><td>
After extensive negotiating with the local authority for power purchase from the electricity grid, Monument has decided to construct a new powerline form Kuala Lipis as the only option for the increased reliable power requirement.
Monument has indicated the bio-oxidation plant build will be an EPC execution with Monument providing the management.
Accommodation will be in surrounding communities.
</td></tr>
</table>

Item	Comment
Cost and revenue factors	Process costs were used from historical oxide costs from site and sulphide operating costs are estimated primarily from Selinsing historical production data and market estimates, based on OMC and BIOX® processing design and CES engineering design. The estimated process unit costs are as follows:

Material treated	Deposit	Gold grade (g/t Au)	Process operating cost (US$/t)
Oxide	Selinsing Buffalo Reef	All	8.68^
Old Tailings	Selinsing Buffalo Reef	All	5.41*
Transition*	Selinsing	All	17.26
	Buffalo Reef	All	17.26
Fresh/Sulphides	Selinsing	All	17.26
	Buffalo Reef	All	17.26

^ 2018/19 FY planned oxide cost of US$8.68/t

** 2018/19 FY planned oxide cost of US$5.41/t*

Mining costs are based on historical data during high mining activity developed from the existing contract.

The all-up mining operating cost was estimated to be US$1.99/t mined.

The mining capital cost was absorbed by contract mining.

Administration costs are fixed at US$1.72 million per year and are inclusive of mining, plant and all other administration costs including refining costs.

Capital costs US$52.93 million were estimated by Monument and others as follows:

- Process capital costs: US$34.9 million
- Power upgrade costs US$4.9 million
- Mine rehabilitation totallingUS$0.6M
- Sustaining costs: totalling US$ 2.5M
- Mining Capitalised costs including waste cutback and TSF construction, access roads and River diversion for Buffalo Reef totalling US$ 9.2M
- Old tailings Process upgrade US$0.1M
- Communications and training US$0.8M

All costs in US$.

Royalties averaging 10% plus tenement fees were applied to all gold produced, at expiry of each current tenement.

Item	Comment
Revenue factors	Monument supplied a gold price of US$1,300/oz. This was applied as real and flat forward in the financial model.
Market assessment	Monument has completed comprehensive market studies, including likely refiners. Gold is freely traded, and the price is set by the LME. A comprehensive marketing study was completed as part of the PM LLC 2013 NI 43-101 Technical Report. The selling of gold is straight forward.
Economic	The discount rate in the Monument financial model was set at 8%. A financial sensitivity study was undertaken to evaluate capital expenditure, operating costs and gold price. The project was found to be most sensitive to changes in gold price. The key performance indicators from the Monument model are summarised below:

Key performance indicator	Units	Value
All-in cash cost (including royalty)	US$/oz produced	863.67
IRR ungeared	%	49
NPV (at 8%)	US$M	27.56
Net cash flow	US$M	44.55
Initial capital cost[a]	US$M	39.77

[a] Excludes working capital

Item	Comment
Social	A socio-economic study was prepared by Monument. The commentary provides a summary of the socio-economic characteristics of the area at a household level. Monument has a full-time Community Relations Officer engaged in maintaining open communications with the local communities. Monument has advised there are no community or social encumbrances that could obstruct the provision of an MLA from the Malaysian government.

Item	Comment
Classification	The Mineral Reserve is classified as Proven and Probable in accordance with the CIM Code, corresponding to the Mineral Resource classification of Measured for stockpiles and Indicated for ore sources from pit ore material. No Inferred Resources are included in the Ore Reserve estimate.
Audits or reviews	Snowden has completed an internal peer review of the Mineral Reserve estimate.
Relative accuracy/ confidence	It is Snowden's opinion that the Mineral Reserve classification of "Probable" for the deposits and Proven for the stockpiles is reasonable. The lower Probable confidence in this estimate is attributed to the use of Indicated Resources.

15.2.2 Additional information on Selinsing Old Tailings Reserve as at end of March 2018

Section 14.7.6 describes the Old Tailings Indicated Resource from which the Reserve is based. For the Reserve estimation the same Selinsing block model (selinsing_may2016_v1_s2.mdl) was used but with different modifying factors applied based on:

- Old Tailings recovery and reconciliation through the plant
- Historical formation of tailings ponds to estimate bunding waste depletion
- Current and future use of ponds above the underlying tailings.

To date, Old Tailings reclaim has been from dry areas described in Section 14.8.2 and shown in Figure 15.1. Currently, tailings are processed through the CIL plant as follows:

- Tailings excavated then moved by truck to a tailings stockpile
- Tailings moved to solid reclaim pond
- A hydraulic water cannon is then used to break up the tailings material and turn it into a slurry
- The slurry then pumped to a thickener and reclaim plant at 20–25 t/hr.

Attempts to process dry tailings direct into the plant previously caused issues with the high clay content not allowing efficient crusher feed.

Historical records of pond embankments were used to generate a plan view of pond areas, as shown in Figure 15.1. Figure 15.1 also shows some of these pond areas are potentially wet. Many of the wet areas are used for treated water and form an important part for the site environmental management plan, Pond E being registered with the Mine Department for the purpose. Figure 15.2 shows a photograph over some of the proposed tailings reclaim area looking southwest over Pond A.

Figure 15.1 Selinsing Old Tailings area and old construction ponds

Pond area	Desription
Pond A1	Current Reclaim area
Pond A	Current Reclaim area
Pond B	Planned Reclaim area
Pond B1	Planned Reclaim area
Pond C	Planned Reclaim area
Pond D1	Originaly part of a larger pond this part not used for treated water storage
Pond D2	currently being used to store water for hydraulic mining and thickener operation
Pond D3	Dry area not used for for treated water storage
Pond E	final spillway, registered with Mines Dept for effluent discharge. Not Included
Pond E1	Retention pond after TSF water detoxified, water will overflowing to pond E. Not Included

Currently Wet
Currently Dry

Figure 15.2 Photograph looking southwest over Pond A

In order to estimate the reconciled tailings tonnage, grade and ounces from CIL processing, a period of production with known material excavation was selected (June 2016 to end of March 2018).

Figure 15.3 shows the area reclaimed during the reconciliation period obtained by difference of topographical end-of-month surveys for the start and finish of the reconciliation period. The majority of excavation in this period was in Pond A, Pond A1 and Pond B.

Figure 15.3 Selinsing Old Tailings reconciled production area

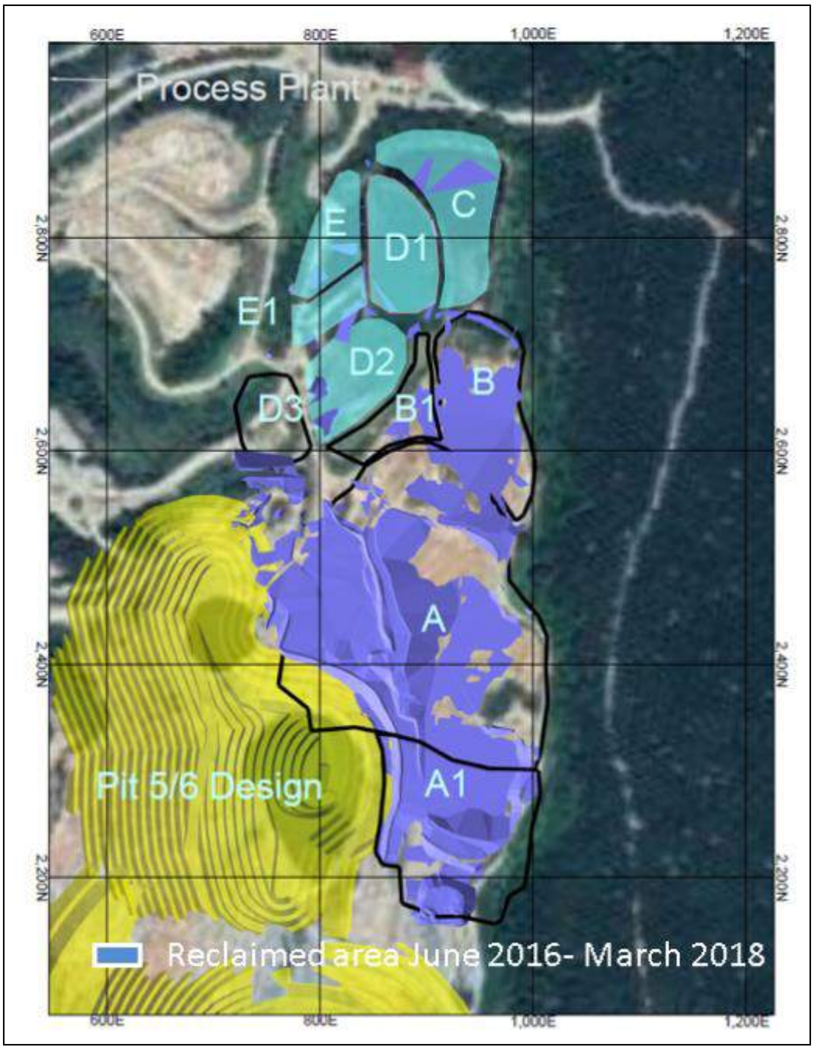


Table 15.6 shows the results for reconciliation for the June 2016 to end of March 2018 period, noting production is reconciled tonnes and ounces to the plant prior to processing.

Table 15.6 Old Tailings production reconciliation June 2016 to March 2018 (>= 0.35 g/t Au)

	BM*	Production	Production vs BM
Volume (m^3)	167,438	101,484	61%
Tonnes	241,110	146,137	61%
Au g/t	1.05	0.90	86%
Ounces Au	8,139	4,210	52%

** Based on: Geological model "selinsing_may2016_v1_s2.mdl". Survey topographical differences between the following: 616_m_mastertopo.dtm (July 2016) and 318_m_mastertopo.dtm (30 March 2018).*

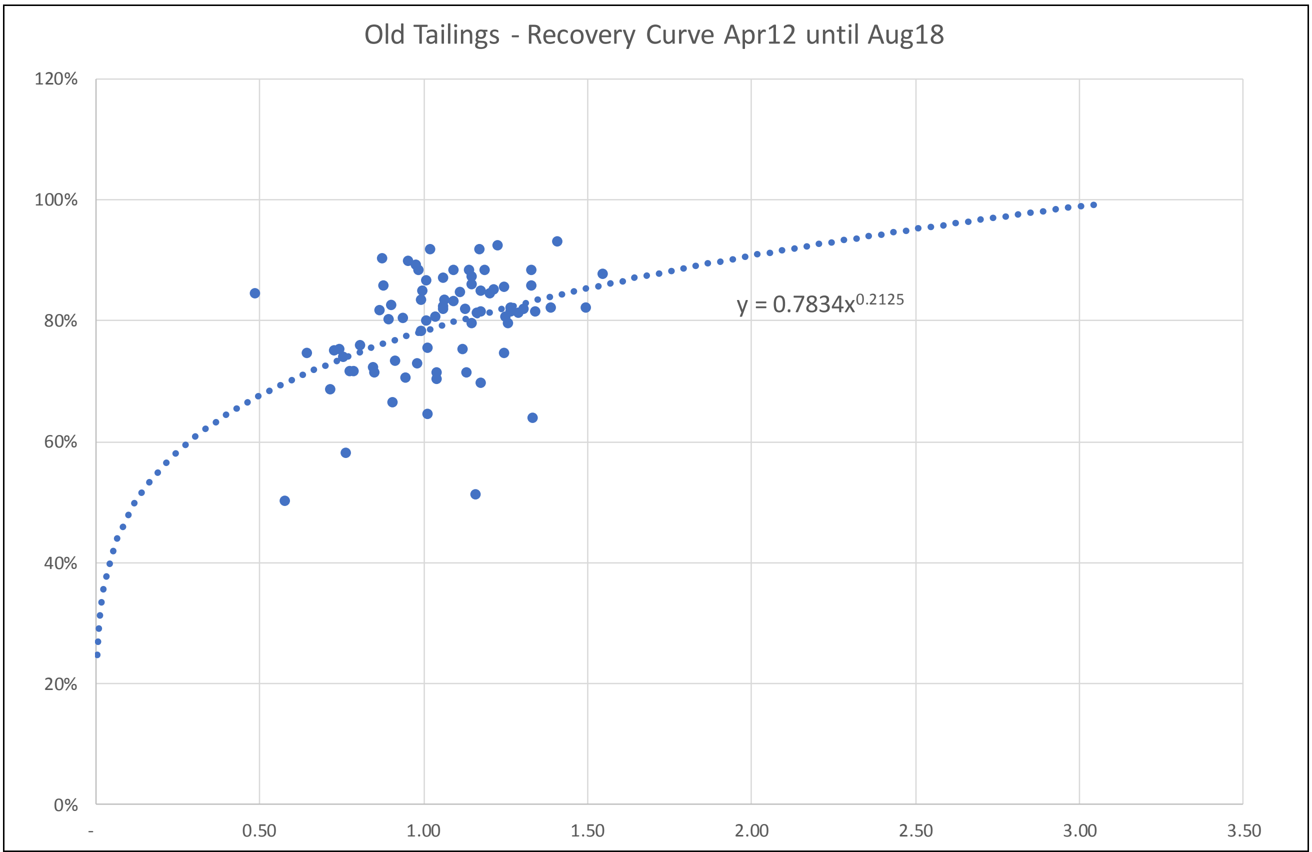
The mined area was compared to a 0.35 g/t Au cut-off grade. This cut-off grade is currently used by site and is adopted as the Reserve cut-off grade. Table 15.7 shows the planned costs used in the FS for tailings based on 2018/2019 financial year forecasts.

Table 15.7 Old Tailings production costs

Item	Unit	Cost
Old Tailings – reclaim	US$/t	1.03
CIL cost	US$/t	2.74
Electrowinning/Gold room	US$/t	1.02
Tailings	US$/t	0.23
Detoxification	US$/t	0.23
Warehouse and distribution	US$/t	0.13
Refinery cost	US$/t	0.03
Total	**US$/t**	**5.42**

Figure 15.4 shows the historical grade vs recovery curve for tailings through the Selinsing Plant. An average value of 74% process recovery based on the Resource grade of 0.76 g/t Au.

Figure 15.4 Selinsing Old Tailings grade / recovery curve



To calculate the Reserve, the following was completed:

- A perimeter for each pond was determined:
  - Each pond separated by perceived waste bunding.
- The perimeter was used to:
  - Cut the tailings model wireframe used for the resource estimation (tails_dec2013.dtm) from Practical Mining (2013)
  - Each resulting digital terrain model (DTM) was then trimmed to the topography as at end of March 2018
  - Each individual DTM was then used to interrogate the model "selinsing_may2016_v1_s2.mdl" above the 0.35 g/t Au cut-off grade

- Conversion factors based on Table 15.6 were then used to generate inventories for each pond area
- Use of wet area ponds was then evaluated depleting these areas form the inventory to form the Reserve.
- The available tailings reserve was then scheduled manually matching April to June 2018 actual data and planned 2018/19 FY production

Figure 15.5 shows the resulting DTMs used for each area and those coloured red not used in the Reserve. Table 15.8 shows the Reserve calculation of tailings being 508.26 kt at 0.71 g/t Au for 11,617 oz. Figure 15.6 shows the resulting schedule of tailings used in the FS. Accelerated production rates could be possible post the SGSP with an upgraded thickener and reclaim plant.

Figure 15.5 Selinsing Old Tailings scheduled Reserve areas

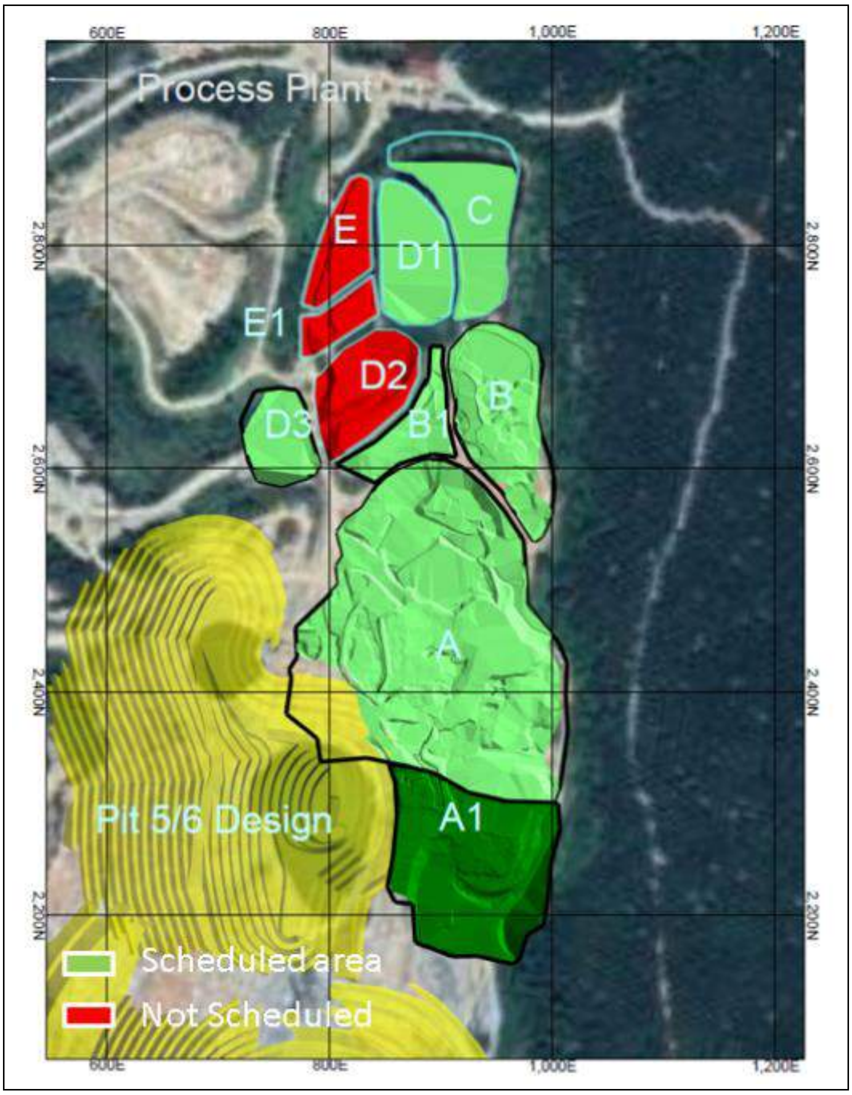


Figure 15.6 Selinsing Old tailings FS schedule

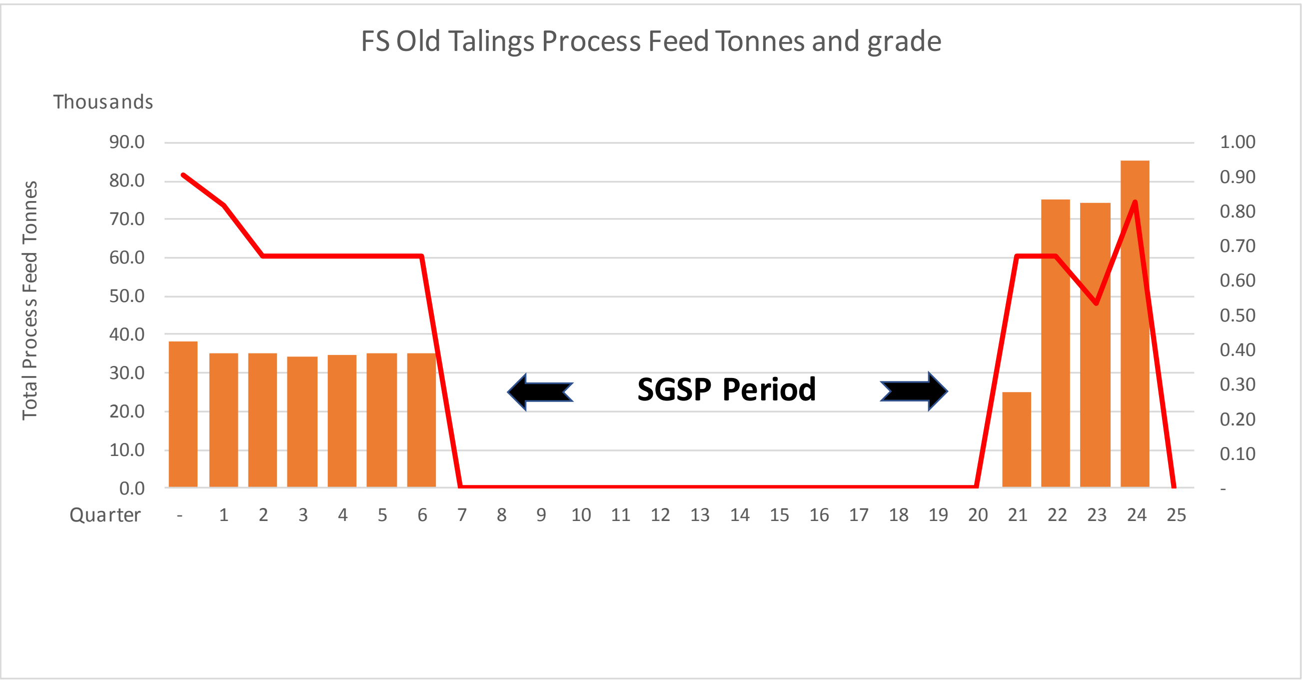


Table 15.8 Old Tailings Reserve calculation

Pond area	In-situ inventory >0.35 g/t cut-off grade				Inventory available based on reconciliation				Reserve				Description
	Volume	Tonnes	Grade (g/t Au)	Ounces	Volume	Tonnes	Grade (g/t Au)	Ounces	Volume	Tonnes	Grade (g/t Au)	Ounces	
Pond A1	68,448	98,566	1.10	3,480	41,486	59,741	0.94	1,808	41,486	59,741	0.94	1,808	Current reclaim area.
Pond A	352,985	508,298	0.79	12,842	213,944	308,080	0.67	6,671	213,944	308,080	0.67	6,671	Current reclaim area.
Pond B	44,696	64,362	0.58	1,209	27,090	9,010	0.50	628	27,090	39,010	0.50	628	Planned reclaim area.
Pond B1	15,612	22,481	0.47	341	9,462	13,626	0.40	177	9,462	13,626	0.40	177	Planned reclaim area.
Pond C	32,602	46,946	0.90	1,363	19,760	28,454	0.77	708	19,760	28,454	0.77	708	Planned reclaim area.
Pond D1	26,169	37,683	0.81	987	15,861	22,840	0.70	513	15,861	22,840	0.70	513	Originally part of a larger pond this part not used for treated water storage.
Pond D2	36,041	51,899	0.55	916	21,844	31,456	0.47	476	Not used				Currently being used to store water for hydraulic mining and thickener operation.
Pond D3	41,833	60,240	1.10	2,140	25,355	36,511	0.95	1,112	25,355	36,511	0.95	1,112	Dry area not used for treated water storage.
Pond E	38,902	56,018	0.66	1,189	23,578	33,953	0.57	618	Not used				Final spillway, registered with Mines Department for effluent discharge. Not Included.
Pond E1	14,858	21,395	0.53	367	9,005	12,967	0.46	190	Not used				Retention pond after TSF water detoxified, water will be overflowing to Pond E. Not included.
Total	**672,145**	**967,888**	**0.80**	**24,834**	**407,386**	**586,638**	**0.68**	**12,901**	**352,958**	**508,262**	**0.71**	**11,617**	

16 MINING METHODS

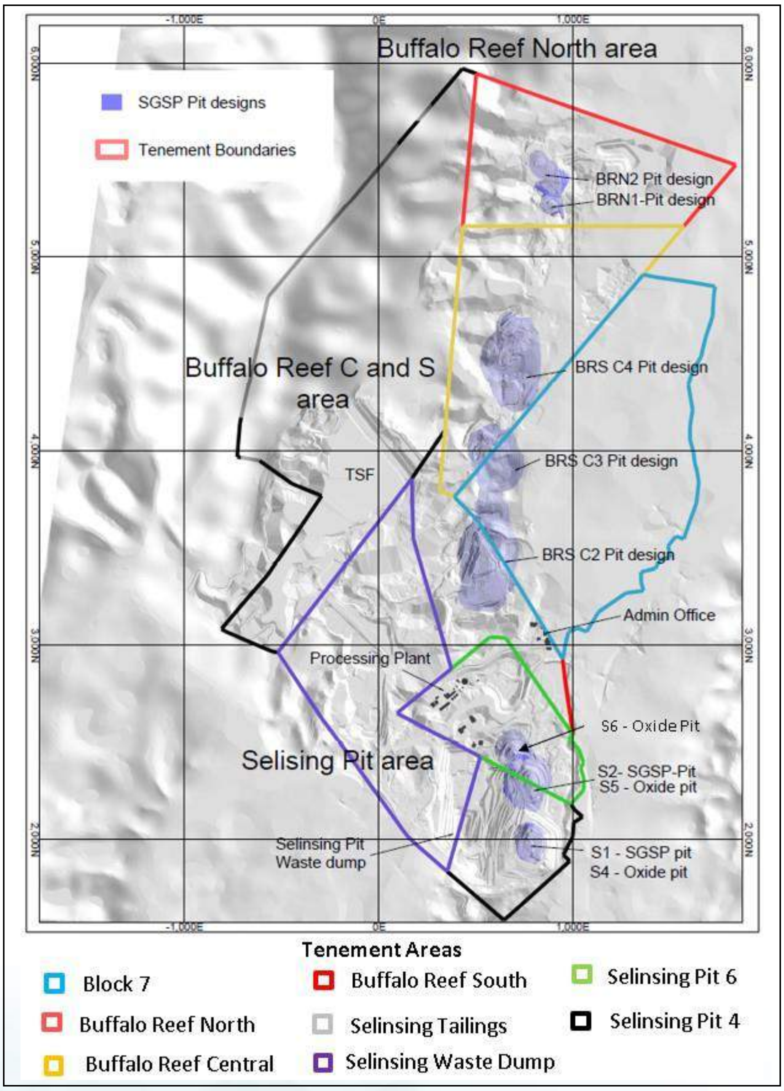
16.1 Overview

The Selinsing and Buffalo Reef deposits are surface outcropping deposits with near surface oxide material suitable for gravity and CIL processing. As such, open pit mining has been carried out for some nine years mining economic oxide material generally above 0.5 g/t Au but considerable material below this grade has also been mined and stockpiled.

The Selinsing deposit is largely sub-vertical dipping and is hosted by a 20–100 m thick shear zone that dips 55° to 75° towards mine grid east (082° true grid).

Mining at Selinsing is more advanced than at Buffalo Reef and has occurred initially as one pit, then as depth increased, twin open pit mining developed in Pit 4 and Pit 5. Figure 16.1 shows a plan view of all mining areas. In both Selinsing open pit areas, oxide mining is largely complete, leaving mainly transition and fresh material to mine. The current mining strategy is to follow the Snowden (Dec 2016) open pit mine designs but only mining and stockpiling low-grade oxide to pit limits within the reserve sulphide pit designs created by Snowden (Dec 2016). Figure 16.1 shows the Snowden (Dec 2016) S1 and S2 designed pits inside which oxide Pit 4 and Pit 5 have been mined to date. For the FS, these pits are referred to as Pit 4 and Pit 5/6 in line with site naming convention. With no planned oxide-only mining, only SGSP (as defined in Sections 1 and 2) sulphide-based mining and processing remains in the Selinsing area.

Figure 16.1 Plan view of the Selinsing Pit existing open pit workings



Mining at Buffalo Reef has occurred since 2012, again mining oxide gravity and CIL treatable material following a pit limit within the Snowden (Dec 2016) pit designs with a plan to cut back to the Snowden (Dec 2016) designs on SGSP approval. There are two main areas of Buffalo Reef comprising of:

- Buffalo Reef South (BRS C2, C3 and C4) includes material in several tenements:
  - Block 7 (tenement yet to be titled but now acquired by Monument)
  - Damar Buffalo Reef South (tenement MC1/111)
  - Damar Buffalo Reef Central (tenement ML12/2012).
- Buffalo Reef North (BRN).

Mining of BRS currently consists of several pit areas with the same naming as the Snowden (Dec 2016) designs of:

- BRS C2 – Buffalo Reef southern area
- BRS C3 and 4 – Buffalo Reef central area.

Mining to date has generally been via small pits influenced by surface topography, with the largest pit developed to date being BRC2.

Unlike the Selinsing area, Buffalo Reef is considered tabular and at BRS ore occurs in a main tabular mineralised zone which is typically 10–20 m thick, along with numerous minor sub-parallel mineralised structures. At BRN, the mineralisation is less continuous and narrower when compared to BRS.

Open pit mining at both Selinsing and Buffalo Reef has historically been carried out by a local contractor (Minetech) who has been involved in the operation since Monument acquired the operation.

A continuation of open pit mining for both the Selinsing and Buffalo Reef areas from the oxide areas down to available fresh (sulphide) areas is considered the most viable mining strategy due to:

- Low to medium ore grades (1.0–2.5 g/t Au) near surface
- Existing open pit knowledge and available expertise
- Well advanced open pit oxide mining
- Open pit minable ore shape.

Some geotechnical concerns concerning stability of the west wall in the Selinsing Pit are discussed in detail in Section 16.2 that affect an open pit mining strategy. These considerations require adjustment to the Snowden (2016) mine design parameters.

The sulphide orebody does extend with depth and some areas have shown high-grade drillhole intersections that, if continue and an attractive mining cost can be established, could enable a possible underground mining strategy at Selinsing. This opportunity is summarised in this Technical Report in Section 24.

As part of the FS, two other scenarios were considered for opportunities:

- Opportunity Case One – Reserve design with Inferred inclusion
- Opportunity Case Two – all Indicated and Inferred open pit potential.

These scenarios are discussed in later sections of this report.

This Section uses reference to Snowden (Dec 2016) with the following subsections:

- Geotechnical.
- Hydrology.
- Mining method selection.

- Mine design:
  - Pit optimisations results
  - Mine layout – open pit scenarios description (Base Case Reserve case, Intermediate Case, and Optimistic Case)
  - Bench height
  - Waste dumps and waste rock management
  - Dump heights
  - Roads, dumps and stockpiles
  - Topsoil stripping and spreading
  - Haul road design and traffic management
  - Stockpiling and reclaiming
  - Pit dewatering and drainage
  - Site layout
  - Drill and blast
  - Load and haul
  - Grade control
  - Other mining activities
  - Open pit work roster.
- Production schedule.
- Mine requirements.
- Underground opportunities.

16.2 Geotechnical studies

The process adopted for geotechnical evaluation requires a description of geotechnical setting in terms of macro and micro geotechnics followed by geotechnical design. The FS included a review of all previous reviews by external consultants leading to the latest review by POB (2018). A list of geotechnical work is listed in Table 16.1.

Table 16.1 Geotechnical studies at Selinsing

Provider	Date	Title	Reference
Golder and Associates	1997	Open Pit Geotechnical Assessment, Selinsing Gold Project, Malaysia	Golder (1997)
Snowden	May 2012	Geotechnical Assessment of Selinsing and Buffalo Reef Project: 3933 – Selinsing Geotechnical Assessment	Snowden (May 2012)
Snowden	October 2012	Project: 3933 – Selinsing Geotechnical Assessment - Selinsing Site Visit Report	Snowden (October 2012)
Snowden	June 2013	Monument Mining Limited Selinsing Mine, Project No. 3933, Slope Stability Management Plan	Snowden (June 2013)
Snowden	January 2016	Geotechnical Review - Presentation	Snowden (January 2016)
Snowden	May 2016	Geotechnical Review of Selinsing and Buffalo Reef Mines Project: AU5192	Snowden (May 2016)
Geomapping Technology Sdn Bhd	January 2018	Geotechnical Review of Selinsing Pit West Wall and Buffalo Reef	Geomapping (2018)
Peter O'Bryan and Associates	June 2018	Selinsing Gold Mine Geotechnical Site Visit April 2018	POB (2018)

In recent years, considerable movement has been experienced in the west wall of the Selinsing Pit 4 and Pit 5 areas with weekly movement exceeding 150 mm as monitored by survey prisms, as well as tension cracking parallel to ore lode strike.

Geotechnical issues, particularly in the Selinsing pit area have been documented since Snowden began its reviews in 2012 and geotechnical work in 2016 including a site visit was the basis of geotechnical assumptions used in the Snowden (Dec 2016) PFS.

Geotechnical assessment of Buffalo Reef is limited due to lack of data. Even though considerable data collection in the form of drillhole structural and face mapping has occurred since 2014 by site personnel, this was not in a useable form as deemed by the recent reviews.

In 2017 and 2018, geotechnical reviews were carried out by Geomapping Technology and then POB. POB (2018) findings are the technical basis for the FS.

The recent reviews consisted of the following investigations:

Stage 1 – Data collation for Selinsing and Buffalo Reef areas:

- Review of current geotechnical data and evaluation history.
- Sorting of relevant data into geotechnical domains.
- Structural analysis, including:
  - Use of dips or other software for visualising joint set orientation
  - Analysis of available data for joint and rock mass strength indications
  - Assigning geotechnical characteristics to domains based on drillhole geotechnical logging, available historical data, and available face mapping data.
- Review of Selinsing pit west wall observation and movement data.
- Presentation of data in a format suitable agreed with Monument for the evaluation stage.

Stage 2 – Evaluation of Selinsing and Buffalo Reef open pit designs to:

- Assess stability of proposed final pit designs as presented in the Snowden (Dec 2016).
- Identify the potential wall steepening in east and west walls for all pits.
- Methodology to include suitable wall stability modelling with all assumptions stated.
- Recommend pit design changes.
- Recommend future geotechnical work associated with the sulphide pit advancement.

The Stage 2 assessments were made for new proposed pit designs as supplied by Monument that were made using Stage 1 data and historical geotechnical review.

Geomapping accomplished much of the data collation as documented in its (2018) report, but essentially the findings in the review by POB (2018) were used as primary reference for this report. POB (2018) reviewed all previous work and available data and presented updated open pit design parameters for use in the FS and the mine design process.

16.2.1 Macro geotechnics Selinsing Pit

In terms of macro geotechnics or description of geotechnical domains, POB (2018) for the Selinsing Pit described three major geotechnical areas which are:

- West wall
- East wall
- North and south end walls.

The observations made by POB (2018) on the west wall during a site visit on 16–19 April 2018 were as follows:

- Wide continuous cracks in the crest
- Noticeable outward bulging phyllites in the central west wall area of Pit 4
- Displacements along shallow to moderately steep east dipping faults and steep dipping faults
- Floor heave on a 30 m batter.

In terms of rock mass description, the following wall characteristics were documented by POB (2018):

- West wall:
  - Surface to current water level at 453 mRL composed of slightly weathered low strength phyllites (<40 MPa UCS) dipping moderately to the east. Phyllites tend to preferentially slab away from undulose foliation/shear planes.
  - Phyllites at some wall locations more intensely sheared and are inferred to have been mylonised and are weaker than the surrounding phyllites (estimates <20 MPA UCS). Mylonite zone containing clay-rich rock gouge was observed adjacent to the floor heave zone.
  - In summary, POB (2018) mentioned the west wall is in a metastable state and west wall rock mass is very weak.
  - A decision not to mine at present was endorsed with further mining and deepening leading to accelerated slope movement.
  - A major component of the damaged portions of the west wall above Pit 4 were due to a waste dump of largely oxide heavily weathered material set close to and forming part of the exiting pit wall.
- East wall:
  - The dominant structural feature of the east wall is a moderately steep east-dipping foliation not linked to any significant instability
  - Did not intersect any significant or unfavourably oriented faults
  - Overall stability appearance was described as adequate
  - POB (2018) noted the east wall had been mined to 70 m depth with overall slope angle of 35°.
- North and south walls:
  - The north and south walls of Selinsing Pit 4 are composed of sedimentary rocks that are more deeply weathered than those exposed in the east and west pit walls
  - The end walls appear to be stable which POB (2018) attributed to sub-perpendicular intersections of walls and foliation fabric
  - POB (2018) noted this area mined to 70 m depth with overall slope angle of 36°

Previous reviews by Golder (1997) and Snowden (May 2016) gave similar descriptions but west wall movement was not as evident as during the POB (2018) review.

16.2.2 Macro geotechnics Buffalo Reef Pit

The observations and descriptions made by POB (2018) for the Buffalo Reef Pits are summarised as:

- BRS C2:
  - Black shales occur regularly and are the source of localised instability
  - Interbedded siltstones and tuffs are the dominant rock types.

- BRS C3:
- Current oxide mining consists of a series of small narrow pits being excavated into the flanks of small hills
  - A small narrow pit observed at the southern end of BRC3 currently 45 m deep and excavated at slope angles of 30°
  - The exposed wall is formed mainly in highly weathered foliated and folded sedimentary rocks (siltstones) appearing to be stable
  - The northern area of BRC3 being excavated through siltstones with slope angles of 34°.
- BRS C4:
  - No observations made.

Snowden (May 2016) cited a lack of geotechnical data; however, indicated the following:

"Buffalo Reef comprises three deposits (South, Central, North) that are hosted within a 200 m wide shear zone parallel to and east of the Raub-Bentong Suture. The shear zone has an apparent sinistral sense, and dips eastwards. A series of discrete fault structures have been interpreted and modelled within the zone; these trend NNW-SSE and dip steeply eastwards.

The dominant rock types within the area are Permian-age argillites and limestones. The host rocks within the shear zone are graphitic shales and minor interbedded fine-grained sandstone and tuffs. Gold mineralisation occurs within a series of quartz veins and fracture zones within the main shear zone, also dipping typically 50° to 80° eastwards.

Rock-mass scale structure in the Buffalo Reef deposits comprises bedding and joint sets. Mapping data has not been reviewed for this assessment, but observations on structure within the pit walls made during the site visit are summarised below:

- *bedding is typically steep-dipping, sometimes tightly folded and reversed;*
- *at least one set of sub-vertical joints with high persistence, typically trending approximately east west, normal to bedding (crosscutting);*
- *at least one set of joints with low to moderate dip and low to moderate persistence, also crosscutting bedding.*

At Buffalo Reef, weathering has typically occurred to 30 m to 40 m."

POB (2018) indicated more geotechnical investigation would be required for full FS standard in the Buffalo Reef area. This work is in progress but the conservative assumptions from previous studies have been adopted for this FS.

For both Selinsing and Buffalo Reef, POB (2018) indicated the following should be described:

- Major rock types:
  - Hard copy plans of rock types for each pit based on in-pit mapping and borehole logging.
- Rock defects and major structures:
  - Repeating Selinsing major defect set identification in BRF with collection of foliation and joint information from existing pit walls
  - The large faults observed in both Selinsing and BRF areas need mapping and digitally modelled.
- Hydrogeology and hydrology:
  - The hydrogeology and hydrology settings for the deposits need better definition
  - Measuring water levels in existing drillholes.
- Geotechnical drilling:
  - Collect core samples for rock property, shear strength testing.

This program of work is now underway.

16.2.3 Micro geotechnics

Detailed geotechnical investigations to determine micro geotechnics or detailed local geotechnical setting have so far been largely limited to observations, but several more detailed studies have been carried out in the Selinsing pit area.

Golder (1997) carried out in-pit mapping and borehole logging as well as strength testing for the Selinsing Pit area. Golder (1997) also conducted kinematic and limit equilibrium analysis.

Snowden (May 2013) reviewed much of the data from Golder (1997), together with other available data presenting geotechnical domains for the Selinsing pit area and assigning rock property and structural data to these domains that can be largely linked to those described by POB (2018). These domains (presented in Table 16.2) can be related to POB (2018) descriptions with FW_F and FW_W relating to Selinsing Pit West Wall and HW_F and HW_W relating to the East Wall.

Table 16.2 Geotechnical domains for Selinsing Pit area (Snowden, May 2013)

Domain	Code
Hangingwall Fresh	HW_F
Hangingwall Weathered	HW_W
Orebody Fresh	OB_F
Orebody Weathered	OB_W
Footwall Fresh	FW_F
Footwall Weathered	FW_W

Snowden (May 2013) reviewed field geotechnical borehole logs that included strength estimates and found them to be largely in agreement with the Golder (1997) UCS laboratory testing. Snowden (May 2013) assigned these values to geotechnical domains for the Selinsing Pit (refer Table 16.3). The logged strength estimates from boreholes were assigned to geotechnical domains by Snowden (May 2013) and are shown in Table 16.4. Snowden (May 2013) concluded that values were in general agreement.

Table 16.3 Golder (1997) UCS test results assigned to geotechnical domains by Snowden (2013)

Domain	No. of samples	UCS (MPa)	Standard deviation (MPa)
HW_F	5	66.0	38.7
OB_F	2	101.3	27.2
FW_F	9	97.6	40.3

Table 16.4 Borehole logging field strength results assigned to geotechnical domains by Snowden (2013)

Domain	Strength (MPa)
HW_F	76
HW_W	15
OB_F	103
OB_W	74
FW_F	78
FW_W*	15

Snowden (May 2013) performed GSI (Geological Strength Index) classifications to the various geotechnical domains for the Selinsing Pit area. These are shown in Table 16.5 .

The GSI is based on:

- The degree of fracturing and “blockiness”of the rock mass
- Defect characteristics (surface roughness and infill material)
- UCS of intact rock.

Table 16.5 GSI assignment to geotechnical domains by Snowden (May 2013)

Domain	GSI	Class
HW_F	53	Fair rock
HW_W	36	Poor rock
OB_F	62	Good rock
OB_W	45	Fair rock
FW_F	53	Fair rock
FW_W	36	Poor rock

Snowden (May 2016) documented a site visit and analysis following recent wall failures in the west wall. It also served as a reference for the Snowden (Dec 2016) PFS. This review relied much on the previous information from Golder (1997) and Snowden (May 2013) but revised the GSI in light of the recent failures in Selinsing pit observed to those shown in Table 16.6.

Table 16.6 Estimated GSI ranges (Snowden, May 2016)

Deposit	Geotechnical domain	GSI range	Comment
Selinsing	Saprolite/Highly Weathered Rock	20–30	All ground above the Oxide boundary
	Footwall	30–40	
	Ore Zone	30–40	
	Hangingwall	45–55	
Buffalo Reef	Saprolite/Highly Weathered Rock	20–30	All ground above the Oxide boundary
	Footwall	30–40	
	Hangingwall	30–40	Not observed: assume same value as Footwall

In 2017, Geomapping (a local Malaysian consultancy) was scoped to collate all available geotechnical data to form a geotechnical model for both Selinsing and Buffalo Reef.

Since 2014, all geological diamond drillholes have been geotechnically logged with structural measurements on oriented core undertaken. A review of this data was undertaken and documented by Geomapping (2018). With no rock testing or strength identification made, only structural information, Geomapping (2018) did not consider this information adequate so only available face mapping data was analysed.

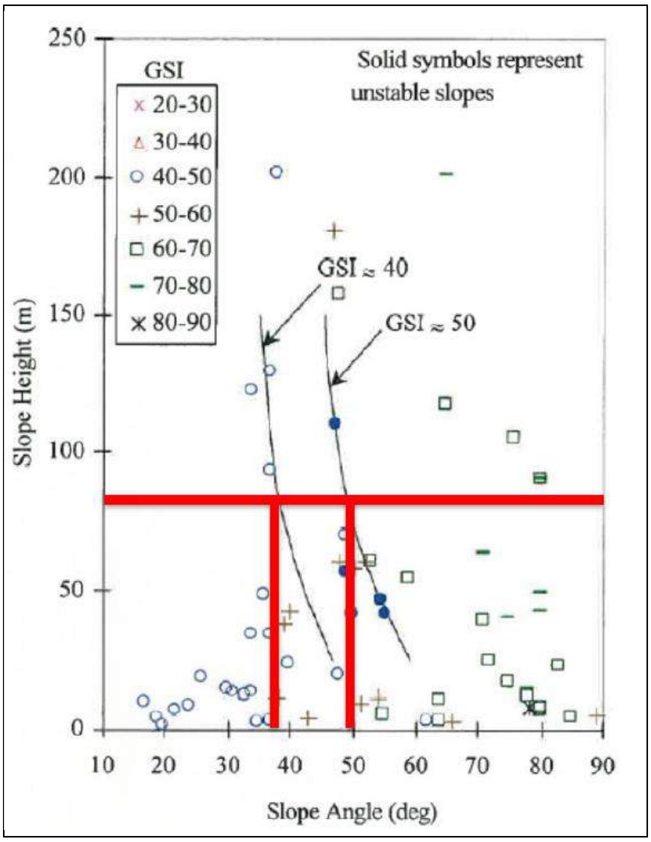
However, Geomapping collated much of the face mapping data to form stereo-plots of available information to show overall structural trends in all areas for possible later kinematic or other stability analysis. Data for Buffalo Reef was again lacking but some initial data collation was possible. In general, some correlations exist to the Snowden (May 2013) and Golder (1997) results with more data available currently for the FS.

16.2.4 Geotechnical design

Recommendations on open pit design parameters have been made several times by various reviewers including Golder (1997), Snowden (May 2013) and Snowden (May 2016) but the most recent review by POB (2013) has been adopted as a guideline for this study.

Previous work by Snowden relied on GSI assessment and empirical assessment to formulate design parameters. Figure 16.2 shows the empirical (experience based) analysis to determine likely slope angles for an 80 m stack height (vertical distance between ramps) for the Selinsing Pit area.

Figure 16.2 Empirical slope design chart for Selinsing Pit area (assuming 80 m bench stack height), Snowden (May 2016)



Snowden (May 2016) assumed similar conditions for Buffalo Reef as Selinsing since no detailed information was available in this area.

Snowden (May 2016) observed in the Buffalo Reef area with limited excavation exposure the main host rock appears like Selinsing Pit 4 footwall argillites and is expected to behave in similar manner. Argillites experiencing shearing along bedding resulting in low shear strength for those parallel to bedding show three structural sets resulting in fissile slabby/blocky rock mass conditions.

These recommendations formed the basis of the PFS of Snowden (Dec 2016) and are shown in Table 16.7 for all pit areas.

Table 16.7 All pit wall design parameters by Snowden (Dec 2016)

Deposit	Geotechnical domain	Bench height (m)	Minimum berth (m)	Batter face angle (°)	Inter-ramp slope (°)	Maximum bench stack height (m)	Overall wall slope angle (°)
Selinsing	Saprolite/Highly Weathered Rock	10	4	45	30	30	27
	Footwall	10	4	50	35	80	32
	Ore Zone	10	4	60	40	80	37
	Hangingwall	10	4	65	45	80	42
Buffalo Reef	Saprolite/Highly Weathered Rock	10	4	45	30	30	27
	Footwall	10	4	50	40	80	37
	Hangingwall	10	4	50	40	80	37

Based on observations previously mentioned in Section 16.2.1, POB (2018) revised the Selinsing Pit area pit design recommendations as shown in Table 16.8. These parameters flatten the west wall problem area but steepen the east wall. This was the basis for Optimisation assumptions for the Selinsing Pit area.

Table 16.8 Recommended pit wall design parameters for Selinsing Pit area by POB (2018)

Wall position	Geotechnical domain	Bench height	Batter angle	Berm width	Inter-ramp angle
West	Completely to highly weathered phyllites	5m	45°	4m	29°
	Moderate to fresh phyllites	5m	60°	6m	29°
East	Completely to highly weathered rocks	10m	50°	5m	37°
	Moderate to fresh limestones	10m	65°	5m	46°
North and south	Completely to highly weathered sedimentary rocks	10m	50°	5m	37°
	Moderate to fresh sedimentary rocks	10m	60°	5m	43°

The flattened west wall allows for removal or cutback of the damaged portions of the wall are which includes part of a waste dump constructed close to the current pit down to 450 mRL depth. Deepening the pit without this occurrence would lead to further instability in the west wall area. POB (2018) indicated this overlying slope is very unstable.

POB (2018) also recommended the following with regards to Selinsing Pit 4:

- The main pit ramp should not be located in the west wall as is currently the case
- Ensure batters with moderate to fresh phyllites are aligned parallel to foliation
- Partially remove the west wall waste dump so the new toe is greater than 50 m from the new pit crest
- Maximise surface drainage away from the pit wall.

These issues are taken onto account in final pit designs with ramp access in the east wall and a significant cutback of the west wall.

For Pit 5, POB (2018) recommended the following:

- Intra-ramp wall angles to be limited to 29°
- Follow all point relating to Pit 4
- Pit ramps not to traverse the unstable west wall area of Pit 4.

For Buffalo Reef, with the accepted lack of reliable geotechnical data, an initial wall design identical to that proposed by Snowden (May 2013) and Snowden (Dec 2016) as shown in Table 16.7, is used for the FS. The data identified as unavailable in Section 16.2.1 is currently being sourced, so that pit walls can be optimised.

16.3 Hydrology and hydrogeology

Hydrology and hydrogeological study information available for the Selinsing operation is very limited, with no specific studies cited.

As mentioned previously, the central Malaysian peninsula has a tropical climate, with the annual temperature ranging between 23°C and 35°C. Annual rainfall averages approximately 230 cm per annum. Peak rainfall periods are September to December and March to May.

The Selinsing and Buffalo Reef properties are approximately 500 m above sea level, and the surrounding area has relatively moderate to gentle relief. Several rivers and natural water courses traverse the tenements with flows to the northeast.

Mention was made in both Golder (1997) and Snowden (May 2016) of hydrology and groundwater issues. Golder (1997) mentioned the Selinsing pit at that stage extended below the water table and flooded when pumps in the nearby Robey shaft were turned off. Golder (1997) indicated this would be consistent with a perennial stream 1 m below the pit workings at that time.

Golder (1997) further discussed the sequence of carbonaceous shales dipping to the east with the limestone unit dipping beneath a nearby river. Potential aquifer conditions were observed in the quartz veining and the limestones. There was direct evidence of limestone being cavernous with cavities up to 2 m in thickness observed during drilling within the pit and to the south of the pit area.

Golder (1997) observed that initial pumping from the Robey Shaft at 840 m^3/day for some six days reduced the groundwater level by 3.8 m. Pumping was then reduced to 290 m^3/day for a period of nine days, during which time water levels recovered.

However, Golder (1997) observed the Robey Shaft was not located in the limestone, but several other observations were made by Golder (1997), which included:

- Drawdown away from the shaft reached a maximum of 11 m at 160 m distance from the shaft
- Drawdown in boreholes significantly less in holes closer to the shaft (0.10 m to 3.77 m) in holes within 100 m of the shaft
- Clear evidence of connection between the exploration drillholes and the shaft
- Preferred groundwater flow along fractures
- Apparent pumping rate of 1,100 m3/day was not adequate to achieve groundwater reduction.

Snowden (May 2016) also made a few observations on groundwater, which included the following:

- *“A small river along the east side of the lease is likely to set the low point of the local groundwater table adjacent to Selinsing Pit 4, with groundwater drainage from the surrounding higher land percolating towards it; the river is at approximately 490 m RL adjacent to the pit. This river could be recharging the groundwater that is draining down into the pit.*
- *The mine ponds and flooded stockpile areas located northeast of Pits 5 and 6 are also potential groundwater recharge sources and are likely to control groundwater levels in their vicinity.*
- *The sheared contact between the main shear zone and the hangingwall limestones is very damp, with seepages evident from most of its exposure in the pit walls. The shear zone trends directly below the river to the south of Pit 4, hence there is likely to be recharge from the river into the fault zone, and then into the pit.*
- *Minor inflows were observed from 25 m long horizontal drain holes in the base of the southeast sector of the Selinsing Pit 4.*
- *Water was ponding in the Selinsing pit at approximately 420 m RL.*
- *Water was ponding in the Buffalo Reef Central pit at approximately 490 m RL.*
- *Evidence noted of surface water draining into tension cracks on the west side of Pit 4; also, rainfall percolating into the waste and low-grade dumps may be draining into tension cracks beneath.”*

POB (2018) did not comment on hydrogeology but indicated that hydrogeological and hydrological settings do need better definition.

Minimal hydrogeological data is available for Buffalo Reef, but water courses traverse the area and a significant water coarse diversion is required in Buffalo Reef South, as mining progresses in the current BRC4 area down into the sulphide areas.

A pump test report for design and construction of a well tube with the objective for hydrogeological survey and to determine the location of groundwater potential was reviewed during the FS (refer Digee Trading, 2009). The test location was to the southeast of Selinsing Pit 4.

Step pumping, constant discharge and recovery tests were conducted from 20 to 24 August 2009. A submersible pump was installed at 46 m depth.

Digee Trading (2009) indicated that:

- *"Based on the drilling work, the grey carbonaceous Shale interbedded with Limestone appears to be the bedrock underlying 4 meters from the top soil at this area. The total estimated discharge from this well is 8,100 gallons/hr''*.
- *"A Step test was analyzed using Hantush-Bierschenk method and the well efficiency (We) is calculated using formula;*

$$We = 1/(1 + CQ/B) \times 100\%$$

Table 16.9 shows values of well efficiency at every water discharge capacity.

Table 16.9 Well efficiency at different step for tube well SGM-1 (Digee Trading, 2009)

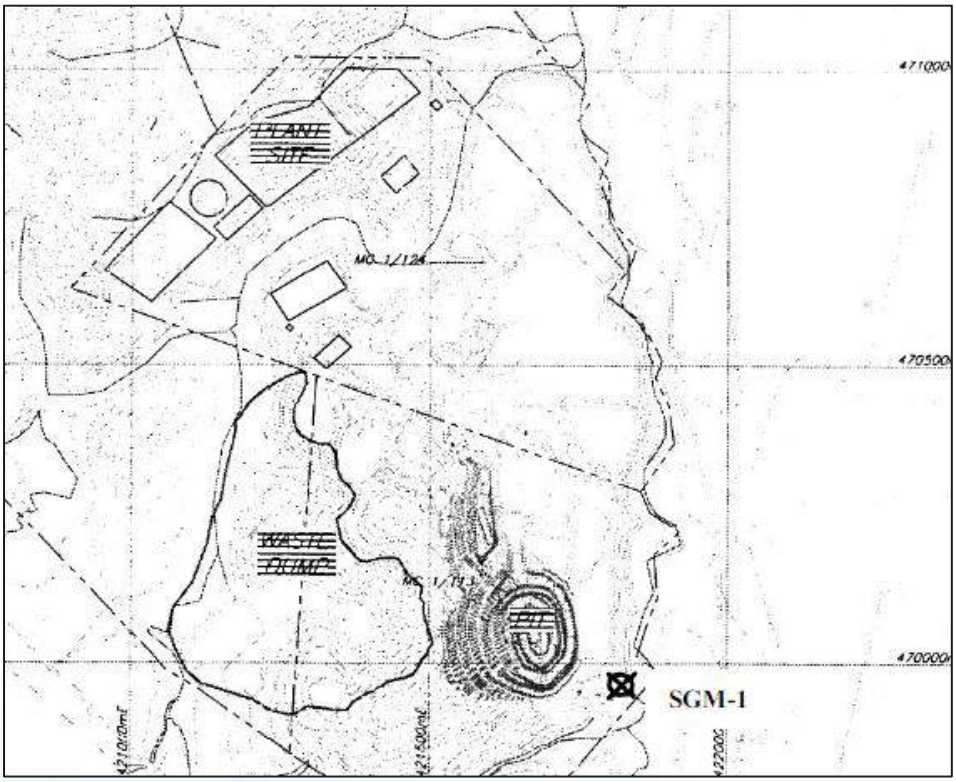
Step	*Q (m^3/hr)*	*Δs*	*S_w*	*S_w/Q*	*W_3 %*
1	4.5	1.1	1.1	0.244	88
2	9.1	0.4	0.4	0.165	78
3	16.0	0.9	0.9	0.150	67
4	20.2	0.3	2.7	0.134	62
5	25.1	0.6	3.3	0.131	57

The method of Copper-Jacob was used to analyse the constant test result. Transmissivity (T) value is 61 m^2/day.

Recovery test were analysed with this recovery method and the T values is 29 m^2/day. The value of T obtained is considered good for hard rock aquifer."

The location of the drillhole for testing was close to a river (refer Figure 16.3) which may influence the result, but indications are that aquifers are present but in this case a pumping rate of 25.1 m^3/hr measured causing a 6.83 m drawdown in 72 hours. This indicated manageable in-pit or borehole pre-mining dewatering may be possible.

Figure 16.3 Location of the SGM-1 pump test location



16.4 Mining method selection

With little change in the Mineral Resource and mining costs, continuation of the open pit mining strategy as described by Snowden (Dec 2016) is to be carried out for both the Selinsing and Buffalo Reef deposits. However, some adjustment was required to the PFS designs to account for the following:

- Geotechnical issues in the Selinsing Pit area.
- Review of open pit optimisation assumptions due to the following:
  - Selinsing overall pit wall assumption
  - Gold price assumption of US$1,300/oz as opposed to US$1,250/oz in the PFS
  - Revised sulphide processing cost of US$20.50/t of ore opposed to US$24.56/t used in the PFS
  - Revised process recovery for sulphide ore of 81% (used in optimisation only) compared to 85% for Buffalo Reef and 87% for Selinsing pit areas.

16.4.1 Mine optimisation

Mine optimisation was carried out in two phases:

1) FS open pit optimisation to identify Mineral Reserves.
2) Indicated and Inferred full potential open pit optimisation for Opportunity Cases One and Two.

The reserve optimisation shells were compared visually to those produced by Snowden (Dec 2016) to determine differences. Physical quantities and optimisation result also compared for both Snowden (Dec 2016) and the FS cases.

Optimisation was carried out using the Deswik software Pseudo Flow open pit optimisation function. Pseudo Flow much more commonly used in the industry with identical functionality to Whittle including the Geovia Whittle Four-X™ used by Snowden (Dec 2016).

All optimisation included mining and processing of oxide as well as sulphide material.

Model preparation

The geological models used were largely unchanged from those used by Snowden (Dec 2016) except for the addition of sulphur modelling in all areas.

The Surpac generated geological models used were:

- "selinsing_may2016_v1_s" (Selinsing model) initially created in 2016 then sulphur added in 2017
- "brscf_apr2016_v2_s" (Buffalo Reef South and central model) initially created in 2016 then addition of sulphur in 2017
- "brn_apr2016_v1_s" (Buffalo Reef North model) initially created in 2016 then addition of sulphur in 2017.

To run the Deswik Pseudo Flow optimisation, the Surpac models were imported in Datamine format then re-blocked for processing, all of which could be done in the DeswikCAD software. Resulting Datamine models were firstly checked against the original Surpac models then re-blocked to a suitable SMU. The re-blocking followed the same as used by Snowden (Dec 2016). The original geological model blocks were sized as follows:

- Selinsing (2.5 mE x 2.5 mN x 1.25 mRL)
- BRN and BRSC (2.0 mE x 2.5 mN x 1.25 mRL).

The re-blocked models used in optimisation were:

- Selinsing (5 mE x 5 mN x 5 mRL)

- BRN and BRSC (4 mE x 5 mN x 5 mRL)

Model properties brought over to the re-blocked model included:

- "Au_mik" – gold grade Selinsing
- "Au" – gold grade BRN/BRC
- "density"
- "OXIDE" – oxidation (oxide, fresh or transition)
- "RESCAT" – resource classification (Indicated, Inferred or Unclassified)
- "s_pct_id" – sulphur grade (not used in optimisation).

After re-blocking, the following occurred:

- Re-blocked models were cut to the most recent topography (surface survey outline)
- Formulas for costs and revenue determined and added to block model
- Modifying factors and reporting layout determined
- Pseudo Flow algorithm ran
- Results were analysed.

For the Indicated-only cases, all Inferred material was set to waste and only Indicated ore used in the processing stream.

Optimisation parameters were largely similar to those used by Snowden (Dec 2016); however, there were several notable changes:

- Gold price: US$1,300 assumed (US$1,250 assumed by Snowden, Dec 2016)
- Sulphide ore processing cost: US$20.50/t ore processed assumed (US$24.56/t ore processed assumed by Snowden, Dec 2016)
- Sulphide ore process recovery: 81% assumed (85% for BRF and 87% for Selinsing assumed by Snowden, Dec 2016)
- Overall wall slope angle for Selinsing pit – POB (2018) guidelines adopted steepening Snowden (Dec 2016) west wall assumptions to 27° for all oxidation types. Snowden (Dec 2016) assumed 27° for Weathered Oxide but 32° for the Footwall and 37° for the Ore Zone.

Final evaluation assumptions for sulphide processing and recovery do differ from the optimisation assumptions, with a lower processing cost of US$17.44/t processed and higher recoveries at 83.7% overall sulphide recovery. Optimisation results would thus be considered slightly conservative to final evaluation assumptions.

Table 16.10 shows the major cost and revenue input parameters used for the FS and Note that final FS evaluation used updated values for process costs and recovery (US$17.26/t for sulphide and 85% for sulphide recovery) not available at the time the optimisation was carried out.

Table 16.11 to Table 16.13 are the assumptions used by Snowden (Dec 2016). Wall assumptions used for Buffalo Reef were as used by Snowden (Dec 2016). However, the wall assumptions used for Selinsing are shown in Figure 16.4, and derived from the POB (2018) recommendations. Deswik software used a "rosette" for input of wall angles at specific locations determined by sectors of circle. For all oxidation stages, the same slope angles are used, as 27° the for west wall area and 31° for all other areas. An area of transition of 10° was allowed for between sectors to avoid sharp corners and inflection points (refer Figure 16.4).

Table 16.10 Input assumptions used for the FS mine optimisation

Item	SEL	BRN	BRS/C	Unit	Source
Assumptions					
Exchange rate	1,300	1,300	1,300	US$/oz	Corporate guidance
Gold price	4.08	4.08	4.08	RM/$US	Corporate guidance
Cut-off grade – Oxide ore	0.40	0.40	0.40	g/t Au	Marginal cut-off grade calculation
Cut-off grade – Transition and Fresh	0.75	0.75	0.75	g/t Au	Marginal cut-off grade calculation
Mining					
Ore mining	1.89	1.71	1.66	US$/t ore mined	Snowden (Dec 2016) – based on 2015 actual
Waste mining	1.89	1.71	1.66	US$/t ore mined	Snowden (Dec 2016) – based on 2015 actual
Depth increment	0.0006	0.0006	0.0006	US$/t/m depth below 500 mRL	Snowden (Dec 2016) – based on Minetech contract
Processing					
Oxide ore gold recovery	75%	75%	75%	% recovery	Average recovery assumption based on 2018 data
Sulphide ore gold recovery	81%	81%	81%	% recovery	FS
Oxide – mill feed ore	8.68	8.68	8.68	US$/t processed	Sel_2018 Projection_2018.03.31
Transition – mill feed ore	20.5	20.5	20.5	US$/t processed	SGSP_OPEX Details 31.05.18
Fresh – mill feed ore	20.5	20.5	20.5	US$/t processed	SGSP_OPEX Details 31.05.18
Administration					
Site-wide costs	1.4	1.4	1.4	US$/t processed	Assumed US$1.4M per annum fixed cost and 955 kt/a production
Selling					
Refining and transport	2.87	2.87	2.87	US$/oz	Snowden (Dec 2016)
Royalty	5%	5%	7%	% NSR	Corporate guidance

Note that final FS evaluation used updated values for process costs and recovery (US$17.26/t for sulphide and 85% for sulphide recovery) not available at the time the optimisation was carried out.

Table 16.11 Wall slope assumptions used in the PFS optimisation by Snowden (Dec 2016)

Deposit	Geotechnical domain	Bench height (m)	Minimum berth (m)	Batter face angle (°)	Inter-ramp slope (°)	Maximum bench stack height (m)	Overall wall slope angle (°)
Selinsing	Saprolite/Highly Weathered Rock	10	4	45	30	30	27
	Footwall	10	4	50	35	80	32
	Ore Zone	10	4	60	40	80	37
	Hangingwall	10	4	65	45	80	42
Buffalo Reef	Saprolite/Highly Weathered Rock	10	4	45	30	30	27
	Footwall	10	4	50	40	80	37
	Hangingwall	10	4	50	40	80	37

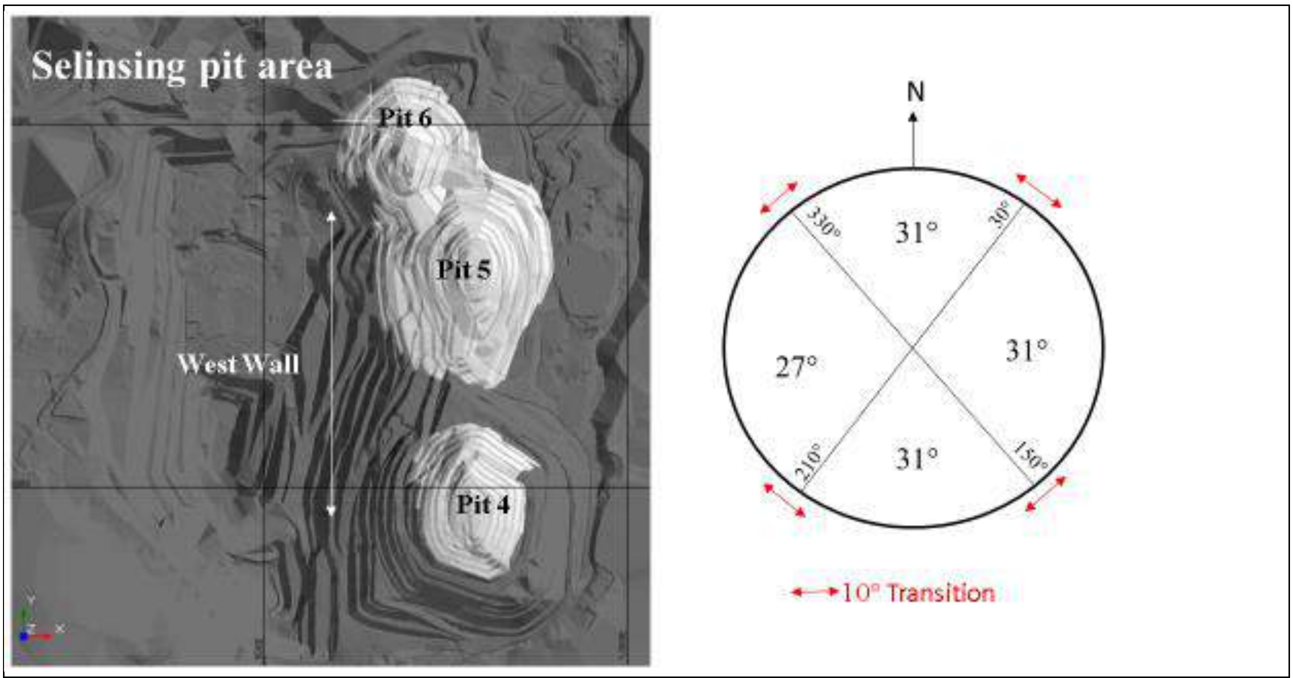
Table 16.12 Recovery and cut-off grade assumptions used for optimisation by Snowden (Dec 2016)

Rock type	Deposit	Grade (g/t)	Recovery (%)
Oxide	Both	<1.00	68.7
		1.00–1.50	74.1
		>1.50	79.5
Transition/Fresh	Selinsing	All	87.0
	Buffalo Reef	All	85.0

Table 16.13 Cost assumptions used for optimisation by Snowden (Dec 2016)

Item	Selinsing	Buffalo Reef North	Buffalo Reef South
Mining			
Waste mining ($/t rock)	1.89	1.71	1.66
Ore mining ($/t rock)	1.89	1.71	1.66
Depth increment ($t/m below 500 mRL)	0.0006	0.0006	0.0006
Processing			
Oxide process cost ($/t ore)	9.73	9.73	9.73
Transition process cost ($/t ore)	24.56	24.56	24.56
Fresh process cost ($/t ore)	24.56	24.56	24.56
Selling			
Refining and transport ($/oz)	2.87	2.87	2.87
Royalty (%)	5	7	7

Figure 16.4 Selinsing slope input for optimisation using a "Rosette" to define areas



Optimisation results

For the Indicated-only Reserve optimisation, results were compared visually to the optimisation shell produced by Snowden (Dec 2016). Figure 16.5 to Figure 16.8 shows the results for Buffalo Reef South Centre area. Table 16.14 shows the Indicated-only results for all areas compared to Snowden (Dec 2016).

Differences were identified with reported optimisation physicals in both Selinsing and Buffalo Reef South. These are attributed to changes in surface topography in all areas and in the case of Selinsing, a different wall design assumption which in all areas resulted in less inventory in the optimisation. Visual inspection was used to verify optimisations in the remaining mining areas for which close correlation was observed. The reported physicals were largely influenced by surface topography and the different mining model to Snowden (Dec 2016) were considered immaterial.

Figure 16.5 Buffalo Reef FS optimisation results

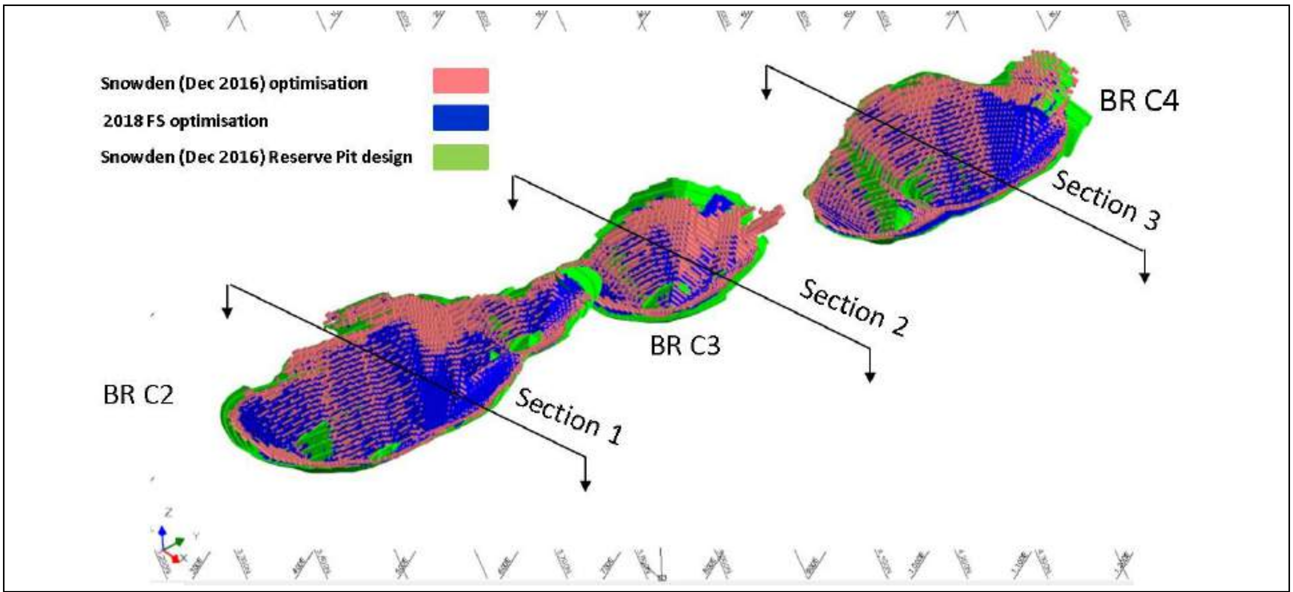


Figure 16.6 Section 1 through BRS C2 showing Buffalo Reef FS optimisation results

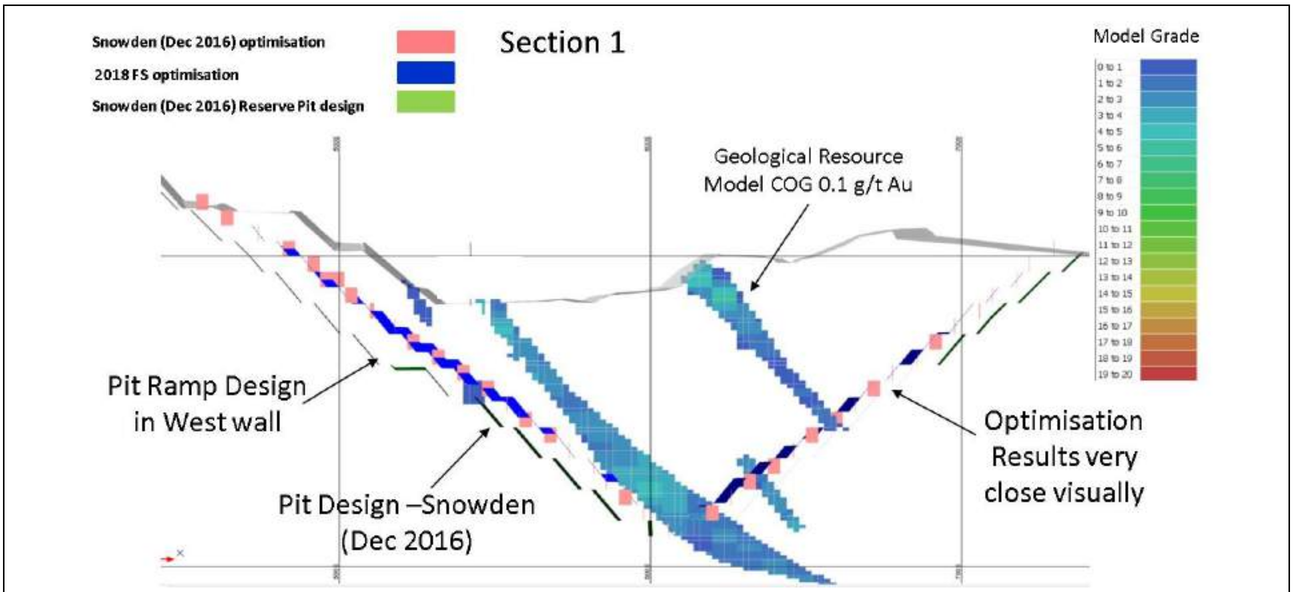


Figure 16.7 Section 2 through BRS C3 showing Buffalo Reef FS optimisation results

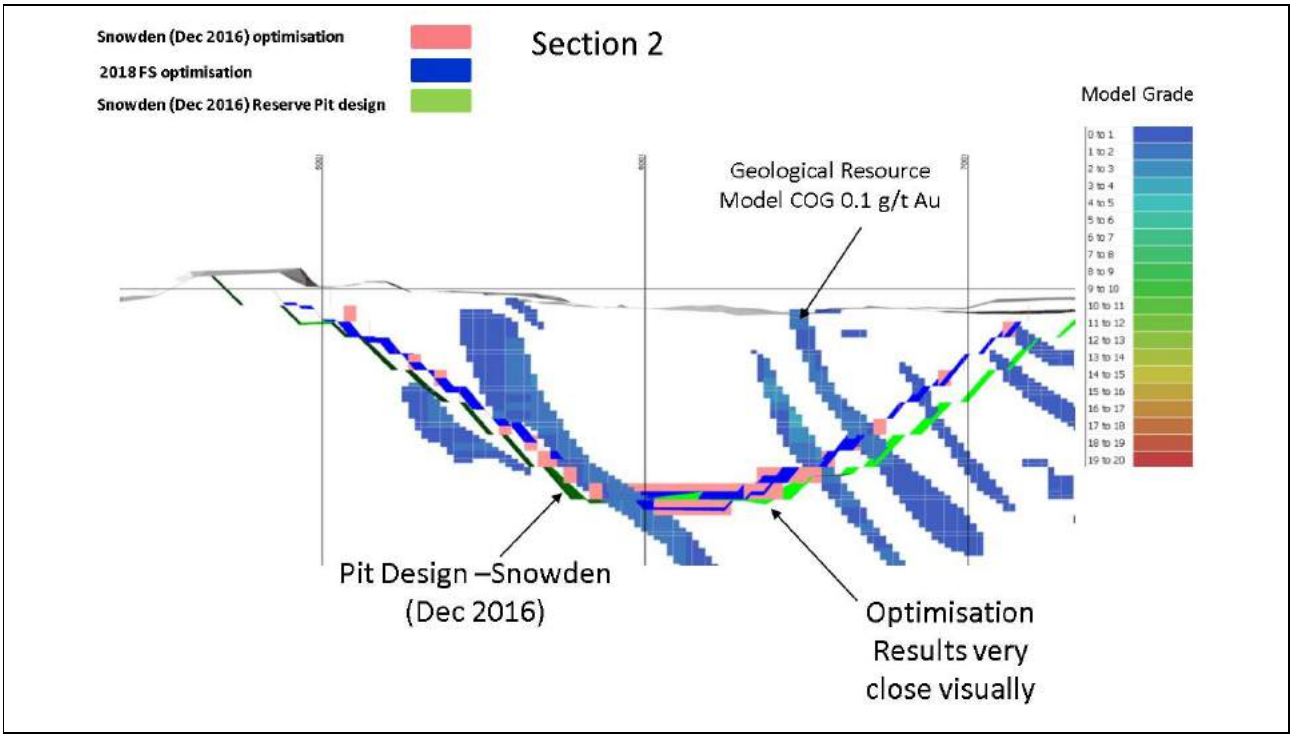
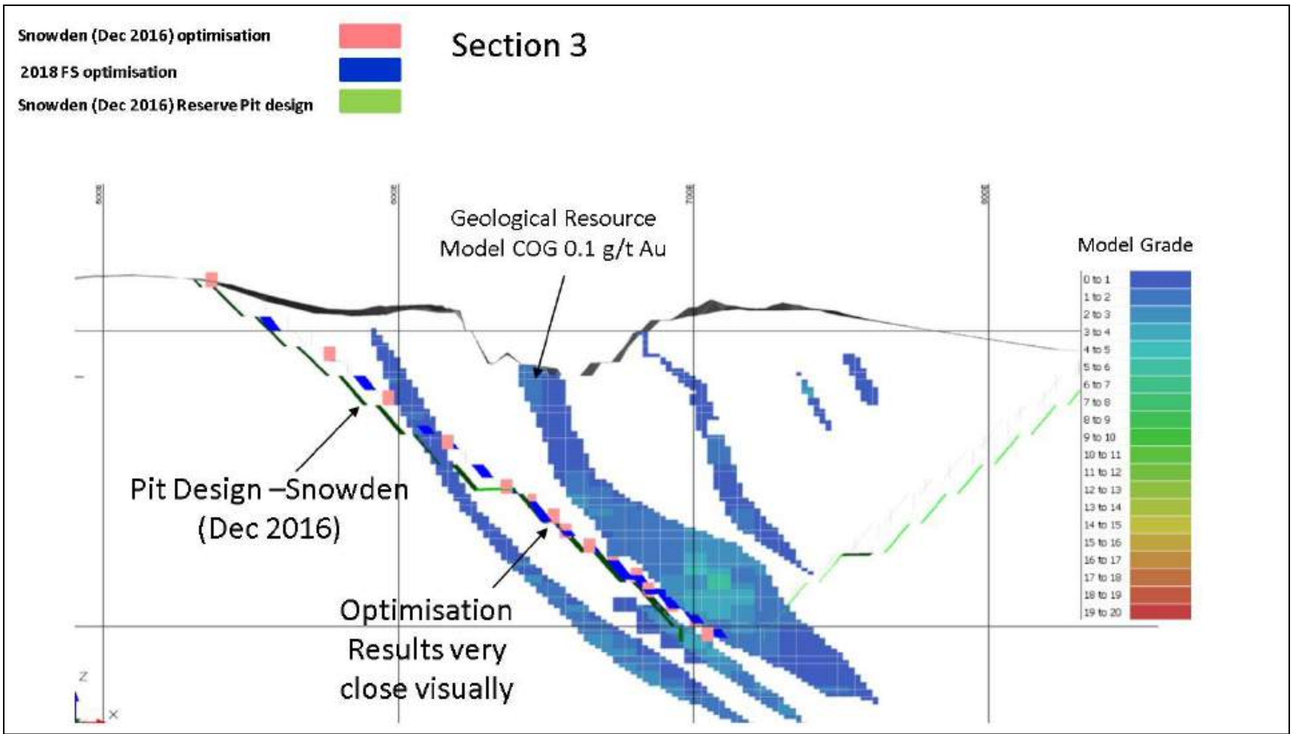


Figure 16.8 Section 3 through BRS C4 showing Buffalo Reef FS optimisation results



With optimisation essentially forming a guide to pit design, visual inspection showed results very similar to the PFS results. Therefore, Snowden (Dec 2016) designs were considered satisfactory for use for reserve estimation. However, a new design is being created for the Selinsing Pit area.

Figure 16.9 and Figure 16.10 show the Snowden (Dec 2016) and FS results for the Buffalo Reef area in graphical isometric view.

Table 16.14 FS optimisation Indicated-only results for all areas compared to Snowden (Dec 2016) PFS

Item	SEL			BRF SC			BRN			TOTAL		
	Snowden (Dec 2016)	FS	Difference	Snowden (Dec 2016)	FS	Difference	Snowden (Dec 2016)	FS	Difference	Snowden (Dec 2016)	FS	Difference
Revenue factor	1	1	1	1	1		1	1		**1**	**1**	
Physicals												
Pit size (kt)	4,594	4,594	4,594	30,418	25,517	-4,901	666	553	-113	**35,678**	**31,984**	**-3,694**
Waste (kt	3,784	3,784	3,784	27,176	22,735	-4,441	500	383	-117	**31,460**	**28,337**	**-3,123**
Strip ratio (w:o)	4.67	4.67	4.67	8.38	8.17	-0.21	3.01	8.35	5.33	**7.46**	**7.95**	**0.49**
Ore – total (kt)	810	810	810	3,242	2,781	-461	166	157	-0	**4,218**	**3,564**	**-654**
Au grade – total (g/t Au)	2.03	2.03	2.03	1.82	1.81	-0.01	1.21	1.16	-0.05	**1.83**	**1.81**	**-0.02**
Contained gold total (koz)	53	53	53	189	161.45	-28	6	5.85	-0	**248**	**208**	**-40**
Recovered metal (kt)	46	46	46	159	130.23	-29	5	4.60	-0	**210**	**167**	**-43**
Ore – Oxide (t)	54	54	54	468	223.08	-245	90	68.51	-21	**612**	**333**	**-279**
Ore – Oxide (g/t)	0.77	0.77	0.77	1.34	1.26	-0.08	1.11	1.01	-0.10	**1.26**	**1.13**	**-0.12**
Oxide contained (oz)	1.34	1.34	1.34	20.16	9.07	-11.10	3.21	2.22	-0.99	**25**	**12**	**-12.58**
Ore – Sulphide (t)	756	756	756	2,775	2,558.37	-217	77	88.80	12	**3,608**	**3,231**	**-377**
Ore – Sulphide (g/t Au)	2.12	2.12	2.12	1.90	1.85	-0.05	1.32	1.27	-0.05	**1.93**	**1.88**	**-0.06**
Sulphide contained (oz)	51.53	51.53	51.53	169.51	152.04	-17.47	3.27	3.63	0.36	**224**	**195**	**-29.13**
Economics												
Revenue	57.3	57.3	57.3	200	169	-30	6.5	6	-1	**264**	**218**	**-46**
Mining cost	8.8	8.8	8.8	51	45	-6	1.1	1	-0	**61**	**56**	**-4**
Processing cost	19.1	19.1	19.1	73	54	-18	2.8	2	-0	**95**	**69**	**-25**
Administration cost	0	0	0	-	5	5	0	0.27	0	**-**	**6**	**6**
Selling cost	3	3	3	15	12	-2	0.5	0.43	-0	**18**	**15**	**-3**
Total operating cost	30.9	30.9	30.9	138	117	-21	2	4	2	**171**	**146**	**-24**
Unit cost (US$/oz)	675	675	675	868	896	28	849	881	32	**814**	**874**	**61**
Operating cash flow	26.00	26.00	26.00	62.00	52.64	-9	2.00	1.93	-0	**90**	**71**	**-19**

Table 16.15 Optimisation results for Indicated only as well as Indicated and Inferred (Snowden, Dec 2016)

Item	Indicated only	Indicated and Inferred	Difference
Physicals			
Pit size (kt)	35,678	58,996	65%
Waste (kt	31,460	53,136	69%
Strip ratio (w:o)	7.46	9.07	22%
Ore – total (kt)	4.219	5,860	39%
Au grade – total (g/t Au)	1.83	1.98	8%
Contained gold – total (koz)	249	374	50%
Recovered metal – total (kt)	210	317	51%
Ore – Oxide (kt)	612	819	34%
Au grade – Oxide (g/t)	1.26	1.26	0%
Ore – Sulphide (kt)	3,607	5,042	40%
Au grade – Sulphide (g/t)	1.93	2.10	9%
Economics			
Revenue ($M)	263.5	397.3	51%
Selling cost ($M)	17.9	26.3	47%
Oxide processing cost ($M)	6.0	8.0	33%
Sulphide processing cost ($M)	88.6	123.8	40%
Mining cost ($M)	60.9	103.7	70%
Unit cost (US$/oz)	825	827	0%
Operating cash flow ($M)	90.0	135.5	51%

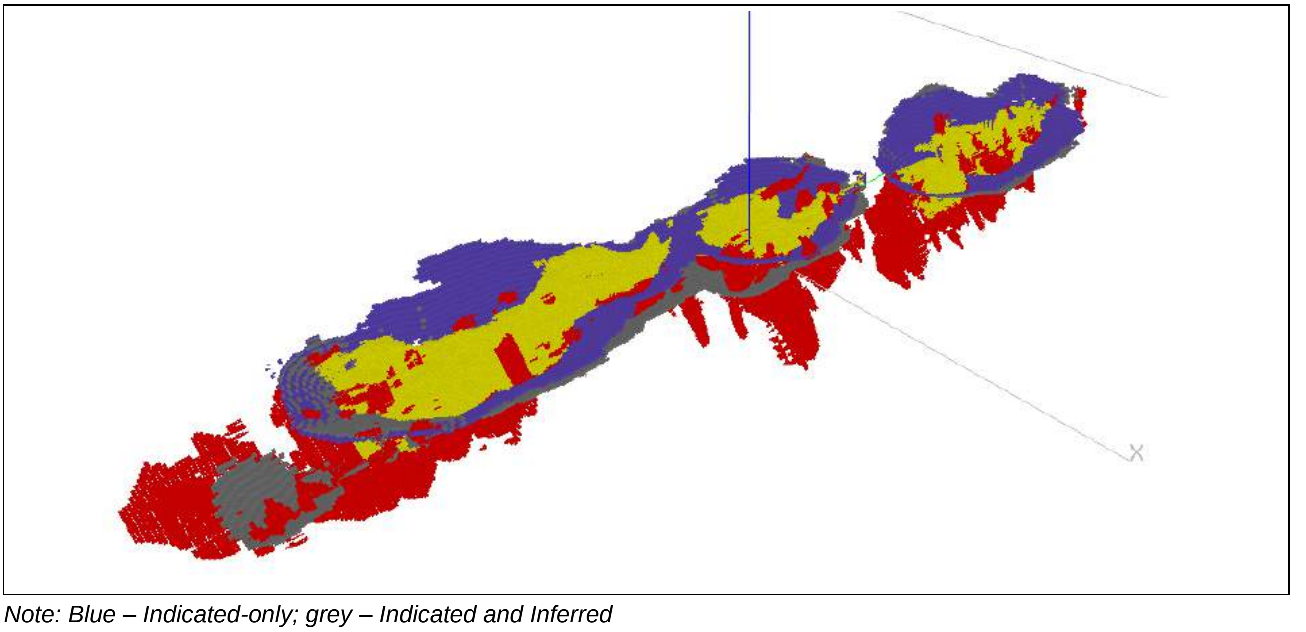
Table 16.16 FS results for Indicated and Inferred by area

Item	Selinsing	BRN	BRS/C	TOTAL
Revenue factor	1	1	1	
Physicals				
Pit size (kt)	27,654	810	35,221	63,686
Waste (kt	26,285	583	31,058	57,927
Strip ratio (w:o)	19.20	2.57	7.46	10.06
Ore – total (kt)	1,369	227	4,163	5,759
Au grade – total (g/t Au)	2.32	1.15	1.75	1.86
Contained gold – total (koz)	102	8.42	234	345
Recovered metal – total (kt)	83	6.63	189	278
Ore – Oxide (t)	149	100.81	374.99	624
Ore – Oxide (g/t)	0.58	0.99	1.27	1.06
Ore – Sulphide (t)	1,220	126.30	3,788.09	5,135
Ore – Sulphide (g/t)	2.53	1.28	1.80	1.96
Economics				
Revenue	107,340	8,617	245,644	361,601
Mining cost	47,809	1,411	63,201	112,421
Processing cost	26,307	3,464	80,911	110,682
Administration cost	2,327	386	7,077	9,791
Selling cost	5,604	622	17,737	23,964
Total operating cost	82,047	5,883	168,927	256,857
Unit cost (US$/oz)	994	888	894	2,775
Operating cash flow	25,292.83	2,734.13	76,716.73	104,744

Table 16.17 FS results for Indicated and Inferred optimisation compared to Snowden (Dec 2016)

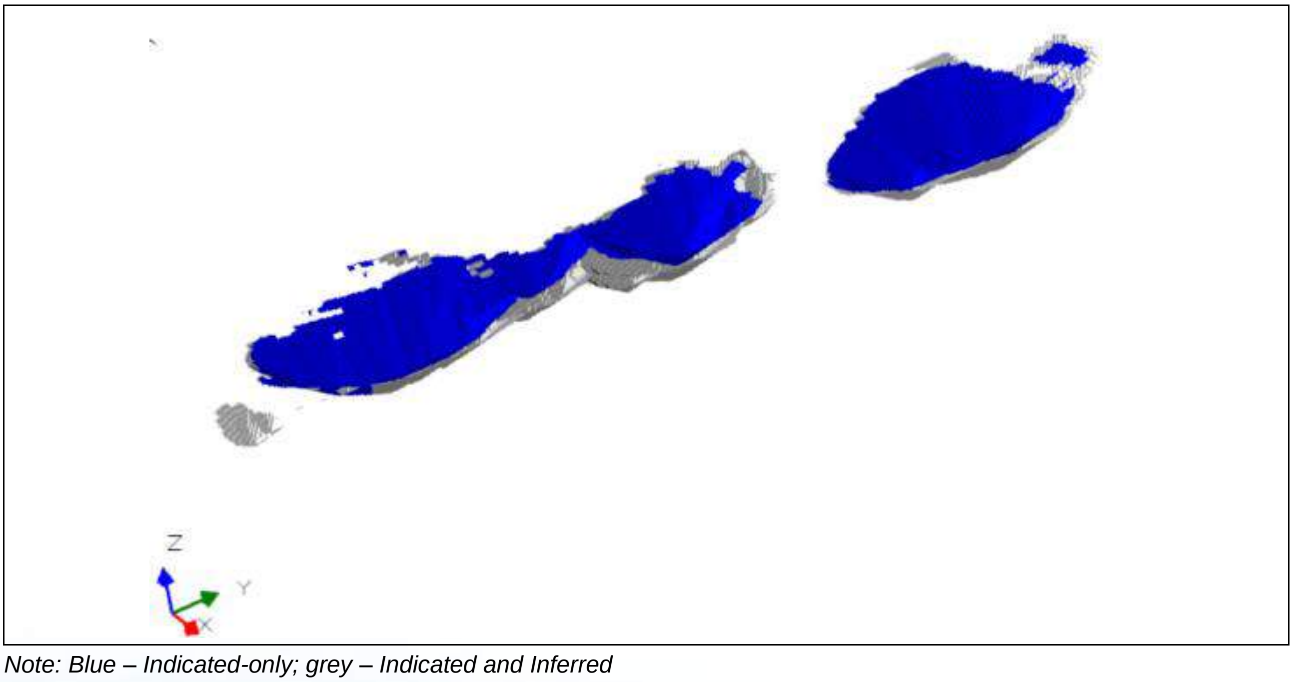
Item	Waste (kt)	Ore (kt)	Au (g/t)	koz	Total mined (kt)
Snowden 2016 (Indicated and Inferred optimisation)	53,136	5,860	1.98	374	58,996
FS optimisation	57,927	5,759	1.86	345	63,686
Difference	**4,791**	**-101**	**-0**	**-29**	**4,690**
Mined depletion estimate (Dec 2016 to Mar 2018)	1,258	193	1.57	9.76	1,451

Figure 16.9 View of Indicated and Inferred as well as Indicated-only optimisation shells for BRF SC (Snowden, Dec 2016)



Note: Blue – Indicated-only; grey – Indicated and Inferred

Figure 16.10 View of Indicated and Inferred as well as Indicated-only optimisation shells for BRF SC as determined by the FS



Note: Blue – Indicated-only; grey – Indicated and Inferred

Optimisation sensitivity

Sensitivity of gold price was considered for the Base Case optimisation. Figure 16.11 shows the results obtained for the Base Case highlighting the variance for Revenue factor (essentially gold price). A gold price of US$1,300 was assumed for the study and results at revenue factor 1. Snowden (Dec 2016) carried out a similar analysis for Indicated-only as shown in Figure 16.12.

Figure 16.11 Effects of varying revenue factor (gold price) on the FS optimisation results for contained ounces

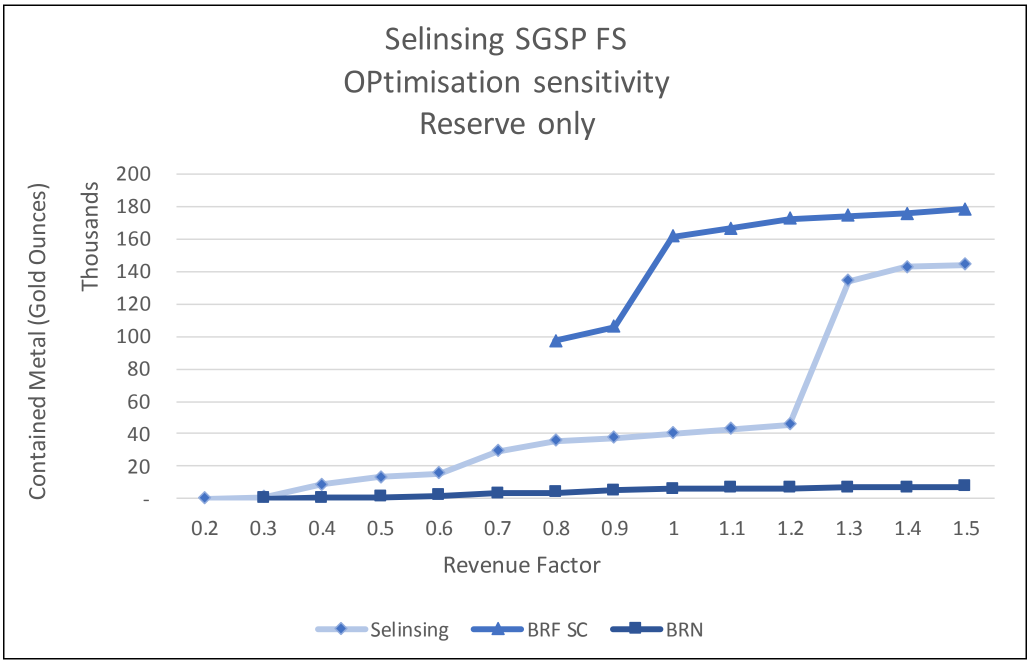
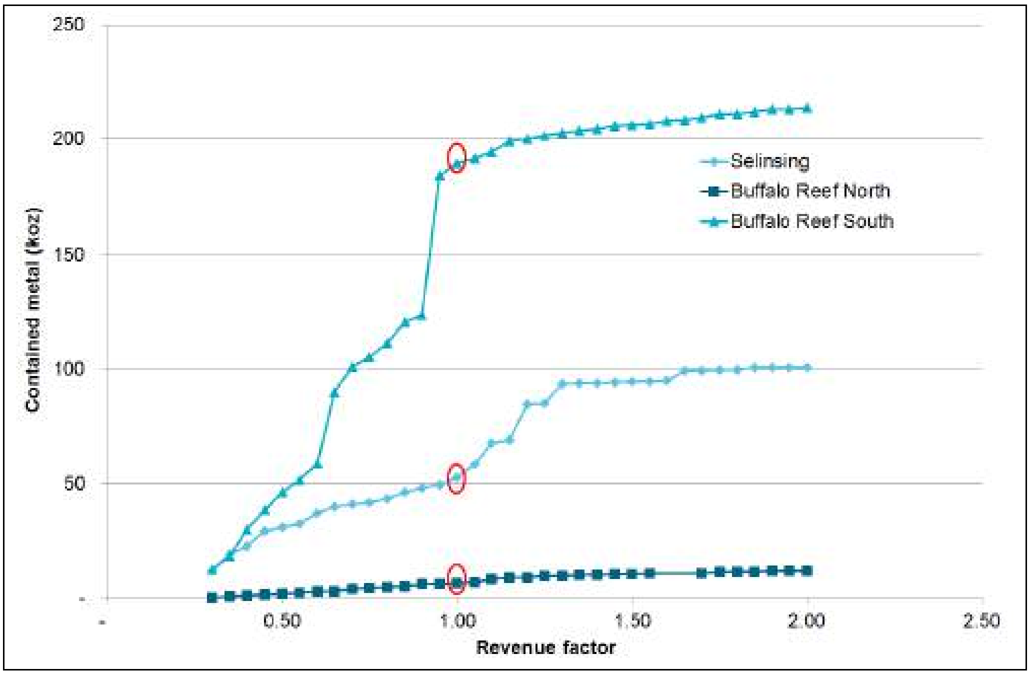


Figure 16.12 Results from Snowden (Dec 2016) for sensitivity on Indicated-only optimisation



16.5 Mine design

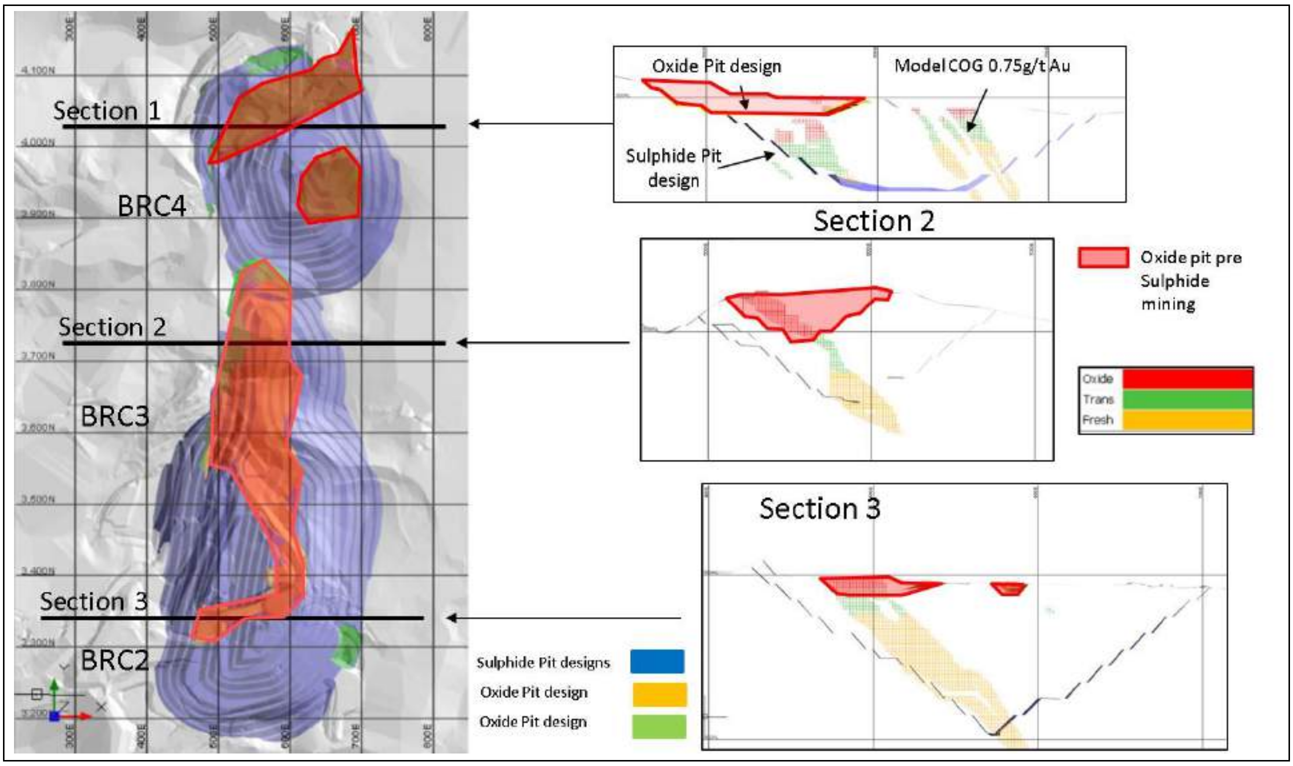
Determination of the open pit mining strategy is largely dependent on existing mining history and proven ability and capability. Pit designs accordingly have been chosen to reflect the current mining contractor capability in terms of machinery availability in Malaysia as well as productivity.

Other mining options such as expat mining contractors have been excluded from the study due to expected high costs. The Minetech mining cost unit rates are approximately 60% of unit cost in Australia.

Minetech is a Malaysian contracting company largely specialising in civil construction around Malaysia and as such has a large fleet of the trucks and other fleets used at the Selinsing site. Minetech source their labour locally including immigrant low cost labour which together with their available fleet enable the relatively low mining cost compared to the Australian mining industry.

The current mining strategy for mining oxide has consideration for the design selected by Snowden (Dec 2016) for Indicated material only with mining largely carried in areas defined by these pit designs. Once the sulphide ore production commences mining will continue as cutbacks from these areas. Figure 16.13 shows the current BRF open pit oxide mining locations together with the Reserve pit designs.

Figure 16.13 Current oxide mining locations at Buffalo Reef



Results obtained for physicals for the FS shown in Table 16.18. These physicals include all mining including oxide from 1 April 2018 as considered for reserve reporting for the Base Case.

Table 16.18 Obtained mining physicals from designs (assumed mining start from 1 April 2018)

Case	Units	FS
PHYSICALS		
Mined		
Density waste	t/m^3	2.18
Density ore	t/m^3	2.52
Oxide		
Ore tonnes mined	kt mined	483
Contained gold grade	g/t Au	1.13
Contained gold ounces	koz	17.53
Sulphide		
Ore tonnes mined	kt mined	3,436
Contained gold grade	g/t Au	1.96
Contained gold ounces	koz	217
Total ore		
Ore tonnes mined	kt mined	3,919
Contained gold grade	g/t Au	1.86
Contained gold ounces	koz	235
Waste	kt mined	36,003
Total mined (t)	kt mined	39,922
Total mined (bcm)	bcm	16,375
Strip ratio		9

16.5.1 FS Case

This case essentially includes all economic oxide and Old Tailings material as well as material contained within the SGSP sulphide-only available for mining and processing of the classified Mineral Reserves.

In general, mining is to be carried out by drilling and blasting on 10 m bench design with mined on 2.5 m to 5 m lifts utilising the existing Minetech mining contract arrangement. The sulphide production expected to be largely fresh material and an increase in drill and blast requirement is expected from current largely oxide and transition mining.

Selinsing

This FS assumes both the remaining open pit mining areas are mined targeting sulphide material and stockpiling any oxide material for later processing post the SGSP. The FS also includes stockpiled oxide material in the economic analysis.

Following the review by POB (2018), a revised mining strategy and design was required for the Selinsing Pit area.

The revised design shown in Figure 16.14 and Figure 16.15 essentially captures all the Snowden (Dec 2016) Indicated-only design but requires a flatter west wall, necessitating the removal of significant waste from that area inclusive of the current waste dump. Figure 16.14 shows an isometric view of the Selinsing Pit area showing both the Snowden (Dec 2016) PFS and FS revised. Previously, Pit 4 was referred to as Selinsing Pit 1 and Pit 5 Selinsing Pit 2 in Snowden (Dec 2016).

Figure 16.15 shows a plan with section views showing the differences between the Snowden (Dec 2016) PFS and FS designs. The revised Selinsing Pit area requiring an extra 4.45 Mt of material, largely in the west wall cutback but also in ramp required in the eastern wall as also recommended in POB (2018).

Figure 16.14 Isometric view of Selinsing open pit area with FS revised designs

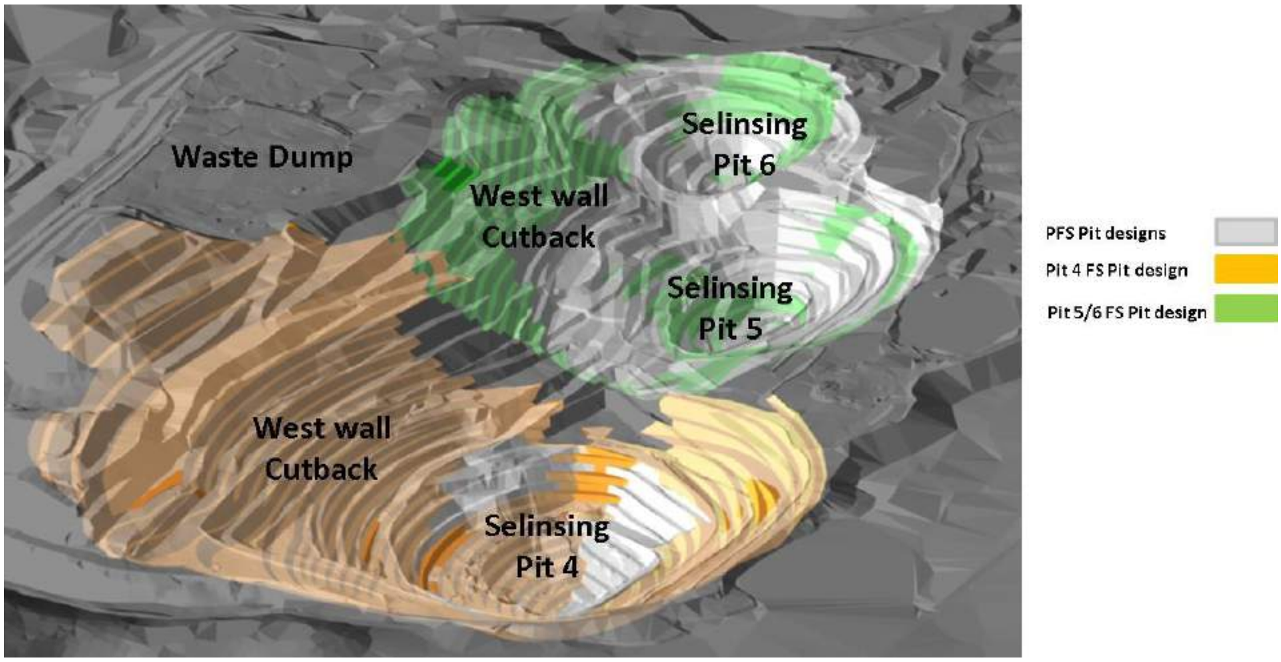
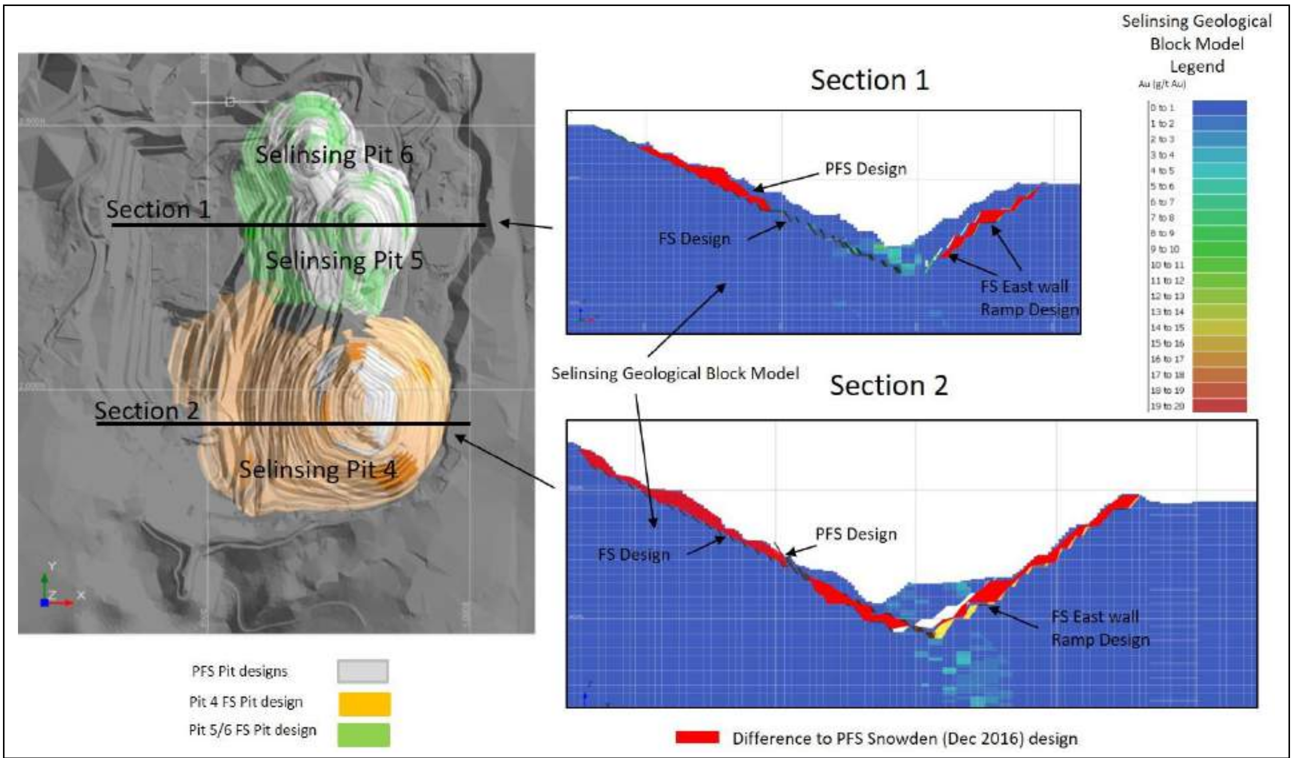


Figure 16.15 Plan and sectional views of Selinsing open pit areas with FS revised Base Case designs



Based on the review by POB (2018) the revised mining sequence for Pit 4 and Pit 5 is as follows:

- Carry out the cutback in the west wall:
  - Drain the current pit pond in Pit 4
  - Excavate the pit wall to 450 mRL to remove damaged rock
  - Partially remove the west wall waste rock dump so a new dump toe greater than 50 m from the new pit crest is left
  - Overall slope of the new relocated waste dump to be less than 22°
  - Maximise surface drainage away from the pit wall.

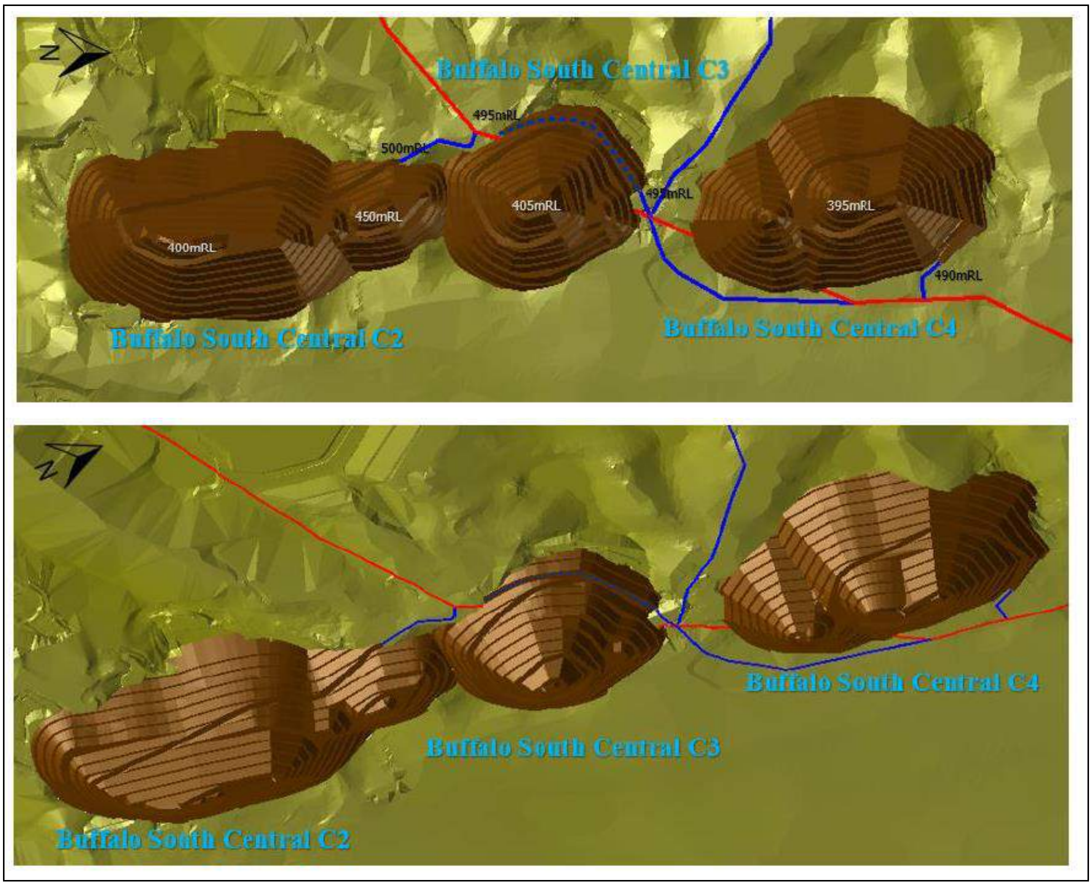
- Recommence mining of Pit 4 and Pit 5:
  - Only after west wall remediation
  - All ramp access to be located away from the west wall
  - Ensure batters within moderate to fresh phyllites are aligned parallel to the foliation.

Buffalo Reef

The proposed mine designs for the Buffalo Reef South Centre area are largely unchanged from Snowden (Dec 2016) for the Indicated only Base Case (refer Figure 16.16). Snowden (Dec 2016) created three major pit designs of BRC2, BRC3 and BRC4. An initial BRC1 design is within BRC2.

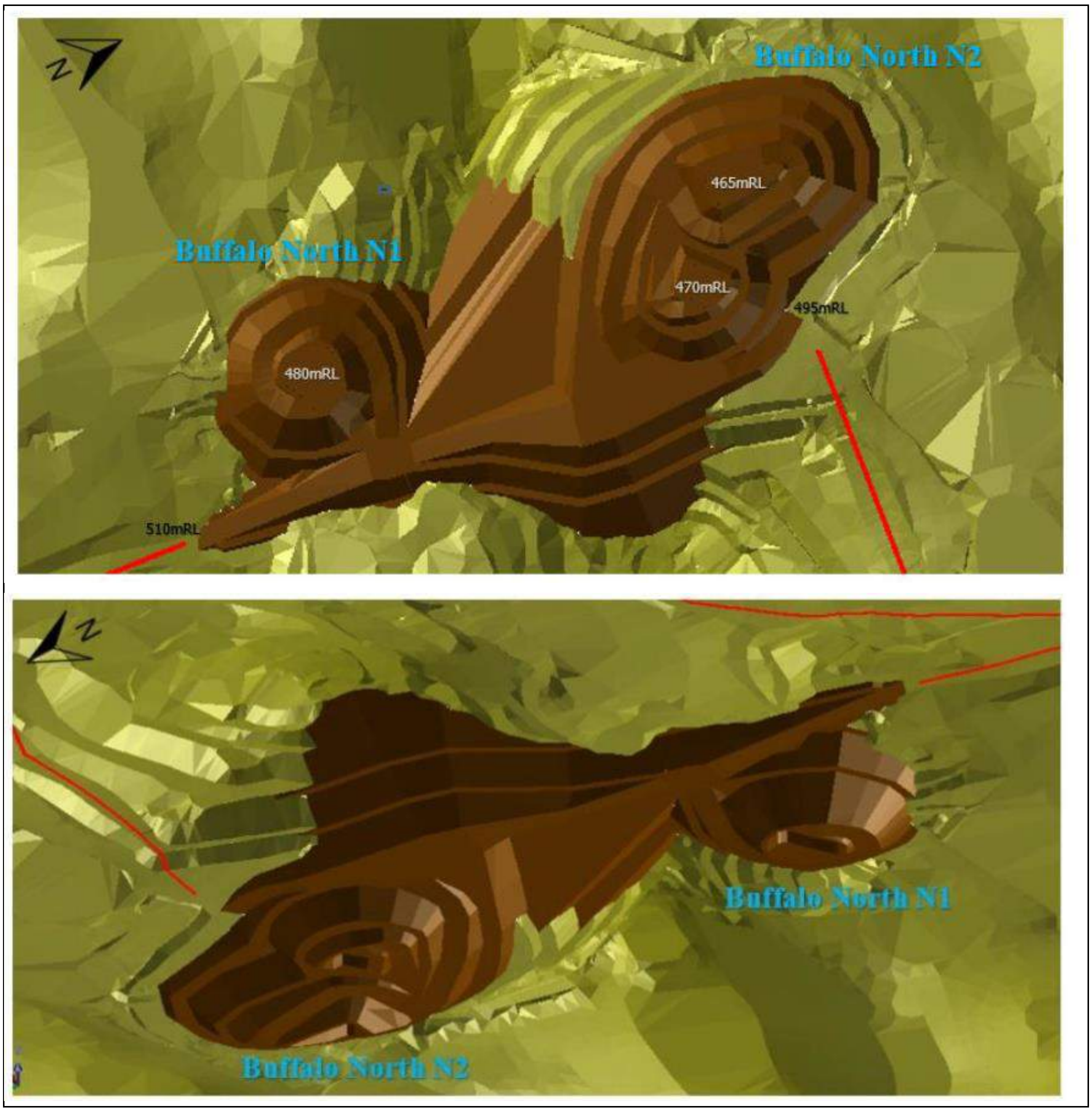
BRC3 will cut the existing haul road, requiring diversion with backfill around BRC3 as shown in Figure 16.16. Figure 16.17 shows the Buffalo Reef North designs as designed by Snowden (Dec 2016).

Figure 16.16 Buffalo Reef Central designs for the Base Case (Indicated-only), Snowden (Dec 2016)



In the Buffalo Reef North area, the contained mineable inventory is insignificant compared to Buffalo Reef SC and Selinsing and as such, only the Indicated-only Base Case designs were assumed for the FS Intermediate Case and Optimistic Case. The incremental Inferred Resource in the area is insignificant. Some mining has occurred in the Buffalo Reef North area since the Snowden (Dec 2016) work, resulting in significantly reduced inventory and different current surface topography.

Figure 16.17 Buffalo Reef North designs used in the FS as per Snowden (Dec 2016)



16.6 Bench height

Assumptions as mentioned in Snowden (Dec 2016) are assumed for the FS. Where possible, current mining has continued with a 10 m bench and ore mined on 2.5 m flitches or lifts.

The geotechnical guidance in both POB (2018) and Snowden (Dec 2016) also calls for 10 m benches except in the west wall of the Selinsing Pit in the waste dump cutback area where a 5 m bench is recommended by POB (2018).

16.7 Waste dump and waste rock management

The waste dump guidance has remained largely unchanged from Snowden (Dec 2016), being:

- Height does not exceed 10 m for visual impact reduction.
- Dump profiles:
  - Angle of repose of 1:1.5 or 34° to ensure geotechnical and erosional stability and allow revegetation.

Waste rock designs and other mining activities require conformance to the guidelines set out in the Mineral and Geoscience Department Malaysia in the approved Operations Mining Scheme.

Testwork has confirmed both PAF (potentially acid forming) and NAF (non-acid forming) waste has been identified at the Selinsing operation. Snowden (Dec 2016) proposed a waste dump design at both Selinsing and Buffalo Reef that encapsulated the PAF with NAF. The dumps are planned to be built with an initial layer of NAF forming the base followed by paddock dumping the PAF with in a PAF cell. The PAF then being surrounded by NAF which would be compacted by haul truck movement. The waste dump design is to incorporate features to minimise the effect of leaching of contaminants."

The waste dump is initially constructed with the natural rill angle of approximately 37°. This is then to be contoured to 20° to allow for slope stability and revegetation. Low grade and ore stockpile dumps are constructed within the context of the mine schedule.

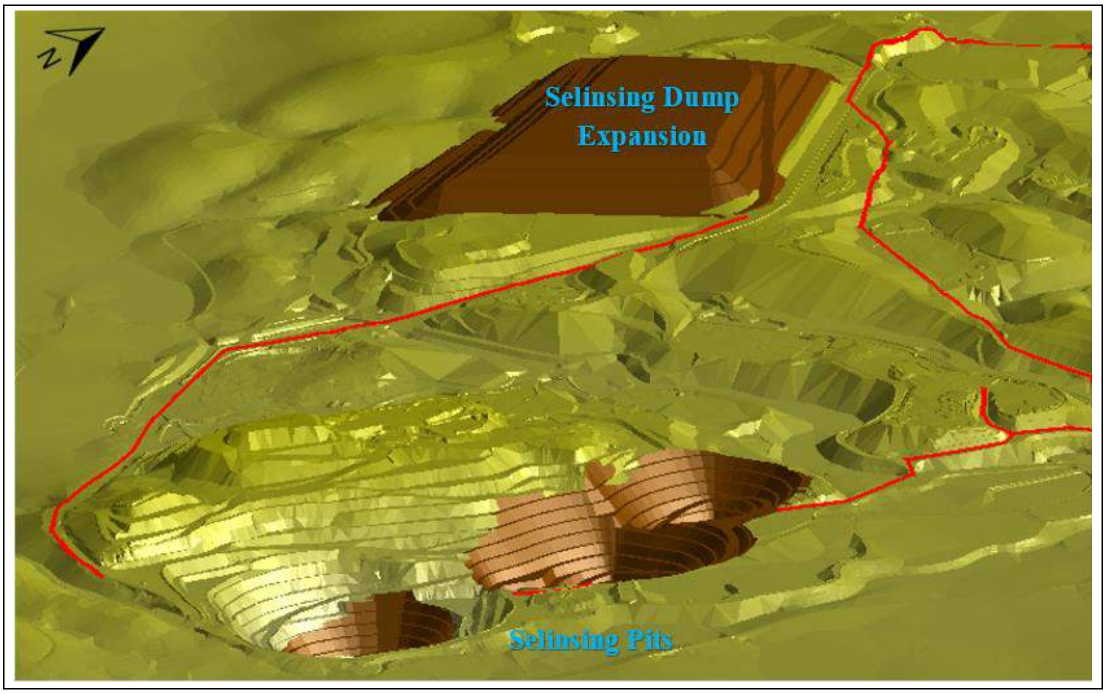
The waste dump is progressed by tipping from a higher level against a windrow and progressively pushing the waste out with a dozer.

Waste dumps have and shall continue to be progressively rehabilitated with topsoil, where possible. Surfaces of dumps shall be contoured to minimise batter scour and ripped at 1.5 m centres to a depth of 400 mm, where practicable. All such rehabilitation work has and shall continue to be carried out progressively. Rock-lined drains are constructed, where required, to ensure excess runoff is controlled and directed down to sediment traps. Good mining practice is executed such that mine depleted sites have and shall continue to be rehabilitated to a compliance at the cessation of mining.

Prior to any pit and dump development being undertaken, vegetation, subsoil and topsoil is removed and stored in a specific location. It has been and shall continue to be re-spread over permanent waste dumps and bare slopes".

In terms of final waste dump designs Snowden (Dec 2016) proposed a new waste dump for the Selinsing area. The POB (2018) review to relocates a large amount of the current Selinsing waste dump and the waste dump proposed by Snowden (Dec 2016), having a capacity of 4.6 Mbcm to a height of 570 mRL will have adequate capacity for the Selinsing pit area FS Case requirements. The dump expansion is shown in Figure 16.18.

Figure 16.18 Proposed Selinsing waste dump expansion Snowden (Dec 2016)



Most waste generated will be from the Buffalo Reef area. Snowden (Dec 2016) proposed a new waste dump to the northwest of BRF SC to service the BRFS SC and BRN areas as shown in Figure 16.19. This dump has a total capacity to 570 mRL of 6.8 Mbcm, and to 610 mRL is 15.6 Mbcm. Thus, a total of 20.2 Mbcm of storage is available based on Snowden (December 2016). Figure 16.19 shows this waste capacity is more than ample for the FS Case.

The proposed waste locations are adequate for the FS Case. Figure 16.19 and Figure 16.20 show the complete site plan with waste dump and haulage route locations by Snowden (Dec 2016).

Figure 16.19 Proposed Buffalo Reef waste dump expansion (Snowden, Dec 2016)

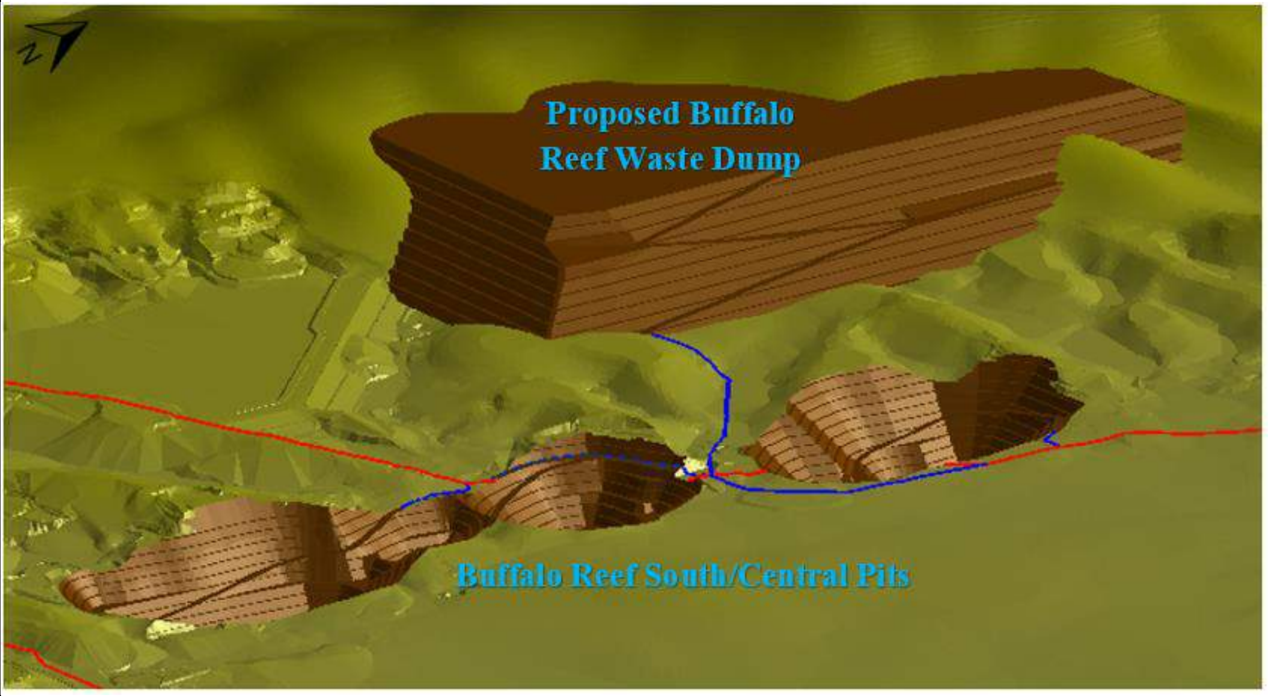
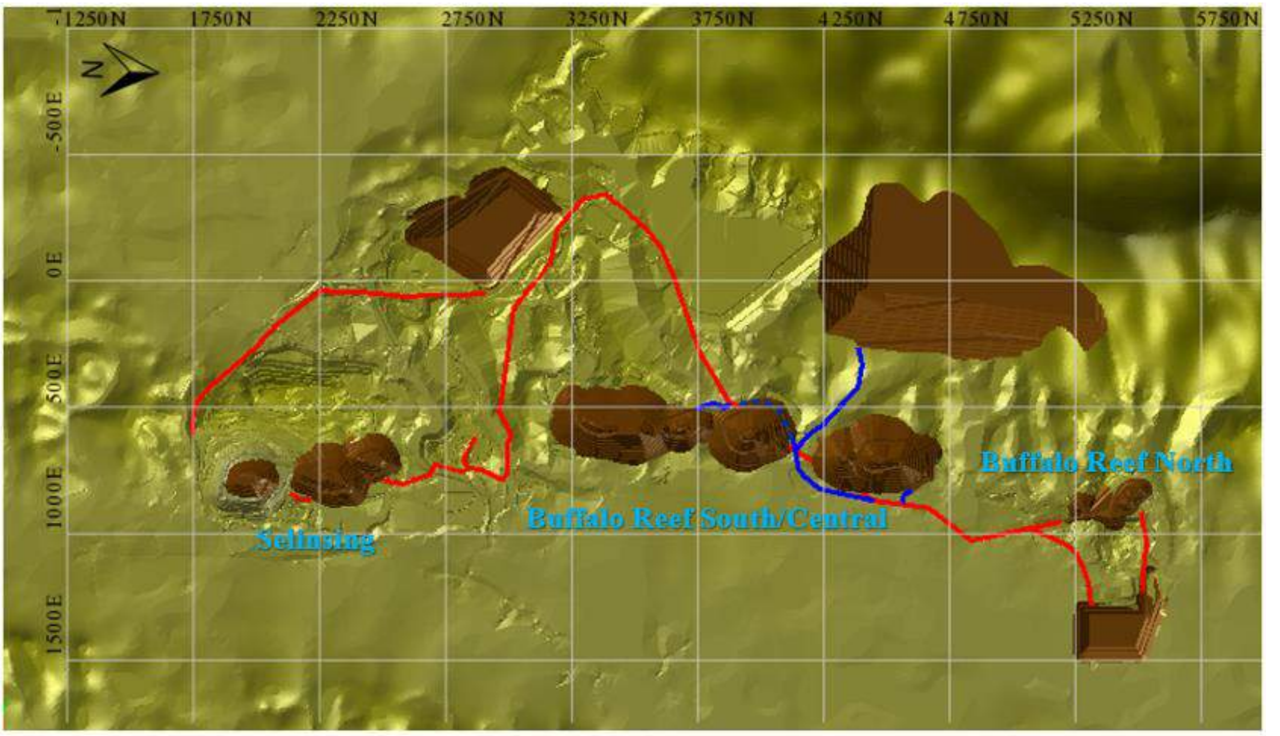


Figure 16.20 Proposed waste dump expansion locations and haulage routes (Snowden, Dec 2016)



16.8 Top soil stripping and spreading

Topsoil stripping is well advanced in most areas of open pit mining due to oxide pit mining but some areas extending to final pit limits area required.

The new waste dump for Buffalo Reef would require stripping and spreading of topsoil. A plan of action for waste dump construction is required which would include the storage of vegetation and topsoil for later permanent rehabilitation of waste dumps.

16.9 Stockpiling and reclaiming

Similar assumptions were made to Snowden (Dec 2016) with stockpiling with approximately 40% of all ore sent directly to the ROM pad is stockpiled and rehandled due to ore blending and scheduling requirements.

16.10 Mine equipment

Mining equipment selection has been largely left as that described in Snowden (Dec 2016). A list of available equipment is shown in Table 16.19. This list was sourced from the Minetech mining contract referenced as Able Return Sdn Bhd (2016) and forms the basis of mining costs.

Table 16.19 Available equipment from Minetech

Item	Description	Capacity
Dump truck	Locally built of road rear dump trucks	7 bcm
Dump truck	Locally built of road rear dump trucks	20 bcm
Drills	Tamrock CHA 1100	89 mm and 76 mm diameter
Excavator		4 bcm
Excavator		2 bcm
Dozer	Caterpillar D9N	
Grader		14 ft
Roller compactor		10 ton
Hydraulic rock breaker	EX 300 fitted	

The contractor is responsible for manning, maintenance and supply of all mining equipment as well as mining associated services including drill and blast.

16.11 Pit dewatering and drainage

Similar assumptions were made as Snowden (Dec 2016) noting water permits for the Selinsing and Buffalo Reef areas are issued by the Mineral and Geosciences Department (JMG) of the Federal Land Authority to the leaseholders. The site water supply is drawn from a local river, from which there is no limit on how much can be drawn.

Snowden (Dec 2016) stated, *"The key operational requirements are to:*

- *Minimise water flows into the pit using perimeter bunds, drains and fill, where practicable*
- *Provide pit pumping capacity to provide an unsaturated rock mass below the bench sub-drill*
- *Maintain pit wall drainage*
- *Provide permanent and temporary sumps capable of handling the peak water inflows*
- *Install settling ponds for the removal of silt prior to discharge off site."*

Section 16.3 shows that from available pump testing results, inflows from aquifers would be manageable with available pumping. All runoff to pits in the regular heavy rainfall periods is also being managed in oxide pits.

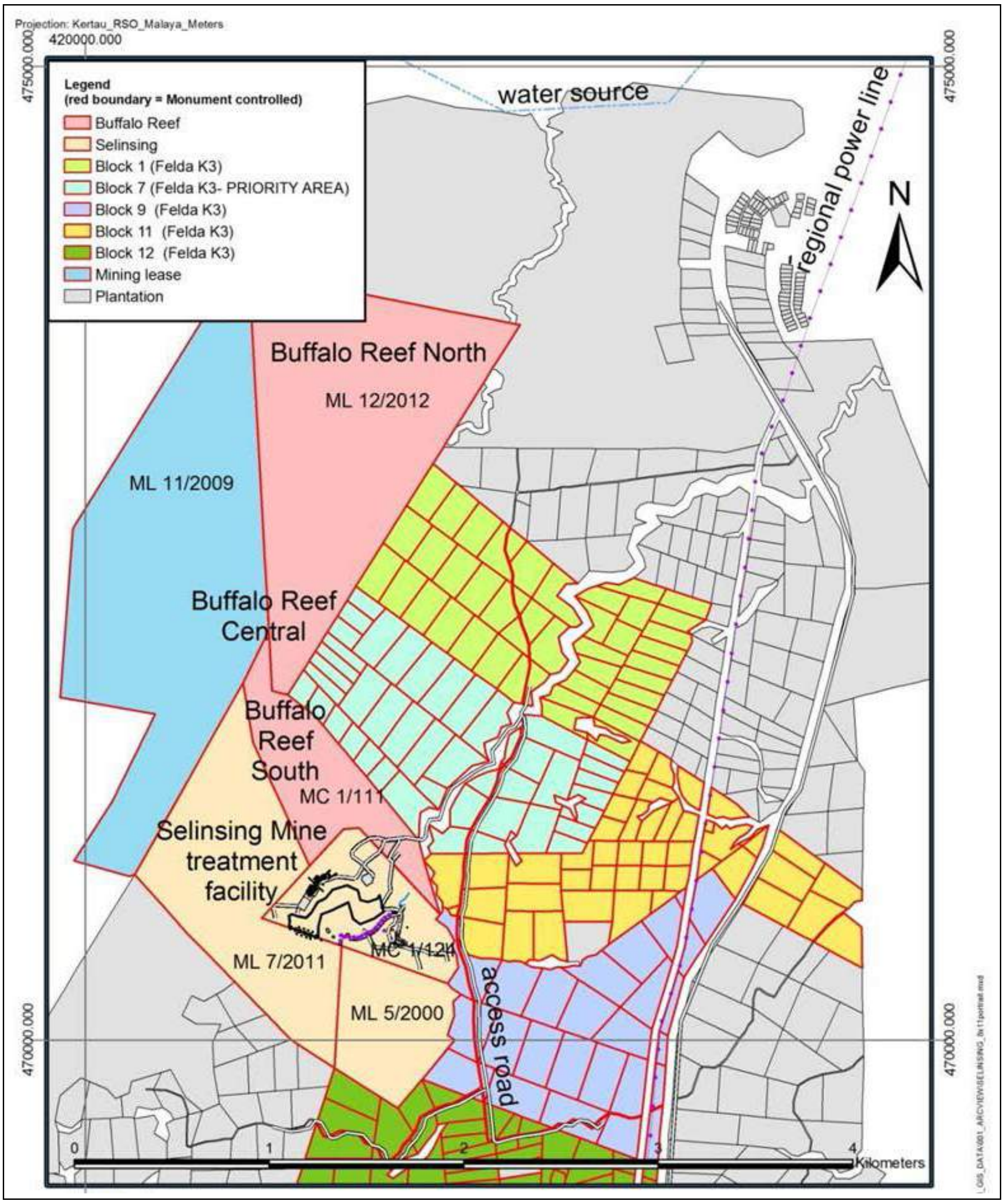
16.12 Site layout

Snowden (Dec 2016) provided a site layout plan in terms of general location of facilities, as shown in Figure 16.21.

Mining related infrastructure includes offices, preparation laboratory, core and sample storage, and mining contractor hard stand.

Proposed processing plant layout and feed arrangements are described in Section 18.

Figure 16.21 Site layout



16.13 Manpower

Manning of the mining fleet will be the responsibility of Minetech, but a team of professional mining personnel will service the mine in terms of:

- Mine planning and scheduling
- Grade control and reconciliation
- Contractor management
- Mining related mine management.

As far as manning is concerned, the mining related manning was observed by Snowden (Dec 2016) is as follows:

- Administration:
  - 13 employees including Site Manager, Site Engineer, Safety Officer, Site Supervisor (Mining) and Site Supervisor
- Maintenance:
  - Eight employees consisting of Chief Mechanic, Mechanic, Mechanic Helper, Diesel Clerk and Tyre Man.
- Operators:
  - 55 operators which consist of Trip Recorder, General Worker, Excavator Operator and Tamrock Drill and Truck Operators.

Increases in contract manpower are expected to meet the planned mining rates.

16.14 Mine infrastructure

16.14.1 Mine services

No major installed services are in use except for dewatering in the open pit areas. Mine offices for mining and geological personnel service the area. Minetech has a maintenance and operating area allocated.

In relation to dewatering and drainage, Snowden (Dec 2016) mentioned the following:

"Water permits for the Selinsing and Buffalo Reef areas are issued by the Mineral and Geosciences Department (JMG) of the Federal Land Authority to the leaseholders. The site water supply is drawn from a local river, from which there is no limit on how much can be drawn.

The key operational requirements are to:

- *Minimise water flows into the pit using perimeter bunds, drains and fill, where practicable*
- *Provide pit pumping capacity to provide an unsaturated rock mass below the bench sub-drill*
- *Maintain pit wall drainage*
- *Provide permanent and temporary sumps capable of handling the peak water inflows*
- *Install settling ponds for the removal of silt prior to discharge off site."*

These are unchanged for this FS.

16.15 Loading and haulage

Snowden (Dec 2016) mentioned the load and haul fleet was selected to achieve close to 8,000–10,000 t/d movement. This was achieved with fleet of 28 x 30-t (7 bcm) and 2 x 80-t (20 bcm) capacity trucks and afore mentioned excavator fleet.

Snowden (Dec 2016) also noted ore is hauled to the ROM pad adjacent to the primary crusher or nearby stockpiles. Waste rock is hauled to separate waste dumps as well as retention walls for water control and tailings storage.

Some concern is evident with capacities far greater than 8,000–10,000 t/day movement are required for the SGSP. Alternatives required included increasing fleet size and operating on night shift.

In relation to stockpiling for the mining of sulphide ore, ore will be stockpiled prior to commencement of the SGSP for start-up treatment as mining is ramped up.

During the SGSP commissioning period, sulphide material will be either stockpiled or fed direct to the ROM and primary crusher.

Generally, it is expected processing would be prioritised on grade but studies to date have shown there may be other issues to be taken into consideration, being:

- Sulphur – management of the BIOX® process
- Stibnite – in areas of Buffalo Reef C4 and can adversely affect process recovery
- Buffalo Reef North Oxide – is high in sulphur to be possibly only treated by the SGSP.

Testwork completed on high stibnite content areas has shown process recovery can be lower but remaining economic and above the 68% Base Case breakeven sulphide processing recovery (refer to Section 22). A discussion of the occurrence of stibnite particularly in the QS (quartz-stibnite) mineralisation type at Buffalo Reef is given in Section 13.4. The areas affected are largely at the base of BRC4 pit and very concentrated (in particular, QS zones). These are mined in the final few months of SGSP production, making up less than 6% of total feed tonnage. Worst case scenarios on final months production could possibly see total project cash flows reduced by 5%.

Sulphur grade effects on processing are largely on the BIOX® process control management with adjustments of lime addition to balance sulphur grades required.

The production schedule has two to five sources of production areas in quarterly periods; hence, areas of high stibnite can be stockpiled for either campaign processing (if process methodology can be found for high stibnite ore) or blending with other sources. High stibnite content ore is processed when the base area of BRC2 and BRC4 are processed in quarters 7 and 17.

Similarly, high sulphur content areas can be blended other areas to smooth out sulphur content.

16.16 Drill and blast

Snowden (Dec 2016) assumed 90% of the ore and 92% of the waste will require drill and blast, with drilling and blasting considerations as follows:

- The entire rock mass requires grade control drilling
- With groundwater present, bulk emulsion explosives will be required
- Drilling is to be carried by Minetech with 89 mm Día rotary blast-hole drills drilling to 10 m depth with 0.7 m to 0.8 m sub-drilling
- The average powder factor is 0.22 kg of explosives per tonne of rock
- Non-electric initiation will be used.

These assumptions were also used in the FS.

16.17 Grade control and reconciliation

A representative sample of the drill cuttings produced from blast holes is used for grade determination. The samples are analysed in the on-site laboratory designated for this purpose.

A grade control model is to be produced for the SGSP to represent not only gold but also sulphur, to assist in BIOX® management. Stibnite and arsenic are also to be tracked in the Buffalo Reef areas as they also can be problematic in ore processing.

Currently, mining data is reconciled on a monthly basis to both grade control models and resource models in use. The data is however limited to total ore comparison with separation of Indicated and Inferred and unclassified portions difficult. To date also mainly oxide and transition are mined.

Table 16.20 shows the reported mined ore to resource model and grade control model reconciliation. Results show positive reconciliation for both tonnes and ounces to the resource model. The Mineral Resource model being used for this study is unchanged.

For choice of mining dilution and recovery for the FS and with largely unknown reconciliation for fresh sulphide, assumptions of 98% recovery and 2% dilution with 0.1 g/t Au grade was assumed. This assumption is less than or more conservative than the reconciliation data available.

Table 16.20 Reported mined ore reconciliation summary for 2017/2018 year-end data for Buffalo Reef

MCF	Project to date			FY 2018 year to date		
	Tonnes	Grade	Ounces	Tonnes	Grade	Ounces
Actual vs Models Reserve*	111%	103%	105%	105%	123%	129%
Ore Mined vs Models Reserve	112%	95%	107%	105%	125%	131%
Ore Mined vs GC Delineated	97%	99%	96%	94%	124%	117%

** Largely daily production records only.*

16.18 Power

Mining requirements for electrical power are limited to electrical power for mining offices and the contractor yard. Section 18 covers the detailed site-wide electrical power upgrade required largely for the processing plant.

16.19 Mine scheduling

The process of mine scheduling involved the following:

- Inventory determination:
  - Determined by 5 m slice by pit for all design options using the Surpac software querying the resource block models on cut-off grade values of 0.4 g/t Au for oxide and 0.75 g/t Au for sulphide.
  - Inventories were constrained to mine surface topography as at 1 April 2018.
  - Separate SGSP (no oxide mining pre-SGSP) cases were generated with the current oxide pit designs were also depleted from the surface topography.
  - Oxide mining schedules before the SGSP (current 2018/19 FY plan) were separately generated.
    - Individual pit inventories by 5 m slice for parameters, including: ore and waste tonnes, bcm, grade, contained ounces, sulphur content, stibnite (Sb) and arsenic (As).
- Scenario schedules:
  - Compiled in Microsoft Excel using standard practice pivot tables and lookup sheets with the following features:
    - Overall automatic mining schedule control by monthly volume assumption and start date and top-down mining at a predetermined fixed rate by volume for each pit.
    - Manual stockpile scheduling.
    - Manual processing scheduling.
    - Input of parameters for:

        - Mining recovery and dilution
        - Mining cost
        - Processing recoveries and production constraints.
    - Output summary sheet generation containing monthly data for:
        - Mining physicals by pit and total mine
        - Mining costs for waste and ore
        - Processing feed schedule physicals adjusted from the manual stockpile schedules
        - This sheet to be fed directly into the cash flow analysis.
- CIL plant-only processing schedules both pre and post the SGSP were manually generated where as SGSP (sulphide only) were generated with the mining schedule also manually at a fixed production rate of 955 kt/a.

The assumptions used for the schedule were:

- Mining dilution 2% at 0.1 g/t Au
- Mining recovery 98%
- Mining costs as described in Section 16.19.2
- Processing plant capacity of 955 kt/a of sulphide ore
- Mining schedule adjusted to meet processing plant requirement
- Mining sequence will follow that of Snowden (Dec 2016), with pits largely sequenced by grade trending northward from Selinsing Pit 4 and Pit 5, the BRC2, BRC3 and BRC4, then BRN.

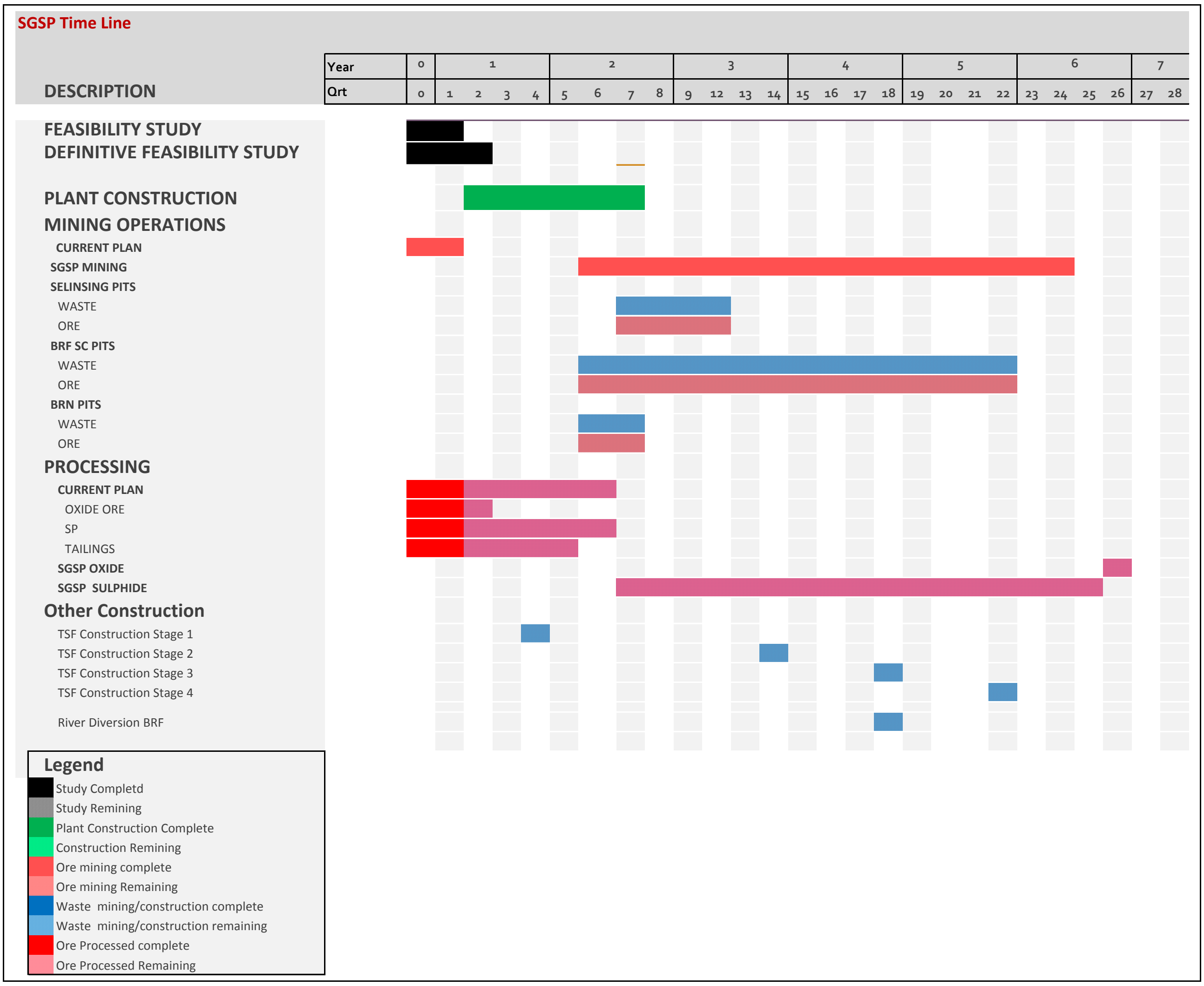
In terms of the SGSP schedule, the key milestones of the mining schedule are:

- Project start – 1 July 2018
- Process Plant construction commencement – 1 January 2019
- Process Plant completion – 1 January 2020.

The mining production need to start in advance of Process Plant completion to stockpile enough sulphide material for processing start-up.

Figure 16.22 details the Selinsing Gold Processing Project timeline which shows the afore mentioned key milestones as well as required (SGSP) mining and processing periods following plant construction and project study phases.

Figure 16.22 SGSP timeline

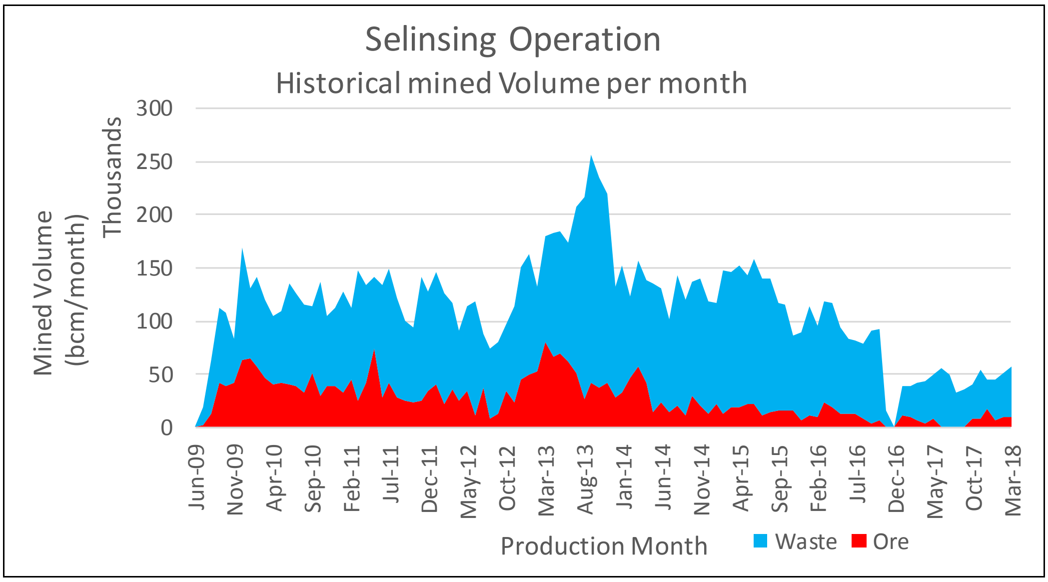


16.19.1 Productivity and constraint assumptions

The key assumption for productivity was achieving the processing plant requirement of 955 kt/a for sulphide ore.

To date, with only limited oxide mining occurring, the situation is largely unchanged. Historically, the best achievement is 4.7 Mt/a of total rock movement in 2013/14 with well over 200 bcm per month of total mining volume achieved in that period. Figure 16.23 showing the historical mining productivities in terms of volume by month.

Figure 16.23 Historical mining productivities in volume by month (based on average density)



The mining schedule was adjusted by individual start date and monthly productivity to match the processing plant requirement. Table 16.21 shows the input constraints for the SGSP three evaluation cases. The monthly individual open pit productivity ranging from 50 bcm to 225 bcm per month. This was achieved historically with Selinsing Pit only being mined in 2013/14. Figure 16.24 showing the SGSP Base Case by volume and by individual pit. Noting BRC2 and BRC3 are divided by tenement required historically for royalty calculation.

At its peak, the schedule requires at least four areas of simultaneous mining to achieve the aforementioned 955 kt/a process plant ore supply.

Table 16.21 Input schedule constraints for the SGSP FS

Pit reference	Total volume	Month start (Month 1…)	Monthly pit rate (bcm)
Selinsing Pit 4	1,765,772	Month 20	114,000
Selinsing Pit 5 and Pit 6	1,434,005	Month 20	92,621
Selinsing Pit 7	0	Month 30	203,000
Buffalo Reef C1	0	Month 19	50,000
Buffalo Reef Ox Pits Damar	0		
Buffalo Reef Ox Pits Felda BL7	0		
Buffalo Reef C2 Stage 2 Pit Damar	1,479,668	Month 19	180,000
Buffalo Reef C2 Stage 2 Pit Felda	948,706	Month 19	115,375
Buffalo Reef C2 Pit Damar	1,798,288	Month 32	140,000
Buffalo Reef C2 Pit Felda BL7	674,795	Month 32	52,558
Buffalo Reef C3 Pit Damar	420,646	Month 37	16,000
Buffalo Reef C3 Pit Felda BL7	2,012,484	Month 37	76,466
Buffalo Reef C4	4,983,985	Month 32	220,000
Buffalo Reef N1	54,922	Month 19	50,000
Buffalo Reef N2	221,704	Month 19	50,000
Buffalo Reef N3	0	Month 32	50,000

Figure 16.24 SGSP Base Case mining schedule by pit and volume

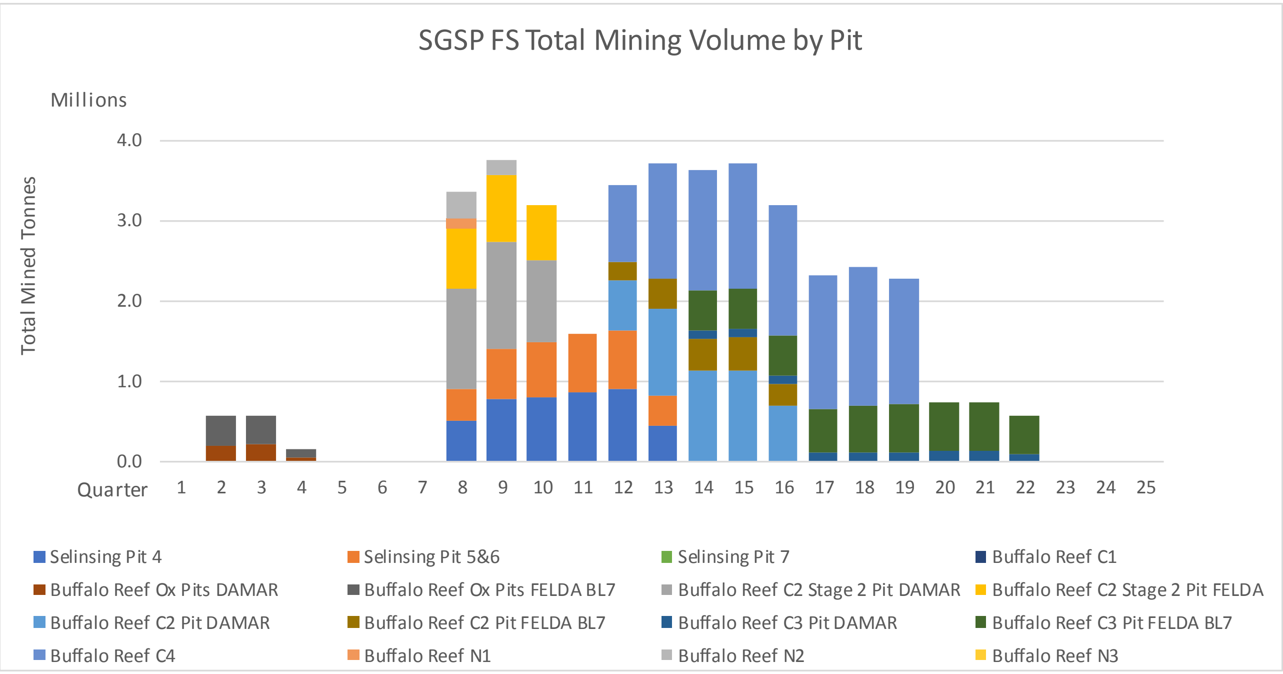


Figure 16.25 and Figure 16.26 show the respective quarterly production for Selinsing and Buffalo Reef respectively. This is different to constraints applied by Snowden (Dec 2016) in their schedule which were:

- Maximum of six 5 m benches or 30 m per year of vertical rate of advance (VROA)
- Maximum mining rate of 10 Mt/a, and 5 Mt/a in the first year
- Delay the mining of Selinsing Stage 1 for one year to enable geotechnical investigations
- Minimise active mining areas at any time.

Differences in the FS are:

- A VROA as presented below.
- Yearly production rate, 10 Mt/a is exceeded in periods from Quarters 6 to 9 and then 10 to 14. In these periods, there are multiple working areas in Selinsing and Buffalo Reef (Quarters 6 to 9) and then BRC2, BRC3 and BRC4 (Quarters 10 to 14).
- Geotechnical investigations in Selinsing Stage 1 and both Selinsing pits occur early in the schedule.
- Where possible, active mining areas are minimised, but several areas are required at any time to meet the schedule.

Yearly vertical advance rates indicated in the schedules as shown in Figure 16.25 and Figure 16.26, excluding waste pushback are as follows:

- Selinsing:
  - Pit 5 – 25 vertical metres per annum
  - Pit 4 – 27 vertical metres per annum.
- Buffalo Reef:
  - BRC2 – 30 vertical metres per annum
  - BRC3 – 50 vertical metres per annum
  - BRC4 – 55 vertical metres per annum.

Earlier production in Selinsing also exceeds the 30 m per year but Figure 16.25 shows nearly all excavation except at the base is a waste cutback of pre-existing open pits (largely the west wall only). The remaining 30 m of vertical extraction on both Pit 4 and Pit 5 is at the limit of the Snowden (Dec 2016) estimate.

Some concern may exist for later production in Buffalo Reef with BRC3 and BRC4 both exceeding the VROA Snowden (Dec 2016) limit of 30 m but both are within other Australasian Institute of Mining and Metallurgy (AusIMM) recommendations that provide for a limit of 40 to 60 VROA.

A total timeframe of three to four years is required in the schedule to mining BRC3 and BRC4. The schedule methodology based on a top-down set rate prioritises BRC2, then BRC3, and then BRC4. Mining BRC3 and BRC4 together at slower rates at the pit base would allow the 30 m VROA to be achieved. However, it should be noted that in this period the total mining requirement is reduced significantly below 10 Mt/a.

Mill feed during these later mining periods utilises at least one month's supply of stockpiled material at the start of each month until the final SGSP production period months when all stockpiles are consumed.

The vertical advance per year in each pit could be reduced with more optimal scheduling between BRC3 and BRC4 using other software such as Deswik MineCAD and Scheduler or Datamine OP with individual SMU scheduling. Snowden (Dec 2016) demonstrate this using "Evaluator" optimisation Mixed Integer based software constrained to the 30 m advance rate.

Figure 16.25 Selinsing Pit quarterly FS production schedule

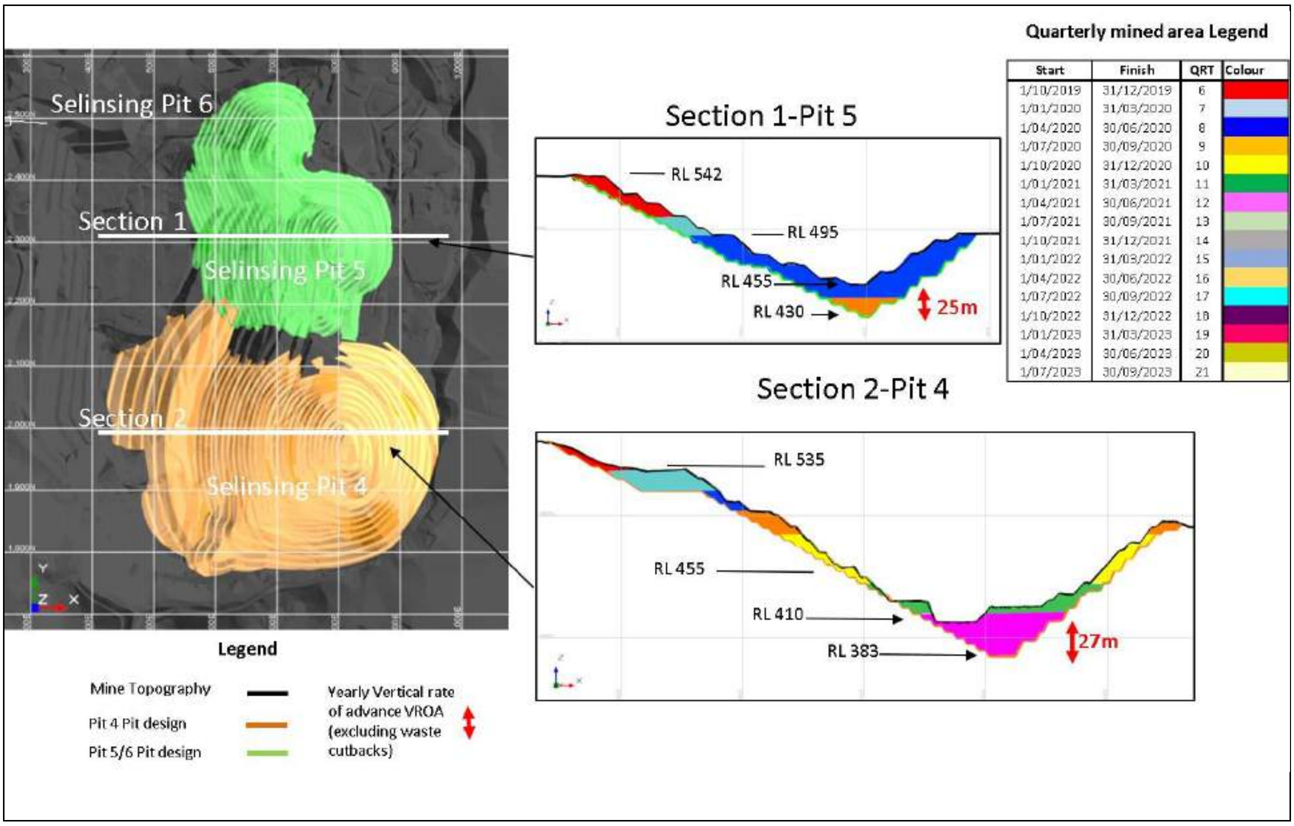
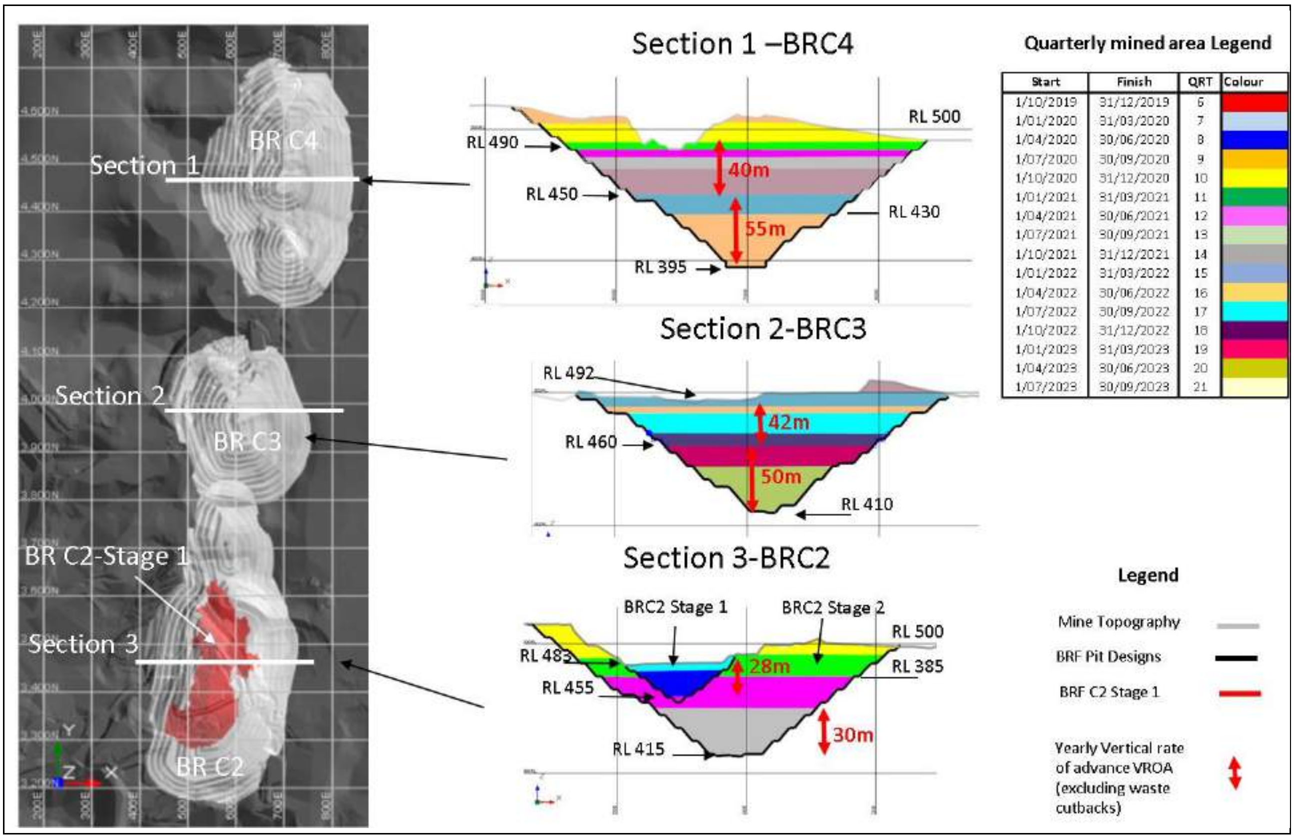


Figure 16.26 Buffalo Reef SC FS Production schedule



In terms of productivity improvement, options such as other contractors with larger equipment were assessed but there are many advantages with staying with current contractor arrangement even though their equipment choices are limited, which for Monument include:

- Low cost
- Ability to source local labour and similar machinery rapidly
- Contractor familiar with Monument requirements for safe operation of the mine and operating systems.

In order to achieve a target of 955 kt/a of ore, as well as up to an additional 9 Mt/a of waste, Snowden (Dec 2016) proposed the following fleet requirements:

- Drilling, based on LOM ore release:
  - Selinsing pits: From one to two drills
  - Buffalo Reef pits: From one to three drills
  - Individual units move from Selinsing to Buffalo Reef as required.
- Loading, based on LOM ore release:
  - Selinsing pits: From one to three excavators
  - Buffalo Reef pits: From one to three excavators
  - Individual units move from Selinsing to Buffalo Reef as required.
- Hauling, based on LOM ore release:
  - Selinsing and Buffalo Reef pits: From six to 18 x 30-t trucks.

An ancillary fleet of bulldozers, graders and three water trucks for dust suppression also provided by the contractor. The core requirement will remain unchanged with the exception of the truck numbers. Negotiations with Monument and Minetech are ongoing but indications are increases in fleet and manning are achievable with continuance of similar pricing.

16.19.2 Open pit mining cost assumptions

Mining costs are largely derived from the Minetech mining contract, which includes:

- Base rate for each area for ore and waste – based on surface haulage to ROM and waste dump
- Increment with depth for haulage
- Increment for drill and blast in waste.

The base rates assumed were from a period of high mining activity largely in the Selinsing pit area but with some Buffalo Reef mining was occurring. The increment for drill and blast in fresh being used as most of the period for the base rate was mining near surface oxide and transition material.

Table 16.23 shows the mining costs assumptions used together with average cost estimate based on average depth and percentage fresh material assumptions.

All contract mining costs were calculated in the schedule file to allow more accurate calculation by bench slice.

Rehandle and stockpile management costs are included in processing costs discussed in Section 21. Grade control and other mining overhead costs are included in the total site administration costs also mentioned in Section 21.

Table 16.22 SGSP mining cost assumptions

Item	SEL	BRN	BRS/C	Block 7	All	Unit
Guidance						
Exchange rate	4.08	4.08	4.08	4.08		RM/US$
Mining						
Mining recovery[1]	98%	98%	98%	98%	98%	% of inventory > cut-off grade
Mining dilution	2%	2%	2%	2%	2%	
Dilution grade	0.10	0.10	0.10	0.10	0.10	g/t Au
Costs						
Base case	7.78	7.05	6.82	6.82	7.01	RM/t mined
Base cost – Ore	1.90	1.73	1.67	1.67	1.71	US$/t mined
Base cost – Waste	1.90	1.73	1.67	1.67	1.72	US$/t mined
Inc with depth (RM)[3]	0.0024	0.0024	0.0024	0.0024	0.0024	RM/t/m depth
Inc with depth (US$)[3]	0.0006	0.0006	0.0006	0.0006	0.0006	US$/m depth
Drill and blast increment for fresh (Waste)[3]	0.26	0.26	0.26	0.26	0.26	US$/t mined
Drill and blast increment for fresh (Ore)[3]	0.26	0.25	0.25	0.25	0.25	US$/t mined
Average pit depth (1/3 down)	45.00	16.67	46.67	46.67	46.67	m depth
Ore mined	672,314	166,566	2,200,020	880,109	3,919,009	t
Waste mined	7,125,410	468,410	20,231,217	8,178,016	36,003,052	t
% fresh rock	60%	60%	60%	60%	60%	% total mined
Total average mining ore cost	**2.09**	**1.89**	**1.85**	**1.85**	**1.89**	**US$/t**
Total average mining waste cost	**2.09**	**1.89**	**1.85**	**1.85**	**1.90**	**US$/t**

Notes:
1. *Reconciliation history.*
2. *Costs to July 2015 (inclusive of drill and blast).*
3. *2016 Minetech contract.*
4. *Reserve-only estimate from FS.*

16.19.3 Mining schedule

The resulting mining schedule for the FS by quarters is shown in Figure 16.27 and shows total tonnes mined by area. Note the Block 7 area is the previous Felda Block 7 area referred to in Snowden (Dec 2016) and now fully acquired by Monument. The period of Quarter 0 to 3 showing only the current oxide mining not included in the SGSP.

LOM start is Quarter 0 representing the now mined in 1 April 2018 to July 2018 period.

Initial oxide-only mining is exhausted by Quarter 2, with no mining until Quarter 6 when SGSP sulphide base mining commences. There will be some ore movement required pre and post the SGSP period with stockpile and Old Tailings material movement required. Figure 16.28 shows this ore feed movement together with all feed material movement including:

- Mined ore to stockpile or direct feed
- Existing stockpiles to direct feed
- Old Tailings to direct feed.

Oxide ore mined during the SGSP period is stockpiled for CIL only processing post SGSP.

Figure 16.27 SGSP mining production schedule of total tonnes mined by area

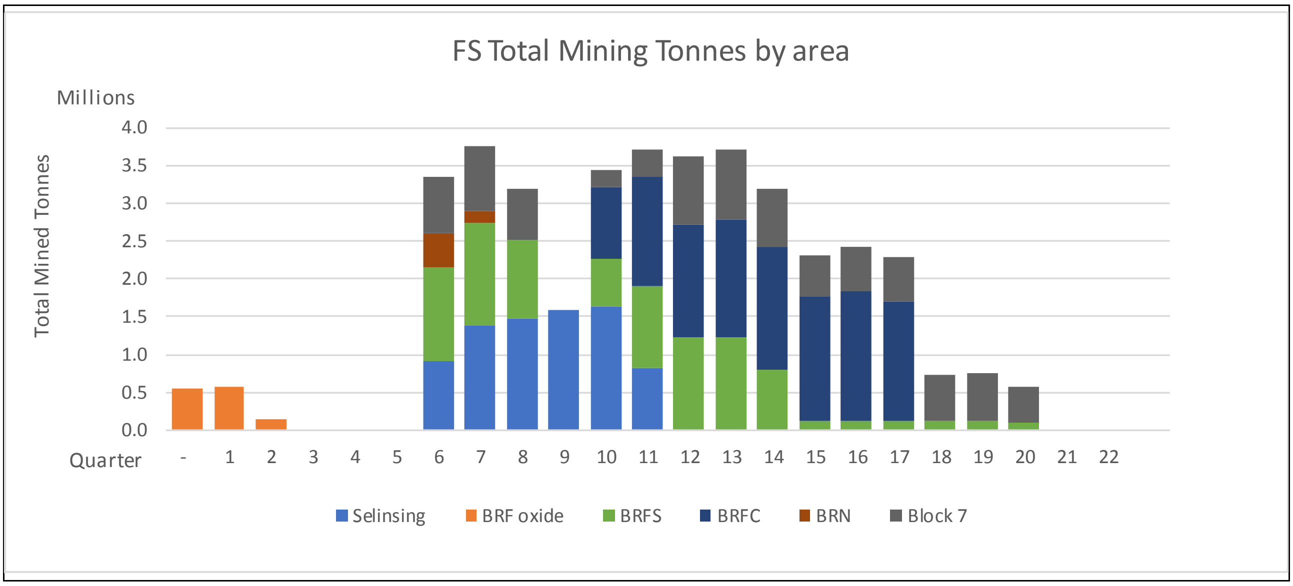
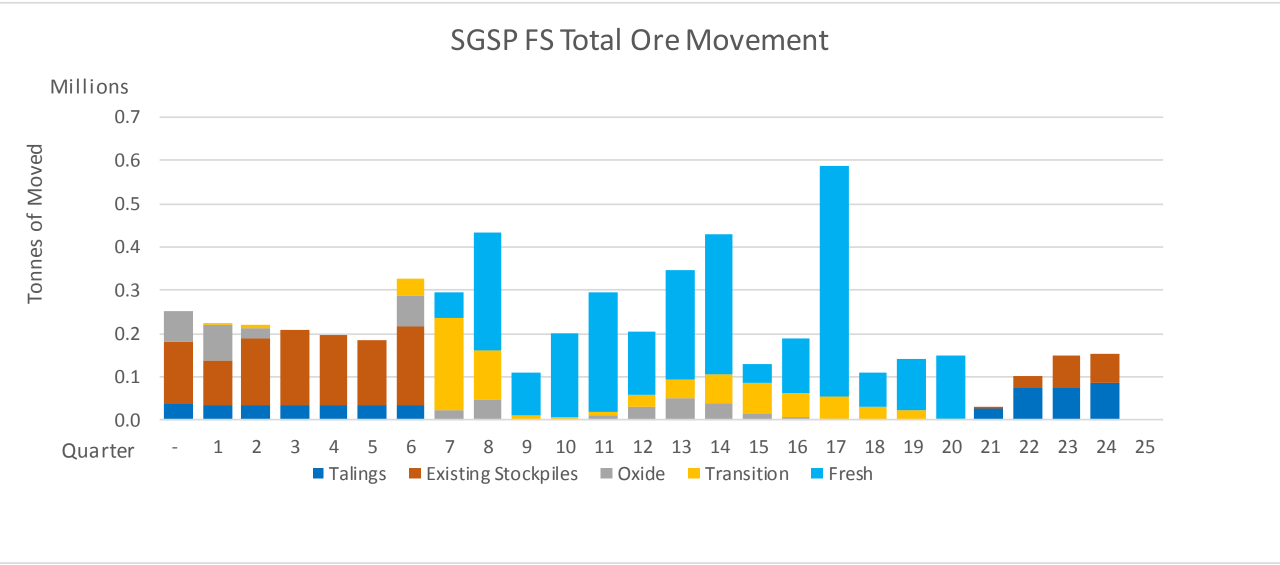


Figure 16.28 Total Processing feed material movement to stockpiles and direct feed



16.19.4 Processing schedule

The SGSP processing schedule commences in Quarter 7 (January 2020 to March 2020) in line with plant construction completion, as shown in the time line of Figure 16.22. Full production capacity of 955 kt/a is planned from commencement. Figure 16.29 shows the Base Case processed tonnes and head grade planned and Figure 16.30 the recovered gold ounces.

Prior to Quarter 7, the CIL processing plant would continue to process available oxide ore, leachable stockpile ore and Old Tailings. The changeover cost and time for CIL to flotation/BIOX® expected to be minimal.

Figure 16.29 SGSP processing schedule of processed tonnes and head grade

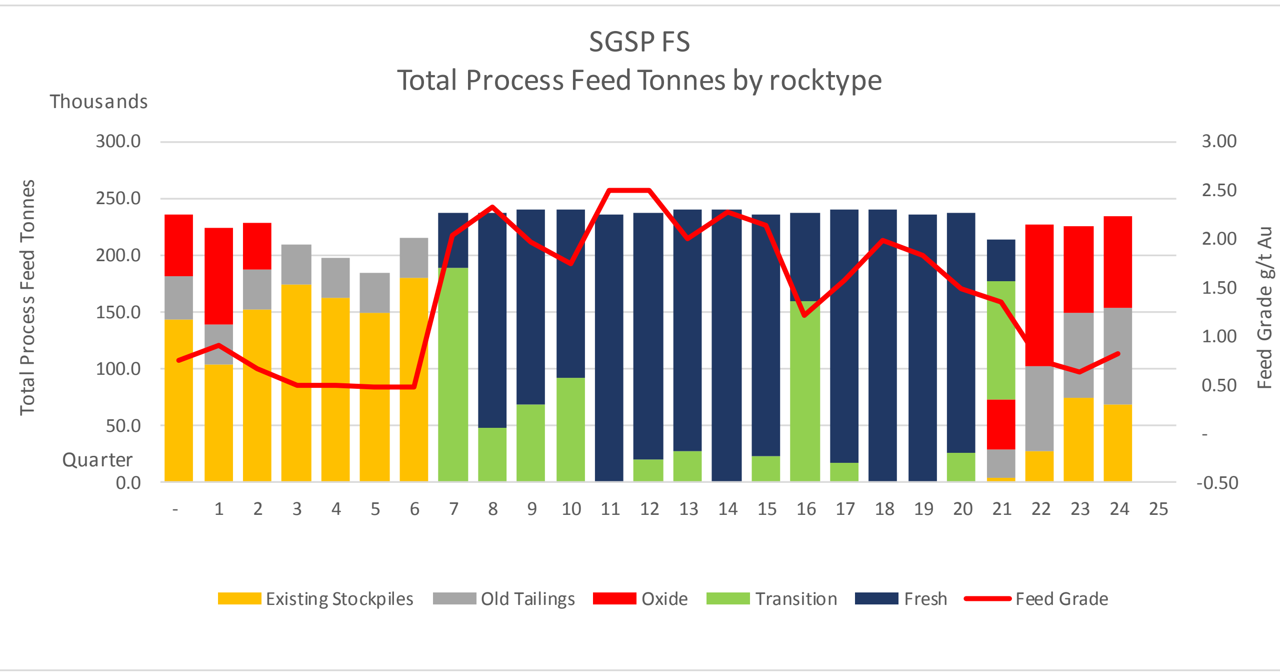
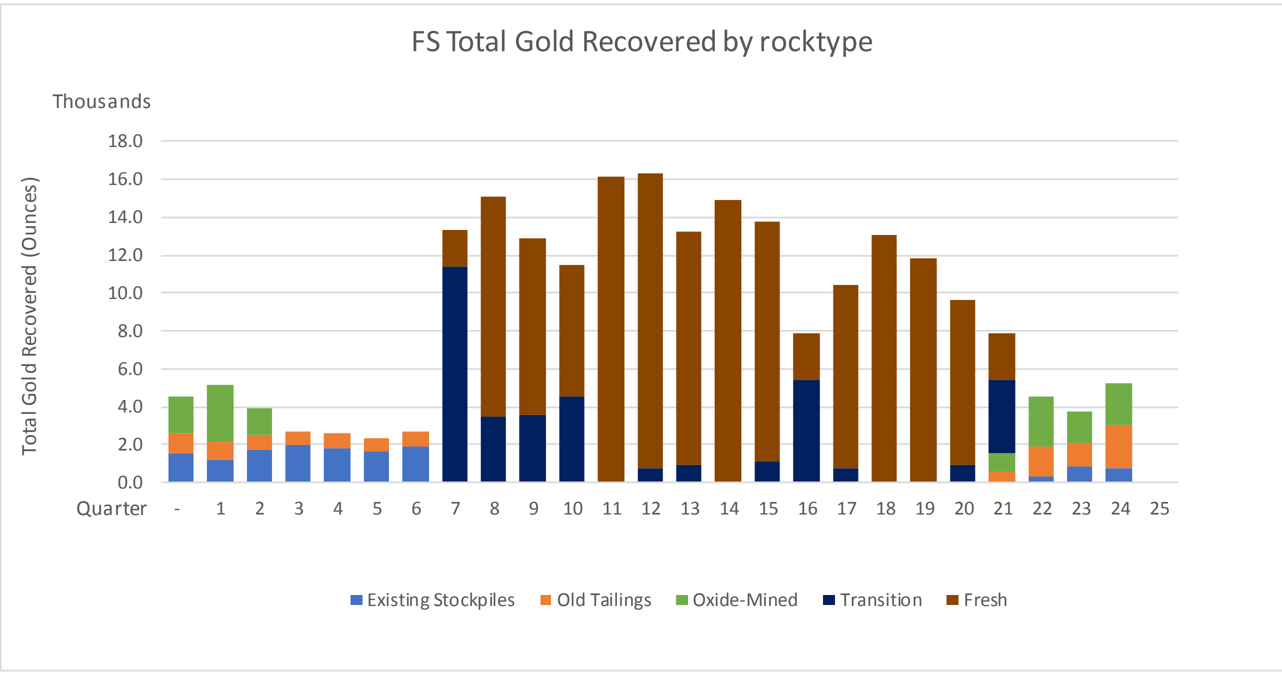


Figure 16.30 SGSP FS processing schedule of recovered gold ounces



16.19.5 Selinsing Process Plant BIOX® and flotation circuit commissioning

A detailed description of the upgraded process plant with BIOX® and flotation circuits added for the FS is given in Section 17.

The mine production schedule used the overall upgraded plant construction timeline and detailed commissioning schedule developed in the Section 18 (Project Infrastructure). SGSP production from the upgraded plant is expected for Quarter 7 (January 2020 to March 2020), during which ramp up to full production in terms of both feed tonnes and recovery is expected in the first month.

The changeover from the CIL only to BIOX® and flotation is expected in Quarter 6, during which production feed is scheduled at 215 kt which is well below the expected 238 kt per quarter full production rate. The timeframe for the changeover is expected to be about one week.

In terms of post-SGSP processing, the remaining stockpiles and Old Tailings at the end of the mine life of the changeover, a similar timeframe to that of SGSP start-up is expected with a simple matter of bypassing the flotation and BIOX® circuits which can be decommissioned concurrently as production from CIL only occurs.

17 RECOVERY METHODS

17.1 Introduction

A key assumption made for continuous production during the SGSP plant upgrade for the treatment of sulphide ores, is that for any oxide stockpiles and other materials such as treatable tailings, the existing CIL based plant would be used until the BIOX® plant changeover occurs.

Any oxide material mined, and stockpiled post January 2020 would again be treated by the same arrangement at the completion of sulphide processing.

The references for this chapter include:

- OMC (Feb 2018) "Selinsing Gold Mine Testwork Results and Process Design Basis ", Orway Mineral Consultants Report No. 7828-03 Rev A
- Outotec (April 2018a) "Outotec Process Design Specification for the Selinsing BIOX® Plant Malaysia"
- Outotec (April 2018b) "Outotec Process Design Package - Selinsing ASTER™ Plant", OMC (November 2018)
- OMC (Oct 2018) "BRC 4 Fresh BV Testwork Program", Orway Mineral Consultants Report No. 7944-02 Rev 0
- SGSP (Oct 2018) "SGSP – Optimisation Tests on BRC 4 FR and Their Extension to Other LOM Pit Ores", M Wort SGMM R&D.

17.2 Plant design

17.2.1 Crushing, grinding and classification

The existing Selinsing Gold Plant is currently treating oxide gold ore. The process includes primary, secondary and tertiary crushing followed by ball milling, gravity concentration and CIL. Future plant feed will contain a sulphide component, which renders the ore refractory, meaning the gold that cannot be effectively recovered by conventional CIL processing. A new processing route has been investigated which incorporates flotation and bacterial oxidation (BIOX®) ahead of the CIL circuit to achieve the target gold recovery.

The existing crushing, grinding and classification plant will remain basically unchanged from its present state apart from some necessary upgrades to conveyors and screens. Capacity will remain basically unchanged.

When the transition and sulphide ores are treated, they are expected to be a little harder than the currently treated oxide ores. This may result in a coarsening of the cyclone overflow from its present P_{80} sizing of 75–94 µm, based on OMC's calculations. Most of the flotation testing to date was done on feeds prepared to a P_{80} of 75 µm. Limited testwork has been performed at varying grind sizes, but anecdotally variations in grind between 60 µm and 100 µm have been found to have little effect on flotation performance.

Comminution circuit

The OMC (Feb 2018) report documents a review of the current and upgraded comminution circuit.

The existing comminution circuit comprises primary, secondary and tertiary crushing followed by two ball mills operating in series. Table 17.1 below presents the design parameters for the comminution circuit (refer OMC (Feb 2018) p3).

Table 17.1 Comminution circuit design parameters

Parameters	Unit	Design			Reference
Plant capacity	**t/a**	**950,000**			**Client**
Feed blend (BR:SEL)	%:%	84:16			Mine Plan
Crushing circuit					
Crushing circuit utilisation	%	66.7			Client
Crushing circuit utilisation	hr/a	5,840			Calculated
Crushing rate	t/hr	163			Calculated
ROM ore top size	mm	500			Client
Crushing product size, P_{80}	mm	8			Modelling
Grinding circuit					
Grinding circuit utilisation	hr/a	8,000			Client
Grinding circuit utilisation	%	91.3			Calculated
Milling rate	t/hr	119			Calculated
Grinding product size, P_{80}	micron	75			Testwork
Comminution parameters		**Buffalo Reef**	**Selinsing Deeps**	**Design**	
Abrasion Index		0.009	0.368	0.066	Testwork/Calc.
Crushing Work Index	kWh/t	7.54–22.7	5.2	13.5	Testwork/Calc.
Bond Rod Mill Work Index	kWh/t	13.7–14.5	18.1	14.7	Testwork/Calc.
Bond Ball Mill Work Index	kWh/t	14.1–16.0	17.1–18.0	15.5	Testwork/Calc.
Ore SG	kg/L	2.66	2.74	2.67	Testwork/Calc.

The sulphide processing plant is being designed for a feed comprising 84% Buffalo Reef (BR) and 16% Selinsing (SEL) ore. The comminution parameters for BR and SEL as listed in the process design criteria are summarised in the table above; the design comminution parameters are the weighted average of the BR and SEL averages.

Crushing circuit assessment

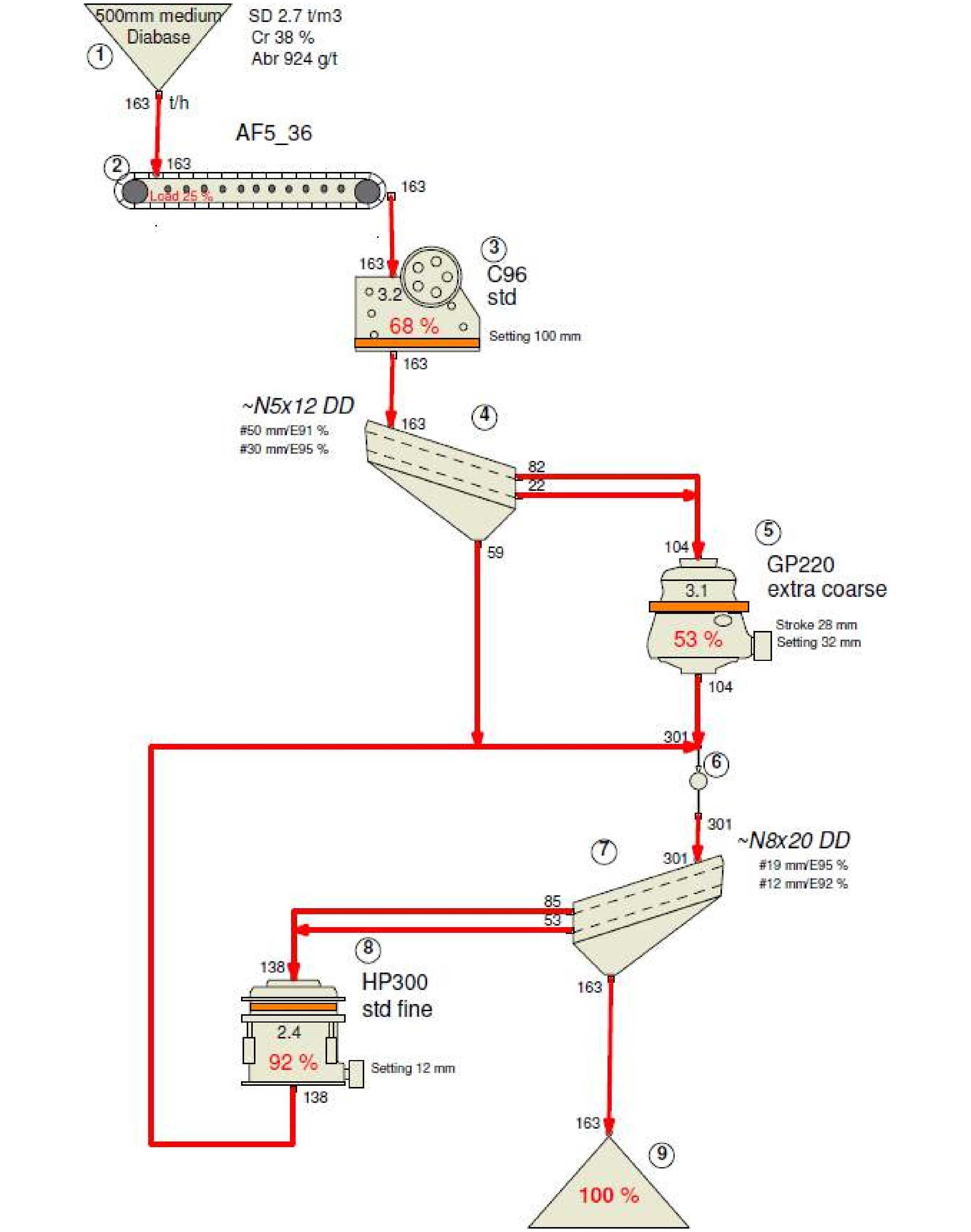
The OMC (Feb 2018) report also describes the crushing circuit as specified in Table 17.2, which presents the specifications of the existing crushing equipment, apart from the product screen that has been replaced with a single deck screen for the current processing of oxide material. For sulphide processing, the specified product screen in this table will be installed.

Table 17.2 Crushing equipment

Parameters	Unit	Details
Primary crusher		
Manufacturer/Model		Jaques JW40ST
Feed opening	inches	42 x 30
Motor power	kW	90
Secondary crusher		
Manufacturer/Model		Terex Automax 1000 L
Cavity		XC Long Throw
Motor power	kW	160
Tertiary crusher		
Manufacturer/Model		Terex Automax 1300 S
Cavity		MC
Motor power	kW	220
Secondary scalping screen		
Manufacturer/Model		Jaques GF/VSDD412HD
Size (W x L)	mm x mm	1,200 x 3,600
Top deck aperture	mm	50
Bottom deck aperture	mm	30
Product screen		
Manufacturer/Model		Jaques GF/VSDD824HD
Size (W x L)		2,400 x 7,200
Top deck aperture		19
Bottom deck aperture		12

OMC (Feb 2018) also describes a crushing circuit simulation conducted using Metso's Bruno® software to evaluate the crushing circuit performance at the sulphide ore design tonnage as shown in Figure 17.1.

Figure 17.1 Crushing circuit simulation



Since the installed crushers are not Metso machines, the closest equivalent units were selected for simulation and therefore, the results should be seen as indicative.

Based on the simulation results, the design crushing rate and product size are achievable using the existing equipment. At a higher crushing rate, the circuit will be constrained by the screens and tertiary crusher. Therefore, a coarser product size would be anticipated if a higher crushing rate was targeted.

Grinding circuit assessment

OMC (Feb 2018) carried out a mill assessment to estimate the maximum grinding power available. The results are summarised in Table 17.3 below.

Table 17.3 Milling assessment

Parameters	Unit	Primary mill	Secondary mill
Mill diameter (inside shell)	m	4.20	3.20
Effective grinding length (EGL)	m	5.20	4.00
Imperial	ft x ft	13.8 x 17.1	10.4 x 13.1
L:D ratio		1.24	1.25
Discharge configuration		Overflow	Overflow
Backing rubber	mm	6	6
Liner type		Rubber	Rubber
New liner thickness	mm	70	70
Ball top size	mm	80	30
Mill speed	%Nc	72	72
Maximum ball charge	%Vol	34	38
Maximum pinion power	kW	1,330	500
Maximum power draw	kW	1,440	540
Installed power	kW	1,600	650

The pinion power is the estimated power draw at the mill shell. This value is approximately 7.5% lower than the motor power input due to drive train losses. Based on the OMC (Feb 2018) ball mill power model, the maximum pinion power available from the two ball mills is 1,830 kW (i.e. 1,330 kW from primary mill and 500 kW from secondary mill). It should be noted that the two ball mills will not draw the installed power at the maximum ball charge.

Table 17.4 below presents the grinding throughput modelling, showing the estimated milling rate and product size of the design feed blend.

Table 17.4 Grinding throughput modelling

Parameter	Units	Buffalo Reef		Selinsing Deeps		Design feed blend 84% BR and 16% SEL
		Low BWi	High BWi	Low BWi	High BWi	
RWi	kWh/t	13.7	14.5	18.1	18.1	14.7
BWi	kWh/t	14.1	16.0	17.1	18.0	15.5
SG		2.66	2.66	2.74	2.74	2.67
Circuit parameters						
Mill feed size F_{80}	mm	8.0	8.0	8.0	8.0	8.0
Grinding product size P_{80}	µm	75	75	75	75	75
Ball mill specific energy – corrected	kWh/t	15.5	17.8	19.7	20.8	17.2
Maximum pinion power available	kW	1,830	1,830	1,830	1,830	1,830
Estimated milling rate @ P_{80} 75µm	t/hr	118	103	93	88	**106**
Estimate P_{80} @ 119 t/hr	µm	76	102	130	152	**94**

Based on the mill feed size F_{80} of 8.0 mm, a milling rate in the range of 88–118 tonnes per hour (t/hr) can be expected when processing these materials while still maintaining the target grind P_{80} of 75 µm.

For the design feed blend of 84% Buffalo Reef and 16% Selinsing, a milling rate of 106 t/hr was estimated at the target grind size P_{80} of 75 µm; this milling rate is lower than the 119 t/hr target. If the target 119 t/hr was maintained a coarser product size of around P_{80} of 94 µm would result. Should a coarser than desired flotation concentrates sizing be produced, space has been allocated for a concentrate regrind mill to be installed between the concentrate thickener and the BIOX® surge tank.

Additional flotation testing at coarser grinds was recommended to quantify the effects of grind size on recovery so that controls can be put in place if required to maintain flotation performance.

The modelling results confirmed that the plant throughput is constrained by the ball mills and as such increasing the crushing rate is not recommended as it generates a coarser crushing product size.

Flotation testwork was conducted on both Buffalo Reef and Selinsing material. Flotation responses were different for samples obtained from different ore zones. The most successful flotation parameters and reagent regime which were selected for design are summarised in Table 17.5.

Table 17.5 Summary of flotation parameters and reagent regime

Parameter	Units	Value	Reference
pH		9.2 to 9.6	OMC (Oct 2018)
Eh	mV (Ag/AgCl)	-95 to -111	OMC (Oct 2018)
Na_2CO_3	kg/t	2 to 4	SGSP (Oct 2018)
PAX	g/t	400 to 500	OMC (Oct 2018)
Na_2S	g/t	300	SGSP (Oct 2018)
$CuSO_4$	g/t	100	SGSP (Oct 2018)

17.2.2 Flotation plant design criteria

OMC (Feb 2018) and OMC (Oct 2018) also reviewed the flotation design.

The design capacity of the plant is 950,000 t/a with a weighted average head grade of 1.92 g/t gold and 0.64% sulphur.

Early in the engineering confirmation testing of gold and sulphide flotation, it became apparent that flotation recovery of sulphides would be complete for most ore types within 20 minutes, with some slower floating variants requiring 25 minutes for completion, commencing with an initial flotation feed pulp density of 35% solids.

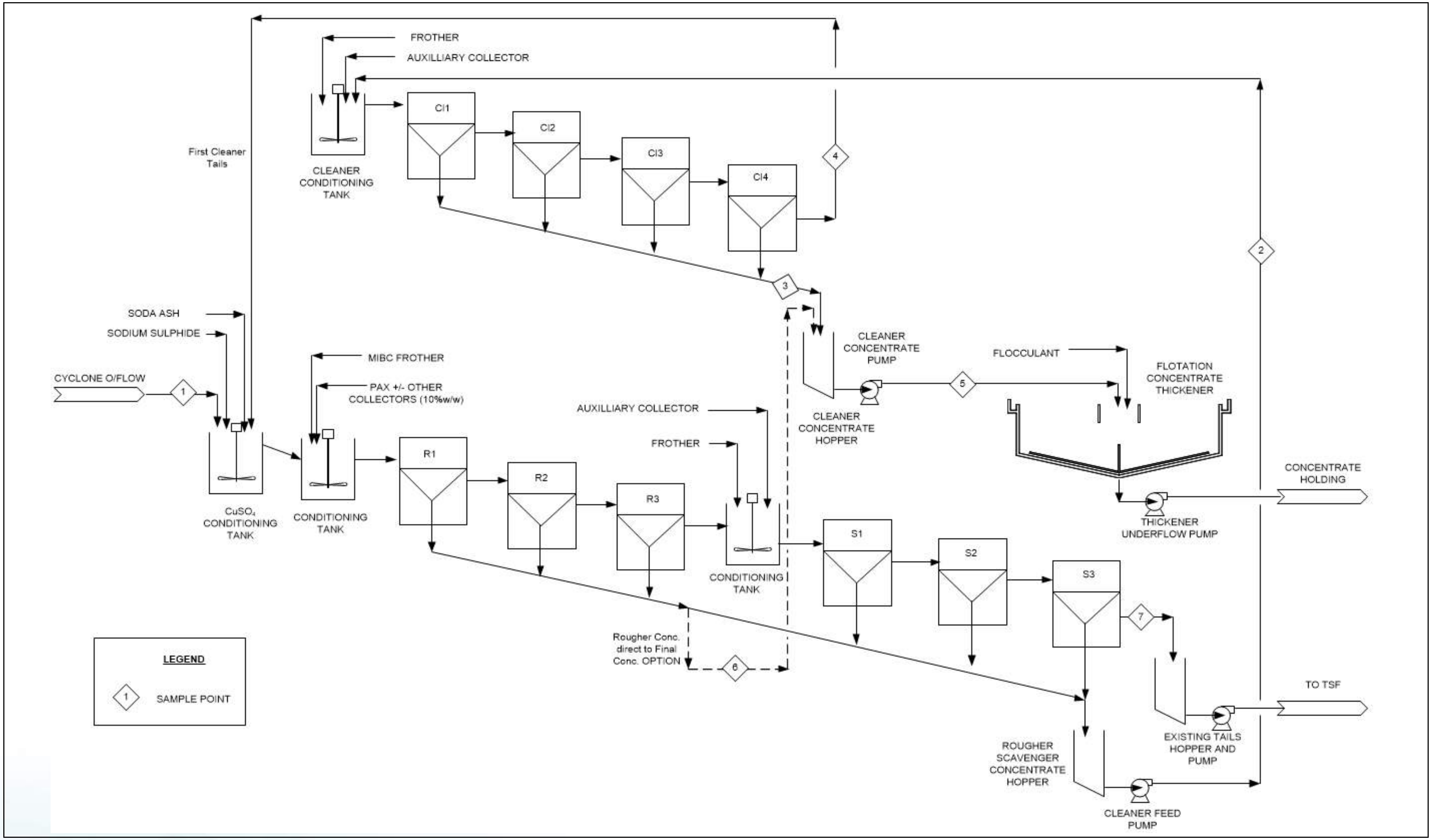
Based on this observation, and using an accepted scale up factor of two, the required rougher flotation capacity for the plant was set at 50 minutes. In a similar way, it was shown at bench scale that cleaning of rougher concentrates at 16% solids could be completed within 12 minutes, defining cleaner flotation residence time in the plant as 24 minutes.

Again, from the results of flotation testing on-site at the Selinsing R&D Laboratory, it became evident that the best performing ores would only require rougher-scavenger flotation to produce concentrate with a high enough sulphur grade to meet BIOX® feed requirements (6% S to 8% S). Accordingly, the conceptual layout of the plant allowed provision of some rougher concentrates to be diverted to final (BIOX®) Concentrate Thickener Feed (see Figure 17.2).

Figure 17.2 is the preliminary layout sketch forwarded to OMC for their consideration when preparing the flotation plant design. The flow streams indicated (1 to 7) are the slurries requiring to be sampled for metallurgical accounting.

Based on the selected mining rate of 950,000 t/a at an assumed 91.32% plant utilisation, together with the known required residence time capacity of the rougher and cleaner flotation banks, the size and number of tanks could then be selected based on the standard capacities of cylindrical flotation tanks as offered by the major equipment suppliers including Outotec, Metso and FLSmidth. A flotation tank design incorporating external concentrate launders was preferred by the on-site team, as it allowed for a slightly smaller gross tank volume to be selected. To minimise the effects of possible slurry short circuiting, a minimum number of six flotation tanks was selected for the rougher-scavenger bank, in accordance with industry practice.

Figure 17.2 Diagrammatic preliminary sketch of the flotation plant circuit



A set of flotation process design criteria (see Table 17.6) was forwarded to OMC, who was contracted to supply the design of the flotation plant. The design criteria originally provided to OMC are listed below, together with the criteria recommended by OMC.

As part of their brief, OMC reviewed the results of the testwork conducted both on site and by specialist external laboratories. The testwork results were used with the existing plant equipment specifications to establish the design basis for the sulphide processing circuit.

Table 17.6 Flotation plant process design criteria

Item	Units	SGMM estimate		OMC calculation	
		Value	Comment	Mass balance	Comment
Solids feed rate, new feed	t/hr	118.8		118.73	
New feed solids SG	g/mL	2.71		2.71	
Feed rate (inc circ. Load from first cleaner)	t/hr	142.56		142.53	
Feed slurry pulp density	% solids	35		30	
Rougher feed residence time	min	50		50	
Rough Scavenger mass-pull	% wt	25		30.01	
Rougher Scavenger bulk concentrate	t/hr	35.64	Maximum	35.63	
Cleaner flotation bank feed rate	t/hr	35.64	Estimated	35.63	
Cleaner feed solids SG		3.01		3.01	
Cleaner bank residence time	min	24		24	
Cleaner bank slurry pulp density	% solids	18		16.5	
Cleaner bank mass-pull	% wt	40		33.26	
Solids flow rate, cleaner concentrate	t/hr	11.88		11.85	
Cleaner concentrate SG		3.4	For 15% S	3.05	For 6% S – 8% S
Solids flow rate, cleaner tail	t/hr	23.76		23.78	
Cleaner tail solids SG		2.75		2.99	
Rougher tail pulp density	% solids	30	Estimated	28.04	
Rougher tail solids SG		2.67	Estimated	2.65	
Rougher tail solids flow rate	t/hr	106.92		106.9	

Based on the above criteria, the rougher-scavenger bank was nominated as six tank cells of 50 m^3 live capacity based on an air hold-up rate of 15%. Copper sulphate solution (5%) will be fed to the grinding circuit as sulphide activator. Two rougher feed conditioner tanks were specified, each giving two minutes residence time. The first tank will receive sodium sulphide and also soda ash if required to adjust pH to 9.0–10.0. The second conditioner tank will receive PAX collector and MIBC or other frother. Between the third and fourth tank, a scavenger feed conditioner will be placed to receive additional PAX collector and any additional frother which may be required.

For the cleaner bank, four tank cells each of 20 m^3 capacity were nominated. The cleaner conditioner at the head of the bank will have two minutes residence time and will receive the auxiliary collector plus any further additions of frother required.

As indicated in their Feb 2018 report, OMC have conservatively recommended an expansion of the supplied capacity for both the conditioners from two minutes residence to three minutes residence, and the capacity of the rougher and scavenger tanks from 50 m^3 each to 70 m^3 each.

17.2.3 BIOX® plant design criteria

Outotec's proprietary logistic model was used to determine a set of operating envelopes for the concentrates from which design conditions can be derived. This allowed for the Selinsing performance scenarios to be predicted based on (i) normal flow, (ii) maximum sulphide, and (iii) maximum throughput.

The composition and mass of the various concentrates over the life of mine were derived from a combination of previous flotation testwork results and the mine production schedule issued by Monument. The Selinsing BIOX® plant is designed as one BIOX® module with three primary reactors in parallel followed by three secondary reactors in series. The BIOX® plant will be required to process an average of 274 t/d of concentrate at a sulphide sulphur feed rate of 16.7 t/d. However, specific production quarters will generate 318 t/d of concentrate at 25.9 t/d sulphide sulphur, and 332 t/d of concentrate at 22.2 t/d sulphide sulphur. The BIOX® circuit basis of design considers these higher input values and sizing of the requisite equipment and utilities has been based on these peak daily concentrates and sulphur tonnage input values in the Outotec design.

A summary of the parameters used by Outotec to establish a series of design criteria for the BIOX® circuit is presented in Table 17.7 as described in OMC (Feb 2018).

Table 17.7 BIOX® basis of design (OMC, Feb 2018)

Parameters	Units	Average	Maximum sulphur	Maximum flow
Concentrate feed to BIOX®	t/day	274	318	332
Sulphur feed to BIOX®	t/day	16.7	25.9	22.2
Sulphur feed grade	%w/w	6.1	8.1	6.7
Pyrite	%w/w	9.3	21.3	14.2
Arsenopyrite	%w/w	3.7	14.3	6.5
Primary reactors in parallel	no.	3	3	3
Primary reactors residence time	days	2.75	2.37	2.28
Secondary reactors in series	no.	3	3	3
Secondary reactors residence time	days	0.92	0.79	0.76
Total BIOX® residence time	days	5.5	4.73	4.56
Sulphide cumulative oxidation				
Primary reactors	%	70.8	67.7	66.4
1st Secondary reactor	%	85.3	83.4	82.4
2nd Secondary reactor	%	91.3	89.8	88.9
3rd Secondary reactor	%	95.2	94.4	93.8
Arsenic cumulative dissolution				
Primary reactors	%	75	72	71
1st Secondary reactor	%	83	80	80
2nd Secondary reactor	%	88	86	85
3rd Secondary reactor	%	96	93	92

BIOX® feed

Rougher concentrate if of sufficient sulphur grade (6–8% S) and cleaner flotation concentrate will be pumped to a high rate thickener of 9 m diameter. Thickener underflow will be pumped to the BIOX® surge tank at 50% solids; the BIOX® surge tank will have 1,004 m^3 live volume that will provide 60 hours capacity. Flotation concentrate stored in the stirred BIOX® surge tank will be pumped to the primary BIOX® reactors with automatic dilution to achieve a 20% solids concentration in the BIOX® feed.

BIOX® reactors

The BIOX® feed slurry will be pumped to the feed splitter ahead of the BIOX® reactors. Dosed quantities of nutrient will be added at the feed splitter from which the slurry will gravitate to the BIOX® reactors. The six BIOX® reactor tanks will be configured as three primary reactors in parallel followed by three secondary reactors in series. At average feed rates the total residence time in the BIOX® reactors will be 5.5 days, split equally between the primary and secondary reactors at 2.75 days each. The BIOX® culture in the reactor tanks will be maintained in a healthy condition by controlling the temperature, slurry pH and dissolved oxygen levels. The temperature will be controlled at 41°C by pumping cooling water through the cooling cools; the slurry pH will be controlled by adding either sulphuric acid or milled limestone to maintain the range of pH 1.3–1.6; dissolved oxygen levels will be maintained by pumping low pressure air through spargers installed at the base of each BIOX® reactor tanks. If the correct conditions are maintained, approximately 70% of the sulphide sulphur in the BIOX® feed will be oxidised in the primary reactors. Slurry in the primary reactors will flow via riser pipes to the secondary BIOX® reactors where further oxidation will take place. After the first secondary reactor it is expected that the sulphide oxidation will have reached 85%; the second reactor is expected to achieve 91% and the final secondary reactor 95% oxidation.

Counter-current thickeners

The duty of the counter-current decantation (CCD) thickeners is to separate the BIOX® solution containing iron, arsenic and sulphur from the residue, which will undergo further treatment to recover the contained gold. There will be three CCD thickeners operating in series: the first CCD thickener will receive the discharge from the third secondary BIOX® reactor along with the overflow from the second CCD thickener; the first CCD thickener underflow will be pumped to the second CCD thickener where it will mix with the third CCD thickener overflow; the second CCD thickener underflow will be pumped to the third CCD thickener where wash water will be added. Overflow from the first CCD thickener will be pumped to the neutralisation tanks; underflow from the third CCD thickener will be pumped to the pH adjustment tanks.

Neutralisation

Acidic overflow solution from the first CCD thickener will be pumped to the six neutralisation tanks. Limestone slurry will be added to partially neutralise the solution to pH 4.5, then hydrated lime will be added to adjust the slurry to pH 7. The neutralised slurry will be pumped to the water recovery thickener from which the underflow solids will be pumped to the TSF and the overflow solution will be recycled.

PH adjustment

The BIOX® residue from the underflow of the third CCD thickener will be pumped to the two pH adjustment tanks. In the first pH adjustment tank, hydrated lime will be added to attain the target pH 7; further adjustment to pH 11 will occur in the second pH adjustment tank.

BIOX® CIL

Slurry from the pH adjustment tanks will be pumped to the new BIOX® CIL circuit, comprising six tanks for a total residence time of 24 hours. Defoamer will be added to control froth formation in the CIL. Cyanide solution will be dosed into the first CIL tank to leach the gold in the BIOX® residue. Oxygen will be added to the CIL tanks to maintain dissolved oxygen levels. Activated carbon for the adsorption of gold will be transferred counter current from tank to tank using airlifts. Loaded carbon will be harvested from the first CIL tank and pumped to the existing carbon recovery screen above the elution column. Batches of 1.5 tonnes of loaded carbon will first be soaked in hydrochloric acid to dissolve any carbonate residues that may have coated the carbon. The activated carbon will then be transferred to the elution column where caustic soda and cyanide solution will be pumped at temperature and pressure to initiate the elution process. Pregnant eluate will be pumped to the two electrowinning cells where the gold be deposited onto the cathodes. The cathodes will be cleaned periodically to recover the deposited gold which will then filtered, dried and smelted into gold bars.

Following the elution process the barren carbon will be pumped to the existing carbon regeneration kiln which will reactivate the carbon prior to reuse in the CIL circuit.

BIOX® CIL detox

The BIOX® CIL slurry will discharge from the last CIL tank into the CIL detoxification tank. A solution of sodium metabisulphite (SMBS) will be added to the slurry along with some copper sulphate solution which will act as a catalyst and low-pressure air providing oxygen for the process; caustic soda or hydrated lime may be added to maintain pH in the range of 8–10. Cyanide and WAD species are converted to cyanate which is further oxidised to ammonia and carbon dioxide. Oxidation of thiocyanate will be limited to 10–20%. The detox slurry will be pumped up to the existing leach tanks which will be converted for use as ASTER™ reactor tanks.

ASTER process

ASTER™ is an acronym for Activated Sludge Tailings Effluent Remediation and is a process that uses certain microorganisms to catalyse the metabolism of cyanide and thiocyanate. In a process similar to the BIOX® process, the micro-organisms are maintained at specific slurry conditions and fed with suitable nutrients. The process works at a pH range of 6.5–8.0, and a temperature of 20–40°C.

The existing three leach tanks will be converted for use as ASTER™ reactor tanks by the installation of air spargers at the base of each tank and variable speed drives to slow down the agitators. Slurry from the detox tanks will be diluted to around 5% solids and pumped to a distributor that will split the feed between the first two leach tanks acting as primary reactor tanks running in parallel. Discharge from the primary reactors will flow into the third leach tank acting as the secondary reactor tank. Discharge from the secondary reactor, now free of all cyanide compounds, will gravitate to the water recovery thickener.

Water recovery thickener

The water recovery thickener will receive slurry from the ASTER™ secondary reactor, the neutralisation tanks and the flotation tailings. Thickened tailings will be pumped to the existing tailings storage facility; thickener overflow will be pumped to the existing process water tank.

17.3 Plant layout

17.3.1 Site geotechnical survey

Geotechnical investigations conducted for the Phase III expansion in 2011 by Get Services Sdn Bhd included borehole drilling with standard penetration tests (SPTs) and plate bearing tests. The subsoil conditions in the plant area varied from stiff to very stiff silty sand at surface, an intermediate layer of completely weathered rock material of very stiff to hard silty sand and a hard layer of highly weathered rock material comprised of very hard silty sand.

In the area of the leach tanks the SPT values reached 50 blows/305 mm from a maximum depth of 1.50 m, indicating a bearing capacity of a minimum 200 kPa. Plate bearing tests in the same area produced maximum allowable bearing capacity ranging from 868 kPa to 1,388 kPa.

Geotechnical drilling was conducted by Get Services Sdn Bhd in 2017 over the area of the proposed sulphide plant to assist in foundation design. Results of the four geotechnical drillholes were summarised as follows in Table 17.8.

Table 17.8 Geotechnical drilling summary

Area		N value*
BH1 – Reagents area		
0–7.00 m	Loose and firm sandy clay and silty sand with a little gravel	4–7
7.50–37.50 m	Stiff to very stiff and hard sandy silt with some gravel	15–50
BH2 – Counter-current decantation area		
0–7.00 m	Loose to medium dense and stiff sandy silt with some gravel	6–20
7.50–19.40 m	Very stiff to hard and dense to very dense sandy silt and silty sand	12–40
19.40–26.25 m	Hard carbonaceous shale	50
BH3 – Flotation area		
0–7.95 m	Stiff to very stiff sandy clay and silty sand with some gravel	12–20
9.00–24.00	Stiff to hard, dense to very dense sandy clay and clayey sand with some gravel	35–50
BH4 – BIOX® reactor tank area		
0–4.00 m	Sandy silt and clayey sand with some gravel	7–8
4.50–20.30 m	Very stiff to hard, medium dense to dense clayey sand and sandy clay with some gravel	15–50
20.30–24.80 m	Carbonaceous shale	

**The number of hammer strikes it takes for the tube to penetrate the second and third 305 mm (6-inch depth) is called the "standard penetration resistance" or otherwise called the "N-value". This methodology commonly used for bearing capacity for structures and machinery.*

17.3.2 Plant layout description

The layout of the new flotation plant and BIOX® facilities shown in Figure 17.3 has been designed with minimum disruption to ongoing operations. Most of the BIOX® facilities will be located over an open area to the north and west of the existing facilities.

Earthwork requirements for the plant expansion will include the excavation of the area to be occupied by the limestone mill, the crushed limestone stockpile area and the new lime silo. A new access road to the TSF and cooling towers will be excavated alongside the pipeline corridor. Excavated material will be used to backfill the fish pond behind the plant office and to raise this area to the same elevation as the existing process plant.

The rougher and scavenger flotation cells will be located close to the secondary ball mill; cyclone overflow will be diverted to a new trash screen ahead of the rougher conditioners. The cleaner flotation cells will be installed further along but at a lower level. The concentrate thickener and BIOX® surge tank will be installed ahead of the six BIOX® reactor tanks, followed by the three CCD thickeners and six neutralisation tanks.

The limestone mill and lime silo will be located across the road at the base of the hill on which the cooling towers will be constructed. Crushed limestone will be stockpiled adjacent to the limestone mill. New reagents storage and mixing facilities, air blowers and compressors will be installed parallel to the BIOX® facilities. The BIOX® MCC will be located with the standby generator in the corner of the new facilities next to the plant access road.

The new BIOX® CIL tanks will occupy some of the footprint currently occupied by the existing workshop. The workshop will be moved to the opposite side of the warehouse when all chemicals are moved to a new facility as part of the additional reagents mixing area. The water recovery thickener will be located adjacent to the BIOX® CIL tanks.

Figure 17.3 Overall plant layout

Flotation

The existing cyclone cluster will be retained to provide the same classified cyclone overflow product of 80% passing 75 microns. The pipework will be modified to redirect the cyclone overflow to the new trash screen ahead of the first of two flotation conditioners. Flotation feed slurry will overflow the first conditioner to the second conditioner and from there to the first rougher flotation cell. Sodium sulphide and soda ash (if required) will be added to the first conditioning tank while potassium amyl xanthate (PAX) and MIBC frother will be added to the second conditioning tank. The three 70m³ rougher flotation cells will be arranged in series and installed at a raised elevation to allow ease of maintenance and to allow a pump level to be installed underneath. Rougher flotation tailings will flow into the scavenger conditioner where further MIBC and PAX are added and from there to the three 70 m³ scavenger flotation cells. Tailings from the final scavenger flotation will gravitate to the flotation tailings hopper and will be pumped to the water recovery thickener. Rougher and scavenger flotation concentrates flow separately to the cleaner flotation cells installed at a lower level to allow gravity feed. MIBC and PAX will be added to the cleaner conditioner ahead of four cleaner flotation cells each of 30 m³ capacity arranged in series. Cleaner flotation tailings will be pumped back to the rougher flotation feed conditioner #1 and cleaner flotation concentrate will be pumped to the concentrate thickener.

Figure 17.4 shows the general arrangement of the flotation circuit with the two flotation conditioners far right flowing into three rougher cells right foreground, followed by the scavenger conditioner in the centre. The three scavenger cells are behind the rougher flotation cells. The rougher and scavenger launders can be seen flowing into the cleaner conditioner and from there to the four cleaner cells. A bypass system will allow high grade (on-spec) rougher concentrate to flow direct into the cleaner concentrate hopper.

Figure 17.4 Overall flotation circuit

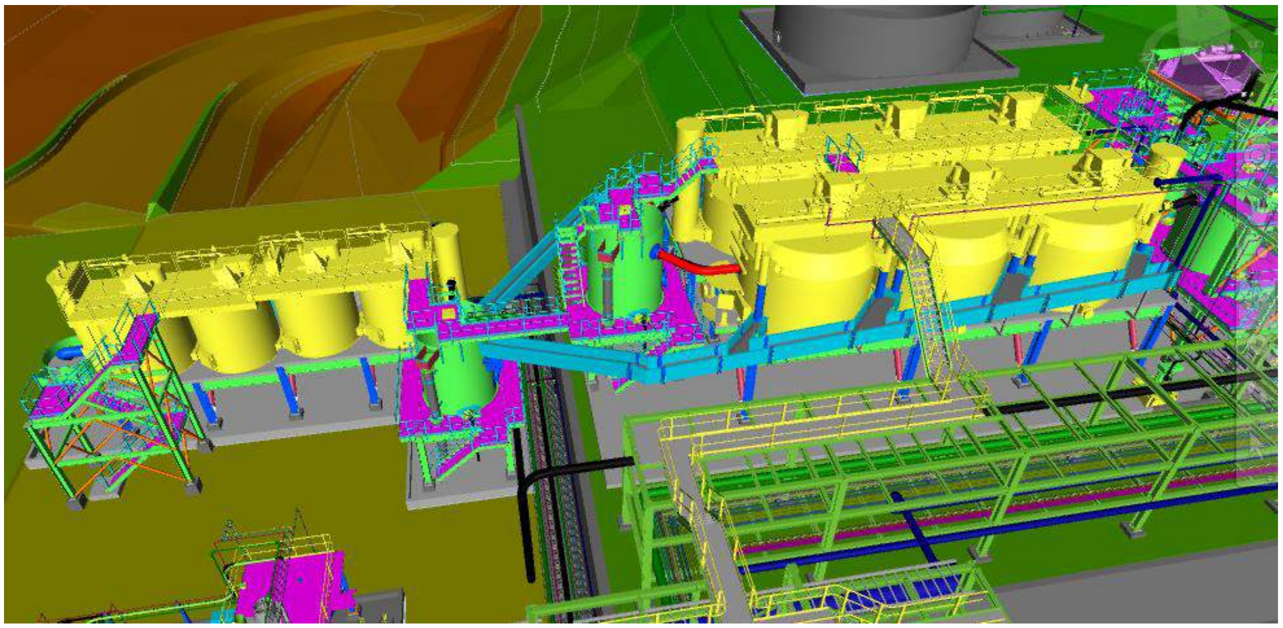
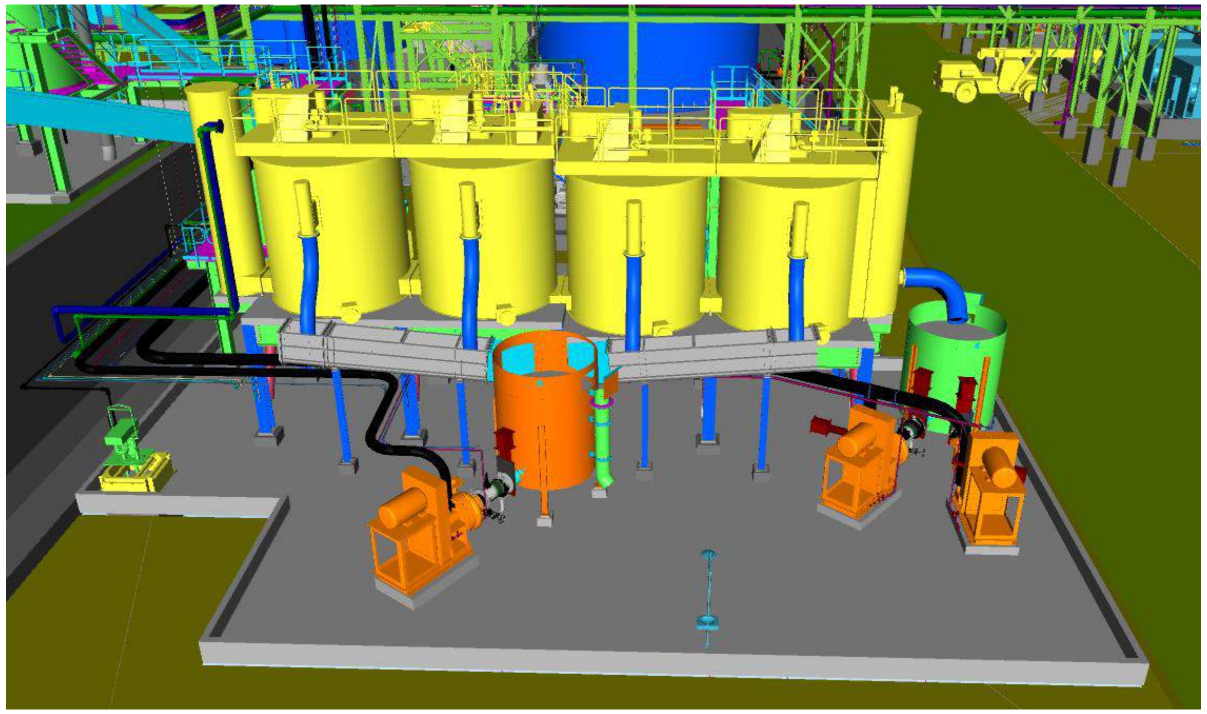


Figure 17.5 shows the four cleaner flotation cells with the cleaner concentrate hopper and pump shown in centre foreground and the cleaner tailings hopper and pump on the right. The cleaner cells are contained within a bunded area with sump pump for recycling any spillages.

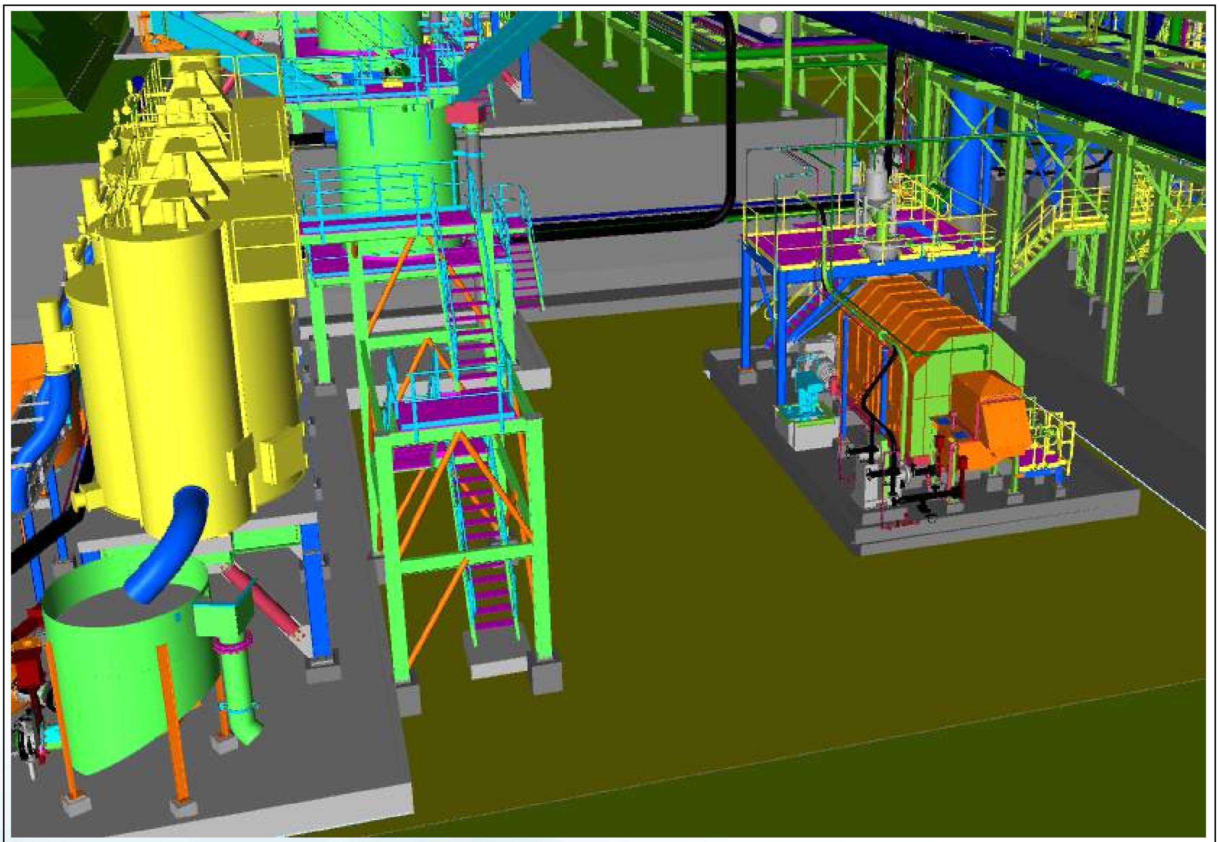
Figure 17.5 Cleaner flotation cells 1–4



Concentrate regrind mill

Space for a concentrate regrind mill as per Figure 17.6 has been allocated at ground level adjacent to the cleaner flotation cells. Should the concentrate regrind mill be required at a later date, it will be operated in open circuit, receiving the concentrate thickener underflow and pumping the mill discharge to the concentrate surge tank.

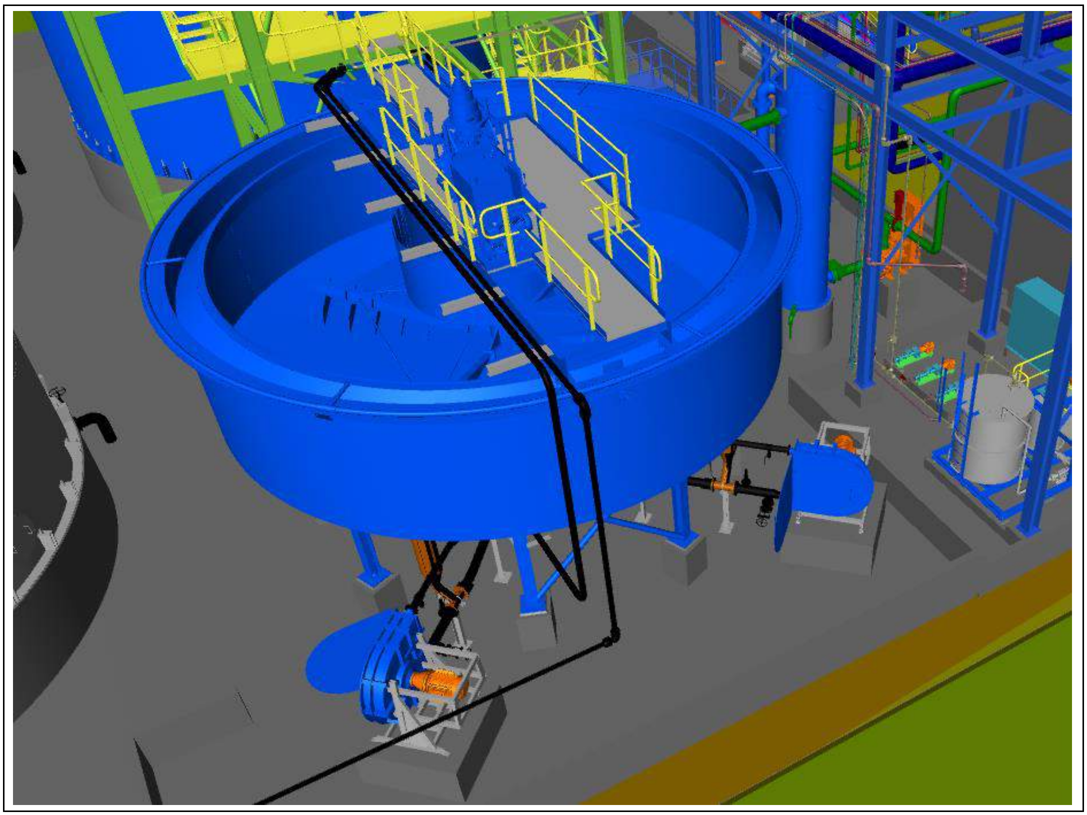
Figure 17.6 Concentrate regrind mill



Concentrate thickener

The concentrate thickener shown in Figure 17.7 is located adjacent to the BIOX® surge tank (refer Figure 17.8). The thickener will be 9 m diameter and constructed of carbon steel. Flocculant will be added at the thickener feed box to aid settling. Concentrate thickener underflow will be pumped to the BIOX® surge tank using one of two peristaltic pumps (one duty, one standby).

Figure 17.7 Concentrate thickener



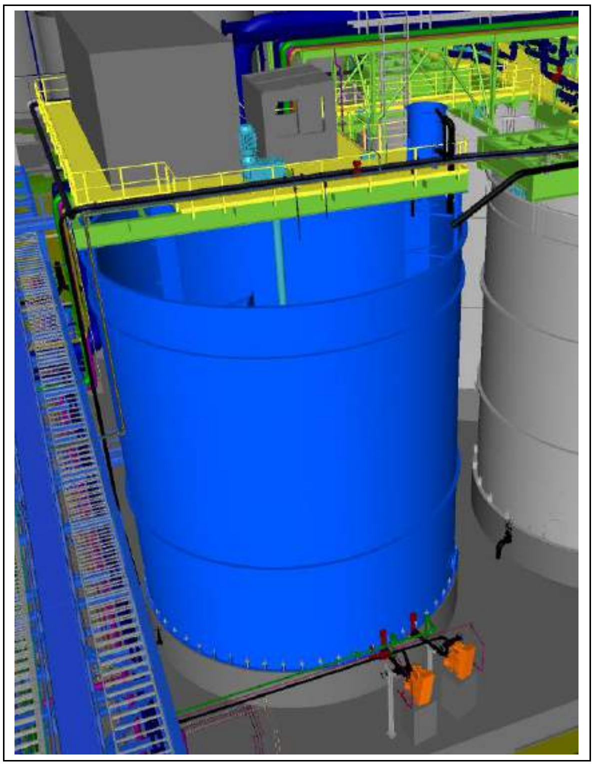
BIOX® surge tank

Figure 17.8 below shows the BIOX® surge tank with the first primary BIOX® reactor tank at right and control cabin with minilab above.

The BIOX® surge tank will have a total capacity of 60 operating hours at the design concentrate feed rate of 274 t/d; the concentrate will be stored at 50% solids. The surge capacity will even out any disruptions from the flotation plant operation and will provide sufficient blending of the concentrate when new sources of ore are processed. The BIOX® surge tank will be located close to the concentrate thickener and adjacent to the primary BIOX® reactors.

Concentrate will be fed to the primary BIOX® reactor tanks by variable speed pumps. Dilution water will be added to the pump suction to dilute the concentrate slurry by means of a nucleonic density gauge from 50% to 20% solids. A flowmeter will record the flowrate which when combined with the density will enable mass flow to be determined for process control and metallurgical accounting purposes.

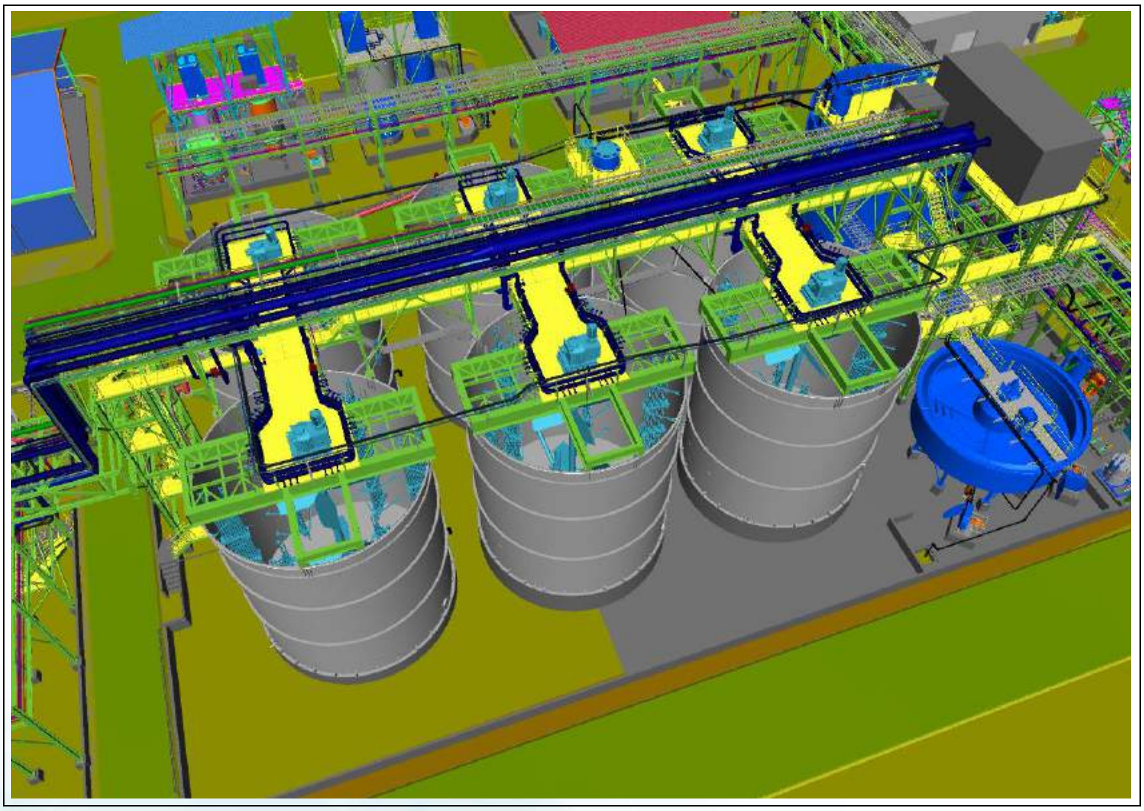
Figure 17.8 BIOX® surge tank



BIOX® reactors

The BIOX® reactors will be arranged as three primary reactors operating in parallel, followed by three secondary reactors in series. The general arrangement of the reactors is shown below in Figure 17.9. The concentrate thickener is shown far right and the control cabin top right with the BIOX® surge tank partially obscured underneath.

Figure 17.9 BIOX® reactors 1–6



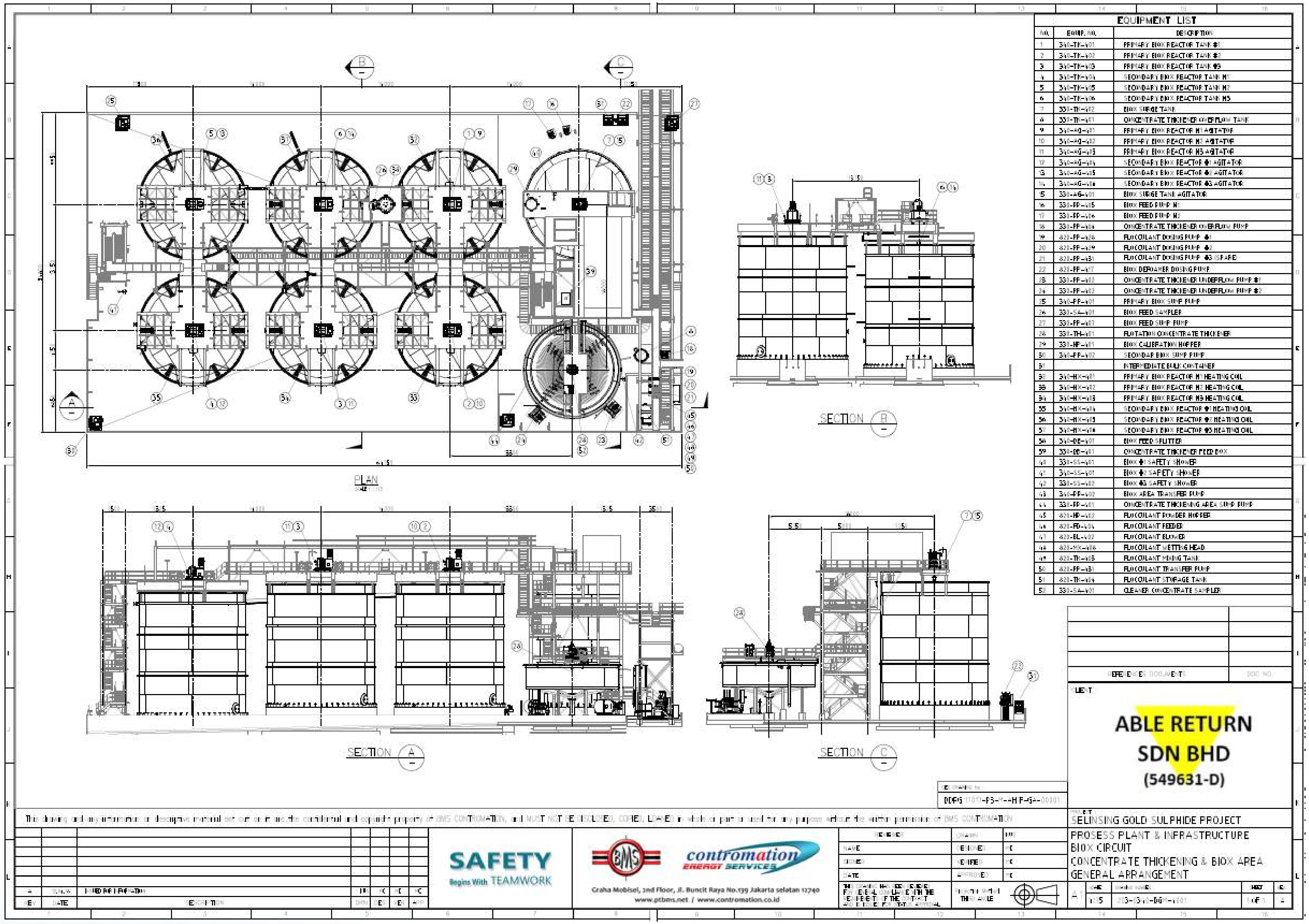
Concentrate slurry diluted to 20% solids will be pumped up to a slurry stream sampler where a representative sample will be taken for metallurgical accounting purposes. After sampling, the concentrate will gravitate to a distributor which will split the slurry into three equal streams with provisions for fourth and fifth streams. Nutrient solution will be pumped to the feed splitter by the nutrient dosing pump.

The primary BIOX® reactors will discharge into launders that will carry the combined flow form the three primary reactors to the first secondary BIOX® reactor. Bypass launders and dart valves will allow the bypassing of one reactor at any one time for maintenance. Slurry will flow from the first secondary reactor to the second and then the third.

The oxidised product from the third reactor will gravitate to the first inter-stage mixing tank of the CCD circuit.

The general arrangement of the BIOX® reactors is shown below in Figure 17.10.

Figure 17.10 BIOX® reactor tanks – general arrangement

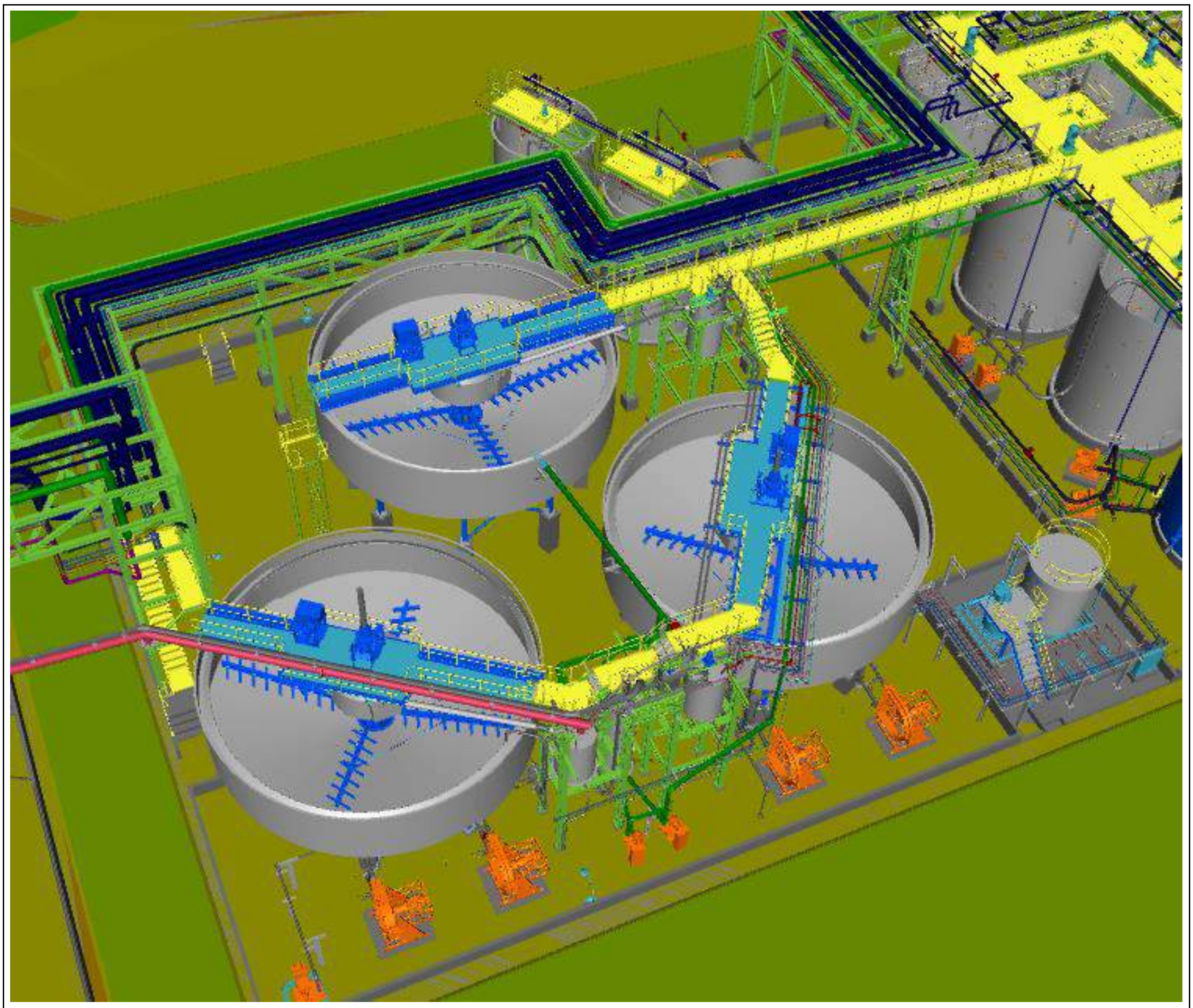


CCD thickeners

The three CCD thickeners are arranged in series. The general arrangement is shown in Figure 17.11 below with the neutralisation tanks far right and the flocculant mixing tank bottom right. The two pH adjustment tanks can be seen in centre background.

The discharge from the final secondary BIOX® reactor tank gravitates to the first inter-stage mixing tank where it is mixed with the overflow from the second CCD thickener. From the first mixing tank, the slurry flows into the thickener feed well where flocculant is added. The first thickener overflow discharges into the overflow hopper from where it is pumped to the neutralisation circuit.

Figure 17.11 CCD thickeners 1–3



The first thickener underflow is pumped to the second thickener inter-stage mixing tank where it is mixed with the third thickener overflow. The second thickener underflow will be pumped to the third thickener inter-stage mixing tank where it will be mixed with wash water. The second thickener overflow will be directed to the inter-stage mixing tank of the first thickener.

The third thickener underflow will be pumped to the first slurry pH adjustment tank and the third thickener overflow sent to the second thickener inter-stage mixing tank.

Neutralisation

The neutralisation tanks are shown below in Figure 17.12 below. The first neutralisation tank receives the overflow from the first CCD thickener. The solution will flow through a series of six neutralisation tanks. The neutralisation tanks will be aerated and agitated. Limestone will be added to the second or third neutralisation tank to raise the slurry to pH 4.5 and lime slurry will be added to the fifth or sixth tank to bring the slurry up to neutral pH 7.0. Recycle pumps will enable slurry to be pumped back upstream to optimise the pH profile and to maximise the neutralisation capacity of the limestone and the lime.

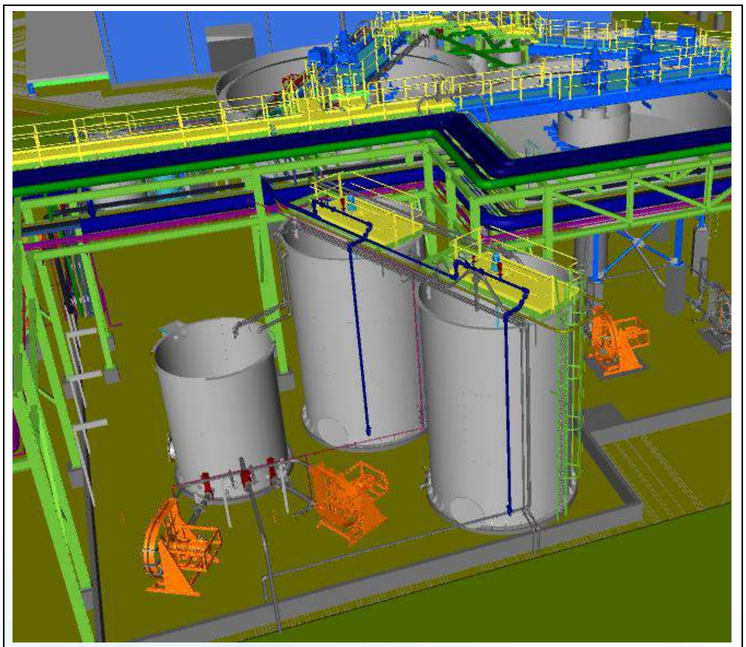
Neutralisation discharge will be pumped to the water recovery thickener; underflow solids will be pumped to the TSF and overflow water returned to the circuit.

Figure 17.12 Neutralisation tanks 1–6

pH adjustment

Figure 17.13 below shows the pH adjustment tanks 1 and 2 that are used to adjust the underflow solids from the third CCD thickener. Hydrated lime slurry will be mixed in the dedicated lime mixing tank using lime supplied from the lime silo. The lime slurry will be pumped to the first pH adjustment tank and mixed with the third CCD thickener underflow to adjust to pH 7. Slurry will overflow to the second pH adjustment tank where further lime additions will adjust the slurry to pH 11.

Figure 17.13 pH adjustment tank 1 and 2

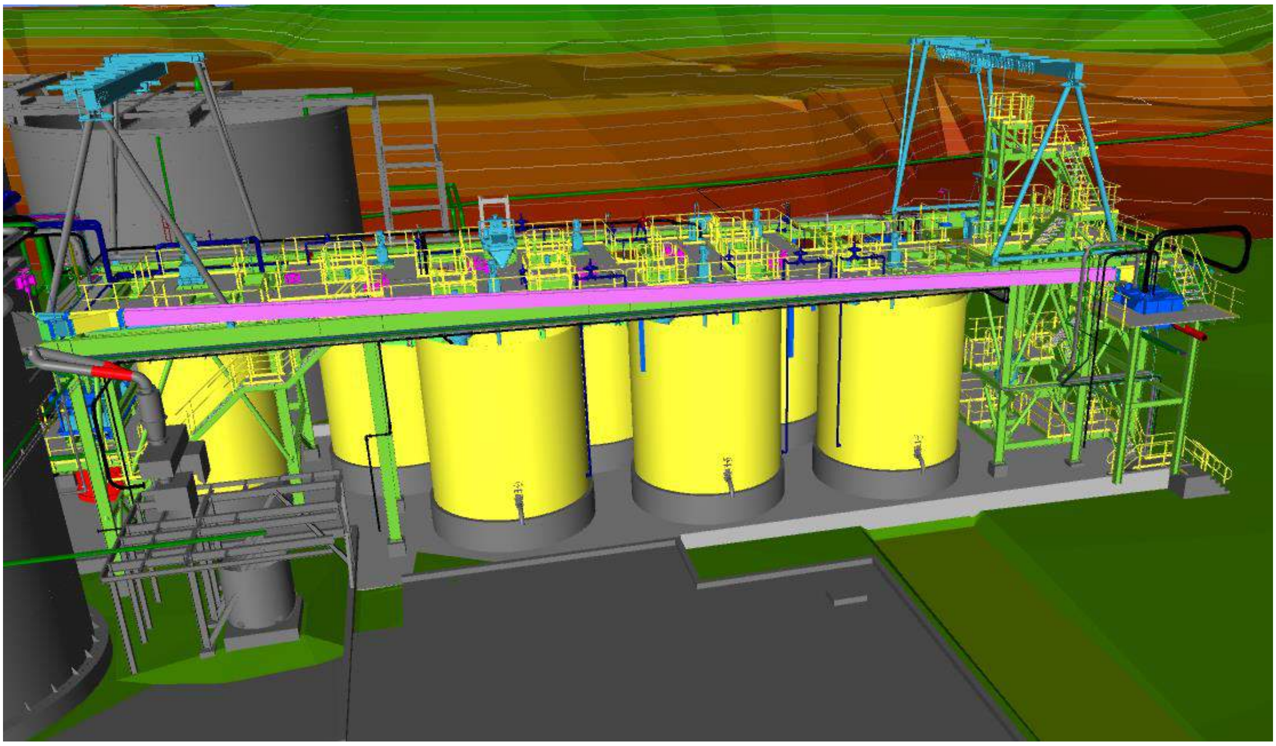


Discharge from the pH adjustment tanks will overflow into the pH adjustment hopper and then be pumped to the BIOX® CIL tanks for CIL process using cyanide.

BIOX® CIL

The BIOX® CIL tanks shown in Figure 17.13 receive the discharge from the slurry pH adjustment tanks. The six tanks will be of 134 m^3 capacity, each 5.5 m diameter x 6.0 m high. Inter tank screens will be installed on each tank; air lifts will convey activated carbon between tanks apart from CIL tank 1 that will have a recovery pump.

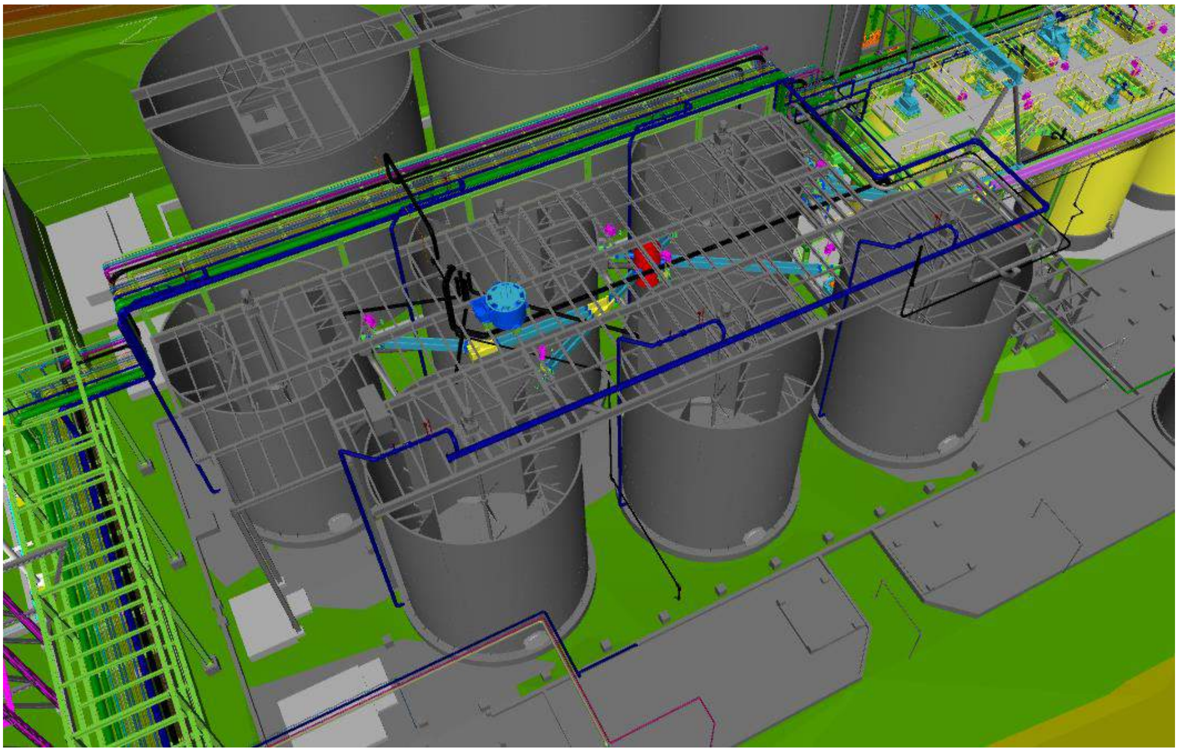
Figure 17.14 BIOX® CIL tanks 1–6



ASTER process

The existing leach tanks shown in Figure 17.15 will be reconfigured as ASTER™ process tanks by installing air spargers at the base of each tank and variable speed drives for the agitator motors. A distribution box will be installed above and between leach tanks 1 and 2. Discharge from leach tanks 1 and 2 as primaries will overflow to leach tank 3 as secondary ASTER™. Discharge from the secondary ASTER™ will gravitate to the water recovery thickener.

Figure 17.15 Old CIL tanks in foreground; old leach tanks for ASTER™ in background



Water recovery thickener

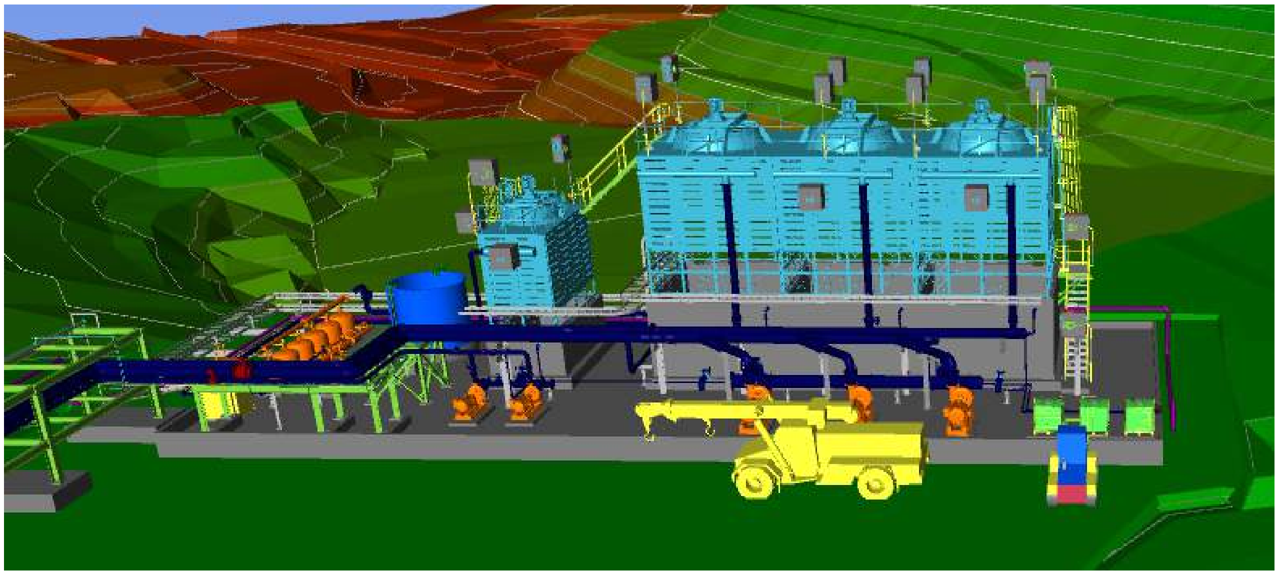
The water recovery thickener will be located close to the BIOX® CIL tanks and will receive the flotation tailings, neutralisation tailings and ASTER™ discharge. Thickener underflow will be pumped to the existing TSF, whereas the thickener overflow will be pumped up to the process water tank (refer Figure 17.16).

Figure 17.16 Water recovery thickener

Cooling towers

The cooling towers will be located on top of the hill behind the limestone mill and lime silo (refer Figure 17.17). The cooling tower location has been selected to avoid the dusty atmosphere that can occur around the crushing circuit during the dry season. Cooling water will be pumped through the cooling coils of the BIOX® reactors to maintain the design temperature of 41°C and circulated through the cooling towers before recycling back to the reactors.

Figure 17.17 Cooling tower for plant (looking west)

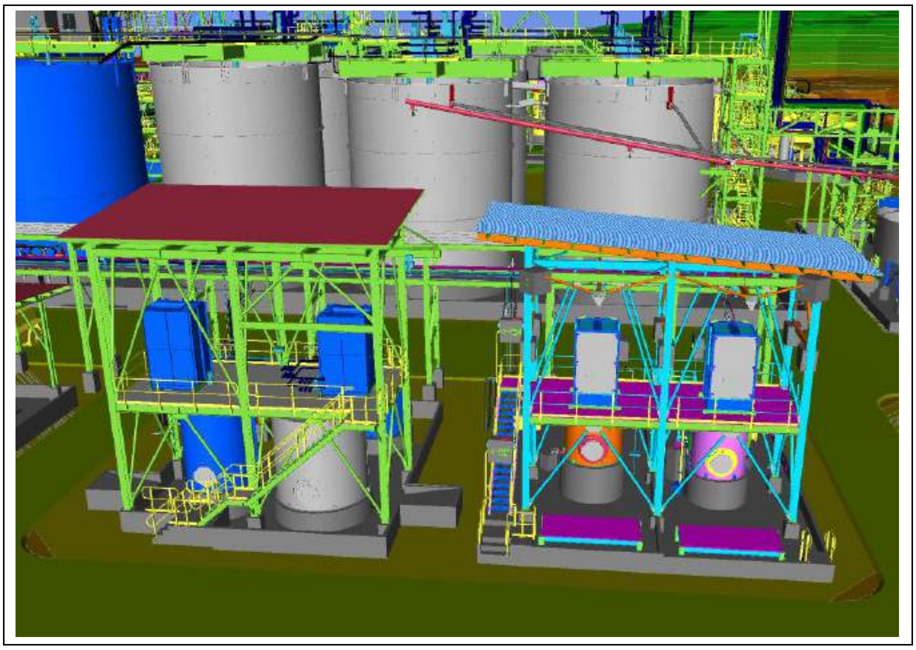


Reagents mixing

It is assumed that Na_2S/NaHS will be supplied in liquid form at approximately 40% mix strength. Exposure of this chemical to heat or contact with acids will result in the release of hydrogen sulphide gas, which is toxic. Therefore, Na_2S/NaHS storage will be kept away from acids, strong oxidising and reducing agents.

The BIOX® reagents mixing area is located close to the reagents warehouse. New mixing facilities will be provided for BIOX® and ASTER™ nutrients as well as for flotation reagents PAX, copper sulphate, Na_2S and tannin (refer Figure 17.18).

Figure 17.18 Reagents mixing



Hydrated lime will be drawn from the lime silo, mixed with water and stored in the lime storage tank at 30% solids. The lime will be used for neutralisation, pH adjustment and CIL. Sulphuric acid will be supplied by bulk tanker and stored in a 15 m^3 tank. Dosing pumps mounted at the tank will pump acid for use in flotation and BIOX® pH control.

Limestone mill and lime silo

The limestone mill and lime silo will be located at the base of the hill on which the cooling towers will be located. A crushed limestone stockpile will be established behind the mill; crushed limestone will be fed to the feed hopper by front end loader. Milled limestone slurry will be pumped across the bridge to the limestone storage tank (refer Figure 17.19).

Hydrated lime will be delivered by bulk tanker to the lime silo, as currently practised. Lime slurry will be pumped at 30% solids to the lime storage tank across the bridge.

Figure 17.19 Limestone mill and lime silo

18 PROJECT INFRASTRUCTURE

18.1 Location and site access

The Selinsing Gold Mine is situated approximately two hours' drive from Kuala Lumpur on a sealed six-lane highway first to Bentong, Pahang State and from there a sealed two-lane road continues north to Raub, approximately 65 km south of Selinsing.

Relative to local communities, Selinsing is approximately 4 km northwest of Kampung (Kg) Sungai (Sg) Koyan in the District of Lipis, Pahang. The project area is surrounded on three sides (north, east and south) by the Felda Sg Koyan 3 Scheme.

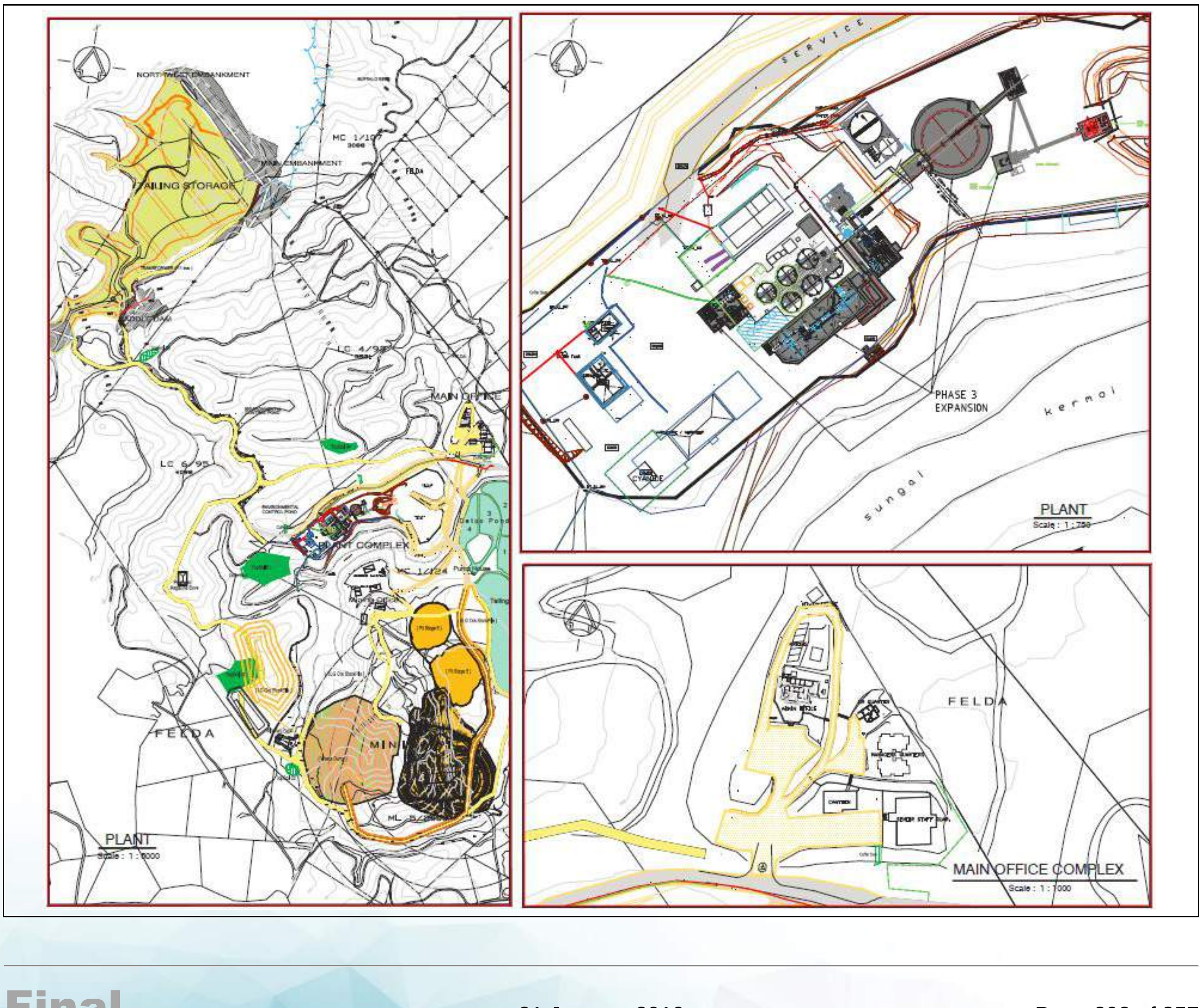
There are very few restrictions for transporting plant components provided they are nominal "wide-load" in nature. At Genting Highlands, 40 km from Bentong, a two-lane one-way 1,000 m tunnel must be considered. Of note, a hydroelectric power plant has just been commissioned where all equipment came through Sg Koyan to Cameron Highlands.

Police or professional escort is available at a nominal cost.

18.2 Existing plant

The existing 3,000 t/d oxide plant consists of three-stage crushing, a 5,000-t surge capacity COS stockpile reclaimed to primary and secondary ball mills in series followed by gravity separation and a CIL circuit. Tails are discharged to the TSF and surplus process water detoxified and discharged to a series of polishing ponds. Assay laboratory, workshop and warehouse are alongside the plant footprint.

Figure 18.1 Left: site overview; Top right: existing plant; Lower right: office and accommodation



18.3 Processing plant upgrade

The upgraded process plant is described in Section 17. Considerable associated infrastructure is also required as part of SGSP in terms of utilities, temporary facilities for construction, new buildings and transport infrastructure.

18.3.1 Process plant upgrade construction schedule

The overall timeline for plant construction is given in Figure 18.2. This shows actual plant construction will take some 12 months following detailed engineering design, with final commissioning during the initial production in Quarter 7.

It should also be noted the HV power construction as detailed in Section 18.4.2 will take until well into Quarter 7 to complete. It is planned to hire a diesel generator plant for commissioning and initial production until the HV power upgrade is complete.

A detailed construction schedule was completed using Microsoft Project and an extract is shown in Figure 18.3 showing the commissioning phase leading to production at full capacity following the allowed one month of commissioning.

Figure 18.2 Timeline extract showing overall plant construction

SELINSING GOLD MINE

SGSP Time Line

	Period:	Month	Days/month:	
Current : 03-Sep-18	Commence:	01-Apr-18	Period:	1
Period Highlight: 5	Completion :	01-Dec-25	Period:	

Year	0	1				2			
Qrt	0	1	2	3	4	5	6	7	8

DESCRIPTION					
FEASIBILITY STUDY		2-Apr-18	20-Sep-18	5.55	#
DEFINITIVE FEASIBILITY STUDY		2-Apr-18	25-Oct-18	6.68	#
PLANT CONSTRUCTION	CN	1-Oct-18	24-Mar-20	17.45	#
Project initiation	CN	1-Oct-18	27-Nov-18	1.87	#
Engineering	CES				
I Plant Process Engiuneering	OMC	1-Oct-18	22-Nov-18	1.71	#
HV Power Supply	TNB	1-Jan-19	24-Mar-20	14.48	#
Detailed Engineering	CES	1-Jan-19	18-Apr-19	3.48	#
Procurement	CN				
Earthworks and Civil	CN	15-Dec-18	11-Jan-19	0.90	#
Plant Engineering	CN	6-Feb-19	9-Mar-19	1.03	#
Infrastucture	CN	15-Dec-18	16-Jan-19	1.06	#
Plant electrical	CN	18-Apr-19	21-May-19	1.10	#
Equipment	CN	12-Nov-18	14-Dec-19	12.84	#
Materials	CN	12-Nov-18	2-Oct-19	10.48	#
Construction	CES				
Services	CES	23-Oct-19	30-Mar-20	5.16	#
Contracts	CES	16-Mar-19	31-May-19	2.48	#
Tankage erection	CES	24-May-19	16-Sep-19	3.74	#
Structural Mechanical and piping	CES	13-Jul-19	19-Oct-19	3.19	#
Electrical and Instrumentation	CES	7-Sep-19	31-Mar-20	6.68	#
Commissioning	CN/CES	21-Nov-19	1-Feb-20	2.35	#
Operation Readiness	CN				
BIOX Plant Establishment	CN	4-Jun-19	3-Jan-20	6.90	#
Plant Start Up	CN	1-Jan-20	1-May-20	3.94	#
Project Demobilisation	CN	1-Feb-20	19-Feb-20	0.61	#

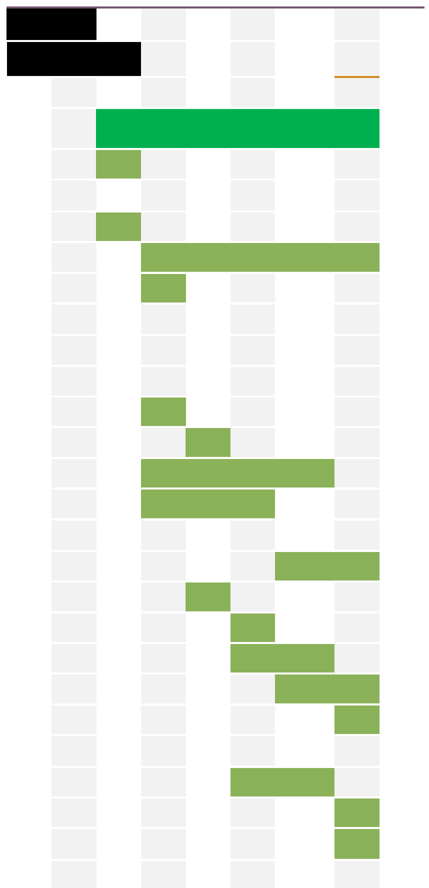
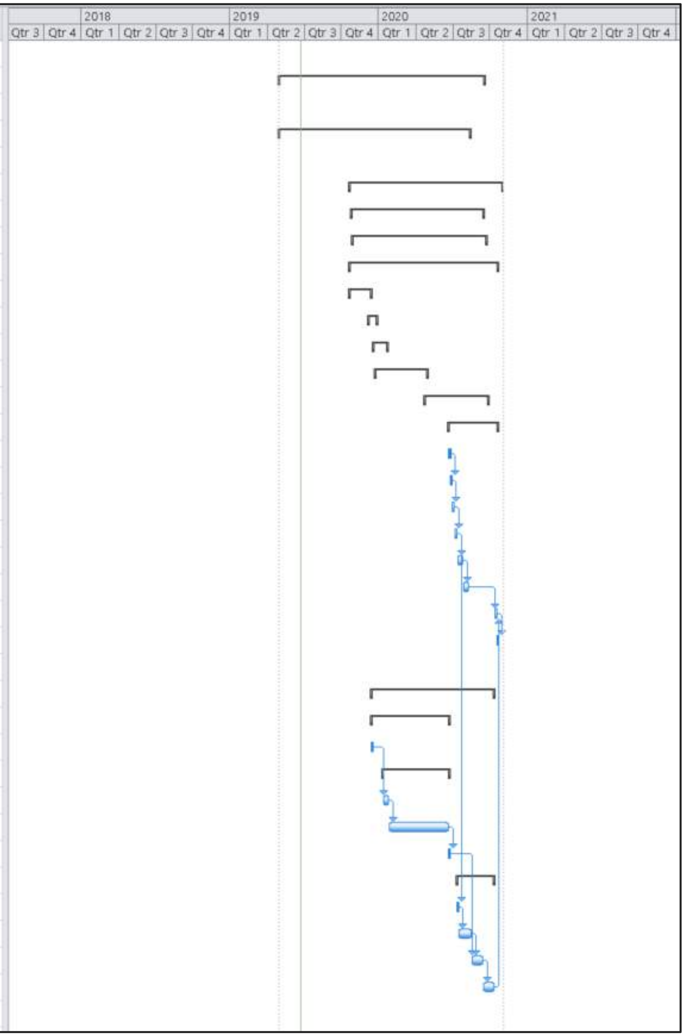


Figure 18.3 Extract from the detailed construction schedule showing the upgraded plant commissioning phase

	Plan Structure	Task Name	Duration	Start	Finish	Predecessors
46						
47	**3**	**Engineering**	**517.89 days**	**Mon 1/10/18**	**Tue 18/02/20**	
166						
167	**4**	**Procurement**	**481.33 days**	**Mon 1/10/18**	**Tue 14/01/20**	
652						
653	**5**	**Construction**	**385.33 days**	**Fri 22/03/19**	**Thu 2/04/20**	
654	**5.1**	**Construction Management**	**332 days**	**Wed 27/03/19**	**Sat 15/02/20**	
661	**5.2**	**Construction Services**	**336 days**	**Thu 28/03/19**	**Fri 21/02/20**	
668	**5.3**	**Construction Contracts**	**373.33 days**	**Fri 22/03/19**	**Sat 21/03/20**	
669	**5.3.1**	**Civils**	**54 days**	**Fri 22/03/19**	**Tue 14/05/19**	
675	**5.3.2**	**Concrete**	**23 days**	**Tue 7/05/19**	**Wed 29/05/19**	
683	**5.3.3**	**Site Tankage Erection**	**34.67 days**	**Mon 20/05/19**	**Sat 22/06/19**	
693	**5.3.4**	**Structural Mechanical & Piping (SMP) Erection**	**129.78 days**	**Fri 24/05/19**	**Sat 28/09/19**	
704	**5.3.5**	**Electrical & Instrumentation**	**160 days**	**Mon 23/09/19**	**Thu 27/02/20**	
716	**5.3.6**	**Commissioning**	**125.44 days**	**Wed 20/11/19**	**Sat 21/03/20**	
717	5.3.6.1	Commissioning - Establish Site & Procedures	4 days	Wed 20/11/19	Mon 25/11/19	701,713,690
718	5.3.6.2	Commissioning - Construction Handover	3 days	Mon 25/11/19	Wed 27/11/19	717
719	5.3.6.3	Commissioning - Dry Commission	7 days	Wed 27/11/19	Wed 4/12/19	718
720	5.3.6.4	Commissioning - Wet Commission	7 days	Wed 4/12/19	Wed 11/12/19	719
721	5.3.6.8	Commisioning - Half Production Rate	14 days	Wed 11/12/19	Tue 24/12/19	720
722	5.3.6.5	Commissioning - Full Production Rate & Hand over to Operations	12 days	Tue 24/12/19	Tue 7/01/20	721
723	5.3.6.6	Commissioning - Close out Package	7 days	Thu 12/03/20	Thu 19/03/20	722,737
724	5.3.6.7	Commissioning - Demobilise	3 days	Thu 19/03/20	Sat 21/03/20	723
725						
726	**5.4**	**Operation Readiness**	**308.33 days**	**Wed 15/05/19**	**Thu 12/03/20**	
727	**5.4.1**	**BIOX Pilot Plant Establishment**	**194.33 days**	**Wed 15/05/19**	**Fri 22/11/19**	
728	5.4.1.1	Operation Team Mobilisation	1 day	Wed 15/05/19	Wed 15/05/19	
729	**5.4.1.2**	**BIOX Pilot Plant Establishment, Cultivation & Training**	**166.33 days**	**Wed 12/06/19**	**Fri 22/11/19**	
730	5.4.1.2.1	BIOX Culture - Establish Facilities	14 days	Wed 12/06/19	Tue 25/06/19	728,386
731	5.4.1.2.2	BIOX Culture - Grow (On The Job Training)	24 wks	Tue 25/06/19	Tue 19/11/19	730
732	5.4.1.2.3	BIOX Culture - Initial Transfer to Plant	3 days	Tue 19/11/19	Fri 22/11/19	731
733	**5.4.2**	**Plant Start-Up**	**94.44 days**	**Wed 11/12/19**	**Thu 12/03/20**	
734	5.4.2.1	Accept Wet Commissioned Plant	4.44 days	Wed 11/12/19	Sat 14/12/19	720
735	5.4.2.2	Ore commissioning	30 days	Sat 14/12/19	Wed 15/01/20	734
736	5.4.2.3	Commissioning Ramp up	30 days	Wed 15/01/20	Wed 12/02/20	735,732
737	5.4.2.4	Plant Through put and Warranty Test	30 days	Wed 12/02/20	Thu 12/03/20	736
738						



18.4 Utilities

18.4.1 Plant water supply options

Sampling and analysis of water carried out for Pit 4, 5, 6 and the TSF showed no evidence of thiocyanate (SCN^-). Tailings return water therefore will be used for initial sulphide plant operation. The water recovery thickener will be used to recover high proportion of process water which will be pumped to the process water tank. Raw water will still be pumped from the Sungai Kermoi river. Excess process water will be discharged to the environment through the existing detox system.

18.4.2 HV Power supply

The processing plant upgrade requires significantly more electrical power than the current plant. Installed power is currently 4.2 MW, whereas 10.4 MW is required for the new plant. Table 18.1 shows the expected peak electrical power demand for the process plant only. This calculation excludes administration and other offices.

Table 18.1 SGSP plant upgrade electrical power requirements based on Monument plant operational cost calculations

Cost code	Plant item	Current power requirement (kW)	Additional power requirement (kW)	Installed power (kW)	Load factor	Availability	Current peak load (kW)	New power installed peak load (kW)
100	Crushing plant	615		615	88%	75%	406	406
300	Ball mill	3,000		3,000	88%	96%	2,534	2,534
320	Flotation	-	1,353	1,353	88%	96%	-	1,143
330	BIOX® feed	-	174	174	88%	96%	-	147
340	BIOX® reactors	-	893	893	88%	96%	-	755
350	CCD	-	235	235	88%	96%	-	199
360	Neutralisation	-	282	282	88%	96%	-	238
370	Water recovery thickener	-	393	393	88%	96%	-	332
380	BIOX® services	-	1,439	1,439	88%	96%	-	1,216
410	pH adjustment	-	43	43	88%	96%	-	36
420	BIOX® CIL	-	317	317	88%	100%	-	279
500	Electrowinning and gold room	50		50	88%	100%	44	44
820	BIOX® reagents	-	202	202	88%	96%	-	171
	Limestone mill	-	450	450	88%	96%	-	380
900	Tailings disposal and water reticulation	100		100	88%	96%	84	84
901	Detoxification	50		50	88%	96%	42	42
920	ASTER™	-	360	360	88%	96%	-	304
950	Laboratory	50		50	88%	96%	42	42
960	Water services	166		166	88%	96%	140	140
970	Air services	92		92	88%	96%	78	78
1,000	Workshop	50		50	88%	96%	42	42
1,001	Warehouse and procurement	50		50	88%	96%	42	42
Total		**4,223**	**6,142**	**10,365**			**3,456**	**8,656**

A new 33 kV dedicated powerline will be installed by Tenaga Nasional Berhad (TNB) from Kuala Lipis to the Selinsing mine site. Monument will construct a new switch room at the site main gate to TNB design. The switch room will house the TNB 33 kV metering equipment and the 33 kV SSU high voltage equipment.

Monument will be expected to pay the full cost of the new 33 kV powerline installation less the contribution from TNB amounting to the equivalent of 6 km of the powerline cost.

The new site 33 kV substation will be built by Monument to TNB specifications and ownership will be transferred to TNB once all electrical equipment is installed and the new powerline is connected. A 15 MVA, 33/11 kV step-down transformer will be installed inside the SGMM consumer building also designed to TNB specifications. Monument will fully own and maintain the 33/11 kV substation.

Two separate overhead lines will connect the 11 kV supply to the existing process plant MCC and to the new 11 kV switchboard for the BIOX® plant. The 11 kV line will be extended on an overhead line to the MCC opposite the limestone mill. A 33-kV HV line will be installed between the laboratory and TNB switch yard to avoid disrupting power supply during the construction of the neutralisation tanks; the existing line to the TNB switchyard will become the standby line once the new powerline is up and running.

Short term site power facility

A 1.5 MW diesel generator set will be used temporarily to power Project Office, contractor offices, ablutions, site accommodation and main administration building as well as construction equipment (i.e. welding sets, blasting and painting machine etc.).

Additional diesel generator sets may be required for initial sulphide production depending on completion of new 33 kV line.

18.4.3 Disposal and drainage

Site accommodation erected for the construction programme will include sewage facilities comprising underground septic tanks. Existing arrangement will be used for waste disposal by local contractors. The existing TSF will be used for the joint flotation tailings, neutralisation tailings and ASTER™ solid tailings from water recovery thickener underflow.

Stormwater drainage for the new plant and access road will be designed according to Malaysian Department of Irrigation and Drainage (DID) standards for 100-year events.

18.4.4 Building and facilities

All reagents will be housed in a new warehouse featuring loading ramps and segregated areas for storing acids and alkaline compounds. Plant maintenance facilities will be rehoused in the space vacated by the existing reagents storage. Process plant office will remain as it is now. Project office will be placed the valley adjacent to the construction site for easy access.

Each office will have air conditioners, power supplies and internet connectivity.

18.4.5 Transport infrastructure

The Sg. Koyan–Kuala Medang mine access road junction leading to the plant area will be upgraded with site crushed limestone and soil then compacted with compactors as its current condition has deteriorated over years of usage. This will help minimise dust spreading around during dry season and make it durable for years to come during plant operation thus less maintenance cost.

18.4.6 Communications

The project office will be fitted out with cable internet connections, Wi-Fi, Skype and telephone to ensure effective communications with HQ, consultants, contractors and suppliers via existing facilities available on-site.

A site radio system will be established (ICOM or Motorola being preferred) for the construction and commissioning stage; UHF frequency being considered acceptable. Communications are required with the existing process plant and mining channels; during mining and plant operation, operators and engineers will communicate using radio system like the existing system used currently.

Licence to operate walkie talkies are required by the Malaysian Communication and Multimedia Commission (MCMC)as they are classified as amateur radio devices, therefore the application should be made accordingly.

Ten additional computers will be purchased from the local IT supplier and it will include all necessary software and hardware to ensure high productivity of the staff.

18.4.7 Temporary facilities for construction

The project office will be established at the Selinsing site, initially making use of available space at the main administration office but moving to temporary buildings at the construction site once the physical work has commenced. Purpose-built construction cabins of various sizes are available in Malaysia as well as modified 20’ and 40’ shipping containers.

A site accommodation camp will be created for construction personnel and placed at the valley adjacent to the main office. Additional houses can be rented in the Kuala Medang or Sungai Koyan townships. SGMM site staff will stay at the existing quarters inside the site as well as at two lodges in Kuala Lipis, Pahang.

Diesel generator sets will be used temporarily to power the project office and site accommodation as well as site machineries such as welding sets, blasting and painting machine etc.

Water will be supplied from the nearby Sungai Kermoi river, licensed for an unlimited amount of water. Use will be made of the existing water reticulation system. River water may be used during the construction of the plant as well as for site accommodation usage.

In terms of lay-down areas, all construction material can be temporarily stored at a valley near the plant office therefore save the need to find other dedicated locations around the site.

18.4.8 Other infrastructure

Fencing around the existing plant will stay in place and Kami Security will continue managing the security of plant area. Temporary fencing will be erected around the construction site to minimise unauthorised access and reduce the opportunity for theft of construction tools and materials. Once the plant is fully constructed, a permanent fence will be erected around the plant perimeter to secure the area.

Fire protection services for the new plant will be designed by a fire protection specialist during the detailed design engineering stage with full compliance to the fire protection code of National Fire Protection Association. New FM200 systems (inert gas dispersion systems) will be installed and used at TNB SSU, SGMM consumer building and MCCs.

18.5 Water storage and usage

The water permit for the plant operation is issued by the Mineral and Geosciences Department to the lease holders.

The site water supply is drawn from Sg. Kermoi and the abstraction rate is limited to 200 cubic meters per hour.

Raw water is used for the laboratory, plant office, welfare facilities, mixing reagents and for specific process requirements. Process water is returned from the TSF and used for milling process.

Domestic water for the administration office and site accommodations is pumped from a bore hole to a 20,000-litre tank.

18.6 Fuel storage and usage

Current diesel fuel storage is unchanged from that mentioned in Snowden (Dec 2016), with diesel fuel contained in certified tanks located inside berms. Spillage is stored temporarily and removed as waste oils by a certified scheduled waste contractor. The tanks are 20,000-litre capacity and are licensed through the Ministry of Domestic Trade and Consumerism.

18.7 Explosives magazine and storage

No explosives are stored on site. All explosives brought on site are consumed in the blasts.

18.8 Contractor facilities

The current assessment of the contractor facilities is unchanged, and the hard stand provides for the repair and maintenance of the contractor's fleet consisting of:

- 18 haul trucks ranging from 20 t to 80 t capacity
- Nine excavators ranging in bucket capacity from 1.5 m^3 to 3.2 m^3
- Various ancillary equipment such as water bowsers, compactors, dozers, graders and light personnel carriers.

18.9 Accommodation

There is accommodation on site for 42 people (Figure 18.4). Mining contractor personnel reside in rented premises in the adjacent communities. The onsite accommodation consisting of twin bedroom apartments largely used by senior and junior management professional personnel and a dormitory style facility largely used by the security personnel.

Figure 18.4 Accommodation and canteen complex (office is the building at the rear)

18.10 Offices

The administration office is adjacent to the accommodations and canteen (Figure 18.5).

Figure 18.5 Administration office

18.11 Security

Security is outsourced to a local provider consisting of eight regular guards and four supervisors, all resident on site. The majority are stationed at the plant area, while some man the main gate entrance and the remainder are on mobile patrol duty. The Security Manager is an expatriate formerly with the Western Australian Police Force.

18.12 Tyres and waste refuse

Used grease and oils are removed from site by a Malaysian Government licensed scheduled waste handler. Used tyres are removed from site to a certified handler.

18.13 Ablutions and septic requirements and sanitation

Ablutions are available at the accommodations and septic tanks are de-sludged as required by a licensed contractor.

18.14 Procurement

The existing procurement system is in place and will be used to manage all purchases of equipment, consumables, scope of work, scope of services and miscellaneous items.

Major purchases are controlled through corporate approval as well as purchase order requisitions.

19 MARKET STUDIES AND CONTRACTS

19.1 Marketing

Gold from Selinsing has, for many years, been marketed through the AGR Refinery located in Perth, Western Australia. The arrangement is approved by the Customs Department of Malaysia. An audit trail for the Malaysian Central Bank (Bank Negra) tracks the export of Selinsing gold as well as when revenue from sale comes back into Malaysia. It is expected by Monument that this arrangement will continue.

19.2 Gold refining transport and sale

The gold will be poured as doré on site and shipped as about 85% gold content as indicated by site performance. The gold will then travel from the mine site to KL International Airport and by there by air to AGR Perth. AGR refine the gold and sell it on instructions at market and the shipper is paid in US$ out of New York, USA, through to Maybank (Malaysia's National Bank) in KL, where it is converted to RM.

20 PROJECT ENVIRONMENTAL APPROVALS AND PERMITTING

There are no environmental liabilities known to Monument at this time.

20.1 Malaysian mining legislation framework

No changes have occurred to Malaysian mining legislation since the previous Technical Report (Snowden, Dec 2016). Snowden (Dec 2016) reported the following:

"The mining industry comes under the purview of the Malaysian Ministry of Natural Resources and Environment (NRE). However, approval for mining related applications is empowered to the respective States in consultation with the federal agencies under the purview of the NRE, such as the Department of Minerals and Geosciences (DMG) and the Department of Environment (DOE).

The two main legal instruments that govern activities relating to mining are the Mineral Development Act 1994 (MDA) and the various State Mineral Enactments. Each State has its own legislation governing mining activities. One of the objectives of the National Mineral Policy project was to harmonise these State Laws and a Model State Mineral Enactment (SME) was prepared. As at end of 2008, 10 States had adopted the SME, including Pahang, the State in which the projects Selinsing Gold Sulphide Project - NI 43-101 Technical Report are located."

20.1.1 Mineral Development Act (1994)

"The MDA defines the powers of the Federal Government on matters pertaining to the inspection and regulation of mineral exploration, mining and other related issues, including environmental matters. The legislation is enforced by the DMG. Under the MDA the Minister may make regulations in respect of any matter which may be prescribed under this Act, this includes environmental protection measures, effluent standards, noise standards, vibration standards and other standards and means to protect the environment, provided such measures do not conflict with any provision of the Environmental Quality Act 1974 (EQA) (refer Section 20.1.3).

Before any development work and mining commences at a project, pursuant to the MDA, the holder of a mining lease is required to submit an Operational Mining Scheme (OMS) for approval by the Director of Mines. One of the aims of the OMS is to demonstrate that mining activities will be carried out with due consideration towards the preservation of the environment and conservation of resources."

20.1.2 State Mineral Enactment

"The SME empowers the States the rights to issue mineral prospecting and exploration licences and mining leases. The administration of the legislation is undertaken by the office of the State Director of Land and Mines (SDLM). In the case of the Selinsing Gold Mine, the Pahang Mineral Enactment 2001 (PME) applies. Pursuant to the PME, the holder of a mining lease for a large-scale operation shall not commence any development work or mining on the land until approval of a mine feasibility study and a plan for rehabilitation and an EIA under the EQA. The Selinsing Gold Mine, including the Buffalo Reef project is classified as a large-scale operation. The SME requires specific rehabilitation actions, inspections, reports, costings and schedules. The SME also stipulates the establishment of a mine rehabilitation fund that includes penalties if not paid."

20.1.3 Environmental legislative approval requirements

"*As previously discussed, some environmental aspects of a mining project are addressed under the MDA and SME, however the primary environmental aspects of mine development in Malaysia are regulated by the Environmental Quality (Prescribed Activities) (Environmental Impact Assessment) Order 1987, which is a subsidiary legislation to the EQA. The EQA regulates the prevention, abatement and control of pollution and the enhancement of the environment.*

The activities undertaken at the Selinsing Gold Mine are categorised as "Prescribed" activities under Schedule 11a and 11b of the EQA Order 1987. These activities are:

- *Mining of minerals in new areas where the mining lease covers a total area greater than 250 ha*
- *Ore processing, including concentrating for aluminium, copper, gold or tantalum.*

Due to the size and scope of the Selinsing Gold Mine Project, environmental related approvals are required under the MDA and the EQA.

Table 20.1 summarises the primary environmental and mining regulatory bodies and legislation applicable to the Selinsing Gold Mine Project."

The approval requirement under the MDA and the EQA are still current statutory requirements.

Table 20.1 Legislation applicable to the Selinsing Gold Mine Project

Step (Some steps may overlap or occur concurrently)	**Regulatory authority**	**Applicable laws and regulations**	**Approvals/permits**	**Comments**
Secure Mining Lease/s	Office of the State Mineral Enactment Director of Land and Mines (SLDM)	State Mineral Enactment	Issue of Mine Lease Approval	The SME empowers the States the rights to issue mining leases. The issuance of licences and leases by the State is subjected to certain conditions and restrictions as prescribed under the SME.
Approval of Operational Mining Scheme (OMS)	Department of Minerals and Geosciences Pahang (DMG)	Mineral Development Act 1994 (MDA)	OMS Approval	The MDA defines the powers of the Federal Government on matters pertaining to the inspection and regulation mining. The legislation is enforced by the DMG. The holder of a mining lease is required to submit an OMS to the Director DMG for approval. This must include management measures for environmental aspects such as Water, Erosion and Mine Abandonment.
Completion of EIA	Department of Environment (DOE) Pahang	Environmental Quality Act 1974 (EQA)	Conditional approval of EIA and approval of: • Environmental Management Plan • Rehabilitation Plan • Erosion, Soil and Sedimentation Control Plan (provided also to Department of Irrigation and Drainage).	The EQA, and its accompanying regulations call for EIA, project siting evaluation, pollution control assessment, monitoring and self-enforcement. Industrial activities are required to obtain the approvals from the Director-General of Environmental Quality prior to project implementation. For mining, an EIA is necessary if: • Mining of minerals in new areas where the mining lease covers a total area greater than 250 ha • Ore processing, including concentrating for aluminium, copper, gold or tantalum.
Secure secondary permits and approvals under EQA as required for Prescribed Activities	Department of Environment (DOE) Pahang	Environmental Quality Act 1974 (EQA)	Approvals and permits as required under *"Part IV EQA 1974 – Prohibition and Control of Pollution."*	Approval for fuel burning equipment installation under *EQ (Clean Air) Regulations)1978*.
Secure Operational Water Resources	Superintendent of DMG	Mineral Enactment 2001	Water Permit	

20.1.4 Permit and approval status

Snowden (Dec 2016) reported previous approvals to that date summarised below. No additional approvals have been granted since that time.

"*The project has been subdivided into three phases. Phase 1 is located on ML5/2000 where the pit has been developed by a previous operation. The second and third mining phases involve the development of two mine pits on ML1/111 and ML12/2012 with production increasing to 1 Mt/a. An expansion and upgrade to the processing plant is also required to accommodate the additional tonnes and ensure compatibility with change in ore feed characteristics. In summary, from a processing perspective, Phase 1 addresses the gravity circuit only, Phase 2 incorporates gravity and CIL at 400 kt, and Phase 3 includes gravity and CIL at 1 Mt/a.*

There have been two EIAs developed for the project:

- *The first was compiled in 2008 and addressed Phase 1 and Phase 2 of the project*
- *The second EIA was compiled in 2012 and was for the plant expansion programs from 400 kt/a to 1 Mt/a, addressing Phase 3 of the project.*

The EMP associated with the EIA for the Phase 3 plant expansion was approved by the DOE on 28 March 2016. It is not anticipated that the project expansion will be delayed or impacted by any further environmental approval requirements at this stage. To implement the Phase IV bio-oxidation process at Selinsing, Monument must first obtain the written approval from the DOE, however a new EIA is not required for this additional process.

A summary and status of the key environmental approvals and permits for the Selinsing Gold Mine Project is provided in Table 20.2 (below)."

Table 20.2 Status of key environmental approvals for the Selinsing Gold Mine Project

Licence description	Company	Issuing office	Issue date	Expiry date	Permit status	Comments
EIA approval for Selinsing gold mining and processing activities (Phases 1 and 2)	SGMM	DOE	04/12/2008	NA	Conditional approval	Conditional approval for gold mining and processing activities
Environmental Management Plan approval (Phases 1 and 2)	SGMM	DOE	08/03/2010	NA	Conditional approval	EMP on ML5/2000, MC1/124, MC1/107, MC1/111, LC6/95 and LC4/93; submitted by SBA Consultant Sdn Bhd
EIA: Expansion of gold processing operation (Phase 3)	SGMM	DOE	12/10/2012	NA	Conditional approval	Increased plant capacity from 360,000 t/a to 1 Mt/a; submitted by SBA Consultant Sdn Bhd
Installation and operation of lab scrubber and stack (No. 1)	SGMM	DOE	06/11/2009	NA	Approved	Reg. No: AKP/018/2009; permissible limit 0.4 gm/Nm^3; stack sampling under MS1596:2003, twice a year; submitted by Cradotex Sdn Bhd
Installation and operation of thermal oil heater and stack (No. 2)	SGMM	DOE	21/12/2010	NA	Approved	
Installation and operation of carbon regeneration kiln and stack (No. 3) and smelting and stack (No. 4)	SGMM	DOE	17/10/2012	NA	Approved	Written approval under regulation 38, *Environmental Quality Act 1978*; submitted by Harmoni Alam Environpro
Installation and operation of diesel generator	SGMM	DOE	07/12/2011	NA	Approved	Certificate no: APB/JANAKUASA/53/2011. Genset 1; 500KVA Scania SP500SCE-S; Genset 2; 160KVA Cummin CS160D5P; submitted by CSK Murni Sdn Bhd
Operational Mining Scheme	SGMM	DMG	11/06/2015	30/10/2018	Conditional approval	Renewal should be submitted one month before the expiry date
Water Permit	SGMM	DMG	11/06/2015	30/10/2018	Approved	The quantity of water to be used shall not exceed 200 m^3 per hour; submitted by SBA Consultant
Permit to Use Explosive	SGMM	DMG	11/06/2015	04/05/2016	Approved	Explosive for each blast shall not exceed Emulite/High Explosives 50 kg, ANFO/EMULSION 2,600 kg, Denotators and Trunkline Delays 120 Rounds; Short Firer Suhaidi Bin Ghapar I/C no. 790310-11-5049
Diesel Storage Permit (50,000 litres)	SGM	Department of Trade and Consumers Affairs	23/09/2015	22/09/2019	Approved	Permit granted on the annual basis and to be renewed one month before expiry date; storage address at MC1/124, Lot 3253, Mukim Ulu Jelai, Km 6 Jln Sg. Koyan-K.Medang

Licence description	Company	Issuing office	Issue date	Expiry date	Permit status	Comments
Registration of High Voltage Transformer 11KV/3440kW	SGM	Energy Commission, Pahang	20/05/2015	19/05/2019	Approved	Permit granted on the annual basis and to be renewed two months before expiry date; installation address at MC1/124, Lot 3253, Mukim Ulu Jelai, Km 6 Jln Sg. Koyan-K.Medang
Registration of Competent Person	SGM	Energy Commission, Pahang	04/02/2016	03/02/2019	Approved	Granted for one year and to be renewed two months before the expiry date
Permit for Buy, Store and Use Sodium Hydroxide (NaOH)	SGM	Pharmacy Department, Pahang	01/01/2016	31/12/2019	Approved	Granted for one year and to be renewed two months before the expiry date
Erosion, Soil and Sedimentation Control Plan (ESCP) Report	SGM	DOE and DID	28/01/2014	NA	Approved	
EMP Report for Stage 3 plant expansion	SGM	DOE	28/03/2016	NA	Approved	
Installation and operation of a stack on a scrubber	SGM	DOE	10/12/2013	NA	Approved	
Installation and operation of one chimney from two furnaces at laboratory	SGM	DOE	30/01/2012	NA	Approved	Written approval under regulation 36 and 38, EQA 1978; submitted by Alloyplas Technology Sdn Bhd
Installation of flotation and sulphide oxidation pre-treatment circuit	SGM	DOE	Application documentation still under review by SGM	NA	Application planning	Meetings to be arranged with DOE requesting written confirmation that the EIA remains intact

20.2 Environmental and social risks

20.2.1 Environmental management system

An environmental baseline study was conducted by SBA Consultant, a consulting company in 2009. In the EIA report, the consultant had identified all critical environmental issues, impacts and possible mitigation for the full-scale open pit mining project and CIL processing plant operation. As reported previously the expansion of processing plant to include BIOX® does require the written approval from Department of environment.

The BIOX® process is considered the key area of change with respect to environmental management compared to previous environmental issues reviewed in previous studies.

Bacteria bio-oxidation (BIOX®) is an oxidation process caused by microbes where the valuable metal remains (but becomes enriched) in the solid phase. In this process, the metal remains in the solid phase and the liquid can be discarded. Bacterial oxidation is a bio hydrometallurgical process developed for pre-cyanidation treatment of refractory concentrates. The bacterial culture is a mixed culture of *Acidithiobacillus ferrooxidans, Acidithiobacillus thiooxidans* and *Leptospirillum ferrooxidans*. The bacterial oxidation process comprises contacting refractory sulphide concentrate with a strain of the bacterial culture for a suitable treatment period under an optimum operating environment. The bacteria oxidise the sulphide minerals, thus liberating the occluded gold for subsequent recovery via cyanidation.

The BIOX® process is a proprietary technology owned by Biomin South Africa. The BIOX® process involves bacterial oxidation in agitated tanks for pre-treatment of refractory ores and concentrates ahead of conventional cyanide leach for gold recovery.

The formation of a stable arsenic precipitate is one of the advantages that the BIOX® process provides over certain other processes through the fixation of dissolved arsenic produced from the bio-oxidation of arsenic sulphide minerals. This is facilitated in a two-stage neutralisation process where the BIOX® liquor pH is increased to pH 4 to 5 using limestone. The slurry pH is then increased to pH 7 to precipitate out the remaining iron prior to solids deposition on the TSF. In conclusion, the BIOX® process has continued to prove itself as an extremely robust and environmentally friendly technology for the treatment of refractory gold orebodies. The technology is ideally suited to any location, be it remote (central Asia or Africa) or near areas with high levels of urban infrastructure (Australia). (Source Biomin Newsletter Second quarter 2014)

Monitoring and risk mitigation measures for bacteria in BIOX® process

Acids and metals, and other chemical products of the sulphide oxidation reactions that are either created or promoted by the bacteria, may be harmful to human health and the environment. Therefore, controls to prevent and contain spills need to be in place to limit exposures or accidental discharges of the untreated by products. Overall, focus is given to proper engineering and management processes, such as the selection of appropriate materials for tank construction and regular monitoring practices.

Risks of accidental release of water suspensions containing high counts of bacteria from the BIOX® Process should be controlled by a spill prevention plan for the mine site. Regular measures should suffice, such as checking fluid levels, monitoring seals around the base of the tanks, and monitoring transfer piping and pumps. Focus should be on spill prevention to control risks to a low level.

A spill prevention plan and active monitoring will be implemented at plant project commissioning to allow control of accidental release of bacteria. In order to determine whether there has been significant discharge of the bacteria used in the BIOX® process, rapid and modern monitoring methods can be put in place at the project site. One method, based on polymerase chain reaction, has been developed to provide rapid and specific detection of *Acidithiobacillus ferrooxidans* and *Leptospirillum ferrooxidans* (Escobar *et al.,* 2008).

20.2.2 Environmental approvals

In terms of Environmental management, the site is governed by approval conditions issued by the DOE in 2012. These resulted from the two EIAs developed for the project.

Snowden (Dec 2016) reported the first EIA was compiled in 2008 and addressed Phase 1 and Phase 2 of the project. The second EIA was compiled in 2012 and was for the plant expansion programs from 400 kt/a to 1 Mt/a, addressing Phase 3 of the project.

Most other environmental impact issues besides the BIOX® issues previously discussed were covered in the Phase 3 EIA. Snowden (Dec 2016) noted:

"*The Phase 3 EIA as required by the EQA, was developed for a production expansion to 1 Mt/a. The Phase 3 EIA was undertaken by independent consultants, SBA Consultants, in May 2012. The document assessed the following aspects associated with the Selinsing Gold Mine revised process, method and production rate that could potentially have significant environmental impacts:*

- *Air pollution*
- *Soil erosion and siltation and sedimentation*
- *Water quality*
- *Noise and vibration*
- *Cyanide management*
- *Solid and hazardous waste (excluding rock from the pit)*
- *Socio-economics*
- *Closure.*

Project activities or aspects ranked in the EIA as having a potential "major adverse impact" were primarily associated with soil erosion, hydrology and drainage, surface water quality, species and populations (terrestrial flora and fauna), forest resources and aesthetic and cultural. The DOE approved the EIA in October 2012. The Environmental Management Plan (EMP) addresses all environmental aspects raised in the EIA. The EMP was later approved in March 2016.

It is not anticipated that the project expansions or the Buffalo Reef project will be significantly delayed or impacted by these non-compliances, or issues associated with the EIA Conditions. The non-compliances identified during the audits have since been addressed or are being addressed."

20.2.3 Audits

Snowden (Dec 2016) reported in relation to external audits that:

"The EIA Approval Conditions were issued by DOE to Selinsing Gold Mine in October 2012. There have been eight regulatory audits undertaken by DOE against the 2012 EIA Conditions – July 2013, November 2013, March 2014, July 2014, November 2014, March 2015, July 2015 and November 2015. Nine non-compliances have been recorded in total. These were in relation to the incorrect storage of wastes, lack of slope protection and risk of erosion and water quality. The eighth and latest audit was conducted by CSK Murni Services Sdn Bhd.

It is not anticipated that the project expansions or the Buffalo Reef project will be significantly delayed or impacted by these non-compliances, or issues associated with the EIA Conditions. The non-compliances identified during the audits have since been addressed or are being addressed."

No further environmental audits were undertaken since the previous Technical Report (Snowden, Dec 2016).

Recommendation

The current EMP, as required by the EIA, was approved by the DOE in March 2016. Snowden (Dec 2016) recommended the EMP should be updated to include environmental compliance for Phase IV technology. They also recommended an entire Environmental Management System (EMS) is established for the project in line with a recognised standard such as ISO14001 Standard for Environmental Management Systems. This will facilitate the identification and management of environmental risks and help to ensure environmental compliance management requirements are achieved.

One important risk is the spill of BIOX® bacteria. The updated EMS will include a spill monitoring and management plant for this risk.

20.2.4 Mine waste management

The mine closure plan and overall waste management plan has not been updated since the previous Technical Report (Snowden, Dec 2016) and the findings of that report deemed sufficient with respect to the following:

"*A Conceptual Closure Plan (Closure Plan) was developed for the Selinsing Gold Mine in December 2011 by AECOM Consultants. The Closure Plan did not incorporate an assessment of the Buffalo Reef project. The Closure Plan will be reviewed and updated to include Buffalo Reef, if, and when, Stage IV goes ahead. Stage IV includes sulphide processing.*

The Closure Plan assesses risk associated with the key closure aspects based on information available at the time. AMD was ranked as a "high risk" for the open pit and waste dumps. Slope stability and re-contouring was also ranked as "high risk" for the TSF. All other aspects were ranked as "low risk" or "moderate risk".

The Closure Plan states that there is limestone for acid neutralisation available on site that can be used for AMD management. The mine has also developed Standard Operating Procedures (SOPs) for waste dump management and waste rock and stockpile management. The SOPs address AMD management and stability and, if complied with, will reduce the risks associated with both these aspects.

Water quality monitoring at the project to date has not identified that there is an AMD issue associated with current mining activities and waste landforms; however, AMD often takes some time to manifest and most of the waste dumps are constructed of oxide and not sulphide materials.

For the Selinsing Gold Mine and the Buffalo Reef project material characterisation is currently being undertaken by the Mineral Research Centre (operated by the Malaysian Department of Mineral and Geosciences) and will be signed off by its principal, Dr Shamsul Kamal Sulaiman. The geology modelling associated with Buffalo Reef to date indicates an increase in sulphide material, in comparison to the Selinsing project. This could indicate PAF material is present and the mine owners are aware of this risk.

The material characterisation results should provide important information associated with the volume and type of PAF that is likely to be encountered during the mining operations. It is crucial this information is incorporated into the mine plan to minimise future AMD risks. Consequently, finalising the material characterisation, particularly for Buffalo Reef, is high priority to reduce potential future risk and costs and to ensure that the mine can be closed and relinquished. Twelve PAF samples were send to the Mineral Research Centre (operated by the Malaysian Department of Mineral and Geosciences) in Ipoh for material characterisation assessment. Based on the testwork, two samples were classified as having potential to generate acid, two samples as uncertain, and the remaining eight classified as non-acid forming."

Assumptions on PAF waste rock management used for the FS are based on the Snowden (Dec 2016) PFS and no reasons are identified for changing those assumptions, with no changes to mine design nor any changes to waste dump design.

20.2.5 Tailings storage facility

Management of the TSF was reported by Snowden (Dec 2016) as follows:

"*Appropriate design and management of mine TSF is crucial. These facilities not only have associated geochemical risks, such as AMD, but also geotechnical/stability, ground and surface water, and dust aspects. There have been numerous catastrophic failures of TSFs that have been well documented in the media.*

The Selinsing Gold Mine TSF has been designed and constructed by specialists, Knight Piésold. The final design covers 45 ha and the facility can hold a maximum of 11 Mt. The facility is comprised of a compacted earth-filled dam, with a clay blanket on the upstream slope and a filter core layer and compacted structural fill on the downstream slope. The initial Stage 1 facility became operational in July 2010 providing capacity for over 16 months of tailings. Expansion of the facility will occur in lifts. The TSF has been expanded as a result of the plant upgrade to accommodate the increased tailings discharge for 10 years of construction. The current design has a dam top elevation of 530 mRL with a capacity of 6.7 Mt of tailings, bringing the total TSF operational life to 10 years. To date, an Embankment Stability Assessment has been undertaken.

There have been no significant issues identified, associated with the TSF to date. Risks associated with cyanide are mitigated due to the cyanide destruction process."

Snowden (Dec 2016) recommended: "*Using a reputable TSF engineering specialist company, such as Knight Piésold, to design and regularly review TSF performance significantly reduces risks associated with geotechnical failure and impacts to ground and surface water. These impacts can have significant environmental consequences and be very costly to rectify."*

Since that period and following on from these recommendations, a series of studies have been under taken by SRK and in-house.

SRK was appointed to undertake a scoping level design to increase the capacity of the existing TSF for the remaining LOM of seven years. A centreline raise of 9 m to a final elevation of 540 m RL was recommended. The embankment zoning will comprise an upstream low permeability zone (Zone A), an internal drainage zone (Zone F) and a downstream structural fill zone (Zone C). (SRK, Feb 2018).

A program of geotechnical investigation was undertaken to address stability analysis gaps including the testing of tailings, borrow material, embankment fill and waste dumps (buttress at saddle dam and south dam). The program is commenced in 23 June 2018 and included:

- CPT (Cone Penetration Test with Pore pressure measurement) drilling at the main embankment to test the strength of the underlying tailings
- SPT (Standard Penetration Test) drilling at the saddle dam to determine the strength of the waste rock placed as a buttress
- Installation of piezometers at the saddle dam
- Excavate test pits at proposed borrow areas at the TSF, Buffalo Reef and Selinsing pits.

The objective of this investigation is to explore the subsurface condition to obtain sufficient and reliable information for the TSF expansion design.

SRK in their proposal of work currently in progress, "*Height confirmation, Geotechnical Investigation and Detailed Design of TSF expansion*", May 2018 (SRK, May 2018), had the following scope of work:

- Raise height confirmation
- Geotechnical and Karst investigation
- Detailed design of initial lift.

SRK designed the first raise of the TSF to an elevation of 533.3 m RL or similar. The initial design includes a spring water management system and new drainage works that will tie in with existing system. To minimise construction cost in any given year, successive TSF lifts will likely be carried in small increments of 2 m or so to stay ahead of the rate of rise of the tailings beach and provide sufficient freeboard in line with safe practice and to satisfy regulatory compliance.

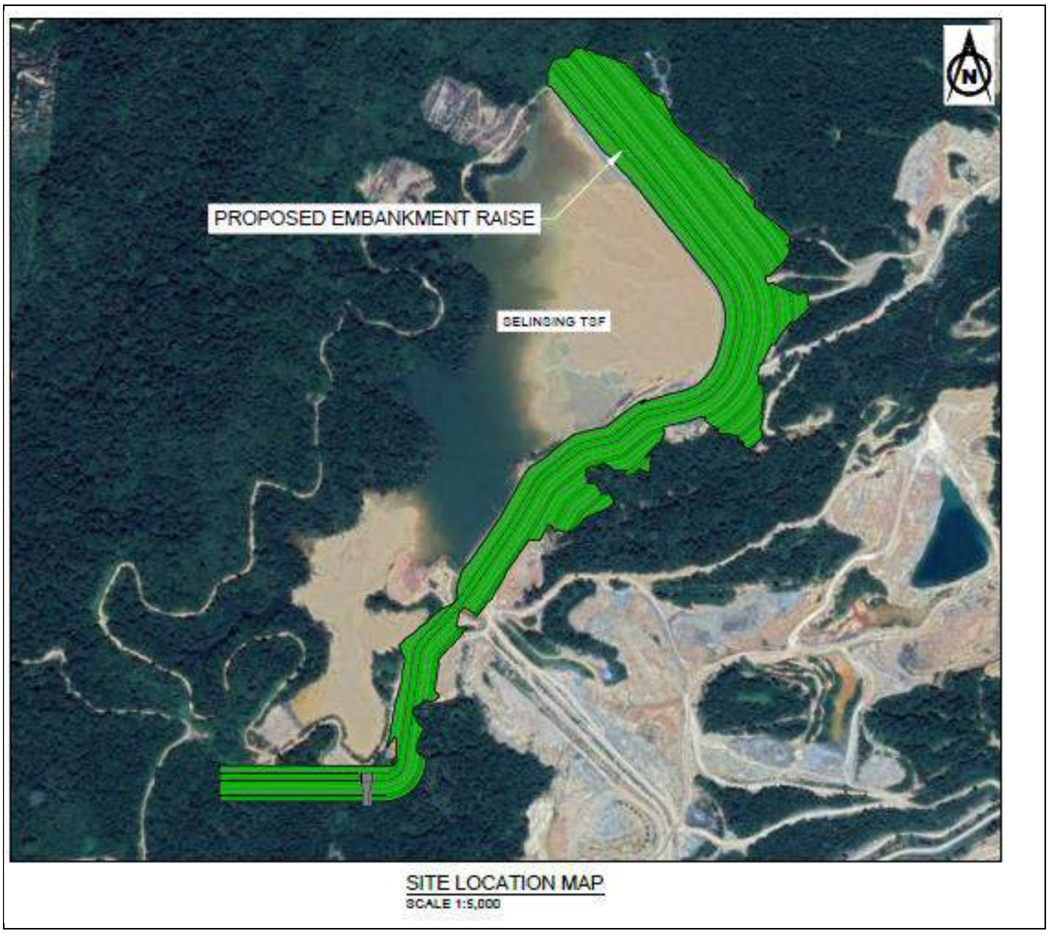
Construction of the spring water management system commenced in September 2018. New spring water ponds and diversion channels were excavated, and the new spring water collection node was constructed and covered with Zone C fill. The old spring water pond was backfilled with Zone C material and compacted to the desired specification. Site clearing at the base of the new embankment was completed, unwanted materials were removed, replaced with Zone C fill and compacted. The local stream was diverted away from the embankment footprint and the powerline was re-routed. Construction of the new finger drains and bulk fill of the Zone C main embankment commenced in January 2019

This initial raise will require an estimated 402,000 bcm of waste material to be extracted from various locations. Initial fill has been the waste rock from the Selinsing Pit V west wall cutback; further waste materials will be excavated from Buffalo Reef and the surrounding area. Regular compaction testing of the embankment fill was initiated. The anticipated capital cost for the first stage will amount to in the region of US$900,000.

Figure 20.1 and Figure 20.2 show the TSF location and planned wall expansion.

Figure 20.1 Site layout showing TSF location

Figure 20.2 Planned TSF expansion areas – SRK (Feb 2018)



20.3 Project water management

20.3.1 Water quality

Water quality was reported by Snowden (Dec 2016) and made the following recommendations:

"*The EIA provides information on baseline surface water quality in the project area, classifying the water as Class I and II of the Water Quality Index (DOE Water Quality Standards). Class I and II represent water bodies of excellent and good water quality. However, this was the water quality on the day the sampling was carried out, and the EIA specifies that the results represent only the conditions prevailing on the day of sampling.*

The mine is located within the catchment of the Sungai Jelai River; there are three rivers and their tributaries flowing within the project site, ultimately all the water enters Sungai Jelai, which is a major river rising in the Cameron Highland catchment.

Impacts associated with inappropriate water management at the project could include downstream sedimentation from erosion, or contamination from chemical and hydrocarbon spills, drainage blockages due to debris, preventing flow, all potentially detrimental to the river ecology. There is a drainage system in place at the project that must be well maintained to minimise downstream impacts."

The proposed BIOX® plant will utilise ASTER™ water treatment technology developed by Biomin. The process will improve water balance around BIOX® sites for a more cost-effective metallurgical operation. The ASTER™ process was developed to deliver an improved and integrated water balance for re-use in BIOX® applications. This simple technology has proven to be an effective process for degradation of tailings dam water at thiocyanate concentrations up to 4 g/L and free cyanide levels as high as 50 mg/L. The ASTER™ process offers a range of advantages to existing destruction processes as it allows efficient thiocyanate and cyanide removals to ≤0.2 mg/L, optimises microorganism performance for tailings feeds containing 4,000 mg/L thiocyanate and 50 mg/L cyanide and offers robust technology that is suitable for remote locations (source: Outotec website).

Snowden (Dec 2016) further reported:

"*There are several EIA approval conditions associated with sediment and erosion control and water quality, including water discharge monitoring requirements. The audits undertaken against EIA conditions identified five of these were in relation to inadequate erosion control measures and water quality. It will be crucial to ensure any drainage and water protection measures are determined and implemented in line with the project expansion and Buffalo Reef project.*

An Erosion and Sediment Control Plan (ESCP) has been developed and approved by the DOE and the Department of Irrigation and Drainage, as required by the EIA Approval Conditions. The ESCP was developed by engineering and environmental consulting engineers, SBA Consulting.

The EIA Approval Conditions specify that groundwater quality should compare to National Guidelines for Raw Drinking Water Quality, Revised December 2010 by Ministry of Health Malaysia. Groundwater monitoring bores have been constructed to monitor groundwater quality through operation until post-closure. Selinsing Gold Mine has a groundwater monitoring program in place, including monitoring locations, parameters and frequency that was approved by the DOE before implementation.

Groundwater monitoring results to date indicate water quality parameters have at times exceeded the National Guidelines for Raw Drinking Water Quality for manganese (Mn) and iron (Fe). The exceedances have been linked to the method used to sample groundwater. In the past, the groundwater in the boreholes has not been flushed prior to sampling, as required. Changes to the sampling program will be implemented and it is anticipated water sample results will reflect this. The boreholes that indicated elevated manganese and iron are located on the mine lease. DOE are aware of the exceedances via the EIA audits. There has been no indication from off-lease water monitoring that the water containing elevated manganese and iron has been discharged from the project."

Recommendation

Snowden (Dec 2016) recommended: "*Design and implement a surface water management and erosion and sediment control monitoring plan to incorporate the expansion and ensure ongoing compliance against EIA approval conditions*."

20.3.2 Water supply

Water permits for the Selinsing and Buffalo Reef areas are issued by the Mineral and Geosciences Department of the Federal Land Authority to the lease holders. The site water supply is drawn from a local river; the quantity of water to be used shall not exceed 200 m^3 per hour.

Recommendation

Snowden (Dec 2016) recommends a "*Review and update the site water balance model to ensure adequate water is secured for the project expansion and Buffalo Reef project development."*

20.4 Mine closure, remediation and reclamation and decommissioning

The SGMM conceptual closure plan framework comprises of pre-closure, closure and post-closure stages. It is a non-rigid, dynamic process by which continuous improvement on closure plans and mitigations can be developed and implemented to achieve regulatory compliance and cost effectiveness.

During the current pre-closure stage, closure planning considerations will be made in terms of information available, the scale of operations, and mine planning to better facilitate the transition from mine’s operation towards closure.

At the closure stage, decommissioning for machinery and plants takes effect. Final re-contouring and cover of disturbed areas takes place. During the closure stage, monitoring and maintenance scope and time line will be established to assess the stability and effectiveness of the closure plan.

The post-closure stage will primarily comprise of the detoxification plant, monitoring and maintenance activities. The detox plant is expected to treat and detoxify the residues contained in the existing TSF and detox pond prior to their ultimate closure. Monitoring will provide information on the process of the closure plan to address closure issue such as acid mine drainage, slope stability and re-vegetation of disturbed land. The time frame required for this stage shall be in agreement with local regulatory committee or agency to achieve compliancy. For SGMM, it is expected that the post-closure period will be for a two-year period.

Progressive rehabilitation will take place as project areas become available. However, decommissioning and closure of most facilities will be at the end of mine life. A final closure plan providing detailed plans and closure criteria will be developed prior to the planned closure date in order to meet regulatory submittal requirements. Closure execution is expected to occur over a period of one to two years. Post-closure monitoring and maintenance is expected to be for a two-year period to demonstrate the closure criteria has been met and the closure process is complete.

Clause 20 of the MDA focuses on mine abandonment and it stipulates the requirement for submission of a written notice and mine closure/abandonment plan. The written notice shall be submitted to Jabatan Mineral and Geosciences (JMG) three months before mine closure, while mine abandonment plan shall be submitted to JMG for approval within one month of mine closure. The mine abandonment plan shall address safe closure of abandoned mines and waste retention areas, as well as fencing off every mine shaft or audit to ensure public safety. The holder of the proprietary mining licence or mining lease shall continue to be responsible for the due compliance of this provision until JMG confirms in writing that the work has been properly executed.

Snowden (Dec 2016) reported:

“*The Selinsing Gold Mine, including the Buffalo Reef project, is classified as a large-scale operation. Pursuant to the PME, the holder of a mining lease for a large-scale operation shall not commence any development work or mining on the land until approval of a plan for rehabilitation is made under the EQA. The SME requires specific rehabilitation actions, inspections, reports, costings and schedules. The SME also stipulated the establishment of a mine rehabilitation fund that includes penalties if not paid.*

Additionally, EIA Approval Condition No’s 37 to 39 address premature closure of the project, specifying that the project proponent is responsible for the closure and rehabilitation of the site, for safety and environmental purposes. Consequently, appropriate financial provision is required. Estimated closure costs associated with the Selinsing Gold Mine Project, as determined in the 2011 Conceptual Mine Closure Plan developed by AECOM were 16,127,000 (MYR) or approximately A$5.3 million. Mine rehabilitation funds have been deposited for ML12/2012 (RM100,000), ML5/2000 (RM10,000) and ML11/2009 (RM10,000) as requested by the Pahang State Government.

The State Government has since revised the fund rates for mine lease renewals and a sum of RM300,000 will now be required for each lease.

It is understood that rehabilitation bonds for the Selinsing Gold Mine Project are up-to-date and accurate.

The intention of a Conceptual Closure Plan is to be used as a tool for mine closure, decommissioning and rehabilitation of all main mine facilities. Mine closure planning is a process that is initially conceptual and becomes more detailed as the project progresses and more information comes to hand. As specified in the Closure Plan, these Plans will be reviewed annually at a minimum, to ensure accuracy, currency and to provide information for project budgeting.

A Conceptual Closure Plan is required by and approved under the SME. A Conceptual Closure Plan was developed by AECOM approximately five years ago but did not address the closure requirements and costs associated with the Buffalo Reef Project."

20.4.1 Recommendation

Snowden (Dec 2016) recommended: "*To ensure risks and requirements associated with mine closure and planning do not impact on the development of the mine, it is a high priority to review and update the Closure Plan, accordingly, incorporating the Buffalo Reef project area. The revised Closure Plan should include all input from studies on PAF material, as reviewed on an annual basis to comply with the commitments of the Closure Plan required by and approved under the SME."*

20.5 Social

20.5.1 External relations

The external relationships are an important aspect of successful project delivery. In the broader sense and in support of national mineral policy and for self-interest, the emphasis is given to the contribution of the mining activities to the socio-economic development of the country through the development of efficient, responsible and sustainable projects. This includes the enduring relationship between the mine operation and all stakeholders that were impacted or involved in the project.

Stakeholders

The key stakeholders that are continuously engaged during the project life and their interest are listed in Table 20.3 below.

Table 20.3 List of key stakeholders

Category	Stakeholders	Interest
Government – State and Federal	Department of Mineral and Geosciences (JMG)	Regulatory compliance – Mining monitoring
	Department of Environment	Regulatory compliance – Environmental monitoring
	Department of Health and Safety	Regulatory compliance – Health and safety
	Pharmacy Department	Regulatory compliance – Consumables and chemical purchase and storage
	Department of Consumerism, Corporation and Domestic Affairs Ministry of Domestic Trade and Consumer Affairs	Regulatory compliance – Diesel consumption
	FELDA	Federal land management
	Pahang State Land and Mine Department (PTG)	Mineral tenement issuance and royalty collector
	Pahang State Development Corporation (PKNP)	Mining Lessee
	Malaysia Royal Police	Explosive permit & monitoring
	BOMBA	Regulatory compliance – Diesel storage
	Sg Koyan Local Clinic	Health support
	Lipis Hospital and Private Clinic	Health support
	Lipis District Office	Local council consultation

Category	Stakeholders	Interest
	State Executive Council	Tenement issuance and policy maker
	Energy Commission	Regulatory requirement – Energy generation and compliances
Community groups	Local village committee	Local impacts and opportunities
	Local Mosque and Religious committee	Local impacts and opportunities
	NGO	Local impacts and opportunities
	Sport, culture and social association	Local impacts and opportunities
	Private landholder	Lease holder
	Local and district community	Local impacts and opportunities
Education and training organisations	School, training college, university	Education, training and research
Key service providers	Telekom Malaysia	Telecommunication provider
	TNB	Energy provider
Vendors and supplier	Exploration, mining, processing, construction suppliers	Parts, equipment, material, consumables
Mining contractor	Minetech	Blasting, excavation and hauling
Consultants	Local and international	Mining, environment, construction

20.5.2 Government relations

The Selinsing gold mining and processing project had received ongoing support by the Malaysian Federal Government and Pahang State Government since its commencement in 2007.

The Malaysian Federal Government via the Ministry of International Trade and Industry had granted a manufacturing licence for the gold processing plant established at Selinsing Project and followed by granting of the pioneer status which the company had enjoyed for a period of five years till 2015. Federal department such as JMG, DOE, DOSH, Energy Commission, Pharmacy Department and KPDNKK had issued the licence to operate and monitoring the mining and processing activities. The provision of attractive fiscal and non-fiscal incentives as well as non-restriction on foreign equity had created a conducive investment climate for mineral development in the project.

Pahang state government had approved the mining leases and proprietary mining leases for it to be operated successfully over the years.

The Selinsing Project has demonstrated its ability to balance the needs of the organisation to create profit as well as operating responsibly and sustainably to create value for all the stakeholders that includes the environment, community and the country. As a result, in 2017 the project had been nominated by the Ministry of Natural Resource and Environment to represent the country for the ASEAN Mineral Award for Sustainability.

20.5.3 External relations program

The project is in line with the principal objectives of the Malaysian Ministry of International Trade and Industry Industrial Master Plan; that is, to promote opportunities for the maximum and efficient utilisation of nation's abundant natural resources.

The Selinsing Gold Mine is considered one of the principal economic activities in the district of Lipis. There has been extensive investment in community programs to share the benefits of mining. Community development strategies and programs have been developed that enhance education, health, environment and socio-economic development. Wages and benefits from employment will continue to be a major economic stimulus to the local community and the State Government of Pahang.

Education programs range from offering scholarships, supporting the school and tertiary education, extra curriculum activities, seminars, workshops and collaboration with tertiary education bodies in regard to educating the university students in industry practices, funding through internship programs with various institutions of higher learning in the country, preparing a skilled workforce for the future. The support benefits more than 50 local schools and institutions and more than 100 various organisations have received benefits from direct contributions to date.

Key programs include:

- Annual recognition for local students that excel in national examinations
- Broad base support of community events and sports programs such as badminton, football and archery
- Support for the orphanage, disability organisations by providing extra funding to purchase necessities etc.
- Supporting events that promote environmental awareness programs at the district and state level
- Organising health surveillance and free medical check-ups for the local community.

The project EIA generally identified positive socio-economic impacts associated with the development and operation of the Selinsing Gold Mine, including employment opportunities for local communities. There are no known significant community concerns or active anti-mine lobby groups associated with the mine or the expansion.

21 CAPITAL AND OPERATING COSTS

21.1 Cost estimation and battery limits

The SGSP battery limits for capital costing reflect the base case scenario for the mining and processing of sulphide ore. The operating and capital costs for the Project are based on the mining, processing and sale of gold doré as defined in the SGSP scope of work in Section 2. All costs in this section are in US$. The SGSP plant capital costs are planned for 2019 and 2020 for an approximate 12-month construction period as described in Section 18.

21.2 Capital costs

21.2.1 Accuracy of estimate

Capital costs are calculated in line where possible within FS limits of accuracy of ±15% unless stated otherwise. No escalation has been allowed for in capital cost estimates.

21.2.2 Basis of capital cost estimate

The oxide processing plant will continue processing stockpiled oxide materials until the BIOX® plant is commissioned in SGSP Q7. Capital costs from the 2018/2019 budget were used for Year 1 sustaining capital and the capital cost estimates for the plant including remaining years sustaining capital costs were estimated by Monument and CES with review by Mike Kitney.

The MYR/US$ exchange rate assumed was 4.1037 based on the previous six months average based on Bank Negara Malaysia FX rates.

The sulphide plant pre-production capital costs in the cash flow model have been allocated at 4% in Year 1 and 96% in Year 2 of the one year of construction.

The capital and operating cost estimates were prepared or advised by the following groups listed in Table 21.1.

Table 21.1 Sources of capital and operating costs

Item	Compiled by
Sulphide process plant	Monument and external parties listed in Table 21.5 (reviewed by Mike Kitney)
EPC management	Monument (reviewed by Mike Kitney)
Mining	Monument (reviewed by Snowden)
Oxide process plant	Monument (reviewed by Mike Kitney)
On-site infrastructure	Monument and external parties listed in Table 21.5 (reviewed by Mike Kitney)
Off-site infrastructure	Monument (reviewed by Mike Kitney)
Environmental	Monument (reviewed by Snowden)
Social	Monument (reviewed by Snowden)
Corporate G&A	Monument (reviewed by Snowden)
Royalties	Monument and regulatory authority
Taxation	Monument

The capital costs have been estimated following a series of indicative prices received from major equipment manufacturers and suppliers. All capital costs have been reviewed by Mike Kitney, who is Monument's internal Qualified Person.

Import duties on capital equipment not sourced but manufactured in Malaysia are high and as such work has been done to minimise, or where possible eliminate importing items not supplied originally or manufactured in Malaysia. On advice from Monument, no import duties have been applied on capital equipment. No escalation has been applied to the capital cost estimate with execution considered imminent.

Plant construction estimate

The plant construction cost has been estimated by Monument with review by Mike Kitney and is shown in Table 21.2.

Table 21.2 Summary of plant construction capital cost estimate

Component	Total/Area (US$) initial	Ongoing (US$)	Contingency (US$)	Contingency %	Total (US$)
Flotation	3,650,297		365,030	10%	4,015,326
BIOX®	13,934,059	600,000	1,393,406	10%	15,927,465
BIOX® – CIL	4,020,542		402,054	10%	4,422,596
Detoxification	50,000		5,000	10%	55,000
ASTER™ process	2,016,853		201,685	10%	2,218,538
Air/Water/Plant services	280,931		28,093	10%	309,024
Piping	866,743		86,674	10%	953,417
Other plant upgrades	898,825		26,407	2.94%	925,232
Detailed engineering	2,820,000			0.00%	2,820,000
First fill	1,630,307		163,031	10%	1,793,337
Owner cost/EPCM	1,298,315		129,832	10%	1,428,147
Total – Plant Construction	**31,466,671**	**600,000**	**2,801,212**	**8.9%**	**34,868,083**

Power upgrade capital cost estimates

The other significant capital cost is the electrical power cost upgrade at US$4.86 million. Table 21.3 shows the total capital cost for the electrical power upgrade.

Table 21.3 Total power upgrade capital cost estimate

Description	Cost (US$)	Contingency 10% (US$)	Total (US$)
MCC Building	126,763	12,676.32	139,440
11kV Diesel	250,011	25,001.07	275,012
630A – Incomer Panel	13,476	1,347.56	14,823
630A – Transformer Feeder Panel	25,148	2,514.80	27,663
630A – Outgoing Feeder Panel (Spare)	8,383	838.27	9,221
BUS VT PANEL	4,874	487.37	5,361
TNB SSU	3,060,126	306,012.57	3,366,138
MCC-01	178,271	17,827.15	196,099
MCC-02	378,635	37,863.49	416,498
MCC-03	382,543	38,254.30	420,797
Subtotal	**4,428,229**	**442,823**	**4,871,052**

Open pit pre-strip capital costs

Capital costs were allocated for open pit capital pre-strip based on:

- Life of the pit over one year
- Two months waste mining costs assumed for each pit in these cases
- Waste mining costs derived from schedule-based costs as described in Section 16.19.22

The mining costs were determined during the scheduling process to account for variable haulage and drill and blast increments in fresh rock. At that time, a FX rate of 4.085 MYR/US$ was assumed this later factored to balance with the assumed FX of 4.1037 MYR/US$.

A total of US$7.665 million has been estimated for the FS capitalised waste mining cost. The schedule of waste capital cost for the Base Case is shown by quarters and pit area in Table 21.4.

Table 21.4 FS waste capital cost estimate

Pit area	Total (US$M)	Quarterly capital cost (US$M)													
		0	1	2	3	4	5	6	7	8	9	10	11	12	13
Selinsing Pit 4	0.61							0.61							
Selinsing Pit 5 and 6	0.79							0.79							
Selinsing Pit 7															
Buffalo Reef C1															
Buffalo Reef Ox Pits Damar															
Buffalo Reef Ox Pits Felda Block 7															
Buffalo Reef C2 Stage 2 Pit Damar															
Buffalo Reef C2 Stage 2 Pit Felda															
Buffalo Reef C2 Pit Damar	1.93											1.24	0.69		
Buffalo Reef C2 Pit Felda Block 7	0.66											0.44	0.23		
Buffalo Reef C3 Pit Damar	0.20													0.20	
Buffalo Reef C3 Pit Felda Block 7	0.84													0.83	
Buffalo Reef C4	2.64											1.71	0.94		
Buffalo Reef N1															
Buffalo Reef N2															
Buffalo Reef N3															
Total (US$M)	**7.66**							**1.40**				**3.38**	**1.86**	**1.03**	

Other initial SGSP capital costs

Other costs associated with the processing plant upgrade for the FS include:

- Closed circuit TV – US$35,000
- New computers – US$50,000
- Entrance road upgrade – US$120,000
- BIOX® training requirements – US$750,000.

TSF construction costs

TSF construction for the FS is described in Section 20. The cost estimate assumes three lifts are required for the SGSP-only period and one for the preceding oxide-only period.

A total of 1.2 Mbcm of material is assumed to be required to be sourced from open pit waste material to complete the 9 m high lift required for the FS Base Case LOM. This forms the bulk of the fill material required in terms of structural fill but some low permeability material is also required.

The cost to produce this structural material is covered by waste mining capitalised costs and is viewed as a partial allocation of these costs to TSF. The contract waste mining costs varies between MYR to MYR 10.5/bcm or US$1.96 to US$2.57/bcm with US$2.27/bcm average. An allocation 33% of the waste mining cost of material required for the TSF construction at US$0.75/bcm cost is costed to the TSF.

The anticipated total TSF cost of US$1.8 million comprises the following:

- Structural fill 1.2 m bcm x US$0.75/bcm = US$900,000
- Other low permeability material drainpipes and miscellaneous item US$225,000 per lift with four lifts x US$225,000/lift = US$900,000.

In the cash flow analysis, a cost is attributed to TSF but the waste mining portion (50% of the TSF total) is subtracted from the total capital costs.

Rehabilitation costs

In line with Snowden (Dec 2016), approximately US$100,000 is assumed for rehabilitation over the LOM. A total of US$600,000 being assumed over the six-year life. US$500,000 is assumed for the SGSP and US$100,000 for the Oxide-only period in 2018/2019.

Buffalo Reef river diversion

A river diversion is required for BRF C4 pit. A total of 330,000 bcm of material is required for the excavation. This amounts to US$500,000 in capital cost based on a near-surface mining cost.

Capitalised sustaining costs

Similar assumptions to Snowden (Dec 2016) are made for the upgraded Processing Plant with associated sustaining costs of US$600,000 per year. US$100,000 is associated with Oxide CIL only production, giving a total of US$2.5 million. Noting only four full years of SGSP production are available for the Base Case.

Other CIL processing capital costs

The capital costs to resume CIL only at the completion of the SGSP period processing is expected to be minimal.

Capital costs to upgrade the Old Tailings reclaim rate are expected to be approximately US$120.61k as recently determined in SGM (Oct 2018).

Total capital costs

The total capital cost for the Base Case is US$52.93 million.

Capital cost contingency

Table 21.2 lists the capital cost contingencies for major capital items which are generally 10% unless otherwise stated. These estimates are based on reasonable estimates for minor changes of scope and material cost alterations. Total capital expenditure for the Base Case is provided in Table 21.5.

Other capital such as waste mining and TSF construction has no specific contingency and is based on site actuals from previous waste movement to date.

Table 21.5 Total capital required for the FS in US$

Item	Total/Area (US$M) Initial	Ongoing (US$M)	Contingency (US$M)	Total (US$M)	Comments
FLOTATION					Estimates by CES and Monument reviewed by Mike Kitney
320 and 330 – Flotation and Conc. Thickener Area	3.65		0.37	4.02	
BIOX®					Estimates by CES and Monument reviewed by Mike Kitney
340 – BIOX®	5.16		0.52	5.68	
350 – CCD	2.11		0.21	2.32	
360 – Neutralisation	1.39		0.14	1.53	
370 – Water Recovery Thickener	0.46		0.05	0.51	
380 – BIOX® Services	4.49		0.45	4.94	
410 – pH Adjustment	0.22		0.02	0.24	
Infrastructure BIOX® Water Supply	0.09		0.01	0.10	
BIOX Licensing		0.60		0.60	Estimates by Outotec reviewed by Mike Kitney
420 – BIOX® CIL	1.63		0.16	1.79	
820 – Reagents	2.35		0.23	2.58	
900 – Tailings	0.04		0.00	0.05	
DETOXIFICATION					Estimates by CES and Monument reviewed by Mike Kitney
901 – Detoxification	0.05		0.01	0.06	
ASTER™ PROCESS					Estimates by CES and Monument reviewed by Mike Kitney
920 – ASTER™ (Use Existing Tanks and Equipment)	0.20		0.02	0.22	
Infrastructure Water Services	1.82		0.18	2.00	
AIR SERVICES					Estimates by CES and Monument reviewed by Mike Kitney
Air Services	0.16		0.02	0.17	
Instrument Air	0.12		0.01	0.13	
Piping Rack	0.87		0.09	0.95	
DETAILED ENGINEERING					Estimates by Monument reviewed by Mike Kitney
Plant Detailed Design	2.82			2.82	
OTHER PLANT UPGRADES					Estimates by CES and Monument reviewed by Mike Kitney
Upgrade existing crushing and milling circuits	0.12			0.12	
Infrastructure Control Room + Power Distribution	0.01		0.00	0.02	
Process plant amenities	0.05			0.05	
Bulk Earthworks – Plant site filled	0.41			0.41	
Warehouse (Estimated, see Structural Worksheet)	0.25		0.03	0.28	
Workshop	0.05			0.05	

Item	Total/Area (US$M) Initial	Ongoing (US$M)	Contingency (US$M)	Total (US$M)	Comments
TRAINING					Estimates by Monument reviewed by Mike Kitney
BIOX® test facility for operator and technical team training (Pilot Plant)	0.75			0.75	
Flotation cells training	0.00			0.00	
CAPITAL SUSTAINING COSTS					Estimates by Monument reviewed by Mike Kitney
Capital Sustaining Cost		2.50		2.50	Assumed US$600k/a full year production + US$100k/a first year
Mine Rehabilitation Cost		0.60		0.60	Estimation by Monument US$100k/a
TAILING STORAGE FACILITIES					Estimation by Monument – costed against capital waste
TSF Construction		0.9		0.9	
POWER UPGRADE					Estimation by Monument reviewed by Mike Kitney
Infrastructure HV Power Supply	4.43		0.44	4.87	
COMMUNICATION					
Internet Connectivity					
CCTV					
Install CCTVs in Process Plant	0.04			0.04	Estimation by Monument
STRIPPING AND CUTBACK					
Selinsing and Buffalo Reef Pit Waste mining capital		7.66		7.66	Estimation by Monument
RIVER DIVERSION (BR)					
River Diversion BRC3		0.50		0.50	Estimation by Monument
OTHER INFRASTRUCTURE					
Upgrade Mine Access Road for Project Construction and Plant Operation	0.12			0.12	Estimation by Monument
FIRST FILL					Estimates by Monument reviewed by Mike Kitney
First Fill and Spares	1.63		0.16	1.79	
OWNER COST					
Owners Costs	0.81		0.08	0.89	Estimates by Monument reviewed by Mike Kitney
Contractors Overheads	0.23		0.02	0.26	Estimates by Monument reviewed by Mike Kitney
Engineering by CES					
Insurances and Statutory Fees (0.48% of Total Project Value; suggested by PIE)	0.26		0.03	0.28	Estimates by Monument reviewed by Mike Kitney
Old Tailings Upgrade	0.12			0.12	Estimation by Monument
TOTAL (US$)	**36.80**	**12.76**	**3.24**	**52.81**	

21.3 Operating costs

21.3.1 Accuracy of operating cost estimates

Plant process operating costs are calculated in line where possible within FS limits of accuracy of ±15% unless stated otherwise. Mining and administration operating costs are also expected to be within ±15%.

21.3.2 Basis of operating cost estimate

The MYR/US$ exchange rate again assumed to be 4.1037 based on the previous six months average based on Bank Negara Malaysia FX rates.

Open pit mine operating costs

The mining operating costs are based on the mining contract and the actual mining costs are used from production records or mining costs are applied going forward from the contract schedule of rates. These are described in more detail in Section 16.19.2

As with capital mining costs, mining operational costs were adjusted to FS FX assumptions by factor allowing for a total mining unit cost of US$1.99/t of material mined. These costs including all mining costs for material delivered to the stockpiles or ROM direct feed.

Process operating costs

The process operating costs are based on plant actuals for the oxide operation as there are several years of production history and costs. The process operating costs for sulphide were estimated by Monument in 2018.

21.3.3 Process operating costs

For the oxide only CIL processing a cost of US$8.68/t processed is assumed based on current Budget Forecasts. These cost inclusive of stockpile handling.

For the SGSP sulphide ore processing stream through the flotation and BIOX® circuits, a unit cost of US$17.26/t processed was calculated based on first principals, assuming one year at full production (950 kt/a) by Monument staff. These costs are also inclusive of stockpile handling.

Table 21.6 summarises the process unit costs.

Table 21.6 Sulphide ore processing unit costs by process and category

BY PROCESS	
Item	**Unit cost US$/t processed**
Crushing	1.82
Milling and classification	3.15
Flotation	2.30
BIOX®	2.29
CIL	2.91
Refinery	0.33
Tailings and detoxification	1.79
Water and air services	1.19
Reagents management	0.36
G&A Overheads	
Laboratory	0.41
Workshop	0.50
Procurement	0.21
Total	**17.26**

BY CATEGORY	
Item	**Unit cost US$/t processed**
Consumables	
Reagents	8.38
Wear parts	0.40
Maintenance	-
Infrastructure	-
Plant equipment	1.81
Mobile equipment	-
Light vehicles	-
Power	3.57
Labour, consultants	1.49
Equipment leasing	1.08
Outsource services	0.53
G&A	-
HSE	-
IT	-
Camp	-
Total	**17.26**

21.3.4 Administration operating costs

Administration costs were compiled by Monument from the 2019 budget.

For the FS, the FY2018/2019 forecast Administration cost assumptions were used. Administration costs of 7.037 MYR being forecast for the period. Table 21.7 shows the FY 2018/19 Forecast Administration costs. These costs are inclusive of mining as well as process related areas. Table 21.8 summarises the cost allocation by function.

Table 21.7 Listing of Administration FY2019 forecast costs

Area	MYR	US$M
Site Maintenance and Development		
Site infrastructure	171,436	0.04
Repairs and maintenance	28,000	0.01
Equipment leasing	830,855	0.20
Vehicles	300,900	0.07
Room and board	501,640	0.12
Subtotal	**1,832,831**	**0.45**
Security		
Labour	604,060	0.15
Outsource services	692,944	0.17
Security supplies	28,000	0.01
Subtotal	**1,325,004**	**0.32**
Health and Safety		
Labour	265,492	0.06
Safety supplies	153,118	0.04
Outsource services	139,600	0.03
Subtotal	**558,210**	**0.14**
Environment		
Labour	75,247	0.02
Supplies	56,359	0.01
Outsource services	198,500	0.05
Subtotal	**330,106**	**0.08**
Site Support		
Site office and general	506,058	0.12
Salaries	1,803,822	0.44
Communications	145,896	0.04
IT support	330,616	0.08
Field travel	204,300	0.05
Subtotal	**2,990,692**	**0.73**
TOTAL ADMINISTRATION COST	**7,036,843**	**1.71**

Table 21.8 Listing of Administration cost allocation to site function

Area	MYR	US$M
Admin cost to Mining	1,407,369	0.34
Admin cost to Process	4,925,790	1.20
Other – Exploration, Development, Construction	703,684	0.17
TOTAL ADMINISTRATION COST	**7,036,843**	**1.71**

21.3.5 Royalty costs

The state royalty calculation is based on a combination of fees and royalty payment based on tenement. Table 21.9 and Table 21.10 show the applicable tenement fees in US$ and royalty rates by tenement.

Table 21.9 Listing applicable tenement fees in US$

Tenement	Location	Tenement fees (PTG land rental @ MYR 1,000/ha/year, PKNP land rental, PTG processing fees, survey fees, renewal report, OMS report) US$					
		2018	2019	2020	2021	2022	2023
ML5/2000	Selinsing	28,804	-	28,804	-	28,804	-
MC1/124	Selinsing pit 7 only	64,594	76,345	48,961	76,345	48,961	76,345
ML11/2009		-	99,097	-	99,097	-	99,097
ML12/2012	BRF C3, C4, BRN	-	-	-	84,457	-	84,457
MC1/111	BRF C1	35,224	28,588	-	28,588	-	28,588
ML7/2011		-	-	-	52,398	-	52,398
Felda Land Application	BRC2 and 3 Felda						

Table 21.10 Listing applicable tenement royalty

Tenement	Location	Applicable royalty rate					
		2018	2019	2020	2021	2022	2023
ML5/2000	Selinsing	5%/10%	10%	10%	10%	10%	10%
MC1/124	Selinsing pit 7 only	10%	10%	10%	10%	10%	10%
ML11/2009		5%	10%	10%	10%	10%	10%
ML12/2012	BRF C3, C4, BRN	5%	5%	5%	5%/10%	10%	10%
MC1/111	BRF C1	5%/10%	10%	10%	10%	10%	10%
ML7/2011		5%	5%	5%	5%	10%	10%
Felda Land Application	BRC2 and 3 Felda	10%*	10%*	10%*	10%*	10%*	10%*

** Assumed rate*

22 ECONOMIC ANALYSIS

The financial evaluation was carried out and the Base Case for Mineral Reserve economic evaluation includes:

- Includes oxide and all economic material as at 1 April 2018
- Includes only indicated material with all economic parameters in line with the SGSP Base Case.

22.1 Key economic assumptions

The Key economic assumptions for all cases are as follows:

- Gold price – US$1,300/oz.
- Exchange rate – US$/MYR 4.1037.
- ISO 4217 assumptions for currency representation for US$, MYR and A$.
- Project start – 1 April 2018.
- Costs as described in Section 21.
- Mining dilution – 2% at 0.1 g/t Au.
- Mining recovery 98%.
- Process recoveries:
  - CIL Oxide – 75%
  - CIL Old Tailings – 76%
  - Sulphide Plant Flotation and BIOX® – 85%.
- Taxation based on expected Malaysian corporate tax without the ECER Tax Incentive.

The ECER (East Coast Economic Region) tax incentive is applicable to the SGSP mining and processes plant capital spend; however, Monument has not had final approval from the appropriate regulatory authority. Hence only a pre-tax assessment is present. Having previously gained approval for the Intec process, approval is not seen as a major issue.

The ECER is a gazetted tax incentive covering the states of Kelantan, Terengganu, Pahang and the district of Mersing Johor. The incentive allows for income tax exemption equivalent to 100% of a qualifying capital expenditure and to be offset 100% against statutory income derived from the qualifying activity.

22.2 FS economic evaluation

The FS evaluation includes processing of oxide material stockpiled and to be mined as well as “Old Tailings” through the CIL plant until January 2020 when the BIOX® based plant is operational and also at the completion of mine life when all remaining stockpiled leachable material is processed again through the CIL-only based plant. Figure 22.1 shows the mining plan and Figure 22.2 the Process Feed schedule for the FS case.

Figure 22.1 shows all material moved including Old Tailings and existing stockpiles. It can be seen that increased movement occurs from Quarter 6 when the SGSP mining begins. Figure 22.2 shows the processing plant operates at consistent rates of 240 kt per quarter with a dip in Quarters 3 to 7, leading to constant production during the SGSP period of Quarter 7 to 21.

Figure 22.1 FS mining schedule by rock type

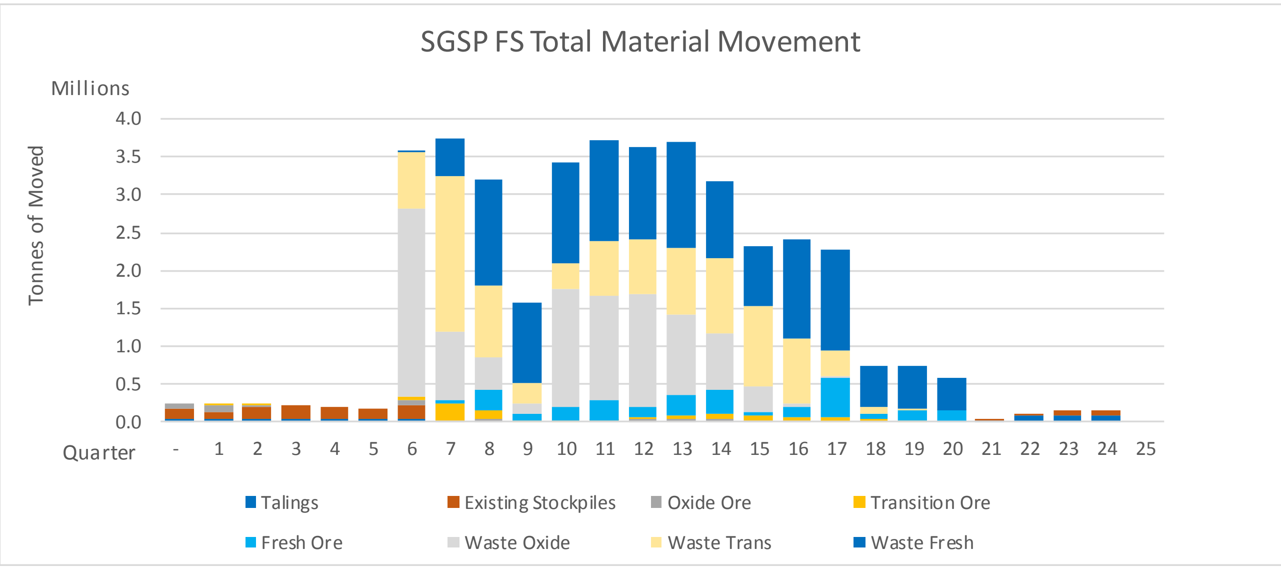


Figure 22.2 FS processing schedule by rock type

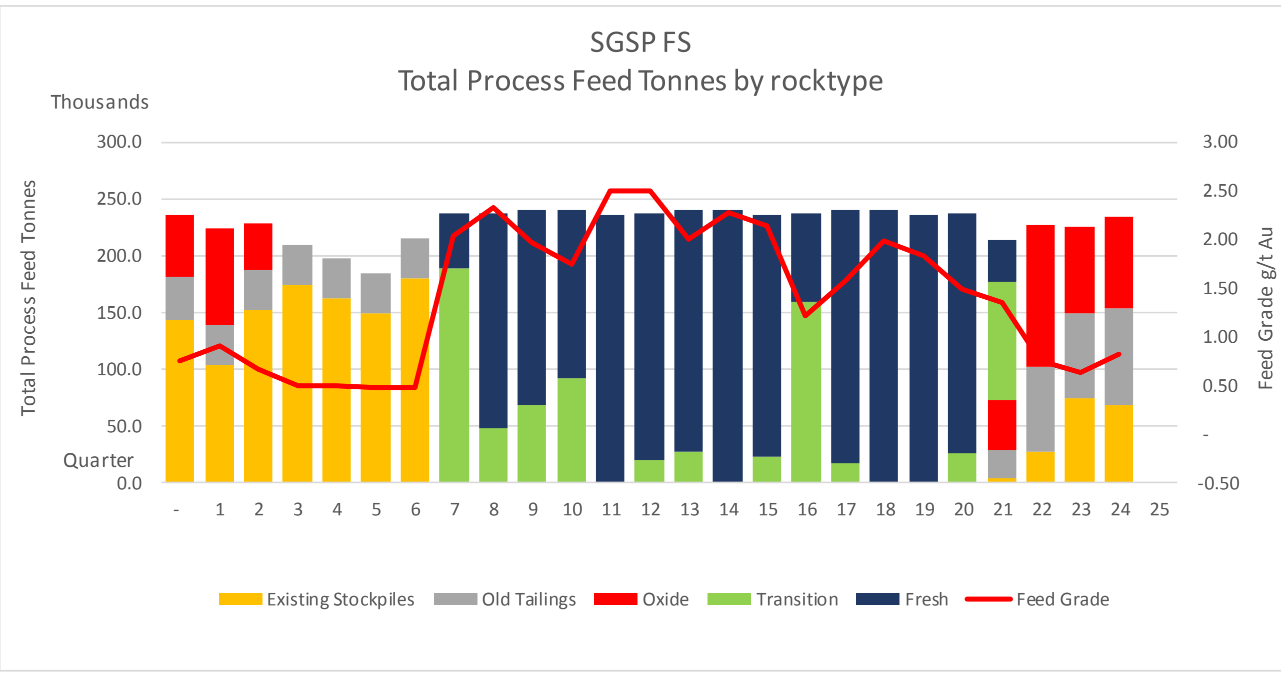


Table 22.1 shows the mining and processing physicals and Table 22.2 the economic evaluation.

The economic evaluation was based on quarterly cash flows with the yearly NPV discount rate divided by 4. The reserve reporting economic evaluation based demonstrating a positive net cash flow of US$44.55 million and NPV (8%) of US$27.56 million. Table 22.3 shows the quarterly cash flow projection and Figure 22.3 the yearly, and Figure 22.4 the quarterly cash flow graphs. These figures show that after the SGSP processing plant capital spend in Year 2, it takes two years for payback on capital occurring in Year 4.

Table 22.1 FS physicals

	FS	Units	Major assumptions and comments
PHYSICALS			
Mined			
Density waste	2.18	t/m^3	Minetech mining contract
Density ore	2.52	t/m^3	Minetech mining contract
Oxide			
Ore (MT)	483	kt	As at 1 April 2018
g/t Au	1.13	g/t Au	As at 1 April 2018
Ounces	17.53	koz	As at 1 April 2018
Sulphide			
Ore (MT)	3,436	kt	As at 1 April 2018
g/t Au	1.96	g/t Au	As at 1 April 2018
Ounces	217	koz	As at 1 April 2018
Total ore			
Ore (MT)	3,919	kt	Inventory including oxide as scheduled
g/t Au	1.86	g/t Au	Inventory including oxide as scheduled
Ounces	235	koz	Inventory including oxide as scheduled
Waste	36,003	kt	Inventory including oxide as scheduled
Total mined (t)	39,922	kt	Inventory including oxide as scheduled
Total mined (bcm)	16,375	bcm	Inventory including oxide as scheduled
SR	9		
PROCESSED			
Oxide (including stockpiles)			
Ore (Mt)	1,748	kt	All oxide processing inclusive of existing stockpiles
g/t Au	0.65	g/t Au	All oxide processing inclusive of existing stockpiles
Recovered ounces	27.44	koz	All oxide processing inclusive of existing stockpiles
Tailings			
Ore (Mt)	508	kt	Reserve estimation at March 2018
g/t Au	0.71	g/t Au	Reserve estimation at March 2018
Recovered ounces	8.77	koz	Reserve estimation at March 2018
Sulphide ore			
Ore (Mt)	3,482	kt	Reserve estimation at March 2018
g/t Au	1.96	g/t Au	Reserve estimation at March 2018
Recovered ounces	186.45	koz	Reserve estimation at March 2018
Total processed			
Ore	5,739	kt	All processing including stockpiles and tailings
g/t Au	1.45	g/t Au	All processing including stockpiles and tailings
Recovered ounces	223	koz	All processing including stockpiles and tailings
Project life	**6**	**years**	

Table 22.2 FS economic evaluation

	FS	Units
FINANCIALS		
Costs		
Capital cost		
Plant construction	34.87	US$ M
Other initial capital areas	4.91	US$ M
Mining capital	7.66	US$ M
Other capital	5.49	US$ M
Total capital	52.93	US$ M
Operating costs		
Mining operational	71.98	US$ M
Processing operational	78.03	US$ M
Site administration	10.31	US$ M
Royalty	31.22	US$ M
Total operating	191.54	US$ M
Corporate tax	1.28	US$ M
Unit costs		
Mining	1.99	US$/t mined
Processing	13.60	US$/t processed
Cash cost/ounce	863.67	US$/oz
Revenue	**290.31**	**US$ M**
Operating expenditure	**191.54**	**US$ M**
Cash flow from operations (EBIT)	**98.77**	**US$ M**
Corporate tax	**11.73**	**US$ M**
Capital investment tax credit	**10.44**	**US$ M**
Operating cash flow after tax	**97.48**	**US$ M**
Capital expenditure	**52.93**	**US$ M**
Cash flow, net	**44.55**	**US$ M**
Discount rate	8	%
NPV	**27.56**	**US$ M**
IRR	49	%
Payback	2.50	years
ROIC	1.11	ratio

Table 22.3 FS quarterly cash flow projection

CASH FLOW	-	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	Total
	USD'000	USD'000	USD'000	USD'000	USD'000	USD'000	USD'000	USD'000	USD'000	USD'000	USD'000	USD'000	USD'000	USD'000	USD'000	USD'000	USD'000	USD'000	USD'000	USD'000	USD'000	USD'000	USD'000	USD'000	USD'000)'000	USD'000
REVENUE																										
Gold Sale	5,562	6,368	4,832	3,288	3,108	2,807	3,219	17,247	19,582	16,742	14,834	20,893	21,084	17,117	19,353	17,864	10,248	13,492	16,920	15,298	12,533	10,008	5,412	4,451	6,045	288,306
Plant Salvage in Mine Closure	-																								2,000	2,000
	5,562	***6,368***	***4,832***	***3,288***	***3,108***	***2,807***	***3,219***	***17,247***	***19,582***	***16,742***	***14,834***	***20,893***	***21,084***	***17,117***	***19,353***	***17,864***	***10,248***	***13,492***	***16,920***	***15,298***	***12,533***	***10,008***	***5,412***	***4,451***	***8,045***	***290,306***
OPERATING COSTS																										
Mining	984	1,113	295	-	-	-	5,009	7,477	6,602	3,503	3,631	5,702	6,000	7,318	6,316	4,597	4,798	4,551	1,455	1,474	1,151	-	-	-	-	71,978
Processing	1,928	1,836	1,871	1,705	1,598	1,492	1,752	4,110	4,110	4,155	4,155	4,064	4,110	4,155	4,155	4,064	4,110	4,155	4,155	4,064	4,110	2,985	1,726	1,720	1,752	78,033
Site Administration	425	404	411	376	354	332	386	428	428	432	432	423	428	432	432	423	428	432	432	423	428	384	408	406	420	10,305
	3,337	3,353	2,577	2,081	1,952	1,824	7,148	12,014	11,139	8,090	8,218	10,189	10,537	11,905	10,903	9,084	9,335	9,138	6,042	5,961	5,688	3,369	2,133	2,126	2,172	160,317
Royalty - Gov't	727	637	483	329	311	281	525	1,527	1,958	1,674	1,623	2,089	2,108	1,712	2,275	1,786	1,025	1,349	1,882	1,530	1,253	1,001	881	445	604	30,015
Royalty - Biox	-	-	-	-	-	-	-	86	98	84	74	104	105	86	97	89	51	67	85	76	63	41	-	-	-	1,207
	4,064	***3,990***	***3,060***	***2,410***	***2,263***	***2,105***	***7,673***	***13,628***	***13,195***	***9,848***	***9,915***	***12,383***	***12,751***	***13,702***	***13,274***	***10,960***	***10,411***	***10,554***	***8,008***	***7,567***	***7,004***	***4,411***	***3,014***	***2,571***	***2,777***	***191,539***
Corporate tax	203	270	270	270	270	-	-	-	-	758	758	758	758	469	469	469	469	987	987	987	987	396	396	396	396	11,728
Less: Capital investment tax credit	-	-	-	-	-	-	-	-	-	(758)	(758)	(758)	(758)	(469)	(469)	(469)	(469)	(987)	(987)	(987)	(987)	(396)	(396)	(396)	(396)	(10,444)
	203	270	270	270	270	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	***1,284***
Cash Generated from Production after tax	1,296	2,108	1,501	608	575	703	(4,454)	3,619	6,387	6,894	4,919	8,509	8,334	3,414	6,079	6,904	(163)	2,937	8,912	7,730	5,529	5,597	2,398	1,880	5,268	97,484
CAPEX																										
Plant construction		231	231	231	231	8,486	8,486	8,486	8,486	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	34,868
Power upgrade		-	-	-	-	609	609	609	609	609	609	609	609	-	-	-	-	-	-	-	-	-	-	-	-	4,871
Communications		-	-	-	-	-	-	-	-	9	9	9	9	-	-	-	-	-	-	-	-	-	-	-	-	35
Trainings		-	-	-	-	188	188	188	188	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	752
Mine development		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Access Road		30	30	30	30	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	120
Waste Capitalized	-	-	-	-	-	-	1,401	-	-	-	3,379	1,857	1,028	-	-	-	-	-	-	-	-	-	-	-	-	7,665
Waste assigned to TSF cost	-	(56)	(56)	(56)	(56)	-	-	-	-	(56)	(56)	(56)	(56)	(56)	(56)	(56)	(56)	(56)	(56)	(56)	(56)	-	-	-	-	(900)
River diversion (BR)	-	-	-	-	-	-	-	-	-	125	125	125	125	-	-	-	-	-	-	-	-	-	-	-	-	500
TSF- Staged lifts		113	113	113	113	-	-	-	-	113	113	113	113	113	113	113	113	113	113	113	113	-	-	-	-	1,800
Mine Rehabilitation		25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	600
Old Tailings Upgrade		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	30	30	30	30	-	-	-	-	120
Capital Sustaining costs		25	25	25	25	150	150	150	150	150	150	150	150	150	150	150	150	150	150	150	150	-	-	-	-	2,500
Cash invested in development	-	*368*	*368*	*368*	*368*	*9,458*	*10,859*	*9,458*	*9,458*	*974*	*4,352*	*2,831*	*2,002*	*231*	*231*	*231*	*231*	*261*	*261*	*261*	*261*	*25*	*25*	*25*	*25*	***52,931***
Change in cash during the year	**1,296**	**1,740**	**1,134**	**241**	**207**	**(8,755)**	**(15,312)**	**(5,838)**	**(3,070)**	**5,920**	**567**	**5,678**	**6,332**	**3,183**	**5,848**	**6,673**	**(394)**	**2,676**	**8,650**	**7,469**	**5,268**	**5,572**	**2,373**	**1,855**	**5,243**	**44,553**
Accumulative Cash flow	**1,296**	**3,036**	**4,170**	**4,411**	**4,618**	**(4,137)**	**(19,450)**	**(25,288)**	**(28,358)**	**(22,438)**	**(21,872)**	**(16,194)**	**(9,861)**	**(6,678)**	**(831)**	**5,842**	**5,448**	**8,124**	**16,774**	**24,243**	**29,511**	**35,083**	**37,456**	**39,310**	**44,553**	
Rate of discount	***8%***																									
NPV	***$27,557***																									
IRR	***49%***																									
ROC	1.14																									
Pay back period	**2.50**	**yrs.**																								

Figure 22.3 FS yearly cash flow

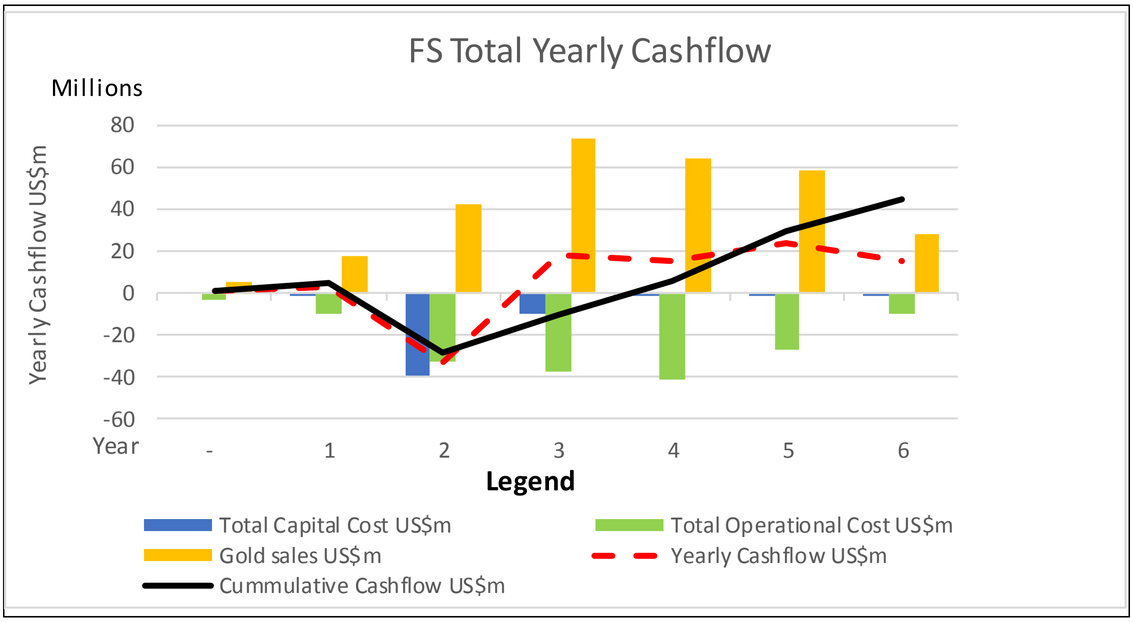
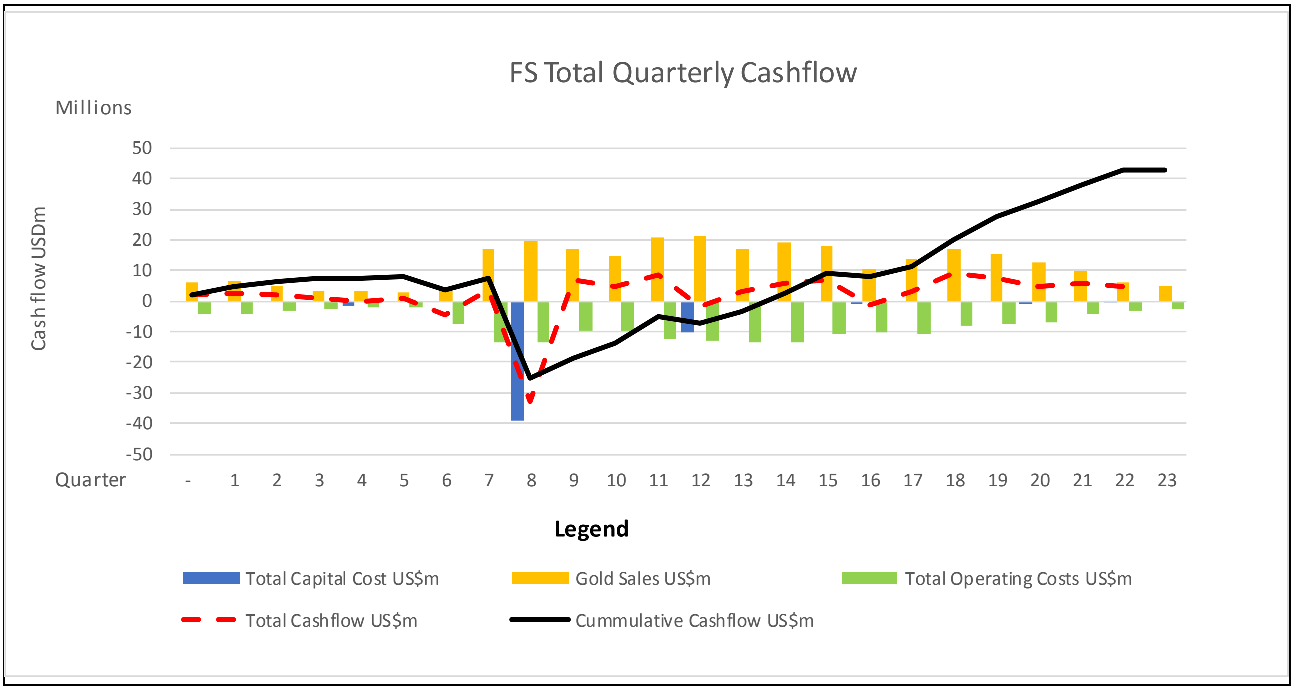


Figure 22.4 FS quarterly cash flow



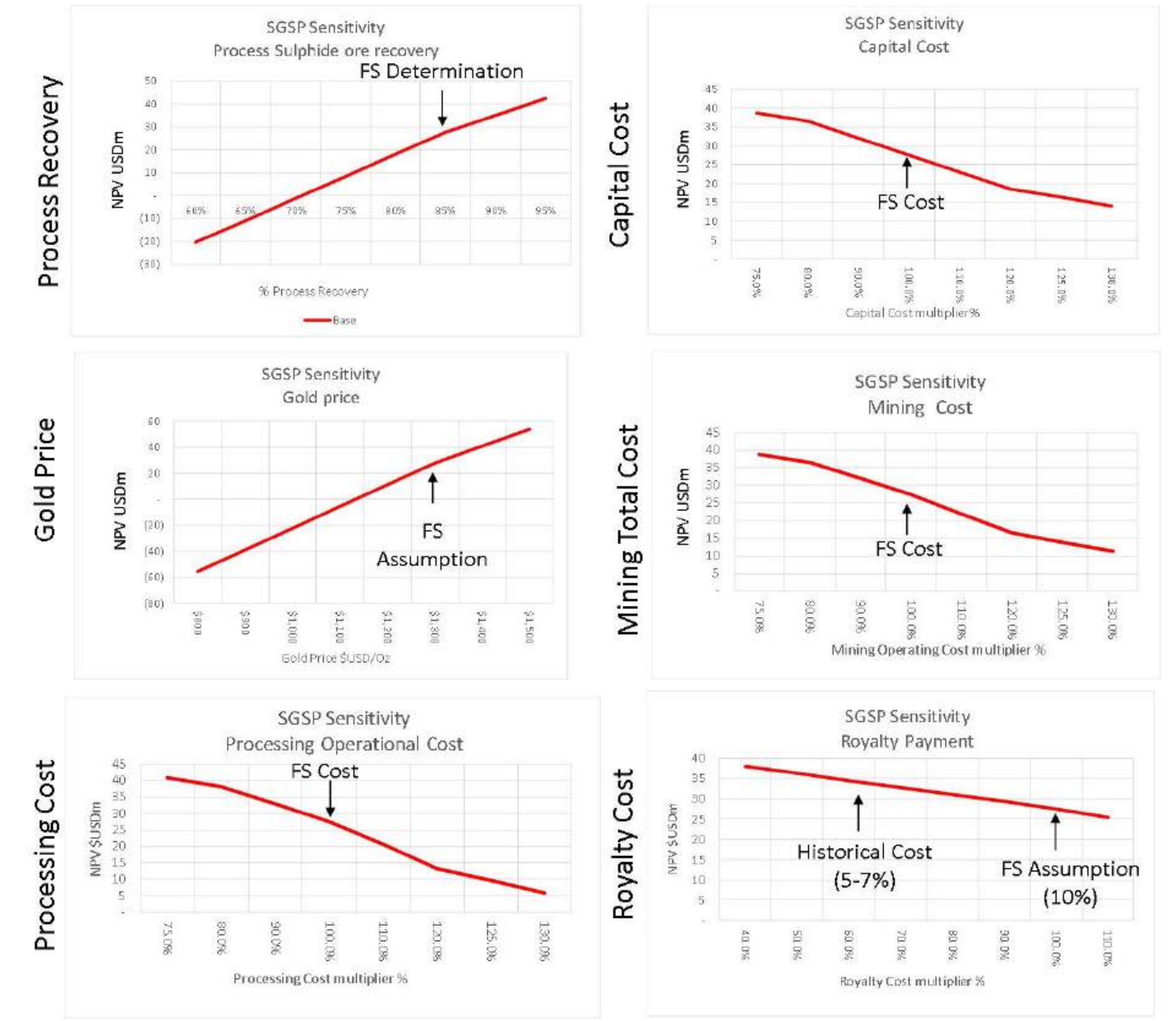
22.3 Sensitivity

For the FS, sensitivities were conducted for:

- Gold price
- Process recovery
- Mining cost
- Processing cost
- Capital cost
- Royalty cost.

Figure 22.5 provides the sensitivities for the FS.

Figure 22.5 FS sensitivities for process recovery, gold price as well as processing, mining, capital and royalty cost



The sensitivity analysis revealing the following:

- Process recovery:
  - The project is sensitive to sulphide process recovery with breakeven NPV (8%) at 70% recovery.
- Gold price:
  - The project is sensitive to gold price with breakeven gold price at US$1,100/oz.
- Capital cost:
  - The project is less sensitive to capital cost with 60% increase possible before a negative NPV is obtained.
- Mining cost:
  - The project is less sensitive to mining cost with 50% increase possible before a negative NPV is obtained.
- Process operating cost:

  - The project is less sensitive to process operating cost with 35% increase possible before a negative NPV is obtained.
- Royalty payment:
  - The project is less sensitive to royalty payment with over 200% increase possible before a negative NPV is obtained
  - A return to previous royalty payment expectation of 5–7%; however, increases the NPV (8%) to US$35 million.

22.4 Economic evaluation conclusion

The evaluation showed the required initial capital of US$39.77 million can be repaid within three years with the project generating an operating cash flow of over US$98.77 million. The actual payback however will not be until Year 4, since the capital injection required for the new process plant does not occur until Year 2.

Total cash cost per ounce of gold is at US$863/oz, giving a margin of US$436/oz to an assumed gold price of US$1,300/oz. This result is largely driven by lower than world industry mining costs in Malaysia.

The FS NPV (8%) of US$27.56 million demonstrates a positive cash flow for the Mineral Reserves included in this study. The inclusion of the ECER tax incentive is also a key attraction for this project, with tax only payable for the pre-SGSP period.

23 ADJACENT PROPERTIES

There is no adjacent properties information to disclose.

24 OTHER RELEVANT DATA AND INFORMATION

Other relevant information to be described in this chapter relates largely to opportunities that exist at the Selinsing Gold Mine operation not included in the FS evaluation and open pit ore reserve materials.

The project opportunities are identified as:

- Inferred Mineral Resource inclusion in open pit mining inventories:
  - Opportunity Case One
  - Opportunity Case Two.
- Underground Mining Potential.
- Early mining of Selinsing Pit 5 CIL leachable ore.
- Accelerated processing of Old Tailings material.

24.1 Inferred potential

For the Inferred Resource potential Opportunity Cases One and Two, the best potential exists at Buffalo Reef. Conversion of Inferred material is considered most likely in this area.

Table 24.1 shows the determined physicals for the Inferred inclusive opportunity Intermediate Case and Optimistic Case.

24.1.1 Opportunity Case One

This case described in Section 16 (Mining Methods) uses the same Reserve open pit design in the FS but includes the Inferred Mineral Resource inside these designs in the inventory. Table 24.1 shows and increase of some 20 koz contained and 16 koz recovered are possible with its inclusion for no extra mining cost.

This potential existing largely at Buffalo Reef where in cases the Inferred Mineral Resource is surrounded by Indicated resource within the reserve pit designs.

Conversion of the material to Indicated is considered highly possible and presents as an ideal LOM case.

24.1.2 Opportunity Case Two

This case also described in Section 16 considers all Inferred open pit potential and Table 24.1 shows and increase of some 130 koz contained and 119 koz recovered from the FS Case.

Section 16 mentioned that in the Selinsing area, the Pit 7 area north of Pit 5/6 addition increased the Gold Contained inventory by 57 koz from the FS to the Opportunity Case Two cases. Issues mentioned with Pit 7 included:

- High waste strip.
- Large waste volume.
- River diversions.
- Interference with “Old Tailings” reclaim:
  - Pit cutback will include “Old Tailings” areas (much of this material scheduled for post-SGSP processing)
  - Relocation of tailings processing facilities required.
- There is uncertainty in resource modelling in pit 7 only.

However, a 72 koz increase occurs in the Buffalo Reef North and South Centre area. This increase represents the bulk of the potential and a much higher certainty of mineral resource conversion exists in this area.

Hence the potential for the Opportunity Case Two is at Buffalo Reef.

Thus, a potential LOM schedule inclusive of Inferred potential needs to be focused on Buffalo Reef with largely Reserve-only material at Selinsing.

Table 24.1 Mining and processing physicals for FS and Opportunity Cases One and Two

Case	Units	FS	Opportunity Case One	Opportunity Case Two
PHYSICALS				
Mined				
Density waste	t/m^3	2.18	2.18	2.18
Density ore	t/m^3	2.52	2.52	2.52
Oxide				
Ore (Mt)	kt	483	589	686
g/t Au	g/t Au	1.13	1.17	1.14
Ounces	koz	17.53	22.15	25.11
Sulphide				
Ore (Mt)	kt	3,436	3,722	4,990
g/t Au	g/t Au	1.96	1.95	2.12
Ounces	koz	217	232.97	339.41
Total ore				
Ore (Mt)	kt	3,919	4,312	5,676
g/t Au	g/t Au	1.86	1.84	2.00
Ounces	koz	235	255	365
Waste	kt	36,003	36,182	61,102
Total mined (t)	kt	39,922	40,493	66,092
Total mined (bcm)	bcm	16,375	16,375	27,814
SR		9	8	11
Processed				
Oxide (including stockpiles)				
Ore (Mt)	kt	1,748	1,855	1,952
g/t Au	g/t Au	0.65	0.68	0.89
Recovered ounces	koz	27.44	31	41.83
Tailings				
Ore (Mt)	kt	508	508	508
g/t Au	g/t Au	0.71	0.71	0.71
Recovered ounces	koz	8.77	8.77	8.77
Sulphide Ore				
Ore (Mt)	kt	3,482	3,776	5,036
g/t Au	g/t Au	1.96	1.94	2.11
Recovered ounces	koz	186.45	199.97	290.94
Total processed				
Ore	kt	5,739	6,140	7,496
g/t Au	g/t Au	1.45	1.46	1.70
Recovered ounces	koz	223	239	342
Project life	years	6	6	7

24.2 Underground desktop study

An underground mining potential desktop study completed in house as part of the FS.

This study concludes that underground mining is not possible based on current Mineral Resources but may be possible in the Selinsing area if:

- A lower cost mining contractor can be found with lower than the worst case Australian based mining cost assumptions used:
  - Reduction by 25% at least required in mining cost.
- The potential resource above 3 g/t Au is increased with exploration:
  - At least 200% increase required
  - A higher cut-off grade could improve cash flow if enough inventory material exists.

Potential may exist at BRF, but it was not evaluated at this stage due to:

- Identified inventory very spread-out requiring extensive development access.
- Identified inventory requiring a minimum mining width of 1 m.
- In both areas:
  - drill coverage is limited
  - geological interpretation has targeted bulk mining for open pit mining and has not been constrained for narrower more selective underground mining.

With the open pit mining operation repaying the process plant capital investment, any increases in available inventory would be considered incremental in value. However, for underground mining, the mining capital investment and operating costs would be higher.

The largest long-term potential for site is expansion of the known Mineral resource at depth for underground mining. The known Mineral resource is very much open at depth and along strike at depth. Increases above 200% (>3 g/t Au) in available underground mining inventory is considered highly probable particularly in the Selinsing area.

24.3 Mining of Selinsing Pit 5

Historical metallurgical testwork on areas previously mined in the Selinsing Pit 5 area have indicated much of this ore is treatable by the current CIL plant. Unfortunately, this data is largely restricted to transition material, so in 2018 a geometallurgical drillhole program was instigated to test fresh sulphide ore in the Pit 5 area for CIL plant leachability.

The testwork was limited in sample size but indicated much of the remaining Pit 5 ore could be treated in the current plant. This mining plan is to:

- Mine Pit 5:
  - Following POB (2018) guidelines for the west wall cutback and pit design.
- Treat identified leachable ore from grade control leachability testing.
- Stockpile unleachable material for the SGSP.

Figure 24.1 shows the FS process schedule and the location of Pit 5 in the process schedule. If Pit 5 were brought forward, processing capacity is available from period 3 but the processing of low-grade existing stockpiles could also be reduced to allow for Pit 5 processing.

Reduction of sulphide-only treatable Pit 5 material would also allow a ramp up of SGSP production rather than the assumed full production rate upon plant commissioning.

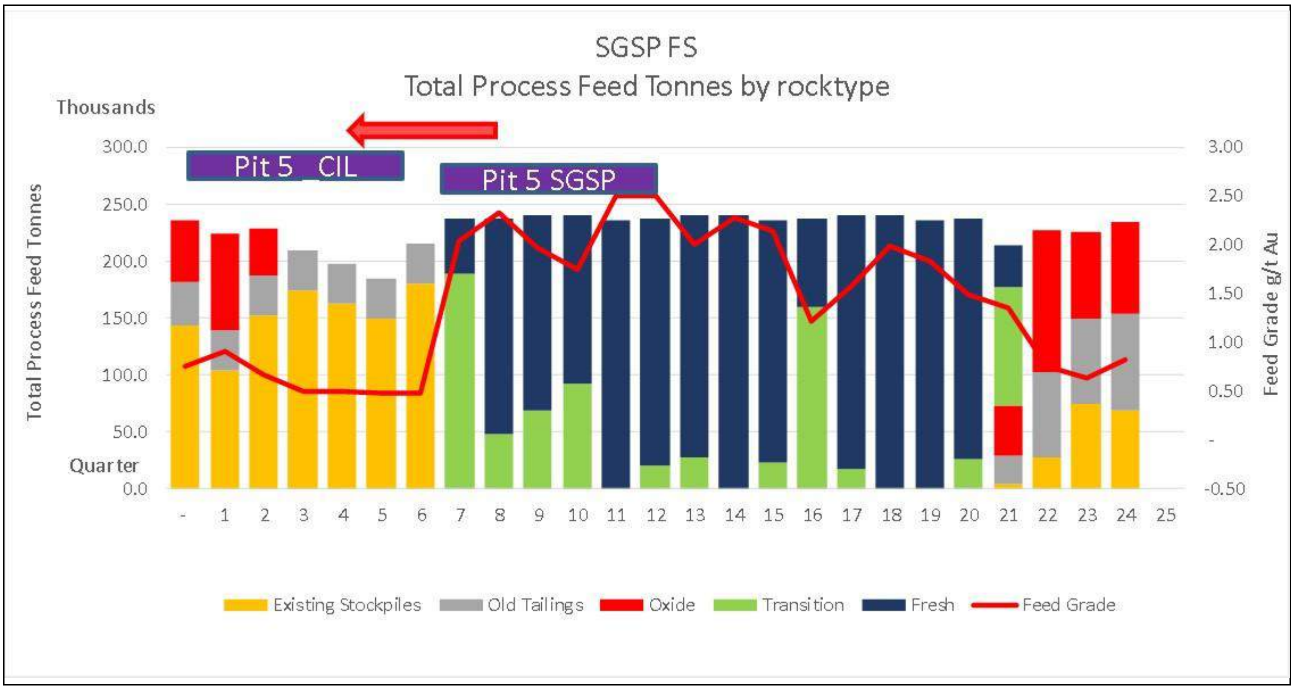
24.4 Accelerated Old Tailings production

As mentioned in Section 15 (Mineral Reserve Estimate), the Old Tailings Reserve could be reclaimed faster with an upgraded thickener and reclaim plant. For the FS, this was planned in the post-SGSP production timeframe.

This could however be brought forward to the pre-SGSP period. Figure 24.1 shows the capability for increased production from Quarter 3 to 7. A capital cost of approximately US$120,600 is required for this project with options are currently being investigated by site personnel (refer SGM, Oct 2018).

A similar cost is expected for an upgrade post the SGSP period.

Figure 24.1 FS process schedule with Pit 5 timing



24.5 Production April – September 2018

The SGM production following Mineral Reserve and Mineral Resource estimation from March 31st to September 30th 2018 is a process feed grade of 473 kt at a grade of 0.94 g/t for 9.9 kozs recovered.

25 INTERPRETATION AND CONCLUSIONS

Economic viability has been proven for the FS Reserve-only inclusive of all CIL treatable material with NPV (8%) US$27.56 million. Sensitivities showing the Project can withstand well over 200% increase in costs but is sensitive to gold price (US$1,100/Oz breakeven) and process recovery (68% breakeven). A standalone SGSP (sulphide-only) Base Case (Reserve-only) has cashflow and NPV (8%) approximately half the FS base case at NPV US$12.2 million but is capable of repaying initial capital costs within four years.

Detailed study work has been carried in many areas as part of the FS. With the geological and mine planning detail largely unchanged from the Snowden (Dec 2016) PFS, most of the detailed work has been associated with metallurgical study and evaluation as well as plant design and construction. There is work demonstrating constructability and feasibility of a plant capable of processing the sulphide ores at Selinsing using a flotation and BIOX® additions to the existing plant.

25.1 Geology and Mineral Resources

The area has been explored over a number of years with both RC drilling and diamond drilling, plus trench sampling conducted across the mineralised veins. The author is satisfied that the drill sample database and geological interpretations are sufficient to enable the estimation of Mineral Resources. Accepted estimation methods have been used to generate a three-dimensional (3D) block model of gold values.

The Mineral Resource estimate is classified with respect to CIM guidelines as Measured, Indicated or Inferred according to the geological confidence and sample spacings that currently define the deposit. Proven and Probable Mineral Reserves are also estimated.

Monument should be able to increase the size and confidence of the Selinsing and Buffalo Reef resource through additional extensional and infill drilling.

25.2 Mining and Mineral Reserves

The mine design and planning work carried out in the FS largely verified the Snowden (Dec 2016) PFS work except in the Selinsing Pit area. In the Selinsing pit area, with ongoing geotechnical issues, a revised more conservative mine design was determined based on the 2018 geodetical study finding in POB (2018).

Similar reserves were reported in this FS to Snowden (Dec 2016) PFS but with the addition of Old Tailings. A key for the project continuance is overcoming the limited supply of suitable feed material for the CIL-only based plant until the SGSP is approved and project execution of the plant construction occurs. The updated reserve inclusive of Old Tailings is thus a welcome inclusion in this study.

Opportunity studies were carried out for Inferred Resource inclusion as well as underground potential.

Inferred resource inclusion in both reserve designs (Opportunity Case One) and full potential (Opportunity Case Two) show a significant increase in inventory is possible in the Buffalo Reef area, but concerns are raised about Inferred inclusion in the Selinsing Pit area with geological confidence and significant pit size increase with the Pit 7 Inferred potential.

The Underground Desktop study showing underground mining potential may be possible in the Selinsing area but will depend on:

- Increases in available mining inventory at suitable grade
- Availability and cost of suitable mining contractors.

25.3 Processing

The detailed metallurgical testwork and process design study work has demonstrated BIOX® and flotation plant additions to the existing plant are feasible for the treatment of the sulphide ores at Selinsing.

The study work includes enough detail to proceed to final plant design stages.

25.4 Environmental and community considerations

No environmental or community issues are apparent to adversely affect the Project.

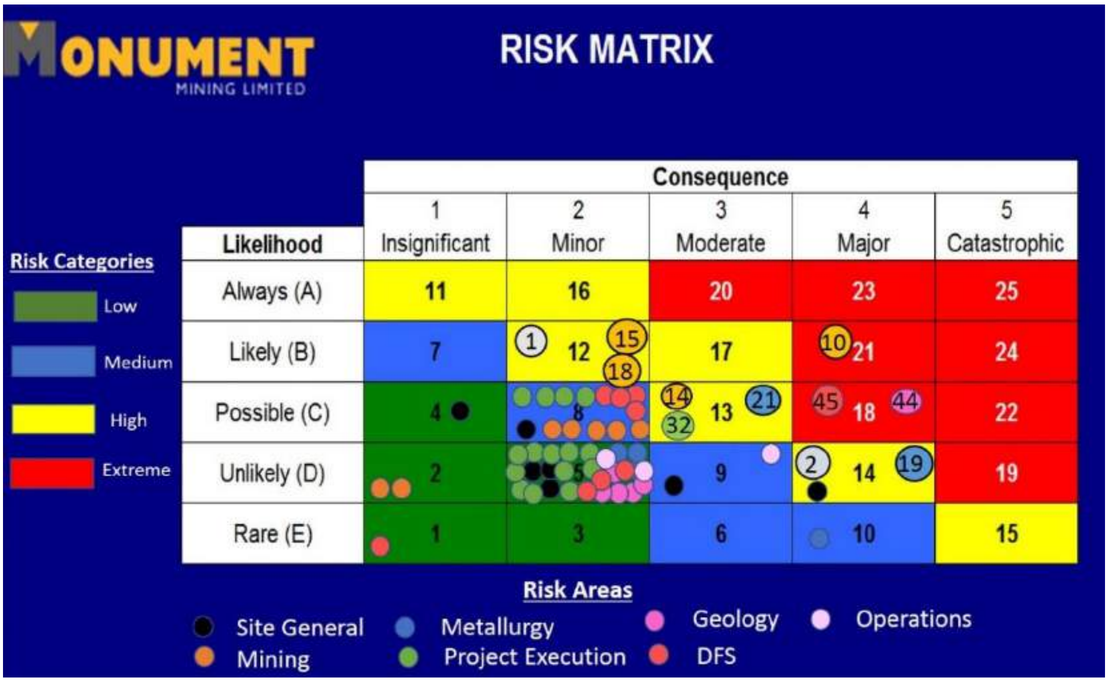
25.5 Economics

The economic evaluation presented in the FS has demonstrated economic viability with NPV (8%) of US$27.56 million. Sensitivities show revenue influenced by the gold price and process recovery are most sensitive, however the project is not so sensitive to cost increases.

25.6 Risks

Risks for the project were well documented in the detailed SGSP Risk Assessment conducted in 2017 and documented in SGSP (May 2017) entitled "Selinsing Risk Assessment Summary and Plan". This formal risk assessment was carried out by members of the Monument site and corporate team at that time using a risk assessment matrix a shown in Figure 25.1; the results also are superimposed on the same figure. This figure shows the majority of risks assessed were in the Medium Risk to Low Risk category, but several were in the High Risk and Extreme Risk category requiring mitigation plans largely based on FS work.

Figure 25.1 SGSP risk assessment matrix and results SGSP (May 2017)



The High Risk and Extreme Risk cases identified in the risk assessment to be largely controlled with dedicated FS work together with (highlighted) FS actions are as follows:

- Risk 2 – "Delay in securing adequate Project financing":
  - FS requires appropriate standard and level of detail to support Project funding
  - This FS is reported and carried out to NI 43-101 standards.

- Risk 10 – "Instability of Selinsing and Buffalo Reef Pit Walls. With no Selinsing pit extended cutback":
  - FS needs to include detailed geotechnical analysis and ongoing geotechnical work program
  - Possible need for fault tree analysis
  - FS to include discussion on other scenarios such as an underground operation in the future
  - Studies by POB (2018) and the Underground Mining Potential Desktop Study.
- Risk 14 – "Mining contract expires June 2018; possible cost escalation":
  - FS to include detailed sensitivity analysis and financial risk assessment associated with mining cost increase
  - Sensitivities on mining cost (refer Section 22 showing significant cost increases (well over 200%) are required to give negative NPV (8%).
- Risk 15 – "Mining at increased depth":
  - FS to include detailed mining operational plan listing key day to day activities and safety management
  - Incremental costs for Drill and Blast in Fresh and Haulage included in the FS mining costs.
- Risk 18 – "TSF spring water issues downstream of main embankment":
  - Ongoing geotechnical review
  - Remedial action plan detailed to include embankment widening if required using available mine waste rock
  - Requires full description in FS
  - Refer Section 20 and studies and ongoing work by SRK.
- Risk 19 – "Metallurgy scale up based on flotation testwork":
  - FS to carry out sensitivity on variability impact on cash flow
  - Refer section 13 for flotation and BIOX® variability testwork.
- Risk 21 – "Incomplete flotation testwork on fully representative variability samples":
  - The following need to be described in the FS:
  - Geological orebody knowledge
  - Geometallurgical project
  - Review by consultant OMC.
  - Refer section 13 and OMC (Feb 2018) and OMC (Nov 2018).
- Risk 47 – "No testwork for BIOX® on Selinsing Pit ore which has more complex geology than BR":
  - Detailed testwork and processing strategy to be documented including:
  - Full discussion in FS if no testwork results available during FS.
  - FS to include a future testwork strategy if results are not available.
  - Refer Outotec (June 2018) which included BIOX® on Selinsing ores.
- Risk 48 – "The requirement for additional flotation pilot plant and testwork to provide a new flotation concentrate feed, more representative of the ore that will be available from the optimised mine plan and revised flotation mass pull, to Outotec for further testwork and confirm feedstock quality parameters":
  - Carry out sensitivity assessment in FS.
  - Define in FS the testwork to be carried out prior to commencement of basic engineering for the plant.

  - BIOX® Pilot Plant is proposed as part of the plant construction phase. Three stages of BIOX® testwork, sensitivities carried in section 13 on process sensitivity.

Since SGSP (May 2017), a separate risk assessment was carried by Monument site personnel scoped on Plant construction SGSP (Aug 2018). The highest ranked risks are similar to the High and Extreme in SGSP (May 2017) are as follows:

- Procurement issues:
  - Weak contract write-up:
  - Control:
    - SGMM to hire personnel that are well versed in contract management.
  - Bottle neck at fabricator to delivery large quantities of steel tanks required:
  - Control:
    - Award tankage steelwork to at least three different contractors via the main contractor to meet deadlines.
  - Escalation of cost due to forex movement:
  - Control:
    - Monument to possibly hedge the exchange rate in advance to minimise the fluctuation.
  - Delay in equipment selection:
  - Control:
    - Prioritise on the long-lead item.
- Construction risks:
  - Concrete quality below specifications:
  - Control:
    - Carry out QAQC on concrete deliveries.
  - Insufficient machinery to move and place bulk earthwork:
  - Control:
    - Contractor to ensure enough equipment availability and to provide advance notice for mobilisation.
- Start-up and operations:
  - BIOX® feed (i.e. flotation concentrate does not meet the specifications):
  - Controls:
    - Extensive testwork on different ore types to forecast expected reagent dosage and response (completed in FS)
    - Blending of ore types
    - Ensure that plant parameters and reagent dosages follow recommended guidelines.
  - Plant operation sabotage by internal and/or external parties:
  - Controls:
    - Installation of new security fences around the plant parameter
    - Installation of CCTV to monitor movement of personnel in and out of plant area.
    - Restrict access to non-production personnel.

The opportunity studies for the inclusion of Inferred Mineral Resources have shown significant increases in mine life with the additional inventory, even with the exclusion of the uncertain Selinsing Inferred MineralResource.

26 RECOMMENDATIONS

26.1 Mining and Mineral Reserves

With the high potential to increase available reserves of sulphide ore and the direct positive impact on cash flow and NPV a high priority is set on:

- Converting the Opportunity Case One Case mineral resource to Indicated at BRF
- Converting the Opportunity Case Two mineral resource to Indicated at BRF.

There is significant exploration potential at the Selinsing Gold Mine Project with the sulphide orebodies open at all directions allowing for potential in both open pit and underground mining. Todate Monument Mining Ltd has demonstrated historical evidence of mineral resource to reserve conversion.

Construction and operation of the Sulphide processing plant may alsoallow treatment of ores form other sources in Malaysia and the Asian region. This depending on treament compatibility and economic distance and other factors for ore and concentrate transport.

Further work optimising the mining schedules in the BRC3 and BRC4 area is also required possibly using Deswik or Datamine software-based scheduling. This should however wait until the mineral resource is updated.

The resource block models in both Selinsing and BRF should be remodelled with more accurate knowledge of geological based model constraints, again ideally when mineral resource conversion occurs.

The geotechnical program for BRF as proposed by POB (2018) must continue to refine open pit design parameters in the BRF SC area.

Other opportunities mentioned in Section 24 must continue to be fully evaluated an acted on as a priority since time is of the essence with ever diminishing stocks of remaining CIL only treatable material as oxide . These opportunities including:

- Advance mining of Selinsing Pit 5
- Accelerated Old Tailings Production.

The underground mining potential evaluation to next phase requires the following:

- Increased available inventory
- Sourcing the ideal mining contractor.

Both these issues relating the underground mining can be progressed with future drilling programs as well as sourcing Asian based contractors and obtaining their engagement in early stages of evaluation.

26.2 Environmental and community considerations

Stakeholder interest in the SGSP must be maintained as an ongoing priority in order to minimise any bottlenecks or holdups in approvals.

26.3 Resource exploration programs

The current strategy for Monument exploration further programs is to add resources for oxide, identifying short-term oxide potential targets, and also add resources for sulphide resources, testing deeper drill targets, open pit and underground.

The program comprises essentially drilling activities but also soil/surface sampling and mapping.

The main programs are show in Figure 26.1 and comprise:

- Oxide extensional drilling at BRSCF and BRN areas
- Sulphide extension drilling program for BRSCF area
- Sulphide confirmation drilling program at North of Selinsing Pit V
- Sulphide infill UG drilling program beneath Selinsing Pit IV.

The oxide Buffalo Reef extension areas just next to the LOM design pits comprise 600 m for 10 holes estimated for the southeast and north extensions of BRN area, and 1,300 m for 21 holes mostly at the northwest of the BRSCF deposits. All holes will have azimuths 270° dipping 50–60° to west, with depths ranging from 35 m to 100 m, average depth approximately 62 m.

The sulphide down dip and strike/down plunge drilling HG extension program for Buffalo Reef area comprises 3,700 m for 18 holes estimated at BRSCF east side, dipping 50–60° to west, with depths ranging from 60 m to 350 m, average depth approximately 207 m. The objective of this drilling is to check deeper extensions of the identified quartz-stibnite high-grade mineralisation and quartz-ankerite high grade mineralisation, aiming to increase the total resource of sulphide material.

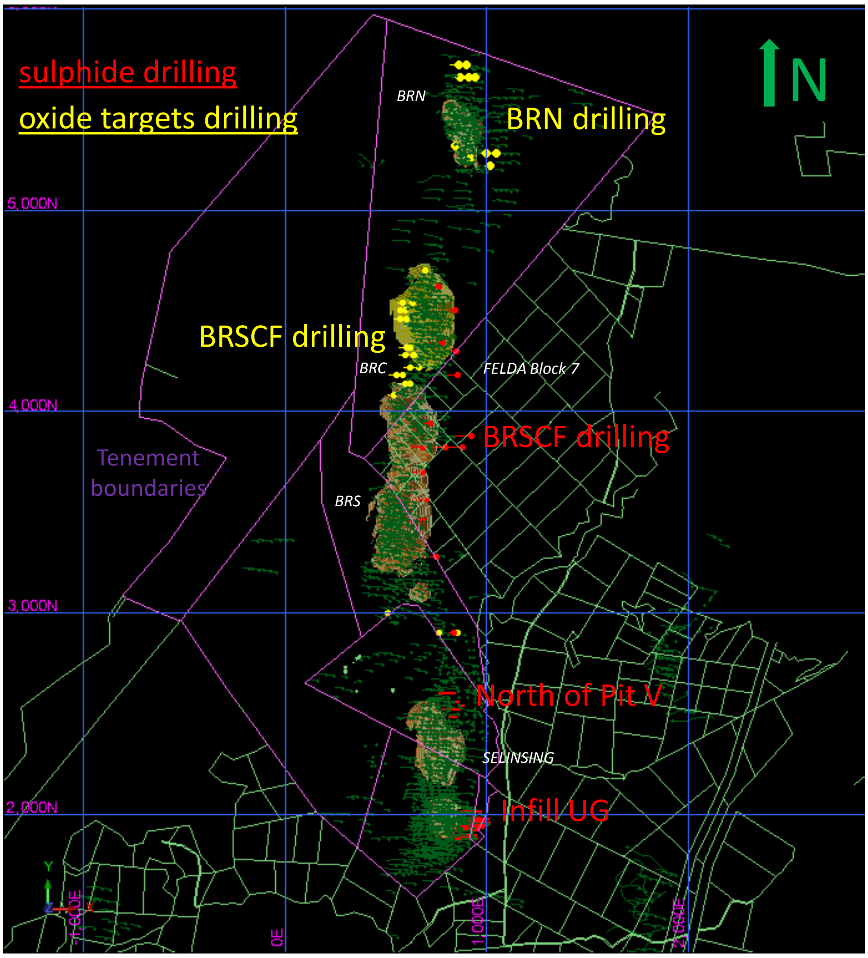
The sulphide confirmation drilling program at North Selinsing Pit V aims to test and get higher accuracy for a high-grade gold area extension. Currently, there is a poor drilling coverage in an area of modelled 55 koz Au.

The Sulphide Infill UG drilling program beneath Selinsing Pit IV is intended to detail a potential UG development area, comprising 3,000 m for 13 holes with depths ranging from 205 m to 285 m, averaging 252 m. Other deep holes will be designed to test strike, down dip and down plunge extensions to the south of Selinsing.

Metallurgical drilling is also planned, in locations to be defined, with an aim to provide sulphide material to be used in metallurgical testwork.

A program for pit mapping at Selinsing and Buffalo Reef current pits has been resumed, and is intended to provide geological/lithological, structural and geotechnical information to support further studies. A soil sampling program at Buffalo Reef area is being planned to cover prospective areas not sampled in the past, aiming to identify potential targets for future drilling.

Figure 26.1 Oxide and sulphide drilling programs in relation to the LOM pits and existing drilling (green traces)



26.4 Metallurgy

26.4.1 Flotation

Follow-up testwork has been recommended for:

- BRF Transition ore variant Gangue Depressant Tests
- Additional flotation testwork is recommended to confirm the recovery at the target concentrate grade for lower head grade samples.
- OMC (Nov 2018) also recommending further refinement of the use of Cu SO_4 addition in flotation.

26.4.2 Plant construction

Items for Plant construction include

- The control plan for items identified in the construction Risk Assessment (refer Section 25.6).
- A detailed Project Execution Plan

27 CERTIFICATES

27.1 Certificate of Qualified Person – John Graindorge

I, John Graindorge, Principal Consultant of Snowden Mining Industry Consultants Pty Ltd, Level 6 130 Stirling Street, Perth, Western Australia, do hereby certify that:

a) I am a co-author of the technical report titled Selinsing Gold Sulphide Project, dated 31 January 2019 (the "Technical Report") prepared for Monument Mining Limited.

b) I graduated with a Bachelor's degree in Geology from the University of Western Australia. I also completed a Post-Graduate Certificate in Geostatistics in 2007 at Edith Cowan University. I am a Member of the Australasian Institute of Mining and Metallurgy and a Chartered Professional Geologist. I have worked as a Geologist continuously for a total of 17 years since my graduation from university. I joined Snowden in 2005 and have been involved in resource estimation and evaluation for 12 years. I have read the definition of "Qualified Person" set out in NI 43-101 ("the Instrument") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements of a "Qualified Person" for the purposes of the Instrument. I have been involved in resource evaluation, including gold projects for at least five years.

c) I have not made a current visit to the Selinsing Property. I previously visited the Selinsing mine site and surrounding tenements in August 2010, however this visit is not considered current.

d) I am responsible for the preparation of sections 7, 8, 9, 10, 11, 12 and 14 of the Technical Report, and contributed to the preparation of sections 1, 4, 23, 25 and 26 of the Technical Report.

e) I am independent of the issuer as defined in section 1.5 of the Instrument.

f) I have no prior involvement with the property that is the subject of the Technical Report apart from the co-authorship of an earlier Technical Report on the Buffalo Reef project dated 2011.

g) I have read the Instrument and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

h) As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Perth, Western Australia, on 31 January 2019.

Signed and sealed John Graindorge

John Graindorge, BSc (Hons), Grad. Cert. Geostatistics, MAusIMM(CP)

27.2 Certificate of Qualified Person – Frank Blanchfield

I, Frank Blanchfield, Principal Consultant and Divisional Manager – Metallurgy of Snowden Mining Industry Consultants Pty Ltd, Level 6, 130 Stirling Street, Perth, Western Australia, do hereby certify that:

a) I am a co-author of the technical report titled Selinsing Gold Sulphide Project, dated 31 January 2019 (the "Technical Report") prepared for Monument Mining Limited. (the "Technical Report") prepared for Monument Mining Limited.

b) I graduated with a Bachelor's degree in Mining Engineering from the University of New South Wales. I am a Fellow of AusIMM. I have worked as a Mining Engineer continuously for a total of 28 years since my graduation from university. I have worked in mine planning and operations of gold mining projects for at least five years.

c) I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("the Instrument") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements of a "Qualified Person" for the purposes of the Instrument. I have been involved in mining and metallurgy related consulting practice for three years, including development and review of nickel laterite projects such as the Koniambo and the Dutwa laterite projects.

d) I visited the Selinsing Property in March 2016.

e) I am responsible for the preparation of sections 15 and 16 of the Technical Report, and accept responsibility for currency and completeness of all sections of the report.

f) I am independent of the issuer as defined in section 1.5 of the Instrument.

g) I have not had prior involvement with the property that is the subject of the Technical Report.

h) I have read the Instrument and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

i) As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Perth, Western Australia, on 31 January 2019

Signed and sealed Francis James Blanchfield

Frank Blanchfield, B.Eng, FAusIMM

27.3 Certificate of Qualiifed Person – Michael Kitney

I, Michael Kitney, Independent Technical Director of Metallurgical Design at Monument Mining Limited, Unit 8, 296 Mill Point Road, South Perth, Western Australia, do hereby certify that:

a) I am a co-reviewer of the Selinsing Gold Sulphide Project - NI 43-101 Technical Report Project Number AU10173 dated 31 January 2019 (the "Technical Report") prepared for Monument Mining Limited.

b) I graduated with an Associateship in Metallurgy from the Western Australian Institute of Technology, and subsequently completed the Graduate Diploma in Extractive Metallurgy at the WA School of Mines. I also completed a Master's Degree in Mineral Economics awarded by Curtin University of Western Australia. I am a Member of the Australasian Institute of Mining and Metallurgy. I have worked as an Extractive Metallurgist continuously for a total of 47 years since my graduation from university. I have read the definition of "Qualified Person" set out in NI 43-101 ("the Instrument") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements of a "Qualified Person" for the purposes of the Instrument. I have been involved in resource evaluation, including gold projects for at least 20 years.

c) I visited the Selinsing Property on multiple occasions between mid-2008 and mid-2016 inclusive.

d) I am responsible for the review of section 21 of the Technical Report.

e) I am independent of the issuer as defined in section 1.5 of the Instrument.

f) I have no prior involvement with the property that is the subject of the Technical Report.

g) I have read the Instrument and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

h) As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all the Project cost information that is required to be disclosed to make the Technical Report not misleading.

Dated at Perth, Western Australia, on 31 January 2019

Signed and sealed Michael Kitney

Michael Kitney, MSc, MAusIMM)

27.4 Certificate of Qualified Person – Fred William Kock

I, Fred William Kock, Principal Metallurgist at Orway Mineral Consultants, Level 4, 1 Adelaide Terrace, East Perth, Western Australia, do hereby certify that:

a) I am a co-reviewer of the Selinsing Gold Sulphide Project - NI 43-101 Technical Report Project Number AU10173 dated 31 January 2019 (the “Technical Report”) prepared for Monument Mining Limited.

b) I graduated with a National Higher Diploma in Extractive Metallurgy from the Technikon Witwatersrand in Johannesburg, South-Africa. I am a Fellow of the Australasian Institute of Mining and Metallurgy (Member number 226667). I have worked as an Extractive Metallurgist continuously for a total of 31 years since my graduation. I have read the definition of “Qualified Person” set out in NI 43-101 (“the Instrument”) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements of a “Qualified Person” for the purposes of the Instrument.

c) I have not visited the Selinsing Property.

d) I am responsible for the review of Chapter 13 and 17 of the Technical Report.

e) I am independent of the issuer as defined in section 1.5 of the Instrument.

f) I have no prior involvement with the property that is the subject of the Technical Report.

g) I have read the Instrument and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

h) As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all the Project cost information that is required to be disclosed to make the Technical Report not misleading.

Dated at Perth, Western Australia, on 31 January 2019

Signed and sealed Fred William Kock

Fred William Kock, (FAusIMM)

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