Monument Mining Limited

Selinsing Gold Mine and Buffalo Reef Project - Malaysia

NI 43-101 Technical Report

Project Number AU5200

14 December 2016

SNOWDEN

www.snowdengroup.com

OFFICE LOCATIONS

Perth

Level 6, 130 Stirling Street
Perth WA 6000
AUSTRALIA

Tel: +61 8 9213 9213
ABN: 99 085 319 562
perth@snowdengroup.com

Brisbane

104 Melbourne Street
South Brisbane QLD 4101
AUSTRALIA

Tel: +61 7 3026 6666
Fax: +61 7 3026 6060
ABN: 99 085 319 562
brisbane@snowdengroup.com

Johannesburg

Technology House, Greenacres
Office Park, Cnr. Victory and
Rustenburg Roads, Victory Park
Johannesburg 2195
SOUTH AFRICA

PO Box 2613, Parklands 2121
SOUTH AFRICA

Tel: +27 11 782 2379
Fax: +27 11 782 2396
Reg. No. 1998/023556/07
johannesburg@snowdengroup.com

Website

www.snowdengroup.com

This report was prepared as a National Instrument 43-101 Standards of Disclosure for Mineral Projects Technical Report for Monument Mining Limited (Monument) by Snowden Mining Industry Consultants Pty Ltd (Snowden). The quality of information, conclusions, and estimates contained herein are consistent with the quality of effort involved in Snowden's services. The information, conclusions, and estimates contained herein are based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by Monument subject to the terms and conditions of its contract with Snowden and relevant securities legislation. The contract permits Monument to file this report as a Technical Report with Canadian securities regulatory authorities pursuant to National Instrument 43-101. Except for the purposes legislated under provincial securities law, any other uses of this report by any third party is at that party's sole risk. The responsibility for this disclosure remains with Monument. The user of this document should ensure that this is the most recent Technical Report for the property as it is not valid if a new Technical Report has been issued.

Qualified Persons:	Frank Blanchfield *B.Eng, FAusIMM* Principal Consultant and Divisional Manager – Metallurgy
	John Graindorge *BSc (Hons), Grad. Cert. Geostatistics, MAusIMM (CP)* Principal Consultant
	Dr Leendert (Leon) Lorenzen *CPEng, CEng, PrEng, FIEAust, FAusIMM, FIChemE, FSAIMM, GAICD* Executive Metallurgical Consultant
Reviewed by:	Lynn Olssen *BSc, Post Grad. Cert. Geostatistics, MAusIMM(CP)* General Manager Operations (APAC), General Manager Geosciences
Issued by:	Perth Office
Doc ref:	161214 Final AU5200 Monument Selinsing NI 43-101
Last edited:	14 December 2016
Number of copies:	
Snowden:	2
Monument Mining Limited:	2

Table of Contents

Figures

Tables

1 SUMMARY

This Technical Report describes the Buffalo Reef and Selinsing Property, a mineral exploration, development, and production area of the Selinsing Gold Mine (SGM) Project, located in the region of Selinsing, in the State/Province of Pahang State in Malaysia. The Buffalo Reef and Selinsing Property is owned by Monument Mining Limited (Monument). The report focus is on the identification of a new sulphide ore process utilising the biological oxidation method, planned to be added to the existing oxide process plant at Selinsing. To this end, prefeasibility studies were undertaken to assess the viability of the combined Buffalo Reef and Selinsing Property using this method, including updated sulphide and oxide Mineral Resource and Mineral Reserve estimates.

The report was compiled by Snowden Mining Industry Consultants Pty Ltd (Snowden) for the issuer, Monument Mining Limited. The components of the prefeasibility study (PFS) were completed by:

- Monument (history, infrastructure, mining and administration costs, environmental status)
- Snowden (Mineral Resource estimates, mine planning, Mineral Reserve estimates, project cashflow)
- Lycopodium Minerals Ltd. (bioleach process design, operating costs and capital costs)

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

1.1 Summary of 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 metres (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 kilometres (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.

1.2 Summary of drilling

As at the end of February 2016, approximately 145,217 m of drilling has been completed across the Selinsing and Buffalo Reef deposits, comprising predominately reverse circulation (RC) (50.9%) and diamond drilling (41.5%). RC and diamond drilling account for approximately 95% of the drilling at the two projects.

Core recovery for diamond holes drilled by Monument at Selinsing and Buffalo Reef averages approximately 94% and 88% respectively. The sample recovery for RC drilling is not recorded.

Diamond drilling included both PQ and HQ diameter with the core cut in half using a diamond saw and sampled based on a 1.5 m sample interval, adjusted to not cross geological boundaries. RC drilling conducted by Monument since approximately 2007, at both Selinsing and Buffalo Reef, was drilled using a 4½-inch face sampling bit. 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.

A number of laboratories have been used to prepare and analyse samples from Selinsing and Buffalo Reef over the project history. The majority of samples included in the 2016 resource estimate 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 ISO17025:2005), with final assay reports signed off by SGS Port Klang. Samples were dried and then crushed using a jaw crusher. The crushed samples were riffle split twice and then pulverised to P90 75 µm. Gold was analysed by fire assay using a 50 g charge with an atomic absorption spectroscopy (AAS) finish.

A systematic or independent quality assurance/quality control (QAQC) program was not applied during the Damar and Avocet drilling and sampling campaigns (carried out prior to Monument) at Buffalo Reef. 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 conducted by Snowden in 2011 between the Damar/Avocet and Monument drilling at Buffalo Reef do not show any material difference or bias.

A random selection of 10 assay certificates, sourced directly from the SGS laboratory in Mengapur, was checked by Snowden 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.

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 current 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 recommends 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.

1.3 Summary of status of exploration, development and operations

Monument plans to follow-up with diamond drilling programs at the Selinsing and Buffalo Reef deposits, focused on defining preferentially sulphide mineralisation at depth below and around the existing pits within gap zones in between the known Mineral Resources that contain little drillhole information, and to convert Inferred Resources into Indicated and/or Measured Resources (proposed “Deep Sulphide Holes”).

A sampling geometallurgical program to be conducted in the BRN, BRC, Felda, BRS and Selinsing deposits is also planned, aiming to define leachable mining blocks to improve mining and plant production.

Details of the planned exploration programs are presented at Section 26.1.

1.4 Mineral Resources

1.4.1 Geological interpretations

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, with priority given to the diamond 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. 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 of the Selinsing and Buffalo Reef gold mineralisation was completed by Monument. 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 was used to guide the interpretation of the gold mineralisation at both deposits. Extrapolation of the interpretation down dip and along strike was limited to approximately half the drillhole spacing.

The interpreted mineralisation at Selinsing comprises a single main mineralised zone which is typically 30 m to 50 m thick (locally up to 80 m thick) in the southern half and narrows in the northern half to 10 m to 20 m thick, along with numerous minor mineralised structures. The interpreted mineralisation at Buffalo Reef South-Central-Felda (BRSCF) comprises a main tabular mineralised zone which is typically 10 m to 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.

1.4.2 Block model and grade estimation

A block model was constructed for each deposit using a parent block size of 10 mE x 20 mN x 2.5 mRL for Selinsing and 8 mE x 20 mN x 2.5 mRL at Buffalo Reef. The 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.

For Buffalo Reef, gold and arsenic grades were estimated using ordinary kriging (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 in both the mineralised domain and the waste domain, along with mixed populations, Snowden elected to use multiple indicator kriging (MIK) to estimate the block antimony grades. Similarly, due to the highly skewed gold grade distribution and mixed populations at Selinsing, Snowden elected to use MIK to estimate the block gold grades. Due to the variable dip of the mineralisation at both deposits, dynamic anisotropy was used to locally adjust the orientation of the search ellipse and variogram models. A three-pass search strategy was utilised for all grade estimates, with the search radii and number of samples based on the results of the variography and a kriging neighbourhood analysis. An initial search of 30 m x 15 m x 7.5 m with a minimum of eight samples was used at Selinsing; a 50 m x 25 m x 10 m search with a minimum of 10 samples was used at Buffalo Reef.

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 declustered) and estimated block grades
- Moving window averages comparing the mean block grades to the composites.

The model validation shows that globally, the block grade estimates compare reasonably well with the input sample data and that, 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.

1.4.3 Bulk density

Based on wax-coated bulk density measurements from diamond drill core from Selinsing and Buffalo Reef, along with hand specimens from the Selinsing pit, Snowden derived bulk density values which were applied to the model based on the oxidation state (Table 1.1).

Table 1.1 Bulk density values applied to the block models

Mineralisation state MINZONE	Oxidation state OXIDE	Bulk density (t/m^3)	
		Selinsing	Buffalo Reef
Waste (0)	Oxide (10)	2.20	2.16
	Transitional (20)	2.40	2.43
	Fresh (30)	2.61	2.67
Mineralised (1000, 2000)	Oxide (10)	2.20	2.20
	Transitional (20)	2.47	2.55
	Fresh (30)	2.66	2.74
Historical tailings	-	1.18	

1.4.4 Mineral Resource classification

The Selinsing 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 the company'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 Mineral Resource estimate is outlined as follows:

- Where the drilling density was approximately 40 m along strike by 20 m down dip (or less), mineralisation within the main mineralised lode 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.

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 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 were classified as an Indicated Resource. The requirement for closer spaced drilling in this area 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.

1.4.5 Mineral Resource reporting

The Mineral Resources for the Selinsing and Buffalo Reef deposits have been reported above a 0.3 g/t Au cut-off grade for oxide material and above a 0.7 g/t Au cut-off grade for transitional and sulphide material. The cut-off grades are based on the results of the mining study.

Selinsing Mineral Resource

The Mineral Resource estimate for the Selinsing deposit is provided in Table 1.2. The Mineral Resource is limited to a pit shell provided by Monument based on a US$1,776/ounce (oz) gold price. The pit shell was used by Snowden to define the likely limits of potential open-pit mining. The mining and cost parameters used by Monument to generate the resource pit shell are not materially different to those used by Snowden for the Mineral Reserves.

Table 1.2 Selinsing Mineral Resource statement, depleted for mining to end of June 2016

Classification	Oxidation	Cut-off (g/t Au)	Tonnes (kt)	Au (g/t)	Ounces (koz Au)
Indicated	Oxide	0.3	90	0.67	2
	Transitional	0.7	90	1.42	4
	Fresh	0.7	3,040	1.98	193
Indicated Total			**3,220**	**1.93**	**200**
Inferred	Oxide	0.3	10	0.84	0.3
	Transitional	0.7	3	1.23	0.1
	Fresh	0.7	540	3.75	65
Inferred Total			**550**	**3.67**	**65**

Note: Small discrepancies may occur due to rounding. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Buffalo Reef Mineral Resource

The Mineral Resource estimate for the Buffalo Reef deposit is provided in Table 1.3. The Mineral Resource is limited to a pit shell provided by Monument based on a US1,776/oz gold price. The pit shell was used by Snowden to define the likely limits of potential open-pit mining. The mining and cost parameters used by Monument to generate the resource pit shell are not materially different to those used by Snowden for the Mineral Reserves.

Table 1.3 Buffalo Reef Mineral Resource statement, depleted for mining to end of June 2016

Classification	Oxidation	Zone	Cut-off (g/t Au)	Tonnes (kt)	Au (g/t)	As (ppm)	Sb (ppm)	Ounces (koz Au)
Indicated	Oxide	BRN	0.3	180	0.99	1,900	270	6
		BRC	0.3	170	0.83	1,600	140	4
		Felda	0.3	260	1.33	2,700	230	11
		BRS	0.3	100	2.10	3,200	560	7
	Oxide total			**700**	**1.23**	**2,300**	**270**	**27**
	Transitional	BRN	0.7	150	1.26	2,200	230	6
		BRC	0.7	310	1.19	2,300	110	12
		Felda	0.7	190	1.64	3,000	330	10
		BRS	0.7	230	2.65	3,000	3,250	19
	Transitional total			**860**	**1.68**	**2,600**	**1,010**	**46**
	Fresh	BRN	0.7	70	1.18	2,300	100	2
		BRC	0.7	990	1.67	3,400	1,990	53
		Felda	0.7	620	1.78	2,900	960	35
		BRS	0.7	1,130	2.12	2,800	1,150	77
	Fresh total			**2,790**	**1.87**	**3,000**	**1,380**	**167**
INDICATED TOTAL				**4,330**	**1.73**	**2,800**	**1,130**	**240**
Inferred	Oxide	BRN	0.3	100	0.81	1,700	120	2
		BRC	0.3	120	1.15	1,600	60	4
		Felda	0.3	70	1.03	1,500	150	2
		BRS	0.3	90	1.14	1,400	190	3
	Oxide total			**370**	**1.04**	**1,500**	**120**	**12**
	Transitional	BRN	0.7	90	1.34	2,300	110	4
		BRC	0.7	140	1.40	2,100	170	6
		Felda	0.7	50	1.54	1,900	150	2
		BRS	0.7	90	1.62	1,700	760	4
	Transitional total			**350**	**1.46**	**2,000**	**290**	**16**
	Fresh	BRN	0.7	30	1.61	2,300	60	1
		BRC	0.7	1,500	1.86	2,800	1,980	89
		Felda	0.7	1,040	1.98	3,300	1,190	66
		BRS	0.7	530	1.59	2,500	630	27
	Fresh total			**3,100**	**1.85**	**2,900**	**1,470**	**184**
INFERRED TOTAL				**3,810**	**1.74**	**2,700**	**1,230**	**212**

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

Stockpile Mineral Resources

Stockpiles at the Selinsing project include 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 in situ 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 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.

Mineral Resources for the stockpiles at the Selinsing Project, as at the end of June 2016, are summarised in Table 1.4.

Table 1.4 Stockpile Measured Mineral Resources, as at end of June 2016

Stockpile name	Stockpile ID	Volume (lcm)	Tonnes (t)	Au (g/t)	Contained gold (oz)
Oxide stockpiles					
Low grade 1 (oxide)	LG1 O	6,885	14,075	1.03	467
Low grade 2 (oxide)	LG2 O	3,189	6,442	0.73	152
Super low grade 1 (oxide)	SLG1 O	2,845	5,349	0.44	76
Super low grade 2 (oxide)	SLG2 O	907,006	1,859,251	0.51	30,747
Super low grade 4 (oxide)	SLG 4	31,378	67,776	0.50	1,090
BR low grade 1 (oxide)	BR LG1 O	186	353	0.33	4
BR super low grade 1 (oxide)	BR SLG1 O	111,268	217,422	0.53	3,739
Oxide subtotal		**1,062,757**	**2,170,668**	**0.52**	**36,275**
Leachable sulphide stockpiles					
High grade 1 (leachable sulphide)	HG1 S	81	175	6.41	36
Low grade 1 (leachable sulphide)	LG1 S	88	190	0.98	6
BR low grade 1 (leachable sulphide)	BR LG1 S	82	166	0.32	2
Leachable sulphide subtotal		**251**	**531**	**2.56**	**44**
Non-leachable sulphide stockpiles					
High grade 2 (non-leachable sulphide)	HG2 S	5,065	10,940	2.71	953
Low grade 3 (non-leachable sulphide)	LG2 S	8,402	16,983	0.97	529
Low grade 4 (non-leachable sulphide)	LG4 S	25,331	54,715	0.95	1,663
Super low grade 3 (non-leachable sulphide)	SLG3 S	748	1,511	0.60	29
BR high grade 2 (non-leachable sulphide)	BR HG2 S	18,536	36,695	2.58	3,045
BR low grade 2 (non-leachable sulphide)	BR LG2 S	22,444	43,206	1.03	1,429
Non-leachable sulphide subtotal		**80,526**	**164,051**	**1.45**	**7,648**
TOTAL		**1,143,534**	**2,335,250**	**0.59**	**43,966**

Note: All stockpiles classified as Measured Resources; lcm = loose cubic metres. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

1.4.6 Drillhole data analysis

Drilling at both deposits 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). The drillhole data was composited downhole prior to running the estimation process using a 1.5 m compositing interval to minimise any bias due to sample length.

Statistical analysis was carried out on the composited dataset for gold (Au) grades at Selinsing and gold, arsenic (As) and antimony (Sb) grades at Buffalo Reef. The statistics show that the Selinsing and Buffalo Reef mineralisation has a positively skewed gold grade distribution with a high coefficient of variation (CV). Top-cuts were applied to gold at Buffalo Reef to minimise the impact of extreme grades on the local block grade estimates, with a top-cut of 20 g/t Au applied for BRSCF and 8 g/t Au applied for BRN.

Variograms for gold for the mineralised domains were modelled to assess the grade continuity and as an input to the kriging algorithms. Due to the application of indicator kriging for the Selinsing model, indicator variograms were modelled for 12 grade thresholds. For Buffalo Reef, normal scores variograms were modelled, with the sill parameters back-transformed. The maximum and intermediate directions of continuity were aligned with the overall strike and down dip directions respectively. The minor direction of continuity was aligned in the true thickness direction.

1.5 Mining

The Selinsing and Buffalo Reef deposits have been mined using open pit methods for seven years and the mining contractor for the mining activities is Minetech Construction Sdn Bhd. The contract is current and they are 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.

1.5.1 Geotechnical

Snowden undertook a geotechnical review of the Selinsing and Buffalo Reef mines in 2016 (Geotechnical Review of Selinsing and Buffalo Reef).

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 developed by Snowden using the current slope stability outcomes and the estimated GSI for the main units; these are summarised in Table 1.5.

Table 1.5 Snowden slope design recommendations for Selinsing and Buffalo Reef

Deposit	Geotechnical domain	Batter face angle	Inter-ramp slopes	
			Face angle	Maximum bench stack height
Selinsing	Saprolite/highly weathered Rock	45°	30°	30 m
	Footwall	50°	35°	80 m
	Ore zone	60°	40°	80 m
	Hangingwall	65°	45°	80 m
Buffalo Reef	Saprolite/highly weathered rock	45°	30°	30 m
	Footwall	50°	40°	80 m
	Hangingwall	50°	40°	80 m

1.5.2 Mine planning

Snowden undertook mine planning for the PFS by undertaking pit optimisation of the Indicated Resources in the block model using GEOVIA's Whittle Four-X™ (Whittle). The model was diluted prior to optimisation with a dilution factor of approximately 10%. Pits and waste dumps were designed resulting in the pit layout illustrated in Figure 1.1. Resultant pit inventories, including the stockpiles at June 30 2016, are summarised in Table 1.6.

Figure 1.1 Overall mining layout with haulage paths

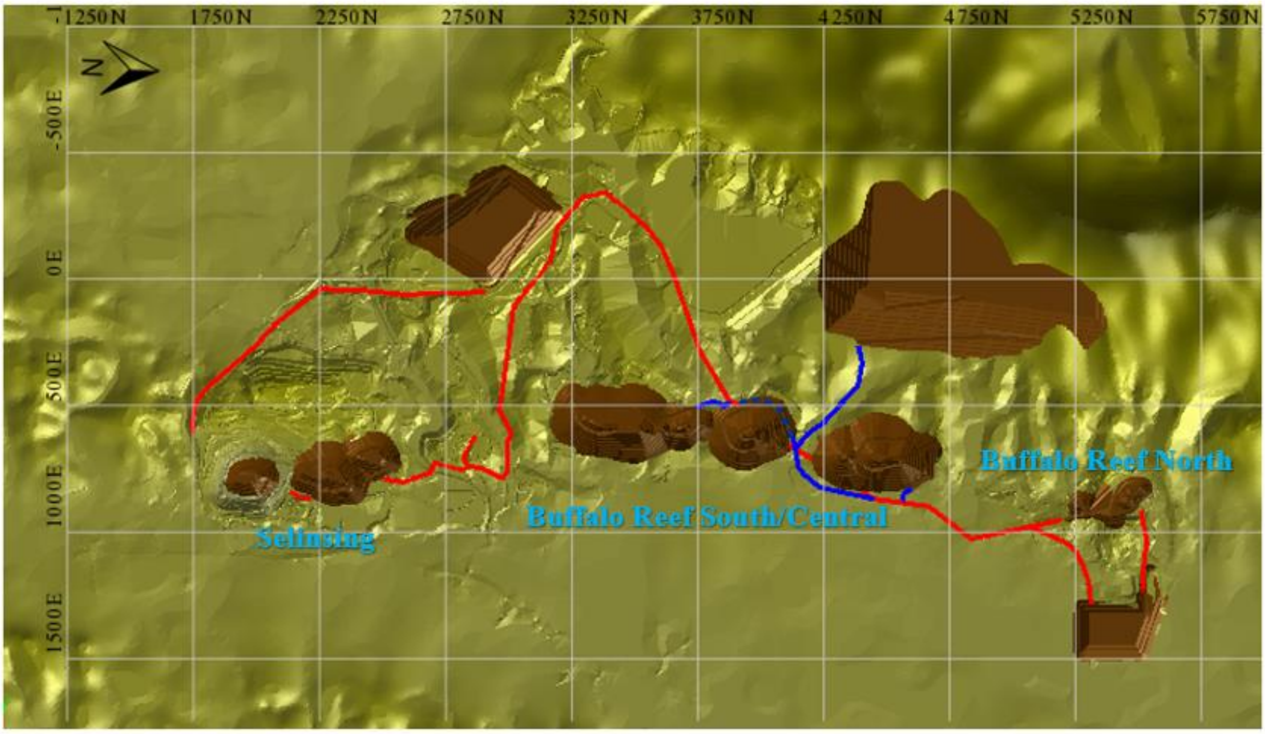


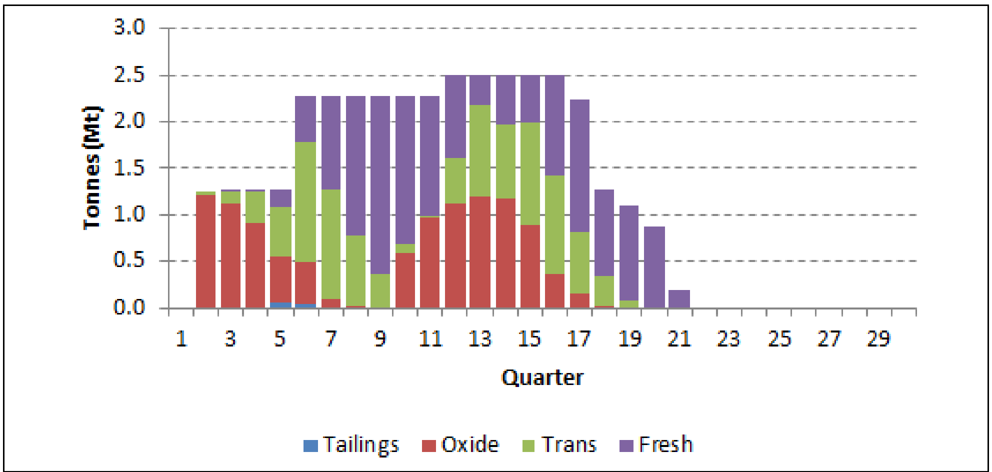
Table 1.6 Total mining inventory for scheduling (all Indicated Resources)

Item	Selinsing	Buffalo Reef South/Central	Buffalo Reef North	Stockpiles	Total
Oxide ore (kt)	8	453	105	2,206	2,773
Au (g/t)	1.20	1.37	1.05	0.52	0.68
Trans ore (kt)	26	623	72	0	721
Au (g/t)	1.98	1.82	1.28	3.58	1.77
Fresh ore (kt)	573	2,125	14	163	2,875
Au (g/t)	2.27	1.91	1.25	1.46	1.96
Total ore (kt)	608	3,201	190	2,369	6,368
Au (g/t)	2.24	1.82	1.15	0.58	1.38
Waste (kt)	3,377	28,252	675	-	32,304
Strip ratio (w:o)	5.6	8.8	3.6		5.1

1.5.3 Production scheduling

The mining quarterly schedule is shown by rocktype in Figure 1.2. The first year of mining, prior to the sulphide circuit commencing, mines only a small amount of in situ material, sourced mostly from the Buffalo Reef surface deposits. The total mining rate in this period is the equivalent of 5 million tonnes per annum (Mt/a). Just prior to the sulphide circuit commencing the mining rate ramps up to 9 Mt/a to 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.

Figure 1.2 Total movement schedule by rock type



The process feed schedule is shown in Figure 1.3 (tonnes) and Figure 1.4 (grade). The first 1.5 years of production mines lower grade oxide resources from in situ and existing stockpiles. The sulphide circuit is commissioned and ramped up in Q7, leading to a dip in production. 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 (>2.5 g/t Au) feed is achieved for about five 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 Process feed schedule

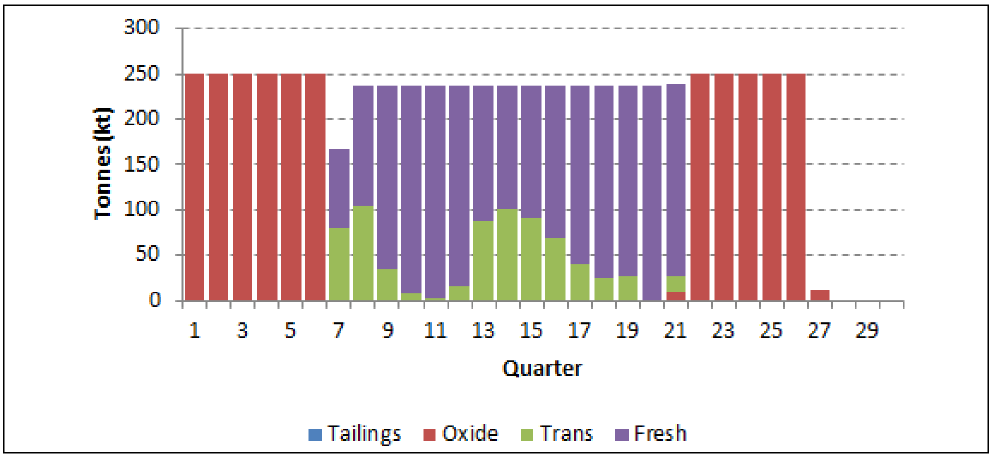
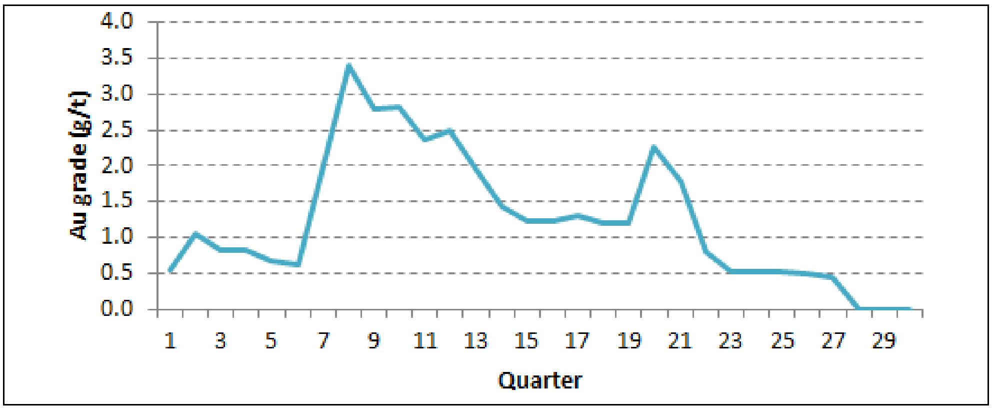


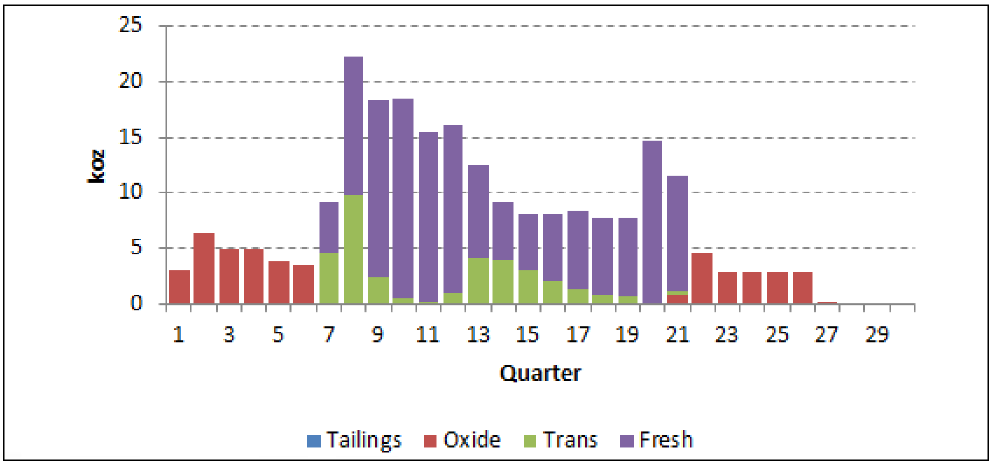
Figure 1.4 Process grade schedule



1.5.4 Product schedule

The gold production schedule shown in Figure 1.5 is quite variable. During initial oxide processing the production rate is approximately 15 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 10 koz/a.

Figure 1.5 Recovered gold schedule



1.6 Mineral Reserves

Snowden assessed the issues that affect the conversion of Mineral Resources to Mineral Reserves as modifying factors that are summarised in Table 1.7.

Table 1.7 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	Snowden prepared the updated Selinsing Mineral Resource estimate in June 2016. 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 Resources and stockpile Measured Mineral Resources used as a basis for the Mineral Reserves are summarised in Table 1.2, Table 1.3 and Table 1.4. The Selinsing and Buffalo Reef Indicated Mineral Resources and stockpile Measured Mineral Resources are inclusive of Selinsing and Buffalo Probable Mineral Reserves and stockpile Proven 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.
Study status	The current NI 43-101 technical report is for a PFS to establish the viability of sulphide ore extraction through the extension of the existing oxide plant to incorporate additional sulphide ore extraction. Another study was completed by Practical Mining LLC in 2012 for the extraction of sulphides from the Selinsing and Buffalo Reef deposits. Snowden has re-evaluated this work using reports from Lycopodium that updated the metallurgy costs and recoveries in 2016. Snowden considers that the mining work completed is of a PFS level of accuracy.
Cut-off parameters	A nominal cut-off grade of 0.35 g/t Au was applied to oxides and 0.75 g/t for sulphides 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 Whittle pit optimisation, staged pit design, production scheduling and mine cost modelling was undertaken by Snowden. 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, block dilution has reduced the recovered ounces by approximately 2% and marginally increased the ore tonnage processed by 5%.
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 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 t/d Au treatment plant in three phases. During 2011, Monument engaged Inspectorate Exploration and Mining Services Ltd (Inspectorate) of Vancouver, Canada, to carry out a metallurgical test program on a selection of diamond drill core material collected from the Buffalo Reef deposit. The extraction for sulphides at Selinsing has been assessed in the engineering study ultimately prepared for Monument by Lycopodium of Brisbane, Australia, and reported by Lycopodium in "Selinsing Phase IV Study", February 2013. The mineralisation modelling and metallurgical testwork indicate that conventional carbon-in-leach (CIL) extraction from oxide ores and bio-oxidation leach for transition and fresh ores can be used to produce gold as Dore. 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

<table>
<tr><th>Item</th><th>Comment</th></tr>
<tr><td></td><td>

Electrowinning
Smelting
Tailings disposal and effluent reclaim
Cyanide detoxification.

Lycopodium applied industry standard methods to prepare the estimate by developing the following components:

Process design criteria – based on the Inspectorate report (ibid)
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 Lycopodium and reviewed by Snowden. 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 rowspan="4">Oxide</td><td rowspan="4">Both</td><td><1.0</td><td>66</td></tr>
<tr><td>1.0 to 1.5</td><td>75</td></tr>
<tr><td>1.5 to 2.5</td><td>83</td></tr>
<tr><td>>2.5</td><td>87</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>87</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>87</td></tr>
</table>
</td></tr>
<tr><td></td><td>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 PFS level of accuracy.</td></tr>
<tr><td>Environmental</td><td>
Rock characterisation was completed in Malaysia and potentially acid-forming (PAF) 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.
</td></tr>
<tr><td>Infrastructure</td><td>
Monument is negotiating with the local authority for power purchase from the electricity grid.
Monument has indicated the bio-oxidation plant build will be a EPC execution with Monument providing the management.
Accommodation will be in surrounding communities.
</td></tr>
</table>

<table>
<tr><th>Item</th><th>Comment</th></tr>
<tr><td>Cost and revenue factors</td><td>Process costs were used from historical oxide costs from site and developed by Lycopodium in 2016 for sulphide as:
<table>
<tr><th>Material treated</th><th>Deposit</th><th>Gold grade (g/t Au)</th><th>Process operating cost (US$/t)</th></tr>
<tr><td rowspan="4">Oxide#</td><td rowspan="4"></td><td><1.0</td><td>7.11^</td></tr>
<tr><td>1.0 to 1.5</td><td>7.11^</td></tr>
<tr><td>1.5 to 2.5</td><td>7.11^</td></tr>
<tr><td>>2.5</td><td>7.11^</td></tr>
<tr><td rowspan="2">Transition*</td><td>Selinsing</td><td>All</td><td>20.00</td></tr>
<tr><td>Buffalo Reef</td><td>All</td><td>20.00</td></tr>
<tr><td rowspan="2">Fresh/Sulphides</td><td>Selinsing</td><td>All</td><td>19.56^</td></tr>
<tr><td>Buffalo Reef</td><td>All</td><td>19.56^</td></tr>
</table>
* Reagent costs should be slightly higher for transitional material especially during flotation
# From Figure 13.5 in Section 13
^ Taken into account historical data (about 1 Mt/a processed) from 17.2 as well as current 2016 year to date oxide cost of $6.83/t
Mining costs were supplied by Monument and developed from the existing contract.
The all-up mining operating cost was estimated to be US$1.90/t mined.
The mining capital cost was absorbed by contract mining.
Monument provided other capital costs, that were estimated by Lycopodium and others as follows:
Process capital costs: US$38.9 million
Closure costs: US$7.8 million
Sustaining costs: US$0.5 million per annum.
Closure costs are included in the valuation model.
All costs were supplied in US$.
Refining costs of US$5.00/t and royalties of 5% were applied to all gold produced, except for Felda which used 7%.</td></tr>
<tr><td>Revenue factors</td><td>Monument supplied a gold price of US$1,255/oz. This was applied as real and flat forward in the financial model.</td></tr>
<tr><td>Market assessment</td><td>Monument supplied a gold price of US$1,255/oz.
Monument has completed comprehensive market studies, including likely refiners.
Gold is freely traded and the price is set by the LME. A comprehensive marketing study was completed as part of the PM LLC 2013 NI 43-101 Technical Report. The selling of gold is straight forward.</td></tr>
<tr><td>Economic</td><td>The discount rate in the Monument financial model was set at 8%.
A financial sensitivity study was undertaken to evaluate capital expenditure, operating costs and gold price. The project was found to be most sensitive to changes in gold price.
The key performance indicators from the Monument model are summarised below:
<table>
<tr><th>Key performance indicator</th><th>Units</th><th>Value</th></tr>
<tr><td>All in cash cost (including royalty)</td><td>US$/oz produced</td><td>830.1</td></tr>
<tr><td>IRR ungeared</td><td>%</td><td>34.8</td></tr>
<tr><td>NPV (at 8%)</td><td>US$ M</td><td>23.1</td></tr>
<tr><td>Net cashflow</td><td>US$ M</td><td>37.2</td></tr>
<tr><td>Initial capital cost[a]</td><td>US$ M</td><td>39.5</td></tr>
</table>
[a] Excludes working capital</td></tr>
</table>

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 that there are no community or social encumbrances that could obstruct the provision of a MLA from the Malaysian government.
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 in-situ 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.

1.6.1 Mineral Reserve 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 June 2016, is provided in Table 15.2. Note that tonnes and ounces have been rounded and this may have resulted in minor discrepancies.

Table 1.8 Selinsing and Buffalo Reef in–situ Mineral Reserve estimate as at June 2016

Classification	Oxidation	Zone	Approximate cut-off (g/t Au)	Tonnes (kt)	Au (g/t)	Ounces (koz)
Probable	Oxide	Selinsing	0.3	8	1.2	0.3
		BRN	0.3	105	1.05	3.5
		BRC	0.3	114	0.91	3.3
		Felda	0.3	234	1.34	10.1
		BRS	0.3	103	1.95	6.5
	Oxide total			**565**	**1.31**	**23.8**
	Transitional	Selinsing	0.7	25	2.02	1.7
		BRN	0.7	69	1.29	2.9
		BRC	0.7	214	1.26	8.6
		Felda	0.7	158	1.66	8.5
		BRS	0.7	232	2.52	18.5
	Transitional total			**698**	**1.80**	**40.4**
	Fresh	Selinsing	0.7	551	2.33	41.2
		BRN	0.7	14	1.25	0.6
		BRC	0.7	719	1.76	40.6
		Felda	0.7	474	1.75	26.7
		BRS	0.7	862	2.22	61.4
	Fresh total			**2,619**	**2.03**	**170.6**
Probable total				**3,882**	**1.88**	**235.4**

Selinsing property Stockpile – Mineral Reserve statement

Mineral Reserves for the stockpiles at the Selinsing project, as at the end of June 2016, are summarised in Table 15.3. Note that tonnes and ounces have been rounded and this may have resulted in minor discrepancies.

Table 1.9 Stockpile Mineral Reserves as at end of June 2016

Stockpile name	Stockpile ID	Volume (lcm)	Metric tonnes (t)	Au (g/t)	Contained gold (Troy oz)
Oxide stockpiles					
Selinsing					
Low grade 1 (oxide)	SEL LG1 O	6,885	14,075	1.03	467
Low grade 2 (oxide)	SEL LG2 O	3,189	6,442	0.73	152
Super low grade 1 (oxide)	SEL SLG1 O	2,845	5,349	0.44	76
Super low grade 2 (oxide)	SEL SLG2 O	907,006	1,859,251	0.51	30,747
Super low grade 4 (oxide)	SEL SLG 4	31,378	67,776	0.50	1,090
Buffalo Reef					
Low grade 1 (oxide)	BR LG1 O	186	353	0.33	4
Super low grade 1 (oxide)	BR SLG1 O	111,268	217,422	0.53	3,739
Oxide subtotal		**1,062,757**	**2,170,668**	**0.52**	**36,275**
Leachable sulphide stockpiles					
Selinsing					
High grade 1 (leachable sulphide)	SEL HG1 S	81	175	6.41	36
Low grade 1 (leachable sulphide)	SEL LG1 S	88	190	0.98	6
Buffalo Reef					
Low grade 1 (leachable sulphide)	BR LG1 S	82	166	0.32	2
Leachable sulphide subtotal		**251**	**531**	**2.56**	**44**
Non-leachable sulphide stockpiles					
Selinsing					
High grade 2 (non-leachable sulphide)	SEL HG2 S	5,065	10,940	2.71	953
Low grade 3 (non-leachable sulphide)	SEL LG2 S	8,402	16,983	0.97	529
Low grade 4 (non-leachable sulphide)	SEL LG4 S	25,331	54,715	0.95	1,663
Super low grade 3 (non-leachable sulphide)	SEL SLG3 S	748	1,511	0.60	29
Buffalo Reef					
High grade 2 (non-leachable sulphide)	BR HG2 S	18,536	36,695	2.58	3,045
Low grade 2 (non-leachable sulphide)	BR LG2 S	22,444	43,206	1.03	1,429
Non-leachable sulphide subtotal		**80,526**	**164,051**	**1.45**	**7,648**
TOTAL		**1,143,534**	**2,335,250**	**0.59**	**43,966**

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

1.7 Metallurgical testwork and design

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 a 1,200 tonnes per day (t/d) Au 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 $8.1 million and was completed in June 2012 on time and at a cost of $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.

During 2011, Monument engaged Inspectorate of Vancouver, Canada, to carry out a metallurgical test program on a selection of diamond drill core material collected from the Buffalo Reef deposit. This report includes:

- A summary of the work completed by Inspectorate and reported in *"Metallurgical Study on the Buffalo Reef Gold Project"* (2011)
- The engineering study ultimately prepared for Monument by Lycopodium of Brisbane, Australia, and reported by Lycopodium in *"Selinsing Phase IV Study"* (February 2013)
- Phase IV capital cost will be in the region of $39.5 million as estimated by Lycopodium and Monument.

The estimated (after implementation of Phase IV) gold recoveries and operating cost going forward are depicted in Table 1.10 below.

Table 1.10 Gold recoveries and operating cost

Material treated	Deposit	Gold grade (g/t Au)	Recovery (%)	Process operating cost (US$/t)
Oxide*	Both	<1.0	66	7.11^
	Both	1.0 to 1.5	75	7.11^
	Both	1.5 to 2.5	83	7.11^
	Both	>2.5	87	7.11^
Transition#	Selinsing	All	87	20.00
	Buffalo Reef	All	85	20.00
Fresh/Sulphides	Selinsing	All	87	19.56^
	Buffalo Reef	All	85	19.56^

** Reagent costs should be slightly higher for transitional material especially during flotation*

 # From Figure 13.5 in Section 13

^ Taken into account historical data (about 1 Mt/a processed) from Table 17.2 as well as current 2016 year to date oxide cost of $6.83/t.

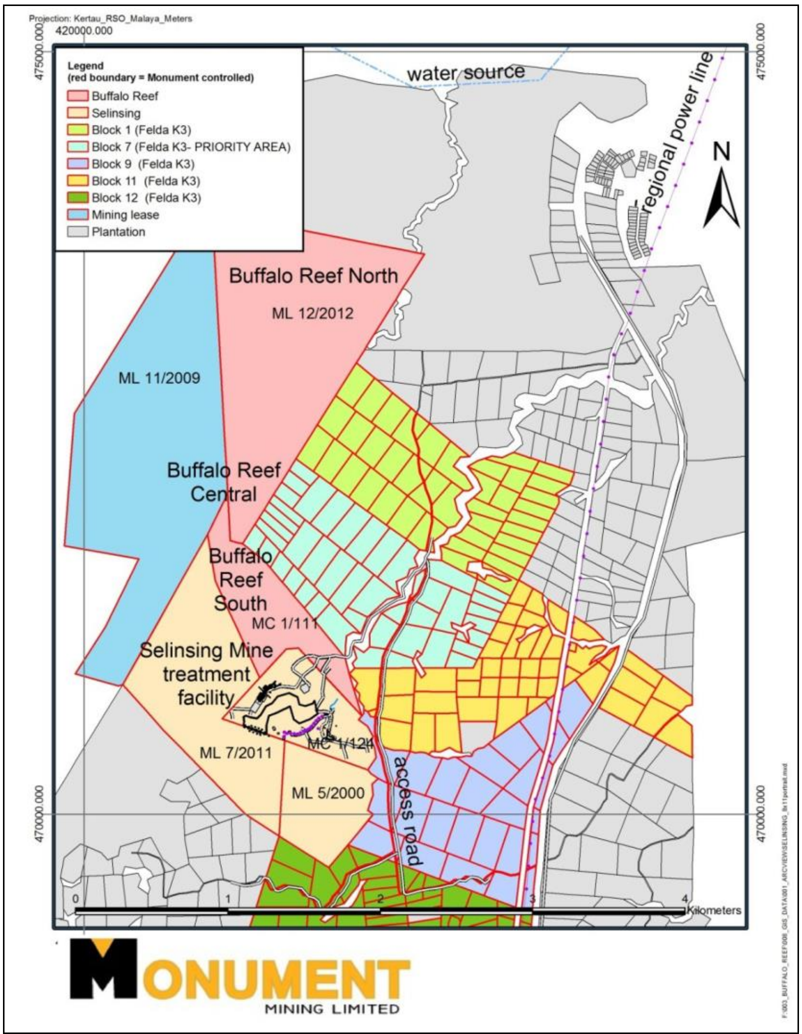
The Qualified Person has provided the summary and write-up for this section based on previous NI 43-101 reports, Selinsing Oxide Process Plant Performance up to date, and recent sulphide testwork completed by Inspectorate and used by Lycopodium to develop a process flow diagram and operating and capital cost for the flotation and bioleach project (Phase IV).

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 PFS level of accuracy.

1.8 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



1.8.1 Power

A 33 KV national grid power line 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 10.7 MW.

1.8.2 Water

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

1.8.3 Mining personnel

Mine personnel consist of expatriates and in-country professionals. Labour employment covenants requiring 50% of all employees to be Bumiputra (ongoing). 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.

1.8.4 Tailings storage areas

Tails storage design was completed by Knight Piesold. Capacity of the current facility is adequate based on 5,434,174 cubic metres (m^3) up to the 530 mRL final crest design.

1.8.5 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 of 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.

1.8.6 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.

Further information regarding the plant is found in Section 17.

1.9 Environmental and social

As part of the Mineral Reserve viability assessment, Snowden assessed key items as part of an environmental review of the Selinsing Gold Mine Project, based on information that was 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.

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.

Future environmental work programs will need to address the implementation of environmental studies that consider all items needed to gain approval for the environmental approvals. The points below summarise the key environmental aspects identified during the review that, if not effectively managed, could impact on future project approval and development:

- Finalise process approval for the bioleach method of gold extraction.
- Finalising the material characterisation program currently being undertaken as a high priority to ensure robust mine plans can be developed, particularly in relation to the sulphidic materials identified at the Buffalo Reef project. This will reduce the potential for significant risk associated with acid mine drainage management.

- There have been no significant tailings storage facility (TSF) issues identified to date. The continued use of a reputable engineering specialist to design and regularly review TSF performance will significantly reduce risks associated with geotechnical failure and impacts to ground and surface water as the project progresses. This reduces the risk of significant environmental and commercial consequences.
- To adequately protect waterways and resources, drainage and water protection measures will require definition, implementation and ongoing monitoring to prevent any future non-compliance in line with the project expansion and Buffalo Reef project.

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 targeting the mine or expansion project.

1.10 Operating and capital costs

1.10.1 Cost estimation battery limits

The operating and costs for the Project are based on the mining, processing and sale of gold dore. The process rates of 1.00 Mt/a for oxide and 0.95 Mt/a for sulphide after an initial ramp-up period for the bio-oxidation sulphide process have been used. All costs in this section are in US$.

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 Lycopodium in 2016. Administration costs were provided by Monument from the 2017 budget.

Capital cost sources

The oxide processing plant will continue processing stockpiled oxide materials until the bio-oxidation 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 Lycopodium, with the EPCM cost excluded and replaced with Monument's internal estimate of the owner costs for management of execution engineering, construction and procurement. The sulphide plant pre-production capital costs in the cash flow model have been allocated 30% in Year 1 and 70% in Year 2 of the two years of construction.

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

- Sulphide process plant: Lycopodium (reviewed by Snowden)
- EPC management: Monument (reviewed by Snowden)
- Mining: Monument (reviewed by Snowden)
- Oxide process plant: Monument (reviewed by Snowden)
- On-site infrastructure: Snowden (reviewed by Snowden)
- Off-site infrastructure: Snowden (reviewed by Snowden)
- Environmental: Monument (reviewed by Snowden)
- Social: Monument (reviewed by Snowden)
- Corporate G&A: Monument (reviewed by Snowden)

- Royalties: Monument
- Taxation: Monument
- Closure and salvage: Monument (reviewed by Snowden).

1.10.2 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 Snowden.

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 is applied to the capital cost estimate.

Bio-leach capital cost estimate

In 2016, Lycopodium revised its 2013 capital cost estimate provided to Monument in its report 3193-STY-001_0_ OPEX_CAPEX_Update Sept., 2016. Table 1.11 and Table 1.12 summarises the outcome of this revision.

Table 1.11 Summary of capital cost estimate (US$, 3Q 2016, 15%+25%)

Primary discipline	Subtotal cost (US$)	Project contingency cost (US$)	Grand total (US$)
A General	292	29	321
B Earthworks	343,344	44,635	387,979
C Concretelwork	638,319	95,748	734,067
D Steelwork	1,024,445	102,445	1,126,890
E Platework	3,793,971	531,156	4,325,127
F Mechanical	12,762,131	1,659,048	14,421,179
G Piping	5,011,106	1,002,221	6,013,327
H Electrical	7,052,417	705,242	7,767,658
J Instrumentation and Control	1,001,130	100,113	1,101,243
M Buildings and Architectural	848,690	127,160	975,849
O Owners	1,164,333	116,433	1,280,766
P EPCM	1,006,396	100,640	1,107,036
Total	**34,646,574**	**4,584,870**	**39,231,444**

Source: Lycopodium, 2016

Table 1.12 Monument capital cost summary estimate (RM, -15%+25%)

Phase IV engineering, design and procurement budget		
Activity	**RM***	**Comment**
Metallurgical review		
2010 test program design	21,000	Review test program design
Process design criteria		
Develop process design	48,000	Incorporate all metallurgical data and SGMM project requirements
Process flowsheet and mass balance		
Ore flows	18,000	
Water balance	18,000	
Flotation and bioleaching circuit	16,500	
Reagent handling	16,500	Process Engineering Consultant
Metallurgical balance	60,000	
Fuel requirements	12,000	
Air requirements	18,000	
Drawing supervision and sign-off	54,000	Crushing; milling; CIL; leach; flotation; bioleaching; carbon handling; services; water
P&IDs and control philosophy		
Develop final piping and instrumentation diagrams	132,000	Based on review of 2007 work
Develop control philosophy documentation	54,000	In consultation with SGMM and Project Electrical Engineers
Equipment list and datasheets		
Review and update 2007 documents	54,000	Will require extension of list
Project design review		
Regular design review meetings with Project Team	270,000	Site meetings: input to plant design, 6 x visit 10 days
Remote review of project design drawings	54,000	4 x hours per 16 weeks
Review final TSF design with consultant	34,318	includes site water balance
Subtotal – Metallurgical design	880,318	
Civil and structural design		
Geotechnical testwork and analysis	72,000	PIII quotation
Foundation design	70,000	PIII quotation
Structural design	80,000	PIII quotation
Subtotal – Civil and structural	**222,000**	
Mechanical and electrical design		
Mechanical and electrical engineer consultants	300,000	Based on previous costs
Subtotal – Electrical	**300,000**	
Design and drafting		
Gravity tails, ball mill 2, CIL and piping	393,000	Overflow Design Services (ODS); estimated 900 hrs
Subtotal – Design and drafting	**393,000**	
Procurement and management		
Salary	1,048,000	RM75,000 for 12 months; SGMM staff time allocated to Phase IV
Monument allocated admin expenses	650,000	5% of total admin cost
Subtotal – Procurement and management	**1,698,000**	
Expenses		
Metallurgical Consultant travel (Perth-Selinsing)	72,000	Estimated: 8 trips x A$3,000
Design Engineer Consultant travel (Perth-Selinsing)	108,000	Estimated: 12 trips x A$3,000
Local Civil and Structural consultants travel	20,000	Estimated: 2 x 10 trips x RM1,000
QAQC from vendor – Specialist Consultant	500,000	Inspections and QAQC test on equipment prior to acceptance
Subtotal – Expenses	**700,000**	
Allowance		
Contingency @ 15%	419,332	
	4,612,650	
US$	**1,107,036**	Forex RM:US$ 1:0.24

Source: Lycopodium, 2016

**Estimate in Malay Ringgit (MR) and a forex of RM:US$ as 1.00:0.24*

The estimated capital cost has also been significantly reduced from the 2013 estimate (US$58.1 million in 2013 compared to the revised estimate of US$39.5 million). The main reasons for this are:

- Foreign exchange rate change since 2013. In 2013, the A$/US$ exchange rate was 1:1.04 and the revised rate used is 1 A$= 0.73 US$ which represents a 30% reduction in all costs with an Australian dollar basis, including equipment prices and engineering costs.
- Concrete supply reduced from a supply rate of US$384/$m^3$ to current estimate of US$180/$m^3$ (including rebar).
- Revised budget quotations were solicited from the major equipment vendors and equipment costs were revised based on recent quotation and current level of competition in the marketplace.
- Re-estimation of the EPCM costs by Monument. In 2013, the EPCM was estimated using a flat 26% of the estimated direct cost for the EPCM. This has been revised to 15% based on current levels of competition in the marketplace.

Other capital expenditure

Monument estimated the capital in *"FY2017_BudgetSummary_V4.xlsx"* as sustaining capital for the process and operation for the 2017 budget, as summarised in Table 21.3.

Table 1.13 Other capital expenditure

2017 budget item	RM
Processing equipment	
Crushing	310,200
Mill and gravity	540,000
CIL	1,045,000
Operation equipment	
Laboratory	35,000
Workshop	18,500
Total processing capital	**1,948,700**

Snowden used a forex conversion of RM:US$ as 1.00:0.24 to estimate this cost as $0.5 million per annum.

Environmental compliance costs were provided by Monument for the Selinsing operation as:

- Closure cost: $3.7 million
- Rehabilitation cost: $2.3 million.

No other capital expenditure was provided by Monument for the Selinsing operation.

1.10.3 Operating costs

Process operating costs

Operating costs for campaign treatment of oxide material going forward will be heavily dependent on the stockpile gold grade. Historical cost data review as provided in Section 17 shows that operating cost is dependent on gold grade as expected. For the oxides and sulphides, the following operating costs provided previously in Table 1.14 are recommended by Snowden.

Table 1.14 Operating costs for oxide and sulphide material treated (after Phase IV expansion)

Material treated	Deposit	Gold grade (g/t Au)	Process operating cost (US$/t)
Oxide#	Both	<1.0	7.11^
		1.0 to 1.5	7.11^
		1.5 to 2.5	7.11^
		>2.5	7.11^
Transition*	Selinsing	All	20.00
	Buffalo Reef	All	20.00
Fresh/Sulphides	Selinsing	All	19.56^
	Buffalo Reef	All	19.56^

** Reagent costs should be slightly higher for transitional material especially during flotation*

 # From Figure 13.5 in Section 13

^ Taken into account historical data (about 1 Mt/a processed) from Table 17.2 as well as current 2016 year to date oxide cost of $6.83/t

Mining operating costs

The mining operating costs were provided by Monument for the project from the mining contract and these were compared to the budget figures (Y2017_BudgetSummary_V4.xlsx) to build up the total mining cost.

Table 1.15 2017 Selinsing budget figures

Category	Item	Total cost MR	Total US$ unit cost	% of cost
Services	Drill, blast and haulage	22,797,105	1.61	89.2
	Assay and laboratory	861,559	0.06	3.4
Mine maintenance	Supplies	86,215	0.01	0.3
	Pit maintenance	591,359	0.04	2.3
Labour		1,214,215	0.09	4.8
Total mining cost		**25,550,453**		**100.0**
Cost per tonne mined		7.51	**1.80**	

The life of mine (LOM) mining services cost provided by Monument is expected to average:

- $1.89/t moved for Selinsing
- $1.71/t moved for Buffalo Reef.

There is provision in the contractor schedule of rates for pit depth and horizontal haulage incremental cost change as a trucking cost increment cost of RM0.03 per BCM per 5 m depth increment, and RM0.0024/t per horizontal metre as in the current contract with Minetech. In addition to this, there is a provision for a cost of $0.13/t moved to cover grade control, supplies, pit maintenance (pumping) and owner labour. The total mining cost is estimated by Snowden to be:

- $2.08/t moved for Selinsing
- $1.90/t moved for Buffalo Reef.

A stockpile cost of $0.79/t moved was considered by Snowden for stockpile extraction.

1.10.4 Administration operating costs

Selinsing provided costs from the 2017 budget (FY2017_BudgetSummary_V4.xlsx) for the administration of the operation.

Table 1.16 Administration operating costs

Item	Total cost MR	Total US$ unit cost	% of cost
Site maintenance and development	3,374,121	0.73	25.6
Security	1,872,327	0.40	14.2
Health and safety	488,908	0.11	3.7
Environment	431,314	0.09	3.3
Site support	7,011,435	1.51	53.2
Total administration cost	**13,178,105**		**100.0**
Cost per tonne processed	13.14	2.84	

A cost of $3.15/t of ore processed was used for the administration costs. This cost does not consider corporate overheads.

1.11 Economic analysis

1.11.1 Project economic results

The following Table 1.17 and Table 1.18 provide the headline results before and after taxation for a gold price of $1,255/oz of gold (base case).

Table 1.17 Project economic model headline results after taxation

Item	Unit	Value at $1,255 /oz Au
Net cash flow	$ M	37.2
NPV_8	$ M	23.1
IRR	%	34.8

Table 1.18 Project economic model headline results before taxation

Item	Unit	Value at $1,255 /oz Au
Net cash flow	$ M	47.7
NPV_8	$ M	31.4
IRR	%	43.5

1.11.2 Key performance indicators

The Project LOM key performance indicators (KPIs) after taxation are presented in Table 22.4 below.

Table 1.19 Selinsing Project KPIs after taxation

Item	Unit	Value
Total value of product sold	$ M	285.2
Cash cost	$/oz	830.1
Total cost	$/oz	1,099.0
Production year payback	year	2.6
Brooke Hunt methodology C1 cost	$/oz	767.4

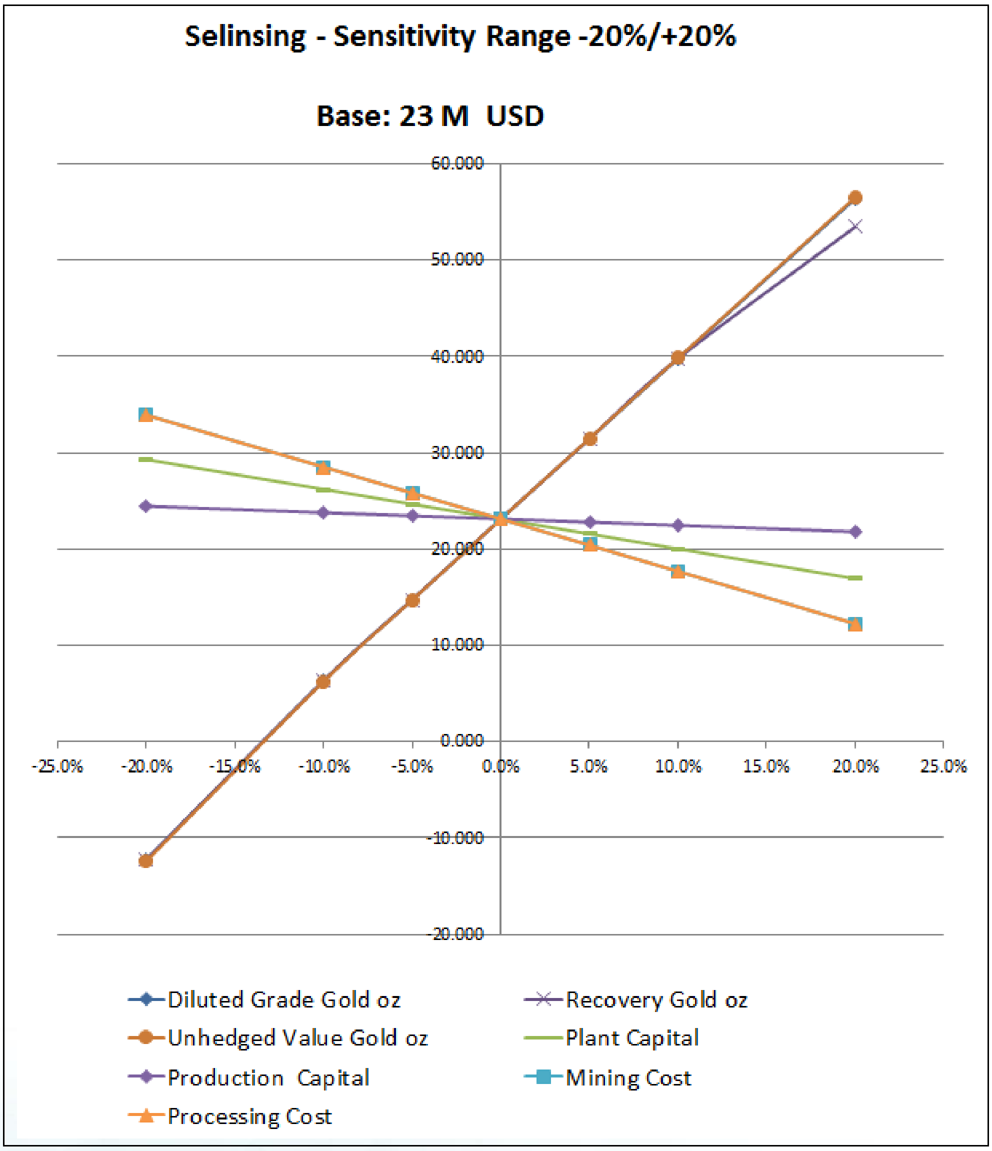
1.12 Sensitivity analysis

The economic cash flow model was used to prepare a sensitivity analysis for the NPV_8 for the Project after taxation. The sensitivity analysis was completed on the following variables:

- Grade of gold
- Recovery of gold
- Price of gold
- Plant capital
- Production capital
- Mining cost
- Processing cost.

The sensitivity chart is shown in Figure 1.7 and it covers a range of variable changes from -20% to +20%. As the variable line increases towards the vertical it indicates the Project is more sensitive to changes of the variable.

Figure 1.7 Cash flow sensitivity graph



A breakeven analysis after taxation was undertaken on the gold price and gold grade for net cash flow. This analysis was conducted on the sensitivity analysis data and provides the gold price which will bring the net cash flow to $0. The results of this analysis are presented in Table 1.20.

Table 1.20 Breakeven analysis after taxation

Item	Unit	Breakeven
Gold price	$/oz Au	1,082
Gold grade	g/t Au	1.20

1.13 Project risks

The Selinsing Gold Mine Project has a long history of successful operation since 2009. In this respect the risks of community disengagement, and environmental non-compliance is low. Political and sovereign risk is considered low in Malaysia.

There is a reasonable production reconciliation comparing the performance of resource models and mining models to recovered gold. The risk of technical malfunction of the resource models is low. There is some risk for the future processing technique using the bio-oxidation method as there is no production history of this method at Selinsing.

A statistical analysis has been carried out on the collected conditional simulation values to determine a real-life net present value (NPV). In addition, because there are a large number of values collected, a probability analysis could also be applied which will give the range of possible outcomes for the Project at different confidence levels.

A statistical analysis for a sample of a population is valid for a sample greater than 80 values. The conditional simulation was run for 10,000 iterations and the results for the statistical analysis are provided in Table 1.21.

Table 1.21 Statistical and probability analysis for the Selinsing Project

Item	Unit	Value	Value
Mean NPV_8	$M		20.7
Standard error	$M		0.115
Standard deviation	$M		11.5
Minimum value	$M		-21.3
Maximum value	$M		61.7
Risk index (CV)	%		56%
Range at 99.7% confidence	$M	-13.3	54.6
Range at 95.0% confidence	$M	-1.9	43.2
Probability of > value	$M	-2.0	98%
Probability of > value	$M	13.0	75%
Probability of > value	$M	20.7	50%
Probability of > value	$M	28.0	26%
Probability of > value	$M	43.0	3%

From the spider chart in Figure 1.7, the Project is most at risk to a lowering of the gold price or the gold grade. The mining and process cost increases are considered to be the next highest risk to the Project when percentage changes are considered. The CV is high, however the risk to the Project failing is considered medium to low.

1.14 Conclusions and recommendations

1.14.1 Conclusions

Monument has demonstrated through metallurgical testing that processing refractory ore through a CIL plant with bio-oxidation pre-treatment as a viable recovery method.

The area has been explored over a number of years with predominately RC drilling and some limited diamond drilling and 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 has been 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 have also been estimated

The authors of this report conclude that a viable project processing the remaining oxide and sulphide ore has been identified at the PFS level and that future studies as feasibility studies be commenced to identify the final process for extraction of gold from the sulphide ore and increase the accuracy of the economic estimate of the project.

1.14.2 Recommendations

The 2016 PFS for the Selinsing Gold Mine Project has identified key areas to optimise and increase the confidence of the Project by:

- Sourcing more sulphide ore to utilise the life of the plant which is currently not optimal
- Investigate metallurgical testwork data from the INTEC process to identify suitable gold recovery and process costs
- Progression of environmental licence permits for the Phase IV expansion
- Update capital and operating cost inputs to a feasibility study level.

2 INTRODUCTION

2.1 Overview

This Technical Report has been prepared by Snowden for 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 9 November 2016 *"Monument Updates Selinsing and Buffalo Reef Resources and Reserves"* (Release #21- 2016), disclosing an updated Mineral Resource and Mineral Reserve for the project. The Mineral Resource was updated by extensional and infill drilling of the deposits, and the results of a PFS that introduces a bio-oxidation leach process strategy for the recovery of the refractory sulphide ores at deposits.

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

The Qualified Persons for preparation of the report and the status of project site visits are shown in Table 2.1.

The responsibilities of each author are provided in Table 2.1.

Table 2.1 Responsibilities of each co-author

Author	Responsible for section(s)	Site visit
Frank Blanchfield	4, 5, 15, 16, 18, 19, 20, 21, 22, 23, 24, 25, 26, 27	Selinsing and Buffalo Reef areas – February 2016
John Graindorge	6, 7, 8, 9, 10, 11, 12, 14, 16	Selinsing and surrounding tenements – August 2010
Dr Leon Lorenzen	13, 17	

John Graindorge is an employee of Snowden, an independent Qualified Person for Buffalo Reef and Selinsing Mineral Resource estimates. Frank Blanchfield is an employee of Snowden, an independent Qualified Person for Buffalo Reef and Selinsing Mineral Reserve estimates. Dr Leon Lorenzen is an employee of Snowden, an independent Qualified Person for the metallurgical aspects of the PFS, including the testwork conducted relating to the Selinsing plant design.

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

The effective date of the report is November 2016.

2.2 Sources of information

The author of this Technical Report is Mr Frank Blanchfield, an employee of Snowden, who is a Qualified Person and accepts the responsibility for the "currency and completeness" for all items in the report.

Snowden has based this PFS on the engineering and geological detail provided in the following studies that were undertaken in 2013 and subsequent investigations in 2015 and 2016.

2.2.1 Applicable documents

Snowden:

- Mineral Resource reports and emails
- Mineral Reserve reports and emails.

Monument:

- NI 43-101 Technical Report: Selinsing Gold Mine and Buffalo Reef Project Expansion – August 2012.

Lycopodium Minerals Pty Ltd:

- Selinsing Phase IV – PFS Capex and Opex Revision – 3193STY001 – September 2016.

3 RELIANCE ON OTHER EXPERTS

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

- Information available to Snowden 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.

The results and opinions expressed in this report are based on the author's field observations and assessment of the technical data supplied by Monument. The author has reviewed all the information provided by Monument and believes it to be reliable.

3.1 Limited disclaimer

Snowden has not researched property title or mineral rights for the Selinsing Gold Project and expresses no opinion as to the ownership status of the property. As such, the description of the property, and ownership thereof, as set out in Item 4 in this technical report, is provided for general information purposes only.

Additionally, Snowden is reliant on reports, opinions, or statements of other experts who are not Qualified Persons, or on information provided by the issuer, concerning legal, political, environmental, or tax matters relevant to the technical report. In particular, Snowden has relied on a desktop review of the Lycopodium report concerning a PFS for integration of a bio-oxidation processing circuit to the existing oxide circuit to recover gold from the Selinsing and Buffalo Reef sulphide deposits.

Where indicated in the report, the source of the information relied upon, including the date, title, and author of any report, opinion, or statement that the author is dependent on is subject to this disclaimer. Where the author relies on such information, the reliance is 100%. The portions of the technical report to which the disclaimer applies are provided in Table 3.1.

Table 3.1 Qualified Person reliance on other experts

Item	Description
Item 4	Property description
Item 5	Accessibility
Item 6	History
Item 13	Mineral processing and metallurgical testing
Item 17	Recovery methods
Item 18	Project infrastructure
Item 19	Market studies and contracts
Item 20	Environmental studies, permitting, and social or community impact

Dr Leon Lorenzen is the Qualified Person for the Items 13 and 17 and Frank Blanchfield is the Qualified Person for the other items of the report in Table 3.1 and they are reliant on the report, opinion, or statements of another expert who is not a Qualified Person, or on information provided by Monument as the issuer, concerning the matters relating to these items in the report.

Snowden is reliant on Monument for the provision of Environmental Studies, Permitting and Social or Community Impact studies and the status of these items. An environmental scientist independent of the issuer was used to review the documents received from Monument. It is the opinion of Sophie Gaunt, who is an employee of Snowden with 20 years of experience in environmental approvals in the mining industry, that the environmental documents received from Monument are current and complete for this technical report.

The results and opinions expressed in this report are based on the author's field observations and assessment of the technical data supplied by Monument. The author has reviewed all the information provided by Monument and believes it to be reliable. Except for the purposes legislated under provincial securities laws, any use of this report by any third party is at that party's sole risk.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Area

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

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

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, three 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

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 hour’s 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 4.1 Selinsing Gold Mine and Buffalo Reef Project – Pahang State, Malaysia

4.3 Type of mineral tenure

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

- Mining leases
- Mining certificates
- Exploration access as Felda blocks.

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 the Company's title. Property title may be subject to unregistered prior agreements or transfers and title may be affected by undetected defect. To the best of the Company'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 the Company'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, Polar Potential Sdn Bhd, Monument acquired 100% of the Selinsing Gold Project with two mining concessions (MC1/124 and ML5/2000), together with a 100% interest in Able Return Sdn Bhd, a Malaysian company holding Malaysian Pioneer status which among other benefits provides a five-year tax break from Malaysian Federal and other taxes. At the time of this report, a third mining lease (ML7/2000) was added to the Selinsing land status controlled by Monument.

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 approximately 4.2 km of controlled property along the regional gold trend.

4.3.4 Summary of mineral tenure

The Selinsing Property encompasses the Selinsing and Buffalo Reef 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 will issue a Proprietary Mining Lease (PML) on the settler's private landholding (currently managed by Felda, acting as manager on the land). 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 property and obligations for mineral tenure retention are shown in Table 4.1.

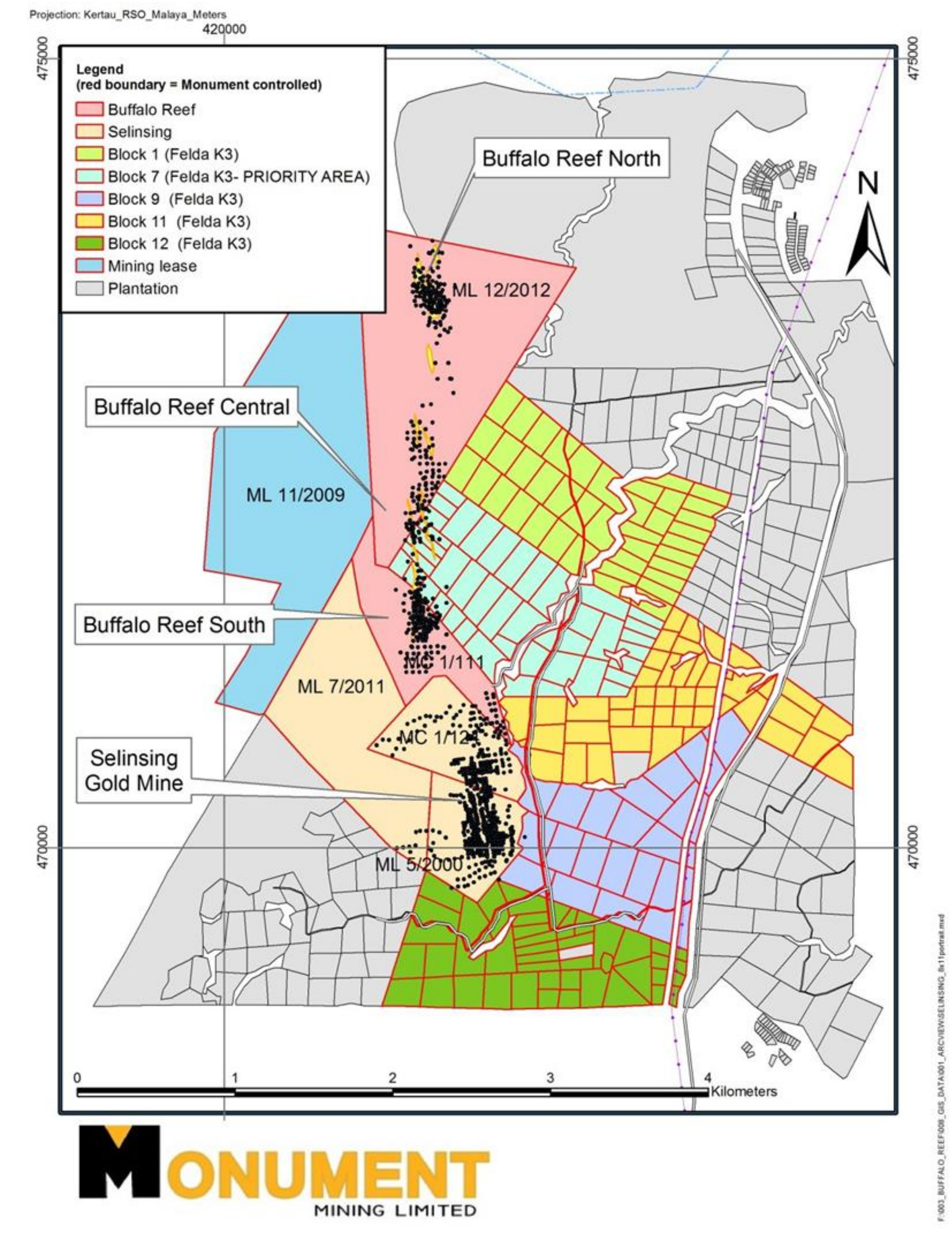
Table 4.1 Mineral tenure retention and expiry dates obligations

Deposit	Lease	Area (ha)	Expiry
Selinsing active	MC1/124	40.3	27/03/2017
	ML7/2011	91.5	28/03/2021
	ML5/2000	32.3	04/05/2018
Subtotal		**164.1**	
Selinsing other	ML11/2009	186.9	28/06/2019
Buffalo Reef	ML12/2012	157	11/02/2021
	MC1/111	43	31/10/2016
Subtotal		**200**	
Felda	Felda Block 7	107.7	31/12/2016
	Felda Block 9	62.3	31/12/2016
	Felda Block 11	34	31/12/2016
	Felda Block 12	34	31/12/2016
Subtotal		**238**	
TOTAL		**789**	

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 (ongoing).

Figure 4.2 Location of property boundaries



4.4 Permit acquisition for proposed work

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

4.5 Royalties, back-in rights, payments, agreements, encumbrances

All mining leases in Malaysia carry a 5% royalty payable to the Malaysian 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)). Tenements granted to Monument have no encumbrances or liabilities with them at the time of this report.

4.6 Environmental liabilities

There are no environmental liabilities known to Monument.

4.7 Risks that may affect access

There are no identified risks that may affect access.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Topography, elevation and vegetation

The Selinsing and Buffalo Reef properties are approximately 500 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 Access

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 tracks 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 Proximity to population centre and transport

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.

5.4 Climate and length of operating season

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 millimetres (mm) per annum. Peak rainfall periods are September through to December and March through to May.

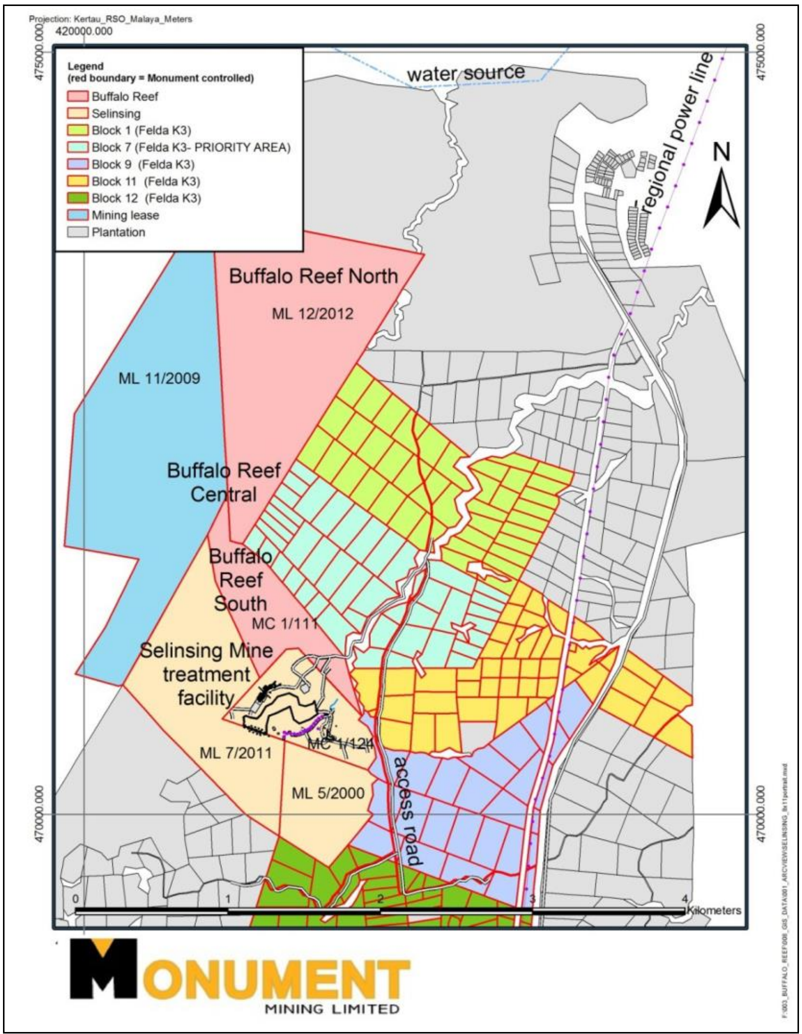
5.5 Surface rights

The current surface rights pertaining to the Selinsing Property are sufficient for all activities relating to the mining activities.

5.6 Infrastructure

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



5.6.1 Power

A 33 KV national grid power line runs past the leases. Power is allocated by the electricity provider and the agreed offtake is currently 3.8 MW. The new expected demand for the plant with a bio-oxidation plant is 10.7 MW.

5.6.2 Water

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

5.6.3 Mining personnel

Mine personnel consist of expatriates and in-country professionals. Labour employment covenants require 50% of all employees to be Bumiputra (ongoing). 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.

5.6.4 Tailings storage areas

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.

5.6.5 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 of 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.

5.6.6 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.

Further information regarding the plant is found in Section 17.

6 HISTORY

6.1 Prior ownership and ownership changes

6.1.1 Selinsing

Historic 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, 246,409 oz of gold was produced as of the end of June 2016. Only 18,155 oz of gold was produced in the financial year ending June 2016.

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 Dore. 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 in 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.

A historical resource at the Buffalo Reef deposit (October 2006) estimated 2.5 Mt grading at 2.26 g/t Au, for a total of 185,300 oz of contained gold. However, this historical estimate is not considered a Mineral Resource or Mineral Reserve as defined under sections 1.2 and 1.3 of NI 43-101.

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 through to 2007, when initial exploration commenced.

The most current drilling 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). Previous development work is discussed in Section 6.1 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 Summary of the Selinsing Property production

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

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).

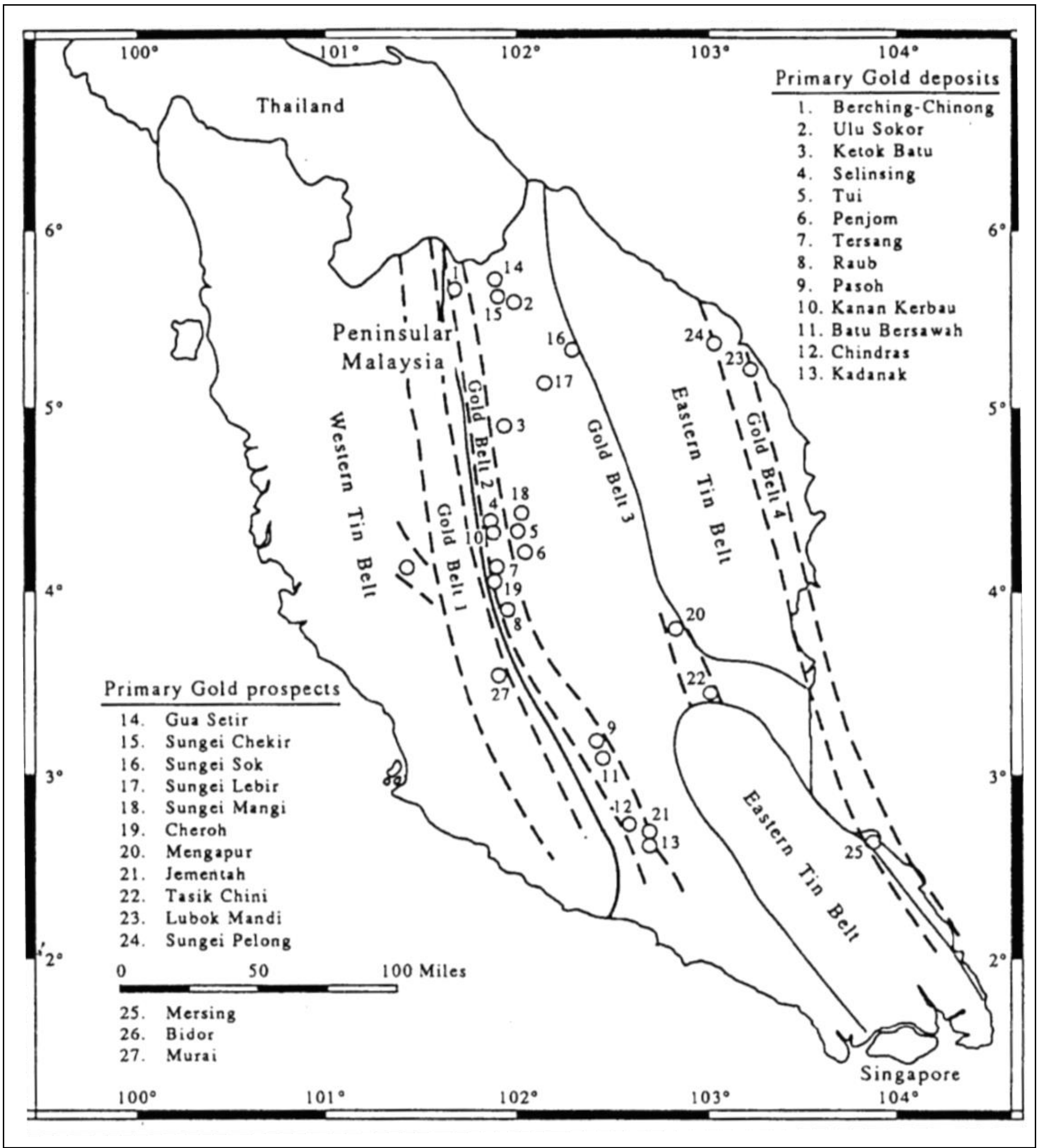
Peninsular Malaysia can be divided into two main regional blocks separated by the Raub-Bentong Line which is a major suture zone. This shear zone divides the Sibumasu Block (Western Block) in the west from the Manabor Block (Eastern Block) in the east (Yeap, 1993). By the late Carboniferous, the Western 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 reinitiated 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

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.

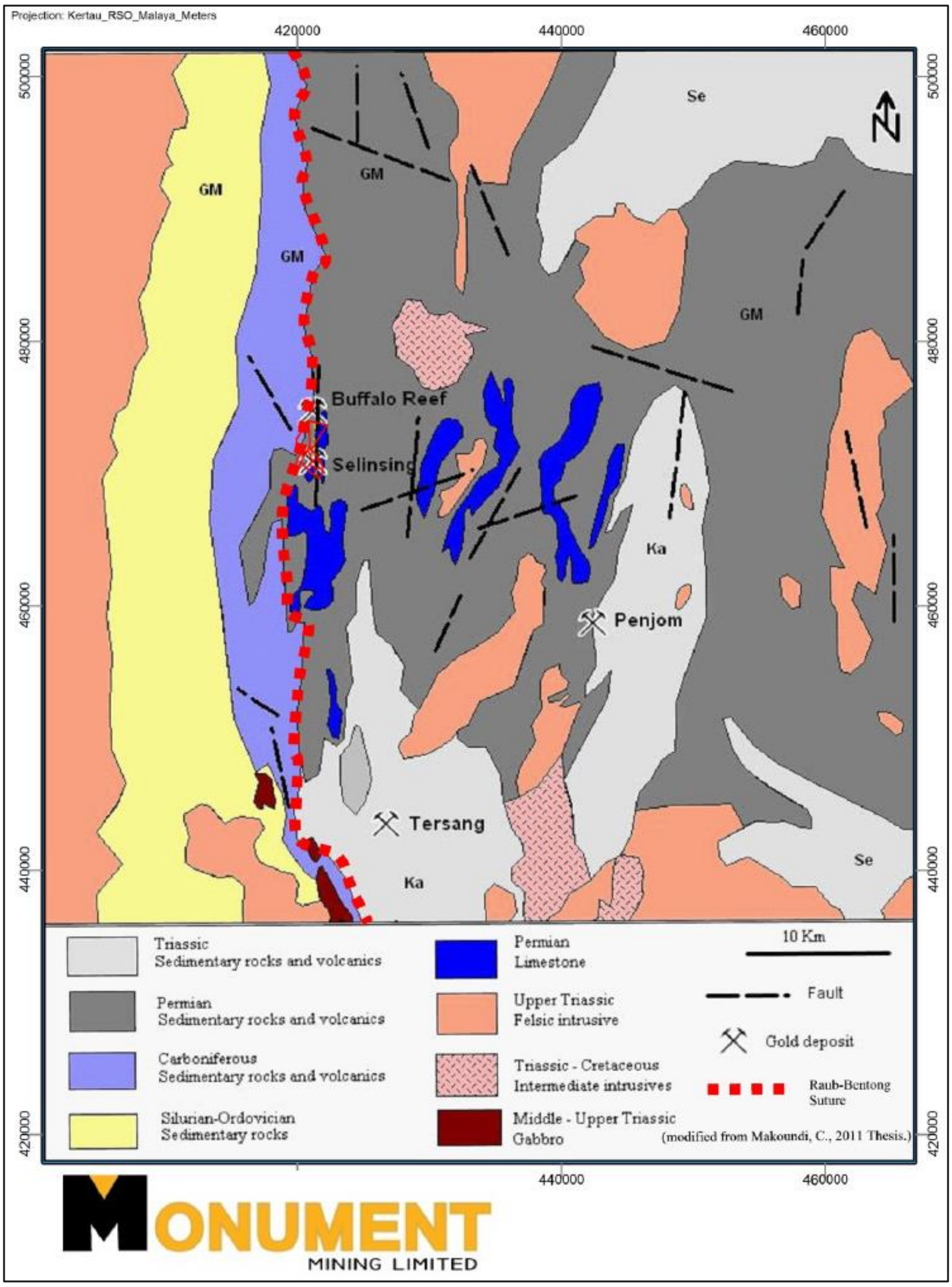
The true thickness of the mine sequence is not well understood. One interpretation is that the mine sequence has a true thickness of about 200 m; however, the footwall contact is not well defined and it is difficult to distinguish between the mine sequences in the field due to the fine-grained nature 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.

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 is associated with these rock types as well as intensive replacement by quartz and calcite gangue minerals.

Figure 7.2 Local geology of the Selinsing area



Mineralisation

The gold mineralisation at Selinsing is hosted within a shear zone that strikes at 350° 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. Pressure and temperature studies on fluid inclusions in quartz from veins suggest that the mineralisation formed at a temperature in the order of 200°C to 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° to 75° to the east and strikes towards a bearing of 330° to 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% to 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 Selinsing and Buffalo Reef primarily comprises drilling, as discussed in Section 10 below. Exploration prior to June 2012 is documented in the May 2013 NI 43-101 Technical Report on the property (Odell *et al.,* 2013). A summary of the exploration, other than drilling, is provided below.

9.1 Pre-2012

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

Damar and Avocet excavated some 139 trenches at Buffalo Reef between 1993 and 2003, totalling approximately 6,800 m (Flindell *et al.,* 2003), however the 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 by 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).

Snowden notes that 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. As such, some of the initial exploration data has been lost.

9.2 Post June 2012

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 Felda area. Monument successfully negotiated with the local landowners to gain access to the Felda settler land between the Buffalo Reef South and Central deposits, enabling drilling to commence in May 2013. The drilling is discussed further in Section 10 below.

As part of the ongoing exploration work undertaken 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. Additionally, pit mapping and sporadic sampling was undertaken to the west of Selinsing Pit 4, where five groups of 1 m horizontal channel samples were collected. The exploration channel and trench samples are primarily used for the planning and targeting of exploration drilling.

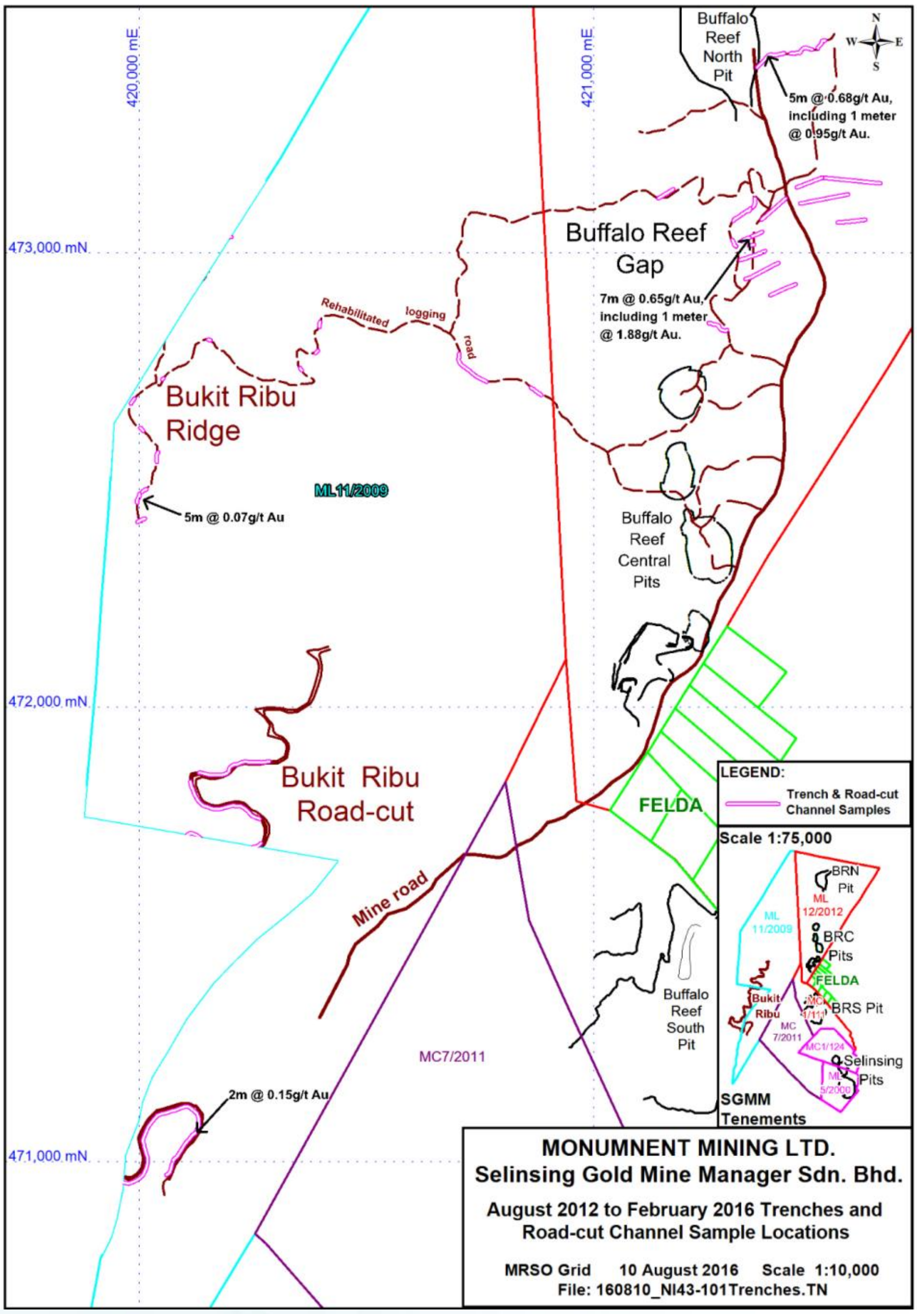
At the Bukit Ribu road-cut, a total of 2,194 m of road side slopes were sampled, with 222 samples collected from three cleared slopes along the road cut. Initially, 5 m reconnaissance-style channel samples were collected, which were later re-sampled at 1 m intervals where anomalous gold values of >0.1 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.

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



9.3 Trench and channel sampling method

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 GPS coordinates for samples recorded. Sample points were later surveyed by the Monument mine surveying team using DGPS 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 4 inch wide by 1 inch 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).

9.4 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 DGPS. 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.

10 DRILLING

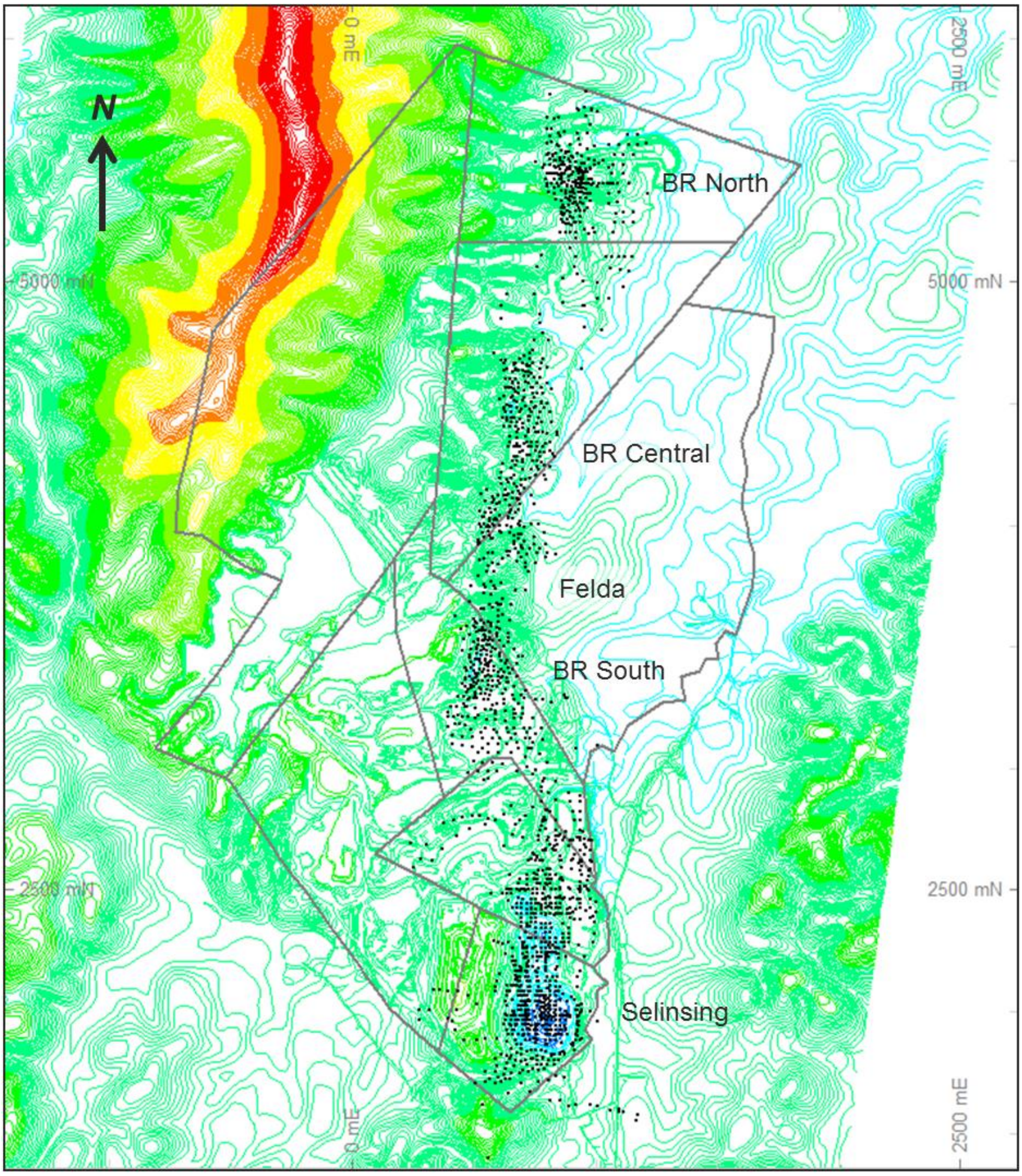
Drillholes with assays received up to the 24 February 2016 for approximately 145,217 m of drilling were completed across the Selinsing and Buffalo Reef deposits, comprising predominately RC (50.9%) and diamond drilling (DD) (41.5%). RC and diamond drilling account for approximately 92% of the drilling 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	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 with 2 m topographic contours



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 Pit 4 and 5 and Selinsing Pit 5 and 6 areas. 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 5 and Selinsing Pit 5 and 6. This program comprised 19 diamond drillholes for 2,596.6 m at Selinsing Pit 5 and 6, and nine diamond holes for 1,441 m at Selinsing Pit 4 and 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 5 for 1,269.6 m and nine holes at Selinsing Pit 5 and 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 5 and the other at Selinsing Pit 5 and 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 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.

The drilling conducted at Selinsing between 2012 and 2015 is summarised in Table 10.2.

Table 10.2 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

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 6 and Felda. Selinsing Pit 5 and 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 drillholes of 70 m each, which were drilled at the Selinsing Pit 5 and 6. In 2013, a 100 m diamond hole was drilled at Selinsing Pit 5 and 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.

Table 10.3 Drilling summary – Buffalo Reef 2012 to 2015

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

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

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 as passed onto 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

No downhole surveys were conducted for the Damar, Avocet or Monument RC drilling at Buffalo Reef prior to 2011. Damar (2002) mentions 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. Snowden notes that the downhole surveying methodology for the diamond drillholes is not stated and as such cannot comment on the accuracy of the measurement technique.

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 not 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 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. Monument note that 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.

Monument indicated that 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

Core recovery for diamond holes drilled by Monument at Selinsing averages approximately 94%. This figure includes intervals from the drillhole 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.

In Snowden’s opinion, 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. However, Monument indicated that sample recovery has not been measured for any of the RC drilling at Selinsing or Buffalo Reef and as such, Snowden is unable to comment on the recovery of the RC drilling.

10.5 Geological logging

Industry standard logging conventions have been used to record information from the drilling at both Selinsing and Buffalo Reef, with the logging procedures 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 may be 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

Recent diamond drilling at Selinsing and Buffalo Reef was undertaken by Monument, using two Desco SP6500SA rigs, which are owned by Monument. Diamond drilling included both PQ 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 being 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 RC

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 drillhole 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 (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 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 drillholes 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 Felda 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 amount of wet samples is not recorded for the Damar drilling.

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

10.7 Author's opinion on the adequacy of sampling procedures

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, Snowden recommends 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

A number of laboratories have been used to prepare and analyse samples from Selinsing and Buffalo Reef. The primary laboratory is SGS (Malaysia) in Mengapur.

11.1 Laboratories used

A number of 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 estimation 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.

Samples submitted to the SGS Mengapur laboratory were analysed for gold, arsenic, silver and antimony. Gold was analysed by fire assay using a 50 g charge with an 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 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 P90 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 P90 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 P90 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 QC.
    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 below.

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 P95 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 13).

Prior to 2007, TRA resubmitted samples from the Selinsing Phase 1 drilling program to the AssayCorp laboratory in Kutching 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 P90 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 P95 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 (ICPMS).

11.3 Sample security

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

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 facility. The core storage facility is maintained by Monument staff and kept secure (Figure 11.1).

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, covering the period June 2012 to February 2016.

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 February 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. Monument indicated that the CRM is currently under investigation and may have been mislabelled.

Control charts plotted for the CRMs show that reasonable accuracy was achieved by the laboratories with a slight negative bias (<5%) present. 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.

Example control charts are provided for CRMs G910_4 and G901_7 in Figure 11.2 and Figure 11.3 respectively.

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)

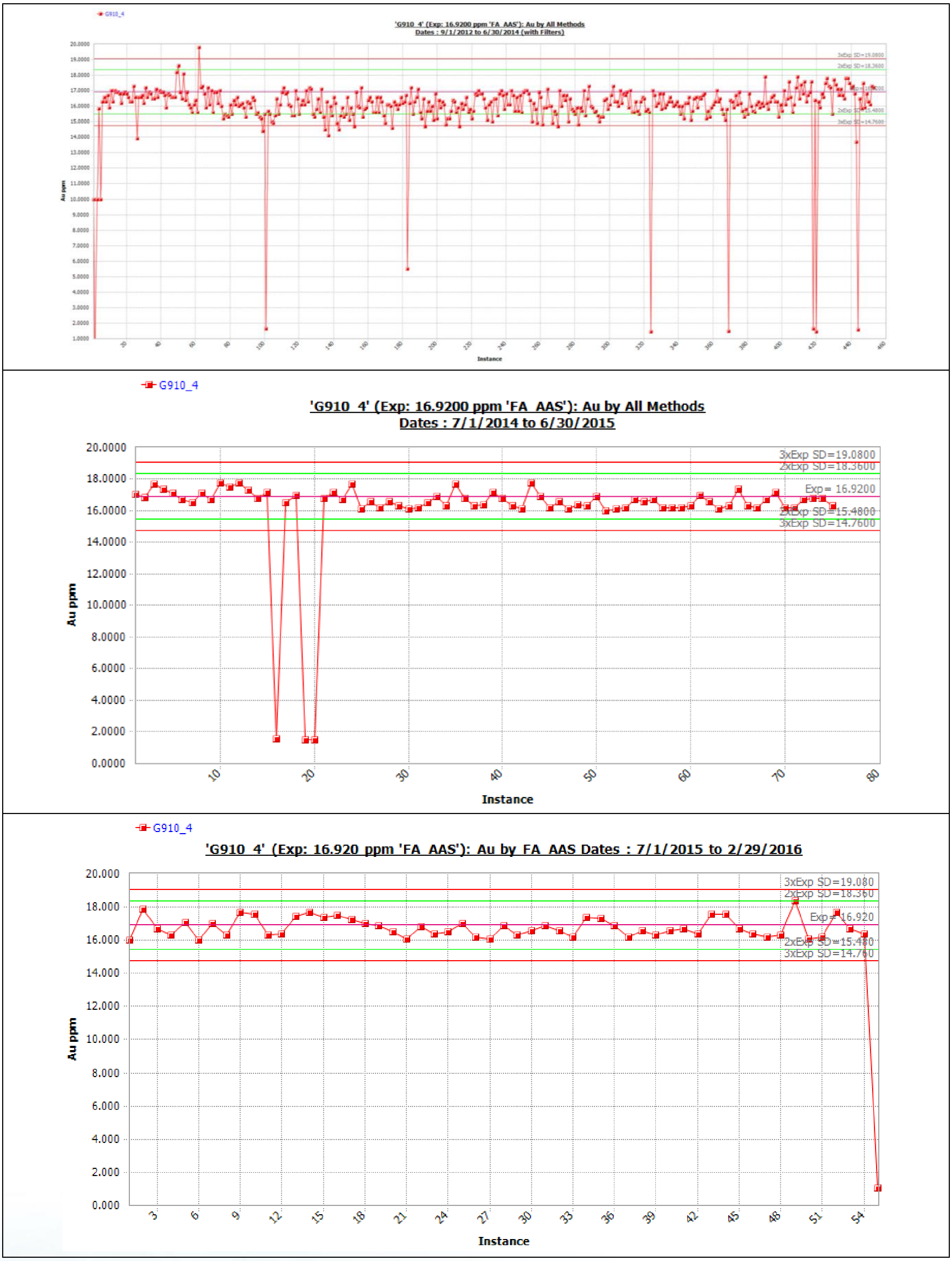
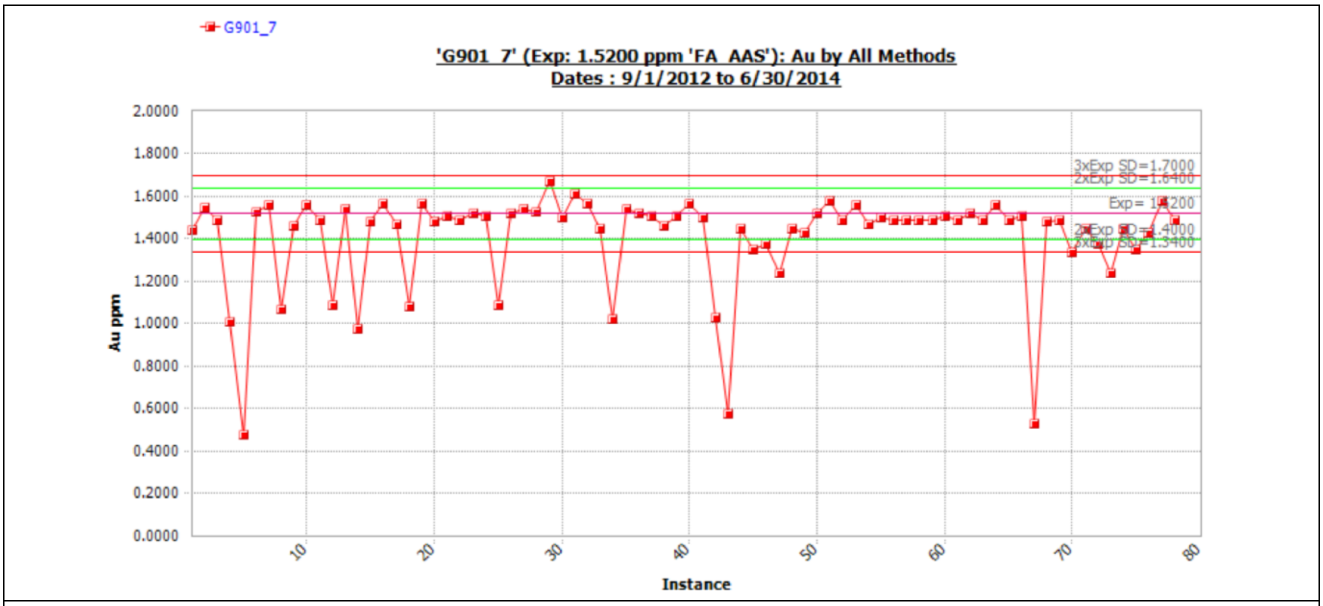
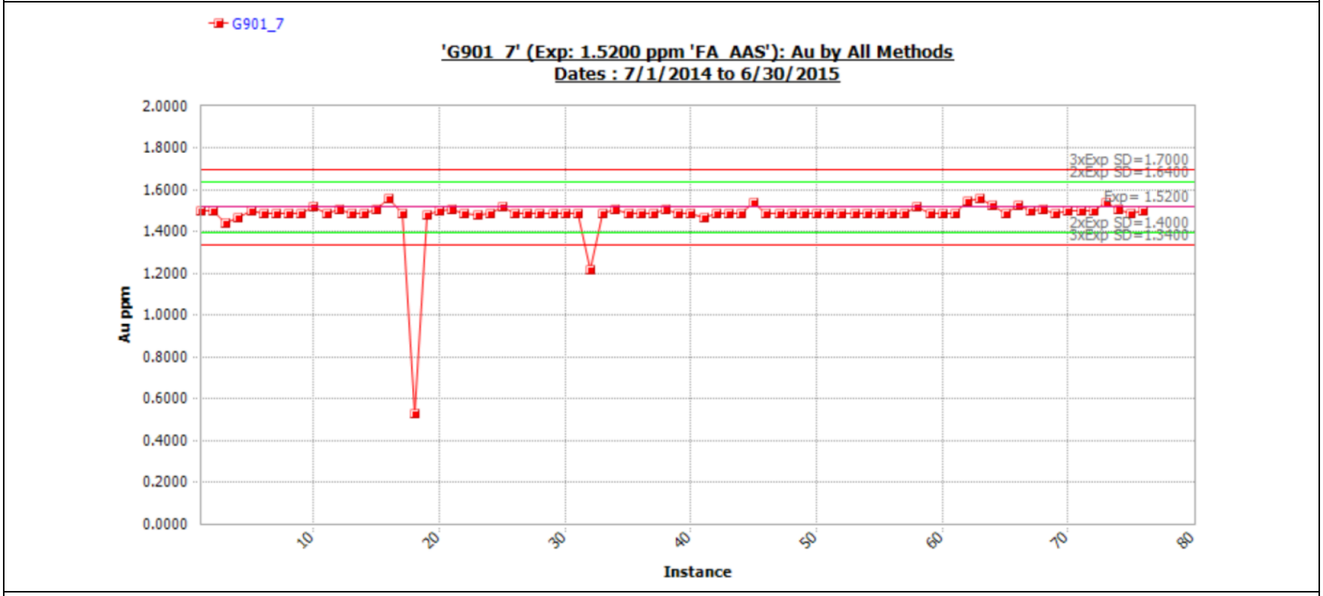
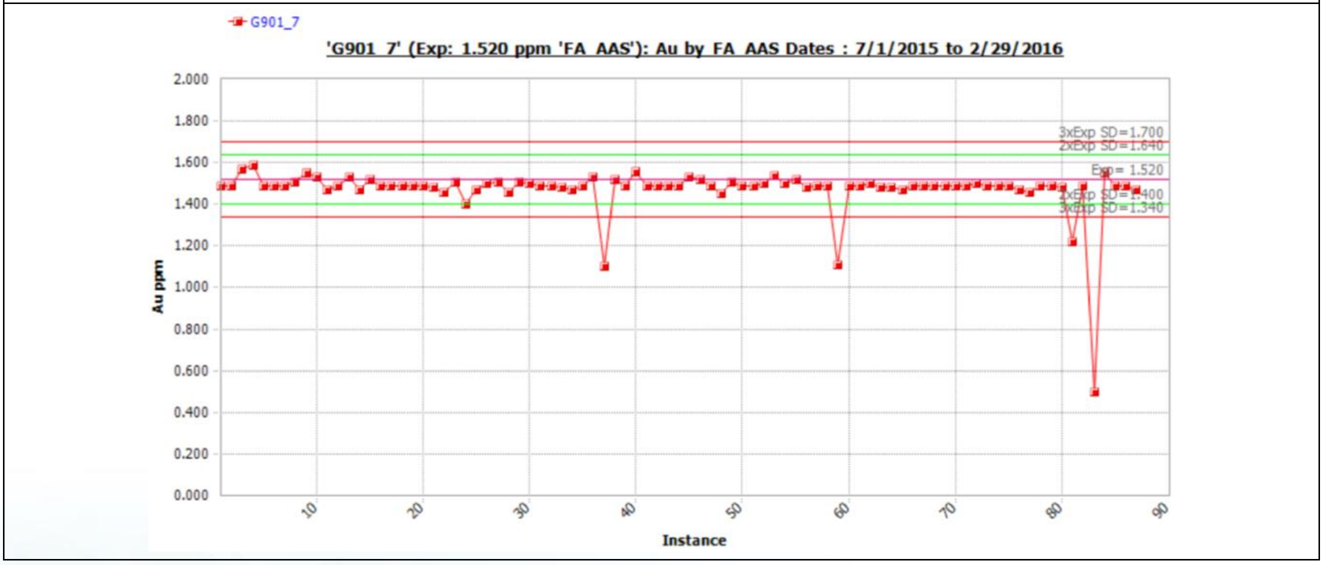


Figure 11.3 Control charts for CRM G901_7 (1.52 g/t Au) for 2012 to 2014 (top), 2014 to 2015 (middle) and 2015 to 2016 (bottom)





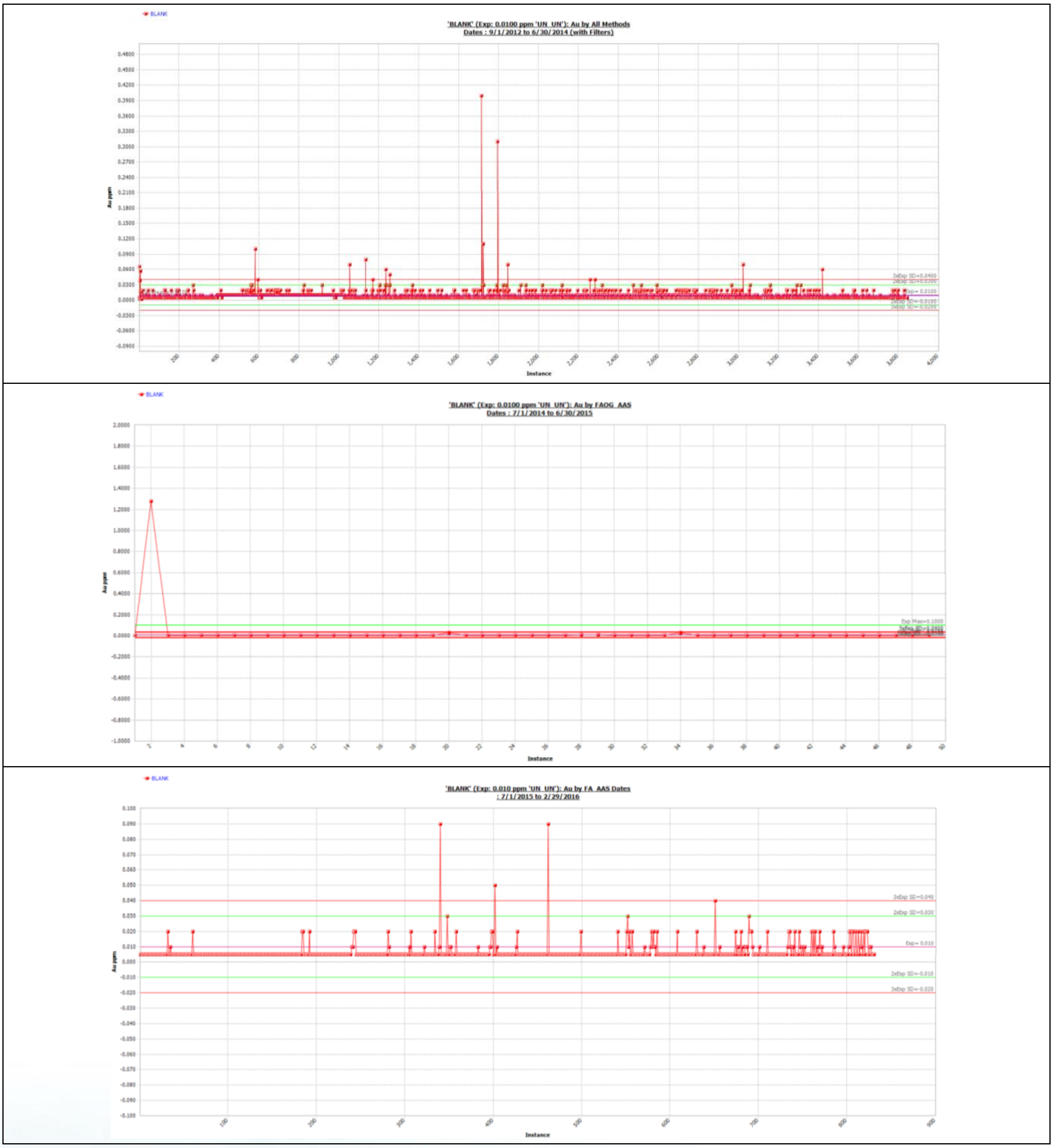


Blanks

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 non-certified.

The majority of the blanks returned a value close to the detection limit (0.01 g/t Au). In Snowden's opinion, the blanks generally report within an acceptable range and show that contamination of samples between 2012 and 2016 is negligible.

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

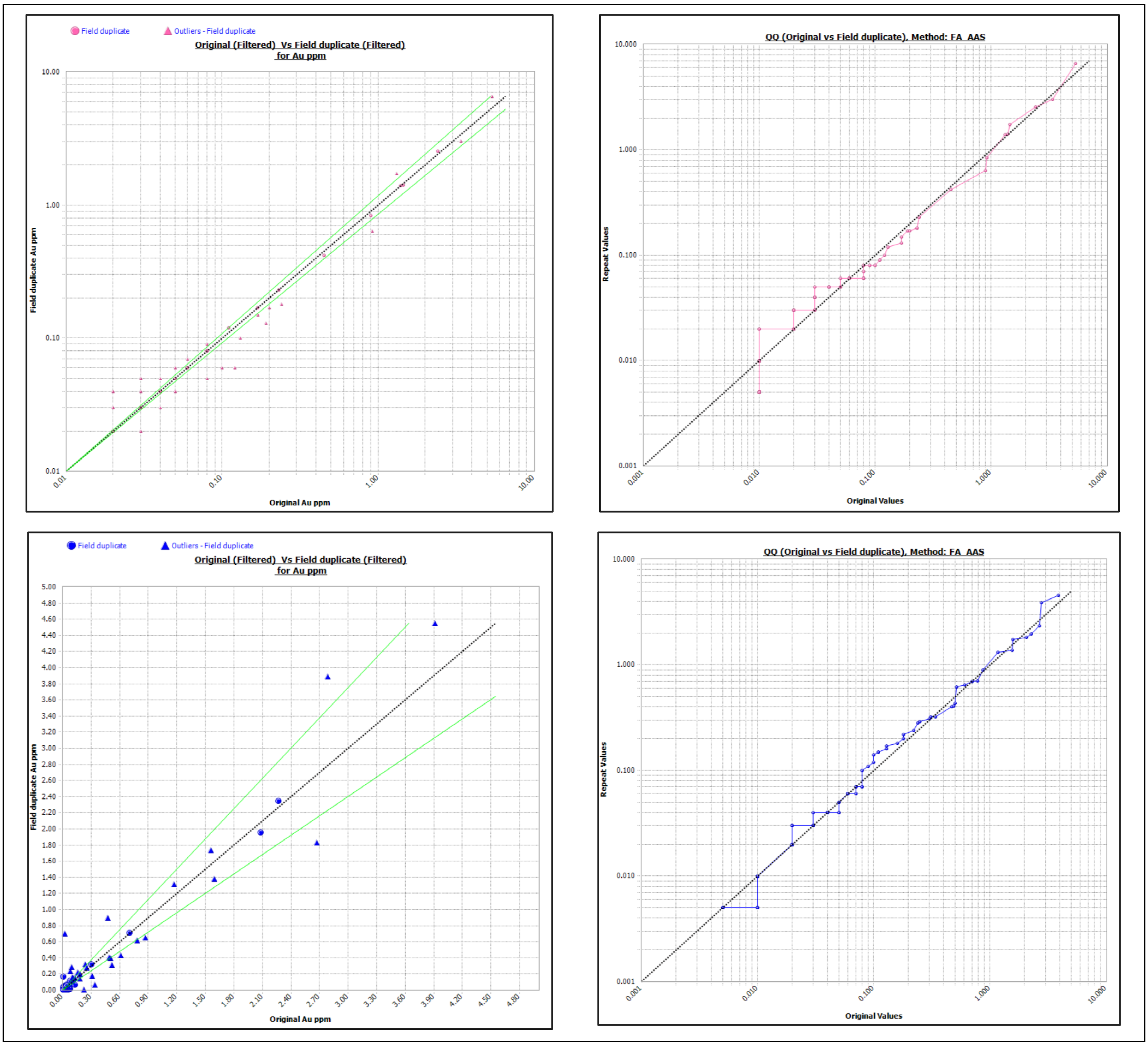


Field duplicates

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 show that the precision (i.e. repeatability), whilst not ideal is a 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.

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 February 2016. Snowden understands that Monument intend to submit duplicate samples, for samples after March 2015, for umpire analysis in October 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

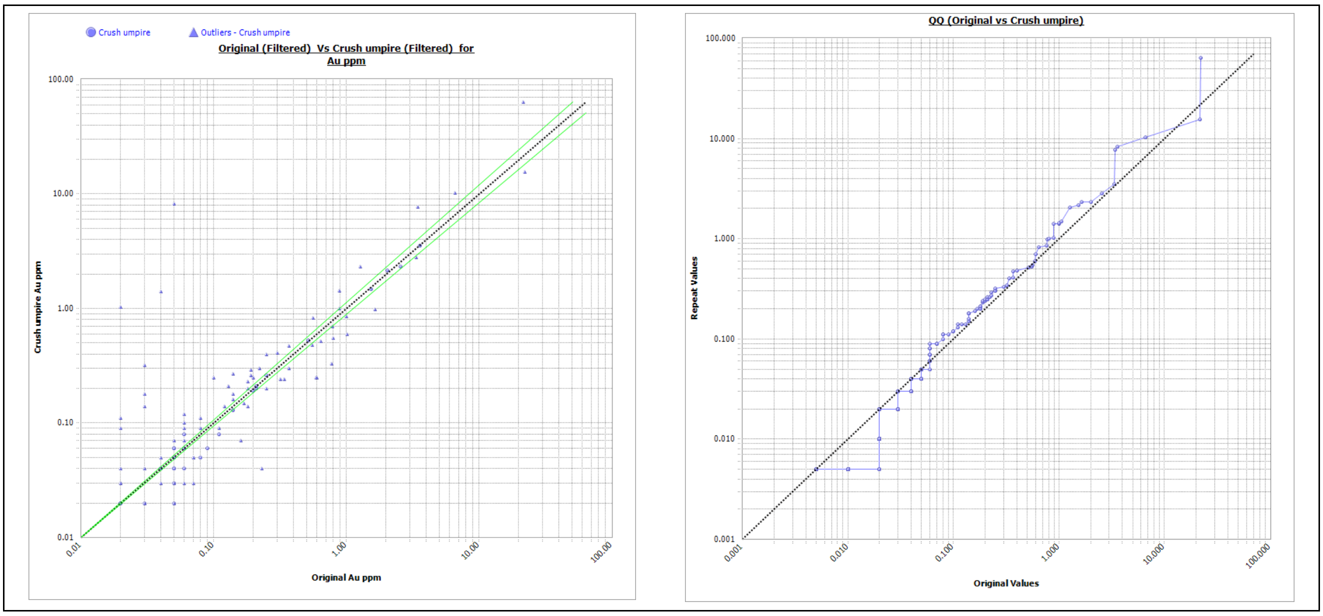
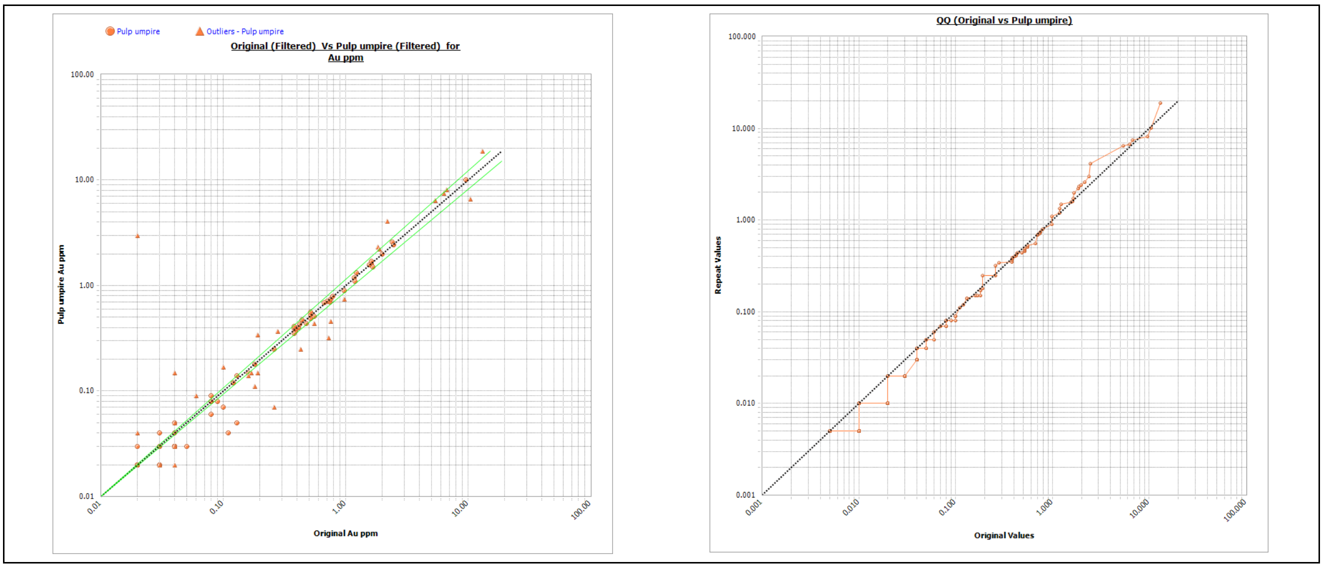


Figure 11.7 Scatterplot and Q-Q plot for umpire pulp duplicates



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 effected 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 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 current 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 recommends 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.

12 DATA VERIFICATION

12.1 Assay data validation

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 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.

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 have been identified in the assay certificate checks. 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 Introduction

This section of the Technical Report summarises all metallurgical testwork conducted at Selinsing and the adjacent Buffalo Reef deposits over the project years, including:

- The geological and geochemical characterisation of the drillhole samples that make up some of the composite test samples
- The location of the drillholes
- The major metallurgy results.

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 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 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.
- Phase III: This expansion began on 6 September 2011 with a budget of $8.1 million and was completed in June 2012 on time and at a cost of $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 Selinsing pits were designed by consultants with Snowden and Monument staff mine engineers.

During 2011, Monument engaged Inspectorate to carry out a metallurgical test program on a selection of diamond drill core material collected from the Buffalo Reef deposit. This section includes:

- A summary of the work completed by Inspectorate and reported in *“Metallurgical Study on the Buffalo Reef Gold Project”* (2011)
- The engineering study ultimately prepared for Monument by Lycopodium of Brisbane, Australia, and reported by Lycopodium in *“Selinsing Phase IV Study”* (February 2013).

For the technical report, Snowden has evaluated the metallurgy, and process engineering of the SGM and the proposed Buffalo Reef mine, adjacent to Selinsing, and considered:

- The original process design criteria
- Testwork done in preparation of the initial process design criteria
- Process plant performance since in production
- Testwork done on fresh samples from Buffalo Reef and Selinsing Deeps in preparation of additional new process design criteria.

13.2 Previous metallurgical testwork and sample characterisation

13.2.1 Summary of metallurgical testwork conducted

A substantial amount of metallurgical testing has been completed on both the Selinsing and Buffalo Reef ores from 1996 to the present. In addition, the Selinsing oxide mill provides actual recovery data from over seven years of consistent gold processing operations. Bio-oxidation pre-treatment studies have recently been completed on Selinsing and Buffalo Reef refractory sulphide ores and a flotation and bio-oxidation plant are being planned as part of the Phase IV plant expansion.

Table 13.1 summarises the major metallurgy studies completed on Selinsing ores. In summary, Ammtec (Perth, Australia), Metallurgical Design (Perth, Australia), and Inspectorate Labs (Richmond, Canada) have completed most of the work to date.

The first Selinsing PFS undertaken in 1998 also listed several other ore characterisation studies that were performed to help support the mine and mill development (TRA, 1998). Some of the initial historical metallurgical studies tested both oxide and sulphide Selinsing ores. The Selinsing composites tested from 2008 to 2012 were sourced from various elevations and likely include transitional or mixed oxide-sulphide ores, which includes bulk samples collected in 2012 from the 457.5 mRL bench in Selinsing Pit 4. Recent metallurgical testing at Selinsing by Inspectorate in Richmond, Canada (2013) focused exclusively on the deeper sulphide ores (i.e. Selinsing Deeps) and examined recovery variability due to elevation, gold head grade, and geochemistry.

Buffalo Reef ores have previously been tested by several investigations (see Table 13.2): Naidu (2005), Avocet (2006), and Cavey and Gunning (2007) and were reviewed in Snowden (2011). The results of metallurgical testing by Monument, since 2010, have been published by Snowden (2011) and Inspectorate (2012, 2013). Recent metallurgical testwork on Buffalo Reef ores from the Central Resource Zone, carried out at Inspectorate in Richmond, Canada and at the Company's bioleach laboratory at Selinsing, are reported later in this section (see Section 13.3).

Ores collected in exploration drillholes at Selinsing and Buffalo Reef were originally logged by geologists according to their geochemical weathering codes as either "oxide" or, as "fresh" (i.e. visible sulphide), which correspond to the two main mill ore feed types as presented in the Mineral Resource and Mineral Reserve tables. The contact between the oxide and sulphide zone is gradational and best approximated by the weathering code ranging from "completely to highly-oxidised" near the surface, to "fresh" (no visible weathering) at depth.

Variably oxidised materials characterised as moderate-oxidation and low-oxidation intensity types occur between the two end members. Oxide ores being processed through the mill are from all weathering code types: completely-oxidised, high-oxidation, moderate-oxidation, and low-oxidation. Most of the more homogenous oxide materials derived from Pit 4 were exhausted at or below the 457.5 mRL bench level, which was exposed in late May 2012. There are a few pods left of variably oxidised material below the 457.5 mRL level at Selinsing (Pit 4). Sulphide and "fresh" mined material was submitted for a quick leach test to determine if it was suitable for processing using the existing oxide mill flowsheet, or if the transition ore needed to be stockpiled for processing at the future bioleach pad in the Phase IV expansion plan.

Table 13.1 Summary of Selinsing metallurgical testwork programs to date

Report no.	Company (report no.)	Report date	No. of variability test composites reviewed	Ore types tested	Geology or alteration type tested	Drillhole type and no. tested	Approximate ore elevation range tested (mRL) mostly in Pit 4 area
1	AMMTEC (#A5293)	December 1996	Unknown	Unknown	Unknown	Unknown	Unknown
2	AMMTEC (#A5477)	1997 (May?)	17	Oxide >Sulphide	Variable	25 RC and 4 core	410 m – 490 m
3	Metallurgical Design	December 2008	-	Oxide	-	Summary Report	-
4	AMMTEC (#A11594)	October 2008	3	Oxide	CAT, Qtz, QS	3 diamond drillholes	484 m – 527 m
5	Metallurgical Design	August 2010	-	Fresh (Sulphide)	-	Summary Report	-
6	AMMTEC (#A112359)	August 2010	4	Fresh (Sulphide)	Qtz, CAT, QS	2 diamond drillholes	448 m – 484 m
7	AMMTEC (#A14150)	August 2012	4	Sulphide	Qtz, MYL, HR	11 diamond drillholes	234 m – 490 m Pit 4: Selinsing Deeps and Selinsing
8	Inspectorate (Richmond, Canada)	Data, 2012	1	Sulphide and TR	CAT, MYL, Qtz, High Carbon, High Sulphide	Bulk surface samples from 457.5 mRL level	457.5 m bench Pit 4: Selinsing
9	Inspectorate (Richmond, Canada) Project #1206609	Inspectorate, 19 February 2013 (revision 1)	6	Sulphide	Qtz, MYL, CAT	9 diamond drillholes	300 m – 432 m Pit 4: Selinsing Deeps

Note:

1. *The AMMTEC December 1996 report is listed as a reference for the June 1998 PFS but was not available for this current review*
2. *Geology type definitions: QTZ = Quartz Vein; QS = Quartz Stockwork; MYL = Mylonite; CAT = Cataclasite; HR = Host Rock = Felsic Tuff Footwall; TR = transitional material between oxide and sulphide (fresh)*

Most of the metallurgical composite samples were characterised by their geological unit or alteration type, and then by their geochemistry and gold head-grade. The composite samples were then tested for initial gravity concentrate of gold (and CIL) and then applied through a secondary CIL leach of the gravity tails. For example, the main geological rock types that host ore at Selinsing include either, "quartz vein" (QTZ or Qtz Vn); "quartz stockwork" (QS); "cataclasites" (CAT); "mylonites" (MYL); or "Felsic Tuff Footwall" (HR). Cataclasite and mylonite are actually structural terms that imply ductile shearing and/or brittle faulting or breccia deformation. Geochemical head grades for most of the composites tested include the averages of grades for gold, silver, arsenic, and organic carbon. More recent testing, from 2010 to the present, also included analysis of total carbon, iron, copper, manganese, total sulphur, and sulphide sulphur.

A geological summary of Selinsing Deeps core composites is listed in Table 13.3.

Table 13.2 Summary of Buffalo Reef metallurgical testwork programs to date

Report no.	Company	Report date	No. of variability test composites tested	Ore types tested	Geology or alteration type tested	Drillhole type and no. tested	Approximate ore elevation range tested (mRL) resource zone tested
1	Naidu (Avocet)	May 2005	Resin in leach; flotation	Fresh > Oxide	Unknown	BRP003; BRP005; BRP012; BRP015 BRP017; BRP018; BRP019	North North South South
2	Avocet	November 2006	unknown	Unknown	Unknown	Unknown	Unknown
3	Ore Quest Consultants Ltd	June 2007	26 quick leach and 8 bottle rolls	Fresh > Oxide	Unknown	RC drillhole samples	North and South
4	Inspectorate (Richmond, Canada)	24 September 2012 (initial results summarised in Snowden (2011)	6; flotation; bio-ox	Fresh Transitional	Argillite, Qtz Vn., Phyllite,	MBRDD001; MBRDD003, MBRDD004	434 m – 449 m North; 438 m – 489 m South; 422 m – 437 m; and 459 m – 461 m South
5	Inspectorate (Richmond, Canada)	In progress: 2012-2013	3; flotation; bio-ox in progress	Fresh Transitional	Felsic Tuff; Qtz Vn	MBRRC167 and MBRRC168	440 m – 467 m and 480 m – 488 m Central Zone

Notes: Listed elevations are in metres RL units

13.2.2 Early metallurgical testwork on oxides

Metallurgical testwork was conducted at Ammtec in Perth during 2005 to 2008 and the work summarised by M. Kitney in a report during December 2008.

The test program was carried out on samples of whole diamond drill HQ cores by Ammtec Ltd of Perth, Western Australia. The samples represented three ore types identified by SGM as “Cataclastic”, “Quartz Stockwork” and “Quartz Vein” materials.

At the time of generating the samples and throughout the execution of the test program SGM had been actively engaged in the design of a 400,000 t/a oxide ore mining and treatment facility at Selinsing. The test program for the Selinsing oxide ore was carried out to confirm the amenability of the ore to treatment for gold extraction and recovery and to provide more complete comminution data for the ore in particular. The scope of the test program included the determination of comminution parameters, the extent of gravity recoverable gold (GRG), response to CIL treatment and the properties of CIL tails pulps.

The scope of the M. Kitney report (2008) covered the following items:

- Summary of the metallurgical results obtained by Ammtec
- Development of essential parameters for the process design.

Table 13.3 Geological characterisation of metallurgical samples from the Selinsing deposit

Composite no.	Drillholes in composite	From (m)	To (m)	Rock type	Weathering code	Approximate m RL elevation range
SelDeep 2/1	MSMDD047	26.0	33.0	Felsic Tuff	Fresh	425 m – 432 m
SelDeep 2/2	MSMDD047	59.0	63.5	Quartz Vein >Felsic Tuff	Fresh	395 m – 399 m
SelDeep 2/2	MSMDD047	65.0	70.7	Mylonite> Felsic Tuff	Fresh	388 m – 393 m
SelDeep 2/2	MSMDD047	72.3	76.9	Mylonite> Felsic Tuff	Fresh	381 m – 386 m
SelDeep 2/3	MSMDD021	168.4	177.2	Mylonite = Felsic Tuff	Fresh	360 m –-371 m
SelDeep 2/3	MSMDD025	153.2	161.8	Mylonite = Quartz Vein > Felsic Tuff	Fresh	360 m – 371 m
SelDeep 2/3	MSMDD025	161.8	168.4	Felsic Tuff = Mylonite	Fresh	350 m – 359 m
SelDeep 2/4	MSMDD025	168.4	186.1	Mylonite>>Quartz Vein	Fresh	340 m – 349 m
SelDeep 2/4	MSMDD021	204.9	208.0	Felsic Tuff	Fresh	340 m – 349 m
SelDeep 2/5	MSMCD002	190.18	191.33	Cataclasite	Fresh	323 m – 324 m
SelDeep 2/5	MSMCD003	187.83	198.08	Mylonite	Fresh	319 m – 329 m
SelDeep 2/5	MSMCD017	188.65	199.07	Felsic Tuff > QTZ Vein	Fresh	316 m – 326 m
SelDeep 2/5	MSMCD007	189.46	200.46	Argillite + Quartz Vein > Mylonite + Arenite sandstone	Fresh	325 m – 329 m
SelDeep 2/5	MSMDD010	182.0	186.9	Argillite	Fresh	325 m – 329 m
SelDeep 2/6	MSMCD003	198.08	210.6	Mylonite>Cataclasite + Argillite	Fresh	309 m – 320 m
SelDeep 2/6	MSMCD017	199.07	210.16	Felsic Tuff > Quartz Vein	Fresh	310 m – 317 m
SelDeep 2/6	MSMDD004	206.95	210.0	Felsic Tuff = Quartz Vein	Fresh	308 m – 310 m
SelDeep 2/6	MSMCD006	194.37	198.26	Cataclasite	Fresh	316 m – 320 m
SelDeep 2/6	MSMCD007	216.49	218.04	Argillite	Fresh	309 m – 310 m
SelDeep 2/6	MSMDD010	192.5	194.0	Calcareous Argillite	Fresh	319 m – 321 m
SelDeep 2/6	MSMCD017	210.16	219.4	Felsic Tuff > Quartz Vein > Argillite	Fresh	295 m – 308 m

Ore samples used in the testwork

The ore samples were obtained from diamond drillholes DD14, DD15 and DD16 and the quantities of each ore type are summarised in Table 13.4. The ores were categorised as being:

- Cataclastic (CAT) – i.e. brecciated;
- Quartz Stockwork (QS) – i.e. comprising fine quartzitic stringers in a brecciated matrix; or
- Quartz Vein (QV) – i.e. distinct quartz veins occurring in the host rock.

The relative proportions by weight of each ore type in the as received samples were:

- CAT 58.5%
- QV 23.4%
- QS 18.0%.

Table 13.4 Selinsing oxide ore samples used in metallurgical testwork

Drillhole no.	Drill core interval		Ore type	Core interval (m)	Interval weight (kg)
	From (m)	To (m)			
SELLDD14	14.20	15.05	CAT	0.85	5.384
SELLDD14	15.05	16.15	QS	1.10	7.000
SELLDD14	16.15	17.15	QV	1.00	7.014
SELLDD15	0.45	3.55	CAT	3.10	14.906
SELLDD15	3.55	4.10	QS	0.55	3.036
SELLDD15	4.10	4.50	QS	0.40	2.528
SELLDD15	4.50	5.85	CAT	1.35	8.106
SELLDD15	16.25	17.70	QV	1.45	6.542
SELLDD15	17.70	18.30	CAT	0.60	3.792
SELLDD15	18.30	19.10	QV	0.80	5.456
SELLDD16	43.65	46.10	CAT	2.45	14.374
SELLDD16	46.10	47.90	QS	1.80	12.184
SELLDD16	47.90	50.15	CAT	2.25	15.196

Head analysis

Each of the separate subsamples from each drillhole was composited by type and assayed for gold content by duplicate fire assays. Other elements were determined by ICP scan. Sulphide sulphur and organic carbon were determined by Leco analysis. The key results are summarised in Table 13.5.

The duplicate gold assays were in fair agreement and on a weighted basis produce an overall grade of 2.18 g/t Au for the oxide ore represented by the three diamond holes sampled. The silver grades were unremarkable and the analyses for organic carbon (Corg) suggested preg-robbing would not be an issue with these ore types. The arsenic and sulphide sulphur (S^{2-}) analyses suggested the presence of some sulphides that could result in the formation of acid species in the presence of air and moisture. The low calcium content suggests these ores will have negligible acid neutralising capacity.

Table 13.5 Summary of chemical analysis of oxide samples

Sample ID	Au ppm	Ag ppm	C_{org} %	S^{2-}%	As ppm	Hg ppm	Ca %
CAT	0.43/0.48	<0.3	<0.03	0.30	4076	1.7	1.42
QS	1.58/1.83	0.6	<0.03	0.65	3914	2.6	0.016
QV	8.28/8.79	3.1	<0.03	0.28	4003	0.1	0.023

Comminution tests

Unconfined compressive strength (UCS) and crushing work index (CWi) tests were performed on whole core specimens. Abrasion index (Ai), Bond rod mill work index (BRWi) and Bond ball mill work index (BBWi) tests were performed on samples of each ore type. The results are given in Table 13.6.

Table 13.6 Summary of comminution testwork results of oxide samples

Sample ID	UCS (MPa)	Strength rating	Ai	Abrasive-ness	CWi (kWh/t)		Mineral SG	BRWi (kWh/t)	BBWi (kWh/t)
					Average	Maximum			
CAT	9.6	Weak	0.039	Very low	4.4	7.3	2.71	11.0	11.5
QS	5.7	Very weak	0.101	Low	4.8	5.6	2.65	12.3	13.8
QV	6.8	Weak	0.152	Low	4.3	5.6	2.64	14.5	18.9

In UCS, the failure was in shear mode in all cases. The UCS data obtained indicated primary crushing system will not require excessive duty.

The moderately low Ai values indicate that wear of comminution and slurry handling equipment will not be excessive. As expected, the quartz vein material will be the most aggressive. The very low CWi values suggest post-primary crushing duties will not be excessive. The results also suggested that achieving a fine crush size should be possible. The SG data were typical of siliceous rock minerals.

The BRWi and BBWi values demonstrate the brecciated ores are more amenable to grinding than those containing discrete quartz grains. The quartz vein material is particularly hard and the high ball mill work index value will have an important bearing on the ultimate capacity of the Selinsing mill.

Gravity recoverable gold determinations

Estimates of GRG were made for each of the samples at grind P80 size of 80 µm. The GRG determinations were carried using a 3” Knelson Concentrator followed by intensive cyanide leaching of the concentrate. The GRG results are given in Table 13.7.

Table 13.7 GRG results of oxide ore samples

Sample ID	GRG (% of head)
CAT	15.96
QS	58.01
QV	79.62

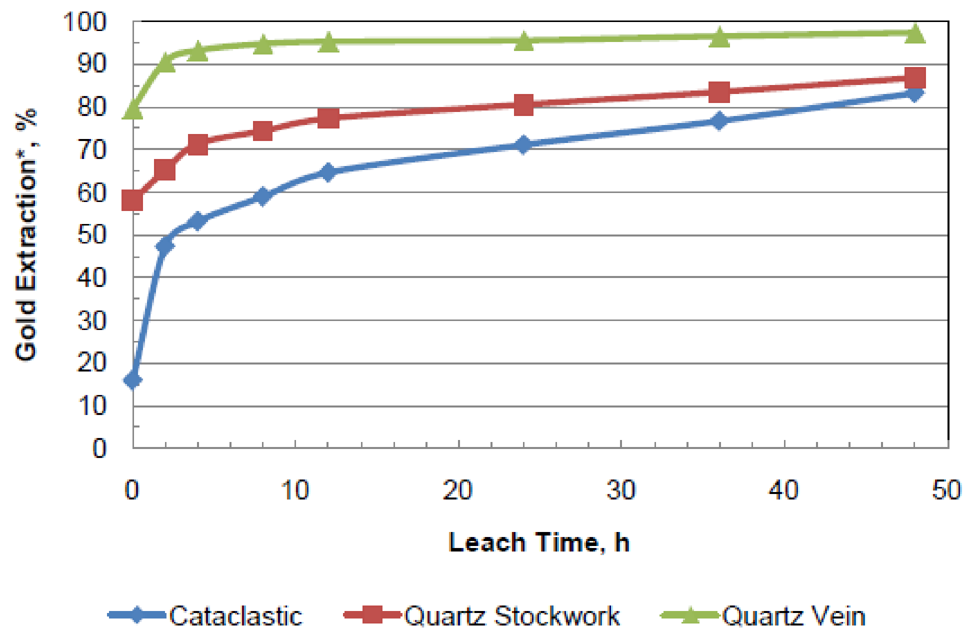
Cyanidation tests

The tails from each GRG tests were subjected to CIL cyanidation in air sparged, mechanically agitated open vats under a set of standardised conditions comprising:

- Solids weight: 3 kg
- Initial NaCN concentration: 0.05%
- pH: 10.5
- Solids content: 40% w/w
- Carbon concentration: 30 g/L
- Leach time: 48 hours
- Air sparged.

The results of cyanidation tests are shown in Figure 13.1.The gold recoveries shown in this figure are the overall gold extractions as the sum of the GRG component and the gold subsequently extracted into cyanide solution. The cataclastic (CAT) material displayed the slowest leach characteristics and least gravity recoverable gold. However, the low head grade of this sample may influence the apparent performance of this ore type. The 36-hour recovery values for each ore type were 76.7%, 83.5% and 96.5% for the CAT, QS and QV samples, respectively.

Figure 13.1 Selinsing oxide ore samples cyanidation response



13.2.3 Previous metallurgical testwork on fresh/sulphides

Three variability fresh gold ore composites from the Selinsing Mine were tested in 2010 by Ammtec Ltd. (Ammtec report no. A12539). The testwork program included the comminution testwork.

UCS testwork was carried out on a single specimen selected from each of the fresh variability composites. The results are given in Table 13.8.

Bond impact CWi determinations were carried out on each of the fresh variability composites, originating from samples in the particle size range –75+51 mm. CWi determinations were carried out using an impact crushability test unit. A subsample of each of the fresh variability composites was tested to determine the Ai value, BRWi and BBWi. The results are summarised in Table 13.9. The results were all similar to that obtained from original oxide testwork listed before (see Section 13.2.2).

Table 13.8 UCS results on fresh ore samples

Sample ID	UCS (MPa)	Failure mode	Strength description
Selinsing CAT 28.20 m – 28.65 m	21.259	Shear	Medium strong
Selinsing QS 31.75 m – 31.86 m	30.492	Shear	Medium strong
Selinsing QV 22.78 m – 22.87 m	124.247	Cataclasis	Strong

Table 13.9 Summary of comminution results on fresh ore (2010)

Sample ID	Ai	CWi (kWh/t) average	BRWi (kWh/t)	BBWi (kWh/t)
CAT fresh ore HQ core	0.0886	5.0	12.2	13.2
QS fresh ore HQ core	0.1958	5.0	13.0	14.6
QV fresh ore HQ core	0.5340	6.8	15.1	18.0

In 2010 and 2011, six Selinsing Deeps composite samples consisting of diamond drill core were collected at approximately 10 m drillhole intervals at elevations that ranged from 300 mRL to 432 mRL and were tested at Inspectorate. The geologic rock types of the samples were characterised as: quartz vein, mylonite, cataclasite, felsic tuff, argillite, and calcareous argillite. The head grades of the six Selinsing Deeps composite metallurgical samples are listed in Table 13.10 to Table 13.14.

All six samples contain lower than average gold head grades compared to the global averages for the August 2012 Measured and Indicated Mineral Resources. The samples also contained moderately elevated arsenic values, low total sulphur, and low antimony. Detailed petrography on metallurgical ore concentrates and two head samples from similar sulphide ore materials has identified microscopic and sub-microscopic native gold and electrum in habits of free grains, on grain boundaries, and as inclusions in arsenic-bearing pyrite, arsenopyrite, stibnite, and in quartz (Zhou, 2011).

Monument tested several composite samples in 2011 and 2012 from the Buffalo Reef deposit at Inspectorate, under the direction of Frank Wright (PEng, Metallurgist and Director for Monument). One sample is core from the north end of the resource, three are core from the south end of the resource, two are blended core samples from the north and south areas, and three samples are derived from the central resource (recently drilled reverse circulation cuttings).

Table 13.10 Selinsing Deeps (Pit 4) composite head grades

Composite	Au (g/t)	Ag (ppm)	As (ppm)	Sb (ppm)	$\%C_{total}$	$\%C_{org}$	%S --	$\%S_{total}$
SELDeep 2/1	1.69	1.21	13,380	25	0.15	0.03	2.30	2.33
SELDeep 2/2	1.63	0.82	9,982	16	0.48	0.05	1.57	1.58
SELDeep 2/3	1.60	0.63	7,956	15	0.53	0.08	1.24	1.24
SELDeep 2/4	1.67	0.54	7,264	12	0.60	0.08	1.39	1.39
SELDeep 2/5	1.43	0.56	6,494	21	1.60	0.11	1.19	1.19
SELDeep 2/6	1.26	0.48	8,041	16	0.94	0.10	1.41	1.41

Source: Inspectorate, 2013

Table 13.11 Geological characterisation of recent Buffalo Reef metallurgical composite samples based on geologic logging

Composite no.	Resource zone	Drillholes in composite	From (m)	To (m)	Rock type	Weathering code
2	North	MBRDD003	44.8	69.4	Quartz Vein> Phyllite	HOX + MOX > Fresh
4	South	MBRDD004	55.2	57.6	Argillite	HOX
5	South	MBRDD004	81.9	84.3	Argillite	LOX and MOX
5	South	MBRDD004	87.8	89.5	Argillite	LOX and MOX
5	South	MBRDD004	90.9	92.7	Argillite	MOX
6	South	MBRDD004	97.9	100.1	Argillite	HOX + MOX
7	South	MBRDD001	65.3	82.4	Phyllite > Quartz Vein	HOX + MOX >Fresh
BR1	North and South	All drillholes above	Most intervals above		Characterised as above	Characterised as above
COMP. C	Central	MBRRC167	17.0	20.0	Felsic Tuff	Fresh
COMP. D	Central	MBRRC168	38.0 40.0 63.0	39.0 42.0 67.0	Felsic Tuff	Fresh
COMP. E	Central	MBRRC168	39.0 62.0	40.0 63.0	Felsic Tuff; Qtz Vn	Fresh

Notes:

1. *Composite 2 from Buffalo Reef North; Composites 4-6 from Buffalo Reef South*
2. *Composites 7 and BR1 include a blend of samples from Buffalo Reef North and South*
3. *Weathering codes: HOX = high oxidation; MOX = moderate oxidation; LOX = low oxidation; Fresh = no oxidation*

Table 13.12 Buffalo Reef North and South composite head grades

Composite no.	Au (g/t)	As (ppm)	Sb	%C_{org}	%S --	%S_T
2	2.08	4273	34 ppm	0.14	1.42	1.42
4	3.14	5155	32 ppm	0.14	1.35	1.37
5	1.59	3054	37 ppm	0.19	0.84	0.86
6	111.0	1147	10.7%	0.19	6.12	6.16
7	2.12	3759	175 ppm	0.43	1.22	1.29
BR1	2.77	4000	102 ppm	0.27	1.32	1.37

Source: Inspectorate, 2012

Table 13.13 Collar coordinates and drillhole orientation for Selinsing and Buffalo Reef composite sampling testing (2012)

DHID	Deposit name	Easting (mine grid)	Northing (mine grid)	Elevation (m)	TD (m)	Azimuth (at 0 m)	Dip (at 0 m)
MSMCD002	Selinsing (Pit 4)	907.009	2056.415	496.252	300.2	278	-70
MSMCD003	Selinsing (Pit 4)	929.965	1973.762	501.248	300.2	278	-70
MSMCD006	Selinsing (Pit 4)	926.202	1916.402	499.328	300.3	278	-70
MSMCD007	Selinsing (Pit 4)	911.752	1870.383	502.675	302.5	278	-70
MSMCD017	Selinsing (Pit 4)	937.304	1936.596	497.874	300.6	278	-70
MSMDD004	Selinsing (Pit 4)	943.418	2029.614	498.467	301.0	278	-70
MSMDD010	Selinsing (Pit 4)	875.807	1822.184	497.308	300.0	278	-70
MSMDD021	Selinsing (Pit 4)	924.891	1901.317	497.809	296.9	278	-50
MSMDD025	Selinsing (Pit 4)	929.606	1955.959	496.371	305.8	278	-60
MSMDD047	Selinsing (Pit 4)	836.687	1990.756	458.403	101.0	278	-90
MBRDD001	Buffalo Reef (S)	584.465	3396.895	505.21	101.9	278	-60
MBRDD003	Buffalo Reef (N)	802.224	5517.315	497.765	110.1	98	-60
MBRDD004	Buffalo Reef (S)	512.397	3373.858	507.397	107.0	98	-60
MBRRC167	Buffalo Reef C	560.331	3994.578	495.372	65.0	278	-60
MBRRC168	Buffalo Reef C	518.893	3925.994	495.857	75.0	94	-50

Notes: Buffalo Reef Resource Areas as defined by Snowden (2011): (S) = South; (N) = North; C= Central; Coordinates listed in Selinsing Mine Grid

Metallurgical results from earlier testwork on fresh/sulphides

In 2013 Inspectorate produced a letter report (*Selinsing Deep Preliminary Process Response - Project #1206609*) in which response of various samples of Selinsing Deeps sulphide ores to cyanidation before and after bioleach pre-treatment was reported to scoping study level of detail.

For the six Selinsing Deeps sulphide composite samples tested at Inspectorate (2013), the grinding was carried out in two steps. After the first stage (P80 of 212 µm), a gravity concentrate was produced with the final grind for other processing completed at P80 of 74 µm. The gravity procedure incorporated a Knelson centrifugal concentrator with the resulting concentrate representing 3% to 5% feed mass forwarded to intensive cyanide (IC) leaching. The Knelson tailing and IC residue were then combined for conventional CIL using a 48-hour retention time. Detailed whole ore cyanidation response for the six Selinsing Deeps composite samples are listed in Table 13.14. In summary, the results confirmed a previously suspected refractory nature to the Selinsing Deeps “fresh” (sulphide redox type) ore samples with corresponding gold recovery ranging from a low of 16.4% to a high of 59.9%. These results suggest that the sulphide ores will be refractory at and below the 432 mRL bench level in the Selinsing Pit area.

Buffalo Reef oxide ores have been tested previously by investigators using bottle roll methods. In previous studies, the results of quick leach tests returned >92% gold recoveries (Snowden, 2011). Historical and recent metallurgy test programs have identified the Buffalo Reef sulphide ores as refractory (Avocet Gold, 1999 and 2005; Snowden, 2011; Inspectorate, 2012).

Table 13.14 Selinsing Deeps (Pit 4), whole ore cyanide response

Selinsing Deeps composite no.	Gold grade (g/t Au)		Gold recovery (%)		
	Calc. head	Final res.	IC	CIL	Total
SELDeep 2/1	1.90	1.19	8.2	29.5	37.7
SELDeep 2/2	1.59	0.85	7.0	25.5	32.5
SELDeep 2/3	1.45	1.34	8.7	7.6	16.4
SELDeep 2/4	1.88	1.32	8.7	25.8	34.5
SELDeep 2/5	1.95	1.08	14.8	26.9	41.7
SELDeep 2/6	2.21	0.69	21.7	38.2	59.9

Notes: IC = Intensive cyanide leaching; CIL = Conventional carbon in leach

Source: Inspectorate, 2013

The flowsheet diagram describing the testwork evaluation for the six Selinsing Deeps samples incorporated a pre-treatment procedure prior to CIL, which consisted of the bacterial leaching of a floatation concentrate. IC leaching of the Knelson concentrate was then performed and the resulting residue was combined with the Knelson tailing without assaying separately. As a result, the gravity recovery is based on the overall calculated head from the entire circuit rather than the gravity circuit alone. The estimated IC leaching gravity recovery ranged from 10% to 25% depending upon the sample and results are similar to that of the whole ore study.

Flotation procedures included using potassium amyl xanthate (PAX) as a collector and did not include cleaning of the bulk concentrate. The kinetic flotation was performed in four stages typically eight minutes each in duration with the first two stages recovering over 90% of the gold and sulphides. The final two stages acted as scavenger cells and recovered <2% of the gold, and <1% of the sulphides. The resulting grades from the open cycle batch of the uncleaned bulk concentrate for the combined first two stages (rougher) ranged from 7 g/t Au to 14 g/t Au, with 9% to 13% S. Due to low gold and sulphide content of the final two stages, only the first two rougher stages were forwarded for bioleaching study. It is likely that flotation tailings will be used for bio-oxidation residue neutralisation, which would result in recoveries as shown in Table 13.14.

Table 13.15 shows the results from separate CIL leaching of the flotation tailing. Gold recovery results ranged from 55% to 89% on calculated feed (float tailing) that contained grades of less than 0.15 g/t Au.

Table 13.15 CIL of bulk float tailing method of recovery for six Selinsing Deeps "fresh" composite samples

Selinsing Deeps composite no.	Calculated head (g/t Au)	Au recovery (%)
2/1	0.13	76.8
2/2	0.09	88.9
2/3	0.13	62.5
2/4	0.14	55.3
2/5	0.06	66.7
2/6	0.08	73.7

Source: Inspectorate, 2013

The corresponding flotation rougher 1 + 2 concentrates of each of the six Selinsing Deeps composite samples were subjected to bacterial leaching. The bioleaching appears to have been completed within 14 days, which is typical for bench scale batch studies. All the samples responded well with the excess of 98% dissolution of arsenic and between 90% and 98% dissolution of sulphides as outlined in Table 13.16.

Table 13.16 Response to bioleaching of flotation concentrate for six Selinsing Deeps composite samples

Selinsing Deeps composite no.	Bioleach (Au, g/t)		Wt. (%) loss	Dissolution (%)		% Recovery (CIL Au)
	Calc. head	Final res.		As (bio)	S (bio)	
2/1	12.6	1.17	44	98.4	93.7	90.7
2/2	10.9	0.79	38	98.9	93.9	92.7
2/3	16.2	1.28	36	98.7	95.5	92.1
2/4	14.1	0.90	32	98.9	94.7	93.7
2/5	15.8	0.68	32	98.8	98.3	95.7
2/6	9.3	0.52	33	98.9	97.1	94.4

Source: Inspectorate, 2013

The preliminary test program suggests that the Selinsing Deep mineral samples that were provided show a poor response to direct gravity and CIL leaching, averaging ~37% gold recovery. Incorporating bacterial leaching of a flotation concentrate significantly improves the CIL recovery as compared to whole ore. The results of a 48-hour CIL gold recovery test on the six bio-residues derived from the above previous process averaged about 93% recovery, indicating a favourable response to bioleach pre-treatment methods. A summary of these results, including bioleaching of the flotation concentrate produced from gravity tailing is provided in Table 13.17.

Table 13.17 Estimated soluble gravity, flotation and bioleach gold recovery (% Au) for six Selinsing Deeps sulphide composite samples

Selinsing Deeps composite no.	IC leach of gravity conc.	Flotation of gravity tail	CIL of bio-residue	Overall recovery (% Au)
2/1	11.7	85.3	90.7	89
2/2	11.3	86.4	92.7	91
2/3	9.8	88.0	92.1	91
2/4	12.7	86.0	93.7	93
2/5	19.1	79.7	95.7	95
2/6	24.6	73.7	94.4	94

The overall process for gold recovery from the Selinsing Deeps sulphide ores composite samples includes gravity treatment followed by bioleaching of uncleaned open cycle flotation concentrate produced from gravity tailing. Results from these processes indicate an estimated range of 89% to 95% recovery. Locked cycle testing, including a separate balance around the gravity circuit, would be required to more accurately determine the overall expected recovery. Additional gold can be recovered by cyanidation of the flotation tailing.

Average gold recovery from oxide material based on previous metallurgical composite testing and oxide processing at Selinsing is 92%. As the Selinsing pit is advanced deeper into sulphide ore, the gold recovery by direct CIL decreases. This decrease in gold recovery relative to elevation has been observed in previous metallurgical sample test studies performed by Ammtec (2012), and from pit floor samples from the 457.5 mRL level collected and tested in the summer of 2012 at Inspectorate.

The results of the bio-oxidation and CIL testing by Inspectorate (2012) on Buffalo Reef refractory ore material are included in Table 13.18. These are the results of CIL-leached tail grade and gold recovery ranges for Buffalo Reef refractory sample material taken from the north and south resource zones following selected pre-treatment procedures with various arsenopyrite/pyrite dominated bulk flotation concentrate (15 g/t Au to 30 g/t Au) from Inspectorate (2012).

Table 13.18 Recovery ranges for Buffalo Reef refractory ore

Description	Tail grade (g/t Au)	Recovery (% Au)
Untreated concentrate	8 to 15	20 to 27
Ultrafine grind to P80<7 µm	~8.3	~38
Two-stage roast (up to 700˚C)	3.5 to 11	52 to 72
Pressure oxidation (220˚C)	6.5 to 11	26 to 42
Bio-oxidation (complete)	1.1 to 1.2	88 to 94

13.3 Most recent metallurgical testwork on sulphide ores (Selinsing and Buffalo Reef ores)

As mentioned before, during 2011 Monument engaged Inspectorate to carry out a metallurgical test program on a selection of diamond drill core material collected from the Buffalo Reef deposit at Selinsing. The Selinsing 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 CIL. Final gold recovery from carbon strip solution and gravity concentrate leach solution is by electrowinning onto stainless steel cathodes.

Monument established the opportunity to access sulphide-rich ores from the nearby Buffalo Reef deposit and instituted a drilling and testing program in 2010.

This section follows on from the previous section on the work completed by Inspectorate and reported in *"Metallurgical Study on the Buffalo Reef Gold Project"* (2011) and the engineering study ultimately prepared for Monument by Lycopodium of Brisbane, Australia, and reported by Lycopodium in *"Selinsing Phase IV Study"* (February 2013).

Metallurgical Design (MD) of Perth, Western Australia, has prepared this review on behalf of Monument.

13.3.1 Buffalo Reef

Sample selection and variability

Monument subsidiary, Damar, produced diamond drill core samples from the north and south resource zones of the deposit to provide a spread of material, which reflected the variability in mineral composition and gold content along strike. Section 2 of the Inspectorate report adequately describes the applied sample receipt, preparation and characterisation procedures.

Inspectorate prepared subsamples for testing as follows:

- Froth flotation of sulphide minerals
- Biological leaching of sulphide concentrates
- Analytical work including:
  - gold by fire assay
  - elemental analyses
  - solids SG
  - petrographic and mineralogical studies.
- Comminution testing – reported earlier.

Metallurgical testwork review

Mineralogical analysis of Buffalo Reef samples was reported by Lehne & Associates and Zhou Mineralogy (2011). The ore was described as containing arsenopyrite and pyrite mineralogy with areas of stibnite and preg robbing carbon, with minor sulphide occurrences of berthierite and tetrahedrite. Gold is strongly associated with arsenopyrite and to a lesser extent pyrite. Gold is present in both solid solution and as sub-microscopic colloidal occurrence. Gravity recoverable gold was indicated to be present with the stibnite but visible gold was not observed in the other head samples.

Table 13.19 summarises the findings of Zhou Mineralogy on examination of Buffalo Reef head and concentrate samples submitted by Inspectorate.

Table 13.19 Buffalo Reef mineralogical characteristics; ore and concentrate

Sample	BR1 head	BRC1 comp.	Comp. 6	Comp. 6 1st Cl conc.
Sample P80, µm	150		150	
Sulphide content, %	Aspy 0.8; Py 2.2	Aspy 7.6; Py 16.5	Aspy 2.15; Py 2.7; Sb 15	Aspy 10.5; Py 4.3; Sb 37.5
Sulphide grain size, µm	300 x 600	<5 to 300	<5 to 300	5 to 200
Sulphide liberation	Well liberated	Well liberated	Well liberated	Well liberated
Gold occurrence	None visible	Microscopic 28% Sub-microscopic 72%	Native 98% Sub-microscopic 2%	Native 95% Sub-microscopic 5%
Preg-robbing carbon?	Not reported	Yes	Not reported	Not reported

Flotation testwork

The flotation testwork program conducted by Inspectorate demonstrated that the Buffalo Reef composite samples responded well to conventional froth flotation.

Bulk gold recoveries of 90% to 95% and sulphide recoveries of 95% to concentrate were achieved at mass pulls of ~10% w/w and grind size P80 of 74 µm. Concentrate gold grades of ~20 g/t Au were obtained but with relatively low concentrate sulphur grades typically 6% to 12% w/w. Additional testwork was conducted in an attempt to upgrade the concentrate (Inspectorate project 1107112), and while gold grades of up to 50 g/t Au and sulphur grades of up to 29% w/w were achieved; this was at the expense of a significant loss in gold recovery.

Figure 13.2 shows the locked cycle flowsheet employed by Inspectorate.

The results show that the combined first cleaner concentrates had a mass pull of 10.3%. Most of the gold and sulphur went into the first pull which had 31.7 g/t Au, 18.1% S and a mass pull of 6.3%. The prorated units give the combined three pulls of the first cleaner concentrate at 19.9 g/t Au and 11.3% total S, with a relatively stable grade of concentrate in each cycle. The combined organic carbon in the final concentrate was approximately 0.7%, which floated primarily in the first pull and which was not much higher than organic carbon in the final tailing at 0.5%. Final tailing gold grades were also steady ranging from 0.15 g/t Au to 0.18 g/t Au with each cycle, resulting in a locked cycle gold recovery of 93.4%.

The flotation tests used potassium amyl xanthate and Aerofroth A208 as collectors, lime, copper sulphate and sodium sulphide as modifiers, and methyl isobutyl carbinol as the frother. Table 13.20 contains a summary of the Inspectorate flotation work.

Inspectorate concluded that, while flotation could be used to produce a concentrate for subsequent pre-treatment ahead of cyanidation, a balance would have to be struck between mass pull (and consequently gold and sulphur grade) and gold recovery.

Selinsing's R&D department has conducted a considerable body of flotation work (see Figure 13.3) since the initial program reviewed above. This work has extended the range of sampling into the Buffalo Reef Central Zone, and has reinforced earlier determinations of flotation performance.

Figure 13.2 Locked cycle testwork flowsheet

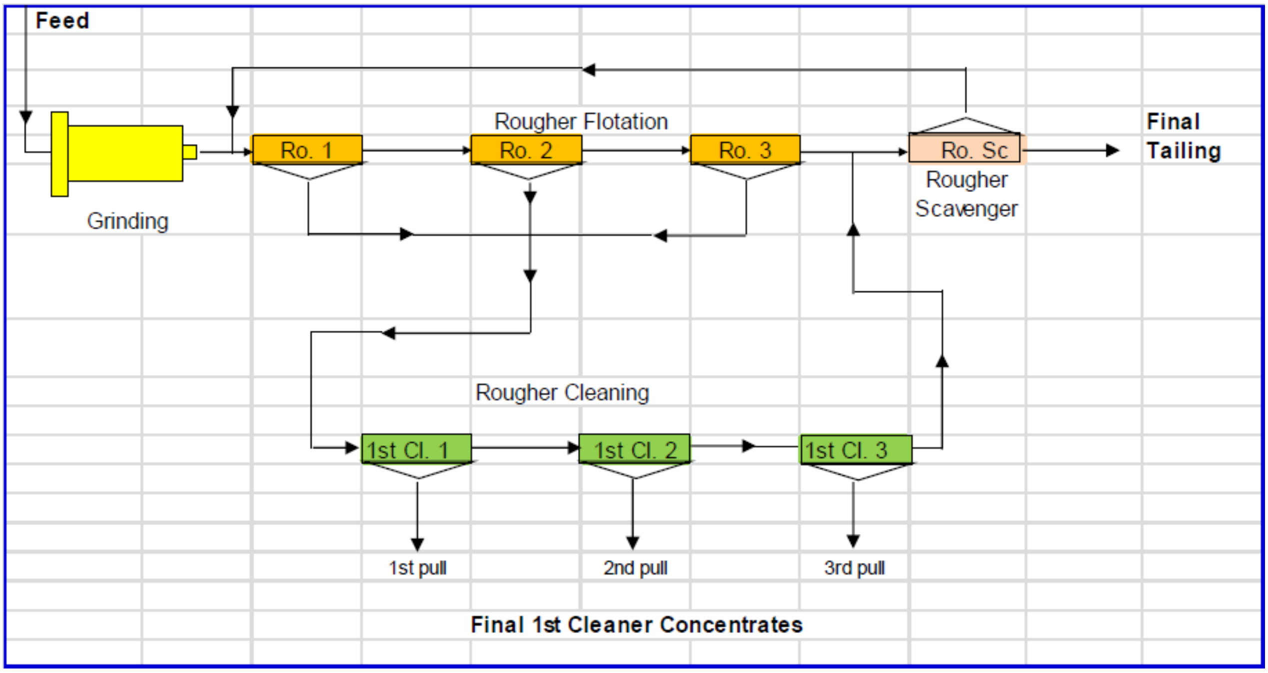


Table 13.20 Flotation data summary

Objective: Vary grind, reagents, and float time to note effect on recovery. Make use of gravity separation on two primary grinds (F4, F8), once after regrind (F9)
Baseline Condition : moderate grind P80~70 u, natural pH, 3X5 =15 min Ro strave PAX/A208 collector, scav 5 min agressively with CuSO4: With F6 increase collector & time + move up CuSO4
Assay Head: 2.06 g/t Au, 1.42 % total S, 0.41% As, 0.14%C(org)

Test No.	Test Description	Grind* P80 (u)	Total Minutes	Calc. Head Au, g/t	Calc. Head %S	Final Tail Au, g/t	Final Tail %S	Rough Rec.(%) Au	Rough Rec.(%) S	Rough Rec.(%) Mass	Bulk Recovery (%) Au	Bulk Recovery (%) S	Bulk Recovery (%) Mass	Comments
Bulk Float Data														
F1	baseline	71	15+5	2.14	1.41	0.26	0.10	75.9	89.8	16.4	90.5	94.4	20.5	baseline grind time = 21.5 minutes
F2	coaser than F1	88	15+5	2.10	1.46	0.29	0.13	75.7	88.8	17.5	89.1	93.1	22.1	coarser grind minor increased loss of Au
F3	finer than F1	43	15+5	2.14	1.42	0.26	0.12	75.1	88.3	17.5	90.8	93.5	23.0	finer grind no significant improvement to Au rec
F4	as F1 with gravity	70	15+5	2.39	1.35	0.27	0.11	88.5	92.3	15.9	90.9	93.5	19.7	Ro. rec includes grav which helps kinetics, not losses
F5	Na2S+more reagent	71	15+5	2.16	1.45	0.21	0.12	91.0	92.5	17.4	92.4	93.6	22.3	sulfidize may help?, but moved up CuSO4 more collector
F6	more time / reagents	71	21+10	1.99	1.37	0.26	0.13	85.9	89.3	12.5	89.3	92.1	17.4	new baseline increase float time reagents move up CuSO4
F7	as F6 except Scav	71	21+10	1.97	1.34	0.23	0.12	87.5	90.2	12.4	90.2	92.6	17.5	more reagents in scav may provide minor improve to rec
F8	as F7 but finer&grav	43	21+10	2.11	1.33	0.20	0.10	90.8	92.3	15.1	92.5	92.3	20.4	finer grind with gravity minor benfit to final tail grade
F9	as F6 with DF250	72	21+10	1.87	1.30	0.28	0.15	80.0	86.3	11.1	87.2	90.3	15.6	DF250 not as good recovery response, lower mass pull

Test No.	Test Description	Au Conc Grade (g/t) vs Ro. Stage Ro. 1	Ro. 2	Ro. 3	Scav. 1	Au Grade g/t Bulk	Au Grade g/t Grav.	Comments
Kinetic Data (15 min Ro+5 min Scav)								
F1	baseline	15.5	7.00	4.47	7.60	9.46	n/a	CuSO4 in scavenger with more collector helps boost recovery
F2	coaser than F1	14.7	6.30	3.33	6.07	8.47	n/a	more mass pull lower grades with coarser primary grind
F3	finer than F1	16.7	6.36	4.25	6.11	8.43	n/a	also more mass, slightly lower grades with fine grind
F4	as F1 with gravity	22.2	5.75	1.91	1.51	11.1	135.6	higher float grades despite grav pretreat(?), higher calc head
F5	more reagents	12.8	15.2	2.24	0.60	8.9	n/a	sulfidize may help kinetics in stage 1+2, may negate need of CuSO4 in Stage 3

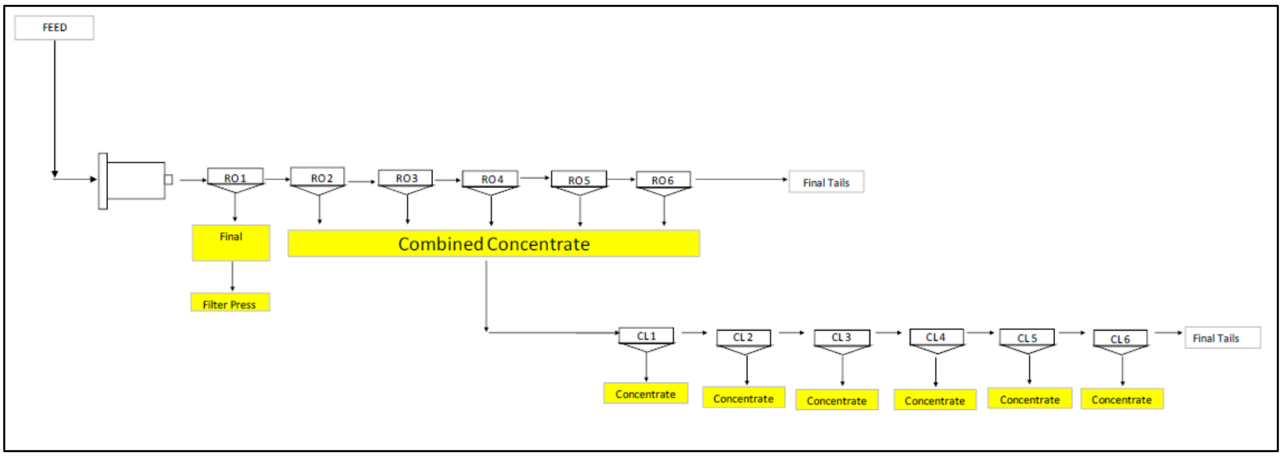
Cleaning Tests No.	Description	Regrind microns	Final Conc. Grades Au, g/t	Ag, g/t	% As	%C(org)	%S	%mass		1st Cleaner Scav. Tail Au, g/t	%Au Distr.	%S	% mass	Comments
F6	4 stage Cl. nat pH	P98~29	71.2	14.4	5.6	15.8	26.3	0.7		9.0	42.5	7.7	9.4	high loss in 1st cleaner tailing, poor upgrade
F7	4 stage pH 10 to 12	P98~30	55.8	10.9	4.2	13.0	37.2	1.0		7.0	29.8	3.8	8.3	higher pH upgrades S but not Au
F8	cleaning similar to F7	P98~28	49.9	12.8	2.0	12.3	33.9	1.2		4.9	22.9	2.8	9.8	
F9	1 stage Cl after grav	P98~30	13.5	-	-	-	10.2	11.1		9.1	37.0	7.6	7.6	

Gravity Recovery Summary

Test No.	Test Description	Grind* P80 (u)	Pan Conc. Au, g/t	Pan Conc. % Au rec	Pan Conc. %mass	Comments
F4	prim with mid grind	70	136	11.1	0.2	comparing F4 to baseline F1 float tailing no improvement to gold recovery with gravity
F8	prim with fine grind	43	212	11.2	0.1	float tail Au grade in F7 is lower than F4 so overall recovery may benefit somewhat from a finer grind. Cleaned conc grade improves
F9	done on Ro. Conc.	<25	381	6.9	11.1	gravity performed on reground rougher concentrate results in slightly improved gold grade to grav conc.

F. Wright Consulting Inc.

Figure 13.3 Selinsing R&D flotation flowsheet



This R&D work, which also included tests on Selinsing sulphides, was reviewed by Orway Mineral Consultants (Orway) of Perth, Western Australia and reported in *"7683-01 Flotation Batch Testwork Review Rev 0"*. Table 13.21 shows the summary of float data prepared by Orway. Test numbers beginning with "S" refer to Selinsing ore. Test numbers beginning with "B" refer to Buffalo Reef ore.

Table 13.21 Selinsing R&D float data

TEST NO.	CUMULATIVE CONCENTRATE MASS PULL (%)							CALC HEAD	CUMULATIVE Au RECOVERY, %						
	R01	CL1	CL2	CL3	CL4	CL5	CL6	ppm Au	RO1	CL1	CL2	CL3	CL4	CL5	CL6
S5F 4C	2.27	7.4	10.6	13.3	14.9	17.3	19.0	7.8	11.5	66.3	88.1	93.0	93.8	94.2	94.4
SDF 33C	9.92	19.5	22.9	28.8	39.7	45.3	50.8	3.4	42.0	93.0	95.3	97.3	98.5	99.0	99.1
SDF 34C	2.48	3.3	4.1	5.4	7.1	8.9	10.1	1.3	68.6	73.4	84.3	91.4	93.3	94.1	94.6
SDF 35C	17.62	28.1	34.5	40.3	48.4	54.4	57.3	8.6	70.4	91.1	96.3	98.7	99.5	99.7	99.7
SDF 36C	5.52	8.5	11.9	13.7	15.9	19.6	21.4	2.3	61.4	79.1	92.8	95.5	97.2	98.1	98.2
SDF 39C	3.68	9.7	12.9	16.1	18.7	21.4	23.5	1.2	52.2	88.4	92.6	94.3	95.0	95.5	95.6
SDF 41C	2.47	3.3	3.9	4.7	5.5	6.1	6.6	1.3	58.9	84.6	91.5	94.1	95.4	95.9	96.5
SDF 42C	4.97	7.3	8.4	9.8	12.8	16.7	19.3	1.7	82.0	96.3	97.5	97.8	98.2	98.5	98.6
SDF 43C	5.62	7.9	9.1	9.7	10.3	11.6	12.3	2.9	78.3	95.0	97.4	98.1	98.4	98.6	98.7
SDF 44C	6.03	7.6	8.6	9.4	10.2	11.0	11.8	2.2	83.5	93.4	96.5	97.3	97.9	98.3	98.6
BRC 30C	9.8	12.0	13.9	15.3	16.9	20.5	25.0	4.4	46.4	69.7	83.5	89.3	92.0	93.6	94.5
BRC 31C	7.27	10.8	11.8	14.2	17.1	20.4	26.5	4.5	24.7	56.3	66.2	83.5	91.5	93.1	94.2
BRC 33C	1.64	4.3	6.4	8.9	13.6	16.4	18.0	5.4	19.9	53.5	73.5	85.1	89.6	90.2	90.4
BRC 35C	3.89	9.7	11.8	13.5	15.9	18.9	20.2	5.0	44.9	88.2	91.9	93.2	93.8	94.1	94.3
BRC 37C	3.82	8.2	10.3	12.3	15.3	21.9	26.2	4.8	56.6	79.1	87.1	88.4	90.1	91.9	92.4
BRC 41C	5.28	11.9	16.0	21.4	24.0	27.4	29.4	6.0	65.8	78.0	80.2	88.2	91.0	91.9	92.3
BRC 43C	4.46	9.9	16.4	19.4	22.3	26.1	27.7	4.9	41.9	60.1	88.7	90.0	91.9	93.8	94.1
BRS 56	3.25	5.6	15.8	37.9	37.9	37.9	37.9	4.1	56.8	67.3	77.6	93.7	93.7	93.7	93.7
BRS 56R	4.78	11.8	34.6	34.6	34.6	34.6	34.6	4.0	65.3	78.3	90.6	90.6	90.6	90.6	90.6
BRS 57	5.7	12.4	41.6	41.6	41.6	41.6	41.6	4.4	68.1	78.7	92.2	92.2	92.2	92.2	92.2
BRS 58	7.09	16.3	38.8	38.8	38.8	38.8	38.8	4.1	76.6	87.5	93.6	93.6	93.6	93.6	93.6
BRS 59	5.54	9.5	31.7	31.7	31.7	31.7	31.7	4.7	68.7	81.5	92.7	92.7	92.7	92.7	92.7
BRS 61	8.87	15.5	35.2	35.2	35.2	35.2	35.2	4.6	78.8	83.8	90.4	90.4	90.4	90.4	90.4
BRS 62	8.49	13.0	36.3	36.3	36.3	36.3	36.3	4.3	81.1	85.5	92.7	92.7	92.7	92.7	92.7
BRS 63	6.59	9.6	24.0	24.0	24.0	24.0	24.0	4.5	78.1	84.3	90.6	90.6	90.6	90.6	90.6

Cyanide leaching and pre-treatment

Whole ore

Inspectorate conducted a series of direct cyanidation tests, which were unsuccessful in achieving significant gold extraction.

Initial work under Project 1005509, included whole ore leaching that consisted of varying grind, use of gravity pre-treatment and CIL procedures. The first set of tests performed on Comp. 2 maintained 2 g/L NaCN for extended leach retention of up to 72 hours. The results show that direct gold cyanidation recovery was very low at less than 5% and there was no improvement to gold dissolution with finer grinding or extended leaching after 24 hours. Longer leach retention times appeared to result in some decrease in solution gold concentration with time. This along with the fact CIL procedures improved gold recovery, indicates there may be a preg robbing component. Gravity pre-treatment of Comp. 2 recovered 16% of the gold into a low grade concentrate (~76 g/t Au). While there was insufficient weight to perform intense cyanidation of the gravity concentrate, the likely premise is it would have a poor response.

Five leach tests were then performed on Comp. 4 whole ore, with baseline conditions using 24-hour CIL leach retention. For the first three tests, the grinds were varied from an 80% passing particle size of less than 44 µm up to 89 µm. All gold extractions versus grind size were similar at approximately 10%. For the remaining tests on CIL, 13 used gravity pre-treatment and recovered approximately 18% of the gold into a low grade concentrate at 130 g/t Au. There was in fact no assurance the gravity concentrate would yield acceptable gold dissolution with high intensity leach methods. Test CIL14 used carbon addition along with a blinding agent sodium lauryl sulphate (SLS) for the natural carbon, and a higher cyanide concentration (5 g/L NaCN). Test C15 used standard cyanidation (2 g/L NaCN). CIL15 procedures resulted in approximately 11% gold recovery versus less than 1% for test C15, indicating while aggressive CIL procedures help the overall gold recovery is still very low.

Based on the above testwork results for CIL14 and CIL15, whole ore CIL leaching with SLS was performed on south zone composites, Comp. 5, 6 and 7, respectively in tests CIL17, CIL18, and CIL21. Early studies have shown the leach does not respond significantly more favourably with finer grinding or extended retention time. Test grinds were not performed and grinds were coarser than targeted at a P80 of ~110 to 155 µm, as well a shorter leach time of 24 hours was investigated. The resulting gold extractions were respectively 3%, 4% and 13% for Comp. 5, 6 and 7.

Another set of samples from Project 1107112 included three individual sample intervals and two composites, one from drillhole 7 and the other from drillhole 8. All the samples were from the south zone and all used CIL leaching with SLS, targeting a moderate grind (P80 = 74 µm), while maintaining 2 g/L NaCN for 53 hours. Details are provided in Appendix 5A of the Inspectorate report and show recoveries of between 4% and 17% gold extraction.

In summary, the best leach recoveries on whole ore were obtained using CIL procedures, with a blinding agent. These resulted in a maximum gold recovery of 21% for Comp. 2 (no blinding agent used) originating from the north zone. For the south zone, the samples appeared slightly more refractory with gold extractions ranging from 3% to 17%. The laboratory work confirmed that leaching pre-treatment methods would need to be evaluated for the arsenopyrite pyrite dominated sulphide material from both ore zones at Buffalo Reef.

Concentrate leaching with pre-treatment

Inspectorate also supervised testing of fine grinding, pressure oxidation (POX) and roasting as alternative methods of pre-treatment of Buffalo Reef sulphides prior to cyanidation.

Table 13.22 shows the results of this work, comparing them to biological leaching.

Table 13.22 Effects of pre-treatment method on sulphide gold extraction using cyanide

Test ID (#1005509)	Description	Tail grade (g/t Au)	Recovery (% Au)
CIL1	Untreated concentrate	9.01	20
CIL2	Ultrafine grind to P80 <7 µ	8.32	38
CIL3	Two-stage roast 550°C/650°C	4.66	68
CIL4	Pressure oxidation (POX) 220°C	10.5	26
CIL-Bio1	Bio-oxidation	1.10	93

This work clearly demonstrates the superiority of bioleach over the alternative methods tested.

Bacterial oxidation testwork

Inspectorate undertook bacterial oxidation (bioleach) testwork as a number of separate projects (*ibid*):

- In Project 1005509, two batch bioleach tests were undertaken
- In Project 1107112, a further two bioleach tests (one partial and one maximum) were done
- In Project 1102204, two continuous bioleach test runs were undertaken.

The bioleach testwork shows that the Buffalo Reef concentrate is amenable to bacterial oxidation and that bioleach improves CIL gold recoveries. Table 13.23 summarises the results.

Table 13.23 Biological oxidation tests data

Project	Test	S (Total) % w/w	As % w/w	Fe % w/w	% w/w Solids (unless noted)	Grind Procedure	Test Time (days)	Sulphide Oxidation % w/w	Acid Addition	Mass Change % w/w
1005509	BLT-1	10.10	2.82	11.40	7.40	Polish regrind	13	98.50	31.5	-28.90
1005509	BLT-2	9.04	3.36	10.37	5.70	Polish regrind	14	99.30	48.6	-33.10
1107112	BLT-1A	12.05	2.42	12.11	6	20 min regrind	13	23.30	58.8	-15.30
1107112	BLT-1B	12.05	2.42	12.11	6	20 min regrind	38	99.40	58.8	-40.40
1102204	CBLT-1	Not recorded	Not recorded	Not recorded	50 g/L	No regrind?	19	Not recorded	Not recorded	Not recorded
1102204	CBLT-2	Not recorded	Not recorded	Not recorded	80 g/L	No regrind?	9	Not recorded	Not recorded	Not recorded

Source: Inspectorate, 2012: "Metallurgical Study on Buffalo Reef Gold Project" Appendix V-B

The recommended industry standard for bioleach feed is a particle size distribution with a P80 of between 75 and 45 µm with a nominal top size of 150 µm and a minimum of minus 20 µm. Target grind size is a balance between a finer grind which increases the oxidation rate but potentially reduces residue settling rates and causes rheological and gas dissolution/diffusion problems. The testwork has been done at 7.4% solids. The effect of increasing percent solids up to a typical commercial scale of 20% has not been shown.

The testwork demonstrates that the following conditions produce acceptable recoveries:

- Feed % solids: 7.4%
- Feed size: Less than P80 of 75 µm
- Residence times 2.1 to 2.4 days primary reactor, 2.1 to 2.2 secondaries.

- Concentrate total sulphur grade 9% to 12% w/w (batch tests).

Commercial scale plants typically operate at feed solids of 20% w/w and concentrate sulphur grades of 19% to 25% w/w. The low percentage solids used in the test bioleach oxidation step would normally lead to the requirement for correspondingly large equipment. For the PFS design purposes, Lycopodium agreed with SGM to base the design on industry typical 20% feed solids concentration to bioleach.

For the batch tests the feed sulphur grade was low at 6% to 12% w/w, compared to other commercial bioleach operations, typically 20% to 25% w/w sulphide sulphur, S^{2-}. The feed S^{2-} grade was not reported for the continuous tests, but are assumed to be similar to those of the batch tests.

The batch tests all showed a mass loss. This is due to dissolution of acid soluble minerals. Typically, this is followed by a mass increase due to gypsum and jarosite precipitation. This does not appear to have happened in the Inspectorate tests possibly due to low feed percent solids, although it may be a function of the mineralogy.

Care needs to be taken in the next phase to accurately measure mass loss or gain, and utilise a tie element such as silicon or gold. The absence of a mass increase may mean that the CIL testing of bioleach residues overstates gold extraction, as gypsum and jarosite precipitation have the potential to inhibit gold extraction.

The CIL of the resulting bioleach residue (Bio1) used two procedures. One incorporated a blinding agent, SLS (CIL2) (sic), and without a blinding agent (CIL2). Both procedures provided a gold recovery of 93%.

Further evaluation of bioleaching of the float scavenger tailing also showed this material responded poorly to direct cyanidation and positively to bioleaching prior to CIL. The use of the float tailing to neutralise bio-solutions indicated some neutralising capacity was available from this stream and that additional gold might be leach scavenged from the float tailing following this procedure. This is shown in test CN1 which resulted in leaching over half the contained gold and leaving a leach residue of the final tailing of 0.09 g/t Au.

A continuous bioleach program was performed by Inspectorate on bulk float concentrate produced from a master composite sample under Project 1102204. This composite was labelled as Comp. BR1 and comprised 25% of north zone (Comp. 2) and the remainder from available south zone composites.

Cyanidation performance of intermediate and final bioleach products did not reach that of the batch data until full oxidation was completed. Steady state operations with fully adapted culture were limited due to a shortage of available sample. The CIL results summary performed on samples pulled from the various stages of bioleaching is outlined in Table 13.24.

Table 13.24 Cyanidation response of continuous bioleach products

Test ID (#1102204)	Description	Tail grade (g/t Au)	Recovery (% Au)
CIL9	First bio-tank	6.94	65
CIL10	Second bio-tank	4.69	72
CIL11	Third bio-tank	4.80	70
CIL12	Bioleach continuous product	0.96	90

13.3.2 Summary of main findings from recent testwork

Buffalo Reef

The Inspectorate work concluded by noting bulk flotation and bioleaching of the float concentrate prior to CIL appears to offer the most promising response in treating the Buffalo Reef sulphide samples that were received.

Most of the sample composites consisted of arsenopyrite/pyrite sulphides, with a lesser amount of samples containing elevated stibnite. Gold distribution is substantially as sub-microscopic content in sulphide minerals. The laboratory program focused on the arsenopyrite/pyrite samples, with only modest scoping level work performed on the stibnite samples. Each type of sample responds differently. Additional work is required in context with the developing resource model, particularly in relation to higher stibnite samples depending how prevalent such ore zones are.

The laboratory program shows that the samples received are of moderate hardness as indicated by BBWi and respond well to standard flotation procedures to make a low grade bulk sulphide concentrate suitable for standard cyanide pre-treatment procedures.

The carbonaceous material noted in the head is also confirmed to be concentrated into the sulphide concentrate. Incorporating gravity recovery in the primary grind circuit does not appear to be necessary for arsenopyrite/pyrite material based on the samples tested.

Bulk flotation showed a good response at moderate grinds of approximately 80% passing 74 µm(200 mesh). Extended rougher/scavenger flotation times may be necessary to maximise recovery, based on the initial work. One stage of cleaning in the bench scale work assisted in rejecting gangue and improving gold and sulphur grades with minimal loss of precious metals. Regrinding prior to cleaning was shown to be detrimental and additional upgrading of concentrate would be challenging without losing recovery. Further work on variability testing, resource mineralogy and testing reagent combinations will be necessary to better characterise the flotation procedures. This includes establishing retention time, and the reagent scheme and dosage.

A conceptual flotation flowsheet developed for the arsenopyrite/pyrite ores concluded no need for gravity pre-treatment, however included the use of a moderate primary grinding with limited flotation cleaning with no regrinding of the bulk float concentrate.

Leaching of the whole ore and corresponding flotation concentrates were shown to be refractory to direct cyanidation procedures, including the use of CIL. Direct CIL cyanidation of the Buffalo Reef sulphides resulted in approximately 22% gold dissolution in a composite sample from the north zone (Comp. 2), and approximately 11% in a composite sample from the south zone (Comp. 4). Mineralogical and chemical analyses of the samples received by Inspectorate suggest this is likely due to two reasons. First, the presence of gold either in solid solution or as very finely disseminated particles within associated sulphides, such as arsenopyrite. Second, the presence of naturally occurring organic carbon in the sulphide zone that results in a “preg robbing effect” of any gold that is dissolved into the pregnant solution.

Of the four cyanide pre-treatment methods investigated on the Buffalo Reef flotation concentrate, bioleaching gave the best response, followed by roasting. Pressure leaching and ultrafine grinding gave gold recoveries of less than 50% and are not recommended for further evaluation. Scoping roaster evaluation using aggressive conditions achieved CIL gold recovery of up to 72% on the resulting calcine. Further modifications to the roasting procedures may improve on these results. Batch bioleaching taken to completion on similar concentrate to that used for roasting resulted in CIL gold recovery on the residue ranging from 88% to 93%, depending on the feed sample source. A continuous bioleach test performed on blended Buffalo Reef concentrate achieved a CIL leach recovery approaching 90% for the bio-residue product. Partial bioleaching based on testing performed during both batch and continuous studies resulted in reduced performance suggesting maximum bio-oxidation of the concentrate is required to optimise gold recovery. Full bio-oxidation also resulted in less reagent consumption during CIL as compared to partial bio-oxidation.

Selinsing Deeps

In its scoping report, Inspectorate concluded that the Selinsing Deeps mineral samples that were provided show a poor response to direct gravity and CIL leaching, averaging ~37% gold recovery. Incorporating bacterial leaching of a flotation concentrate significantly improves the CIL recovery as compared to whole ore.

Inspectorate estimated overall gold recovery could be in the range of 89% to 95%. Locked cycle testing, including a separate balance around the gravity circuit would be required to more accurately determine the overall expected recovery. Additional gold can be recovered by cyanidation of the flotation tailing.

13.4 Original oxide process selection and plant performance

The basis of the design of the current Selinsing Project process plant has been drawn from historical testwork on oxide ore.

13.4.1 Comparative data

Table 13.25 shows a comparison of test results for similar ore types that have been derived from different test programs.

As stated previously, 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 CIL. Final gold recovery from carbon strip solution and gravity concentrate leach solution is by electrowinning onto stainless steel cathodes.

Table 13.25 Comparative metallurgical test data

Parameter	Unit	Values			Source	Notes
		Breccia	Quartz	Stockwork		
Proportion	%	20	10	55	SGM, 2006	
		60	**18**	**22**	**SGM, 2008**	Metallurgical SELDD14-16, 2008
UCS	MPa	97.15			Golder rock tests.xlxs	Single value drawn from Golder geotechnical work
		9.6	**6.8**	**5.7**	**Ammtec A11594, Oct 2008**	
Ai		0.173	0.56	0.055	Dover Consultants, 37060	
		0.039	**0.152**	**0.101**	**Ammtec A11594, Oct 2008**	
SG		2.71	2.65	2.76	Dover Consultants, 37060	
		2.71	**2.64**	**2.65**	**Ammtec A11594, Oct 2008**	
BBWi	kWh/t	11.1	17.4	10.7	DovCon2, March 1997(Amdel N8257)	
		16.9	16.8	15.5	Amdel N8302, May 1997	
		11.5	**18.9**	**13.8**	**Ammtec A11594, Oct 2008**	
Combined data						
Head	g/t	2.38			Snowden, January 2007	Based on ore proportions
		2.18			**Ammtec A11594, Oct 2008**	
Gravity recovery	%	52.32			Ammtec 5477 7.2.2	Based on ore proportions
		67.95			**Ammtec A11594, Oct 2008**	
Overall extraction	%	83.8			Ammtec 5477 7.2.2	Based on ore proportions
		91.8			**Ammtec A11594, Oct 2008**	
Lime to leach	kg/t	0.60			Ammtec 5477 7.2.2	Based on ore proportions
		0.46			**Ammtec A11594, Oct 2008**	
Cyanide to leach	kg/t	1.18			Ammtec 5477 7.2.2	Based on ore proportions
		1.39			**Ammtec A11594, Oct 2008**	

In 2009, mining operations commenced at Selinsing. Since then, Monument developed an open pit mine and construction of a 1,200 t/d gold treatment plant in two 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.
- Phase III: The Phase III expansion began on 6 September 2011 with a budget of $8.1 million and was completed in June 2012 on time and at a cost of $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.

13.4.2 Summary of historical and recent plant performance

Early years

Some production results and costs from the early years of operations are provided in Table 13.26, Table 13.27 and Table 13.28.

Table 13.26 Monthly reagent and power cost

Month	Reagent (US$/t ore milled)	Power (US$/t ore milled)
January 2012	4.67	1.85
February 2012	6.14	1.83
March 2012	4.56	1.58
April 2012	3.55	1.67

Table 13.27 Monthly steel ball and reagent consumptions

Month	Steel ball (t)	Hydrated lime (t)	Sodium cyanide (t)	Activated carbon (t)
January 2012	34.1 (1.20 kg/t)	68.35 (2.40 kg/t)	14 (0.49 kg/t)	1
February 2012	21.9 t (0.80 kg/t)	62.68 (2.3 kg/t)	18 (0.66 kg/t)	19
March 2012	27.45 (0.99 kg/t)	62.82 (2.27 kg/t)	17 (0.61 kg/t)	-
April 2012	36.9* (1.41 kg/t)	63.615 (2.43 kg/t)	15 (0.57 kg/t)	-

**Above budget due to harder material of the fresh rocks*

Table 13.28 2010 to 2011 financial year production data

Production 2011		Q1	Q2	Q3	Q4	Total	Budget	Variance
Crusher	t	95,736	93,286	92,721	91,472	373,215	367,614	101.5
	t/hour	80.4	81.2	79.7	84.4	81.4	80.0	101.8
Ball mill	t	89,834	87,845	87,780	86,540	351,999	367,614	95.8
	t/hour	43.0	41.4	41.8	41.0	41.8	42.0	99.5
Availability	%	94.6	96.1	97.1	96.6	96.1	92.0	104.5
Feed grade	g/t Au	4.08	4.42	4.18	4.58	4.32	3.51	123.1
Tailings grade	g/t Au	0.39	0.28	0.26	0.27	0.31	0.31	100.00
Recovery	%	90.0	93.7	93.7	94.1	92.9	91.2	101.9
Gold recovered	kg	330.9	363.9	343.6	373.3	1,411.6	1,175.9	120.0

Recent years

In Table 13.29, the ore blends processed in 2016 can be seen. The gold reconciliation results for 2016 are depicted in Table 13.30. The production and recovery results as well as financial results of the Selinsing Gold Mine since 2010 until 2016 are presented in Table 13.31.

Some of the processing plant monthly processing data since 2011 were plotted to provide some gold recovery versus grade relationships for various ore blends from historical plant data. These are depicted in Figure 13.4 to Figure 13.7.

There is currently a proposal before management to convert leach tank 1 into a CIL tank to eliminate the risk of gold loss due to preg robbing of organic carbon material present in the ores.

Table 13.29 Ores processing FY2016

Ore blended	Tonnes	Au g/t	% material treated
Selinsing oxide (SLG + LG + HG)	576,993	0.76	58
Selinsing leachable sulphides	57,819	0.90	6
Buffalo Reef oxide	194,149	1.20	20
Buffalo Reef leachable sulphides	15,629	1.04	2
MOSF/old tailings	147,478	0.92	15

Table 13.30 Gold reconciliation FY2016

Description	Year to date			
	Tonnes	Au (g/t)	Au (g)	%
Wet tonnes crushed	893,650			
Dry tonnes milled and reclaimed	992,070			
Bullion			566,734	
Inventory difference			20,800	
Total gold recovered		0.59	587,544	
Reconciled gravity gold recovery		0.21	208.727	23.9
Reconciled CIL gold recovery		0.38	378,816	43.4
Reconciled total gold recovery: milled and reclaimed				67.4
Gold in gravity tail: milled and reclaimed		0.81	802.862	
Gold in final tail: milled and reclaimed		0.29	284,819	
Reconciled gold in milled feed: milled and reclaimed		0.88	872,363	
Mill feed gold: milled and reclaimed		0.98	967,982	
Assay gravity recovery				17.1
Assay CIL recovery				53.5
Assay total recovery				70.6
Grade variance				-11.0

Table 13.31 2010-2016 financial years' production data and financial results

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

(a) Total cash cost includes production costs such as mining, processing, tailing facility maintenance and camp administration, royalties and operating costs such as storage, temporary mine production closure, community development cost and property fees, net of by-product credits. Cash cost excludes amortisation, depletion, accretion expenses, capital costs, exploration costs and corporate administration costs.

Figure 13.4 Recovery curve for Selinsing oxides only

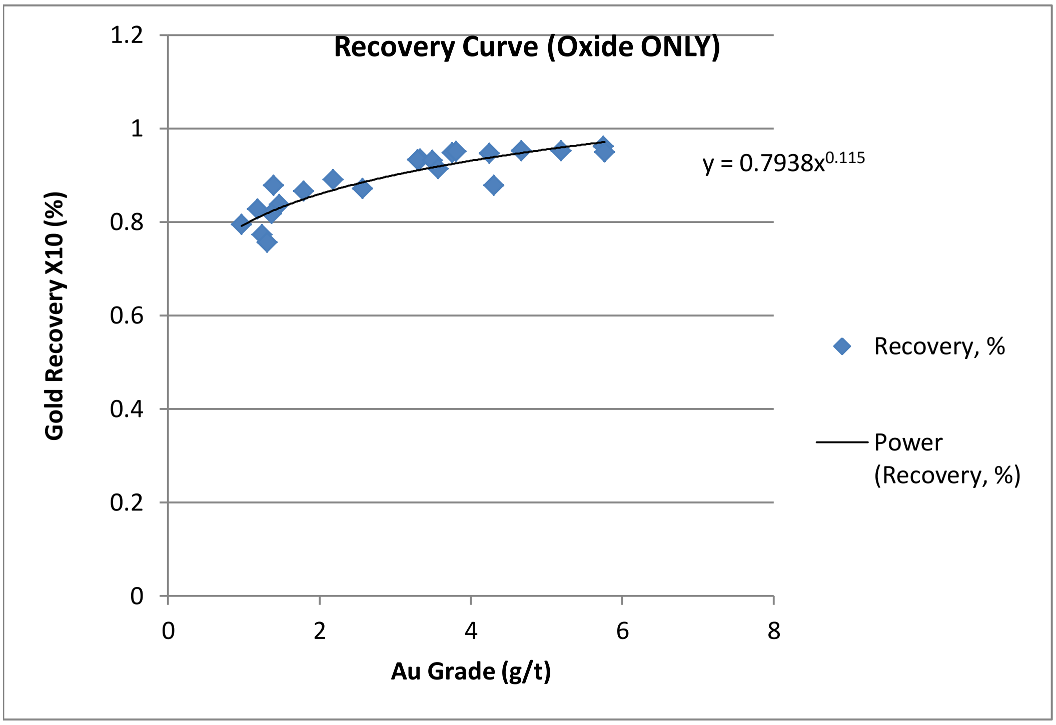


Figure 13.5 Recovery curve for Selinsing and Buffalo Reef oxides

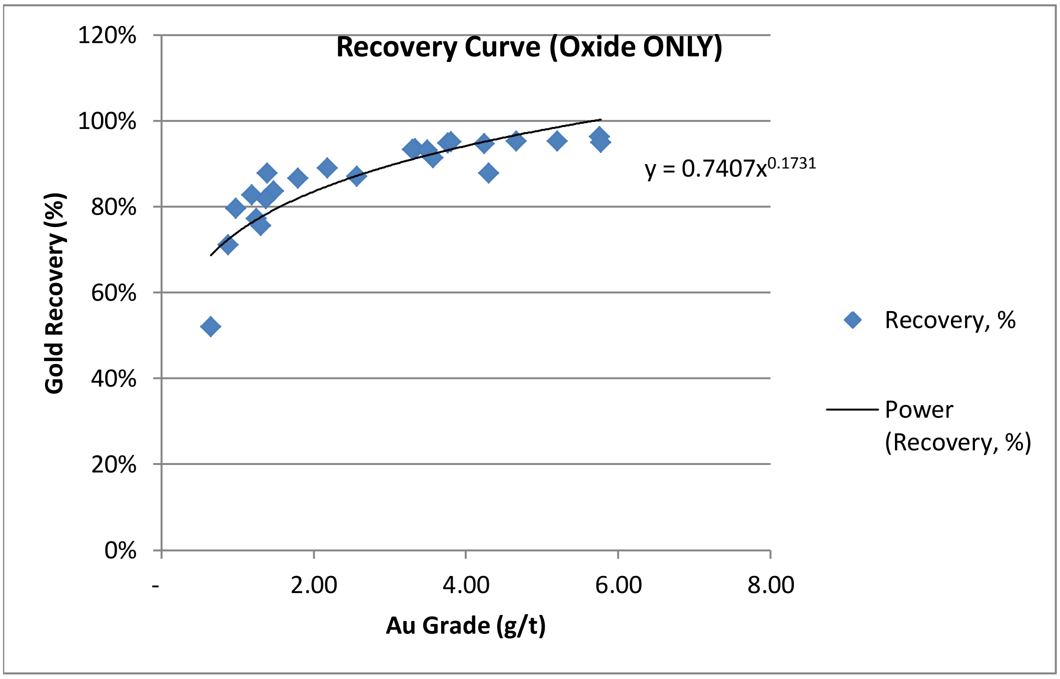


Figure 13.6 Recovery curve for oxides + <15% sulphides

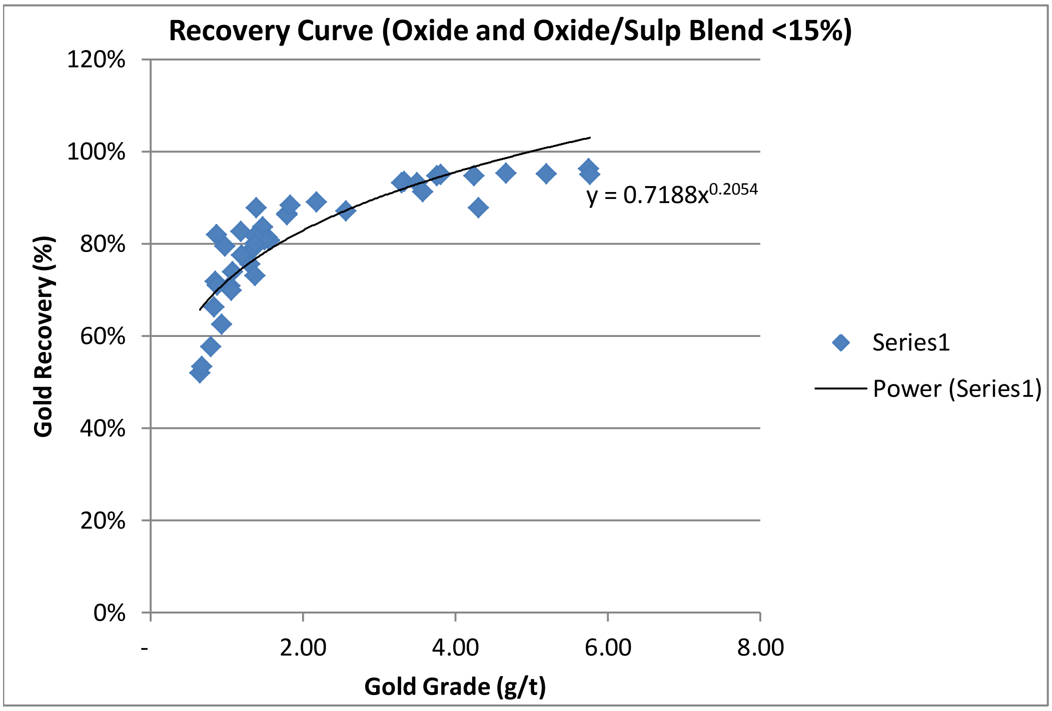
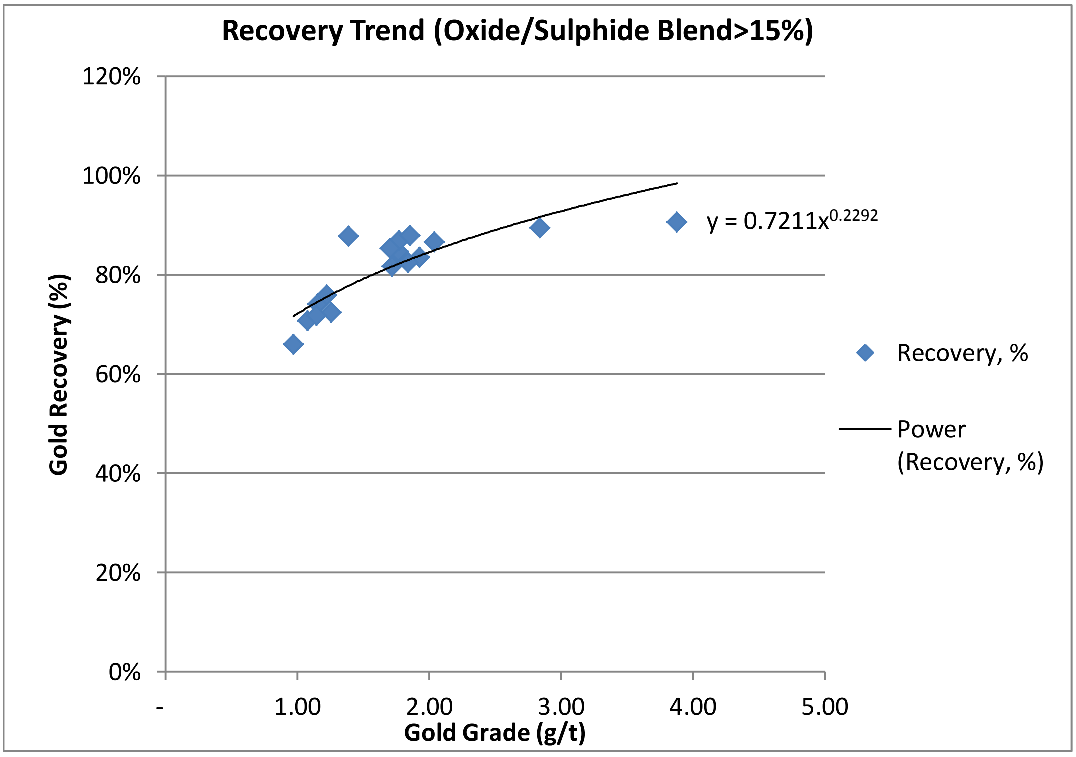


Figure 13.7 Recovery curve for oxides + >15% sulphides



In selecting Buffalo Reef sulphide ore for inclusion in current mill feed, SGM first assessed cyanide leach response by applying a laboratory cyanide leach test to drill core samples. Material with a cyanide extraction of >70% was directed to the run of mine (ROM) to be blended with Selinsing oxides ahead of processing.

13.5 Process selection for sulphide/fresh ore

The principal implications for process design arising from the Inspectorate work in particular include:

- Gold occurrence in both Buffalo Reef and Selinsing Deeps sulphides is sub-microscopic in nature – either as inclusions, or in solid solution
- Bioleaching is the most effective for of pre-treatment ahead of CIL when compared with fine grinding, roasting and pressure oxidation
- The sulphide ores are amenable to conventional crushing and grinding
- The sulphide ores respond favourably to rougher flotation, but less favourably to cleaner flotation where there is a risk of sustaining significant gold losses when attempting to upgrade rougher flotation concentrate.

Lycopodium developed process design criteria for Buffalo Reef sulphide ore based on the Inspectorate findings, which are detailed below. Given superior flotation response and similar bioleach response, it is reasonable to assume that these criteria would apply adequately to Selinsing Deeps sulphide ore.

Table 13.32 provides a summary of the key process design criteria components.

13.5.1 Summary of process description

Existing circuit

The existing oxide ore flowsheet at Selinsing is designed to treat 1 Mt/a of oxide ore and comprises the following process steps:

- Tertiary crushing and screening of ROM ore to produce a -15 mm product
- Primary and secondary ball milling closed with cyclones to produce a nominal product P80 of 75 µm
- Primary gravity gold recovery employing Knelson concentrators within the mill recirculating load
- Primary cyanide leach and secondary CIL with 36 hours’ residence time
- Conventional carbon elution and gold electrowinning to stainless steel wool
- CIL tailings disposal to conventional tailings management facility equipped with supernatant process water recovery
- Cyanide and lime mixing, storage and distribution systems.

Bioleach incorporation at Selinsing

After the Phase IV expansion, the Selinsing plant will incorporate the following unit process operations:

- Three stages of crushing. The existing equipment does not require modification for the Phase IV expansion.
- Primary and secondary ball milling. The existing equipment does not require modification for the Phase IV expansion, with the exception of the incorporation of a flash flotation cell receiving feed from the grinding circuit cyclone underflow and returning tailings to the primary mill feed. The milling circuit will continue to produce a product P80 of 75 µm.

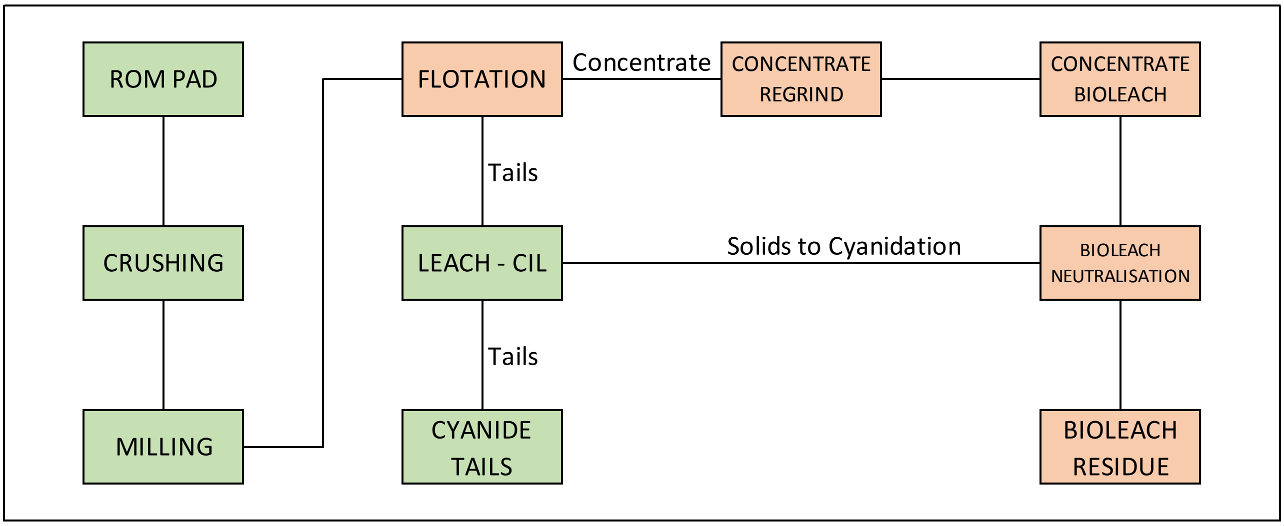
- Gravity recovery circuit. The gravity recovery circuit will not be required for the expansion initially, but will be left in place for possible subsequent use.
- Two-stage flotation, roughing scavenging followed by one stage of cleaning. Testwork has shown that there is no advantage in regrinding rougher/scavenger concentrate prior to cleaning.
- Concentrate regrind and storage. Cleaner concentrate will be reground to a product size P80 of 45 µm prior to bacterial oxidation to maximise the sulphide oxidation achieved.
- Bacterial oxidation reactors consisting of a one train of six oxidation reactors (three primary reactors in parallel and three secondary reactors in series) to produce an oxidised residue.
- A three-stage counter-current decant (CCD) circuit to produce a washed oxidised residue product.
- A liquor neutralisation circuit consisting of six tanks in series, to stabilise arsenic, neutralise acid and precipitate sulphate ions as calcium sulphate (gypsum).
- CIL circuit consisting of three leach tanks followed by six CIL tanks in series. This is existing equipment and does not require modification for the expansion other than the provision of piping tie-ins. The CIL circuit will have excess capacity when treating bacterial oxidised residue. Consideration can be given to utilising some of this capacity for flotation concentrate storage, or for cyanide leaching of flotation tailings.
- Elution circuit with gold recovery to Dore. The existing equipment does not require modification for the Phase IV expansion.
- CIL tailings decant water detoxification. The existing equipment will have excess capacity for the Phase IV expansion and does not require modification.
- Tailings pumping. Flotation tailings will be combined with CIL tailings in the Phase IV expansion. As a result, the tailings pumping and disposal duty will be similar to the existing plant and will not require modification. A portion of the flotation tailings can potentially be used to partially neutralise the acidic CCD wash effluent. However further work is required to define the design parameters if this option is to be included in the final plant.
- Tailings return water reclaim. Water reclaim from the existing TSF will be cyanide containing and not suitable for the flotation or bacterial oxidation sections of the expanded process. As a result, the Phase IV expansion includes a segregated water system. The existing process water tank and pumps will continue to accept cyanide containing water from the existing TSF which will be used for some existing uses including reagent make-up and acid wash and elution requirements.
- A new tailings facility will be required for the neutralisation effluent. The new TSF was not part of the Lycopodium PFS scope and is not included in the design or cost estimate.
- Non-cyanide containing water will be reclaimed from the neutralisation effluent TSF and returned to the mill for use in milling, flotation and bacterial oxidation. Raw water and reclaimed non-cyanide water will be required for the milling circuit, flotation circuit and bacterial oxidation, neutralisation, CCD wash and associated reagents make-up requirements.
- Reagent mixing, storage and distribution systems. The existing lime system will be mothballed. A new lime system is included to service the lime requirements of the entire plant. Additional reagents, nutrients and treatment chemicals are required for the expansion and these are included in the PFS design and cost estimate.

Table 13.32 Bioleach process design criteria summary

Criteria	Units	Value	Source
Ore			
Ore throughput	t/a	950,000	SGMM
ROM ore gold grade	g/t	2	Lycopodium
Overall gold recovery	%	86	Lycopodium
Sulphur grade – nominal	% S	1.2	SGMM
Sulphur grade – maximum	% S	1.37	SGMM
Grinding			
Primary product size – P80	microns	75	SGMM
Regrind product size – P80	microns	45	Testwork
Plant availability	%	91.3	Lycopodium
Flotation			
Mass pull to concentrate	% feed	10	Testwork
Concentrate sulphur grade – design	% S	11.3	Testwork
Bacterial oxidation			
Capacity – nominal	t/a	100,558	SGMM
Plant availability – design	%	95	Lycopodium
Sulphur oxidation	%	91.2	Lycopodium
CCD			
Number of CCD stages	number	3	Lycopodium
Neutralisation			
Flotation tails used for neutralisation	t/t concentrate	3.2	Lycopodium
pH stage 1		4 - 5	Lycopodium
pH stage 2		6 - 8	Lycopodium
Neutralisation reagent		flotation tails/limestone/lime	SGMM
Medium pressure process air			
Air requirement – bacterial oxidation reactors	Nm^3/h	46,500	Lycopodium
Neutralisation	Nm^3/h	11,731	Vendor
Reagents			
Limestone neutralisation	kg/m^3	50	Lycopodium
Nutrient addition – $(NH_4)_2SO_4$	kg/t ore	3.23	Lycopodium
- $(NH_4)H_2PO_4$	kg/t ore	0.6	Lycopodium
- K_2SO_4	kg/t ore	0.72	Lycopodium

Figure 13.8 provides a simplified illustration of the integration of sulphide ore processing into the existing oxide ore flowsheet.

Figure 13.8 Integrated Selinsing bioleach flow diagram



Note: Green = existing process; Red = additional bioleach components

13.6 Overall opinion of the adequacy of the testwork

The Qualified Person has provided the summary and write-up for this section based on previous NI 43-101 reports, Selinsing Oxide Process Plant Performance up-to-date and recent sulphide testwork completed by Inspectorate and used by Lycopodium to develop a process flow diagram and operating and capital cost for the flotation and bioleach project (Phase IV).

The Qualified Person's opinion is 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 PFS level of accuracy.

14 MINERAL RESOURCE ESTIMATES

Mineral Resources reported in Section 14 were 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.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

14.1 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; however, Snowden understands that mining approvals in the Felda area at Buffalo Reef are still to be approved from the relevant authorities, although 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, a limited amount of 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 the proposed bioleach process. Successful commercial scale treatment of the sulphide ore will depend on successful implementation of the bioleach process at the Selinsing processing plant.
- There are no known infrastructure issues.

14.2 Selinsing Mineral Resource estimate

14.2.1 Geological interpretation

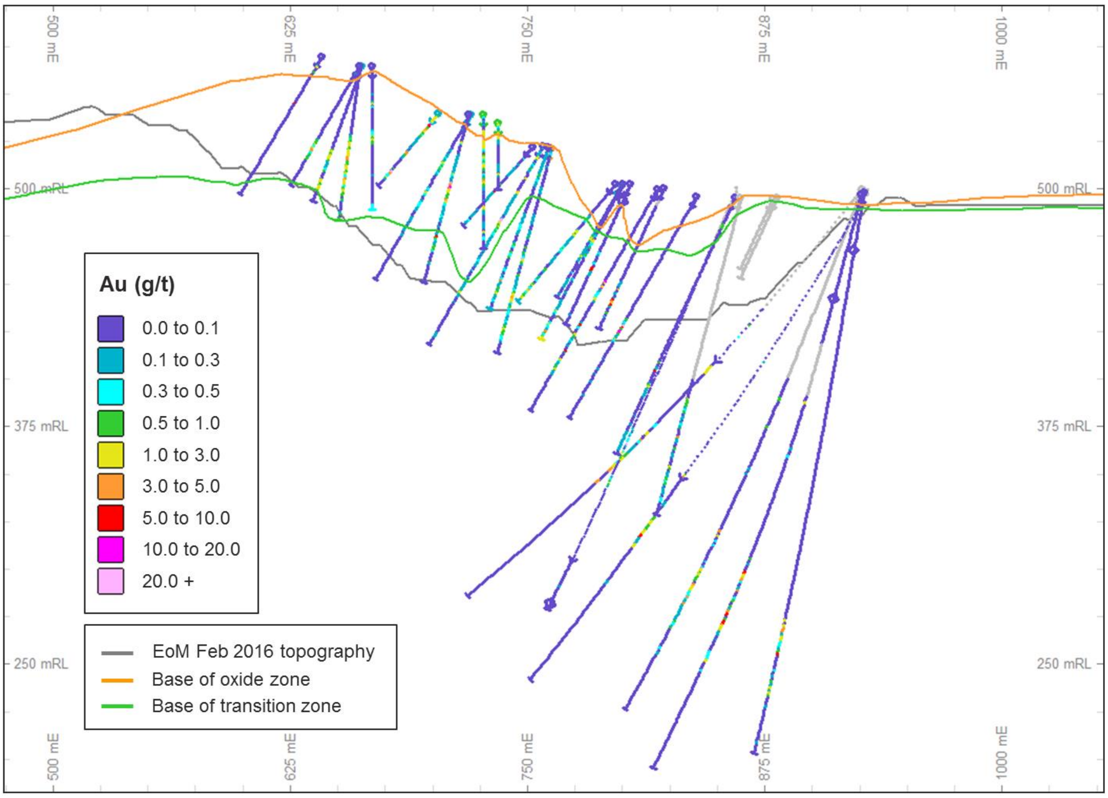
Oxidation

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 14.1.

Figure 14.1 Example west-east section (1920 mN) showing interpretation of oxidation surfaces



Source: Snowden

Mineralisation

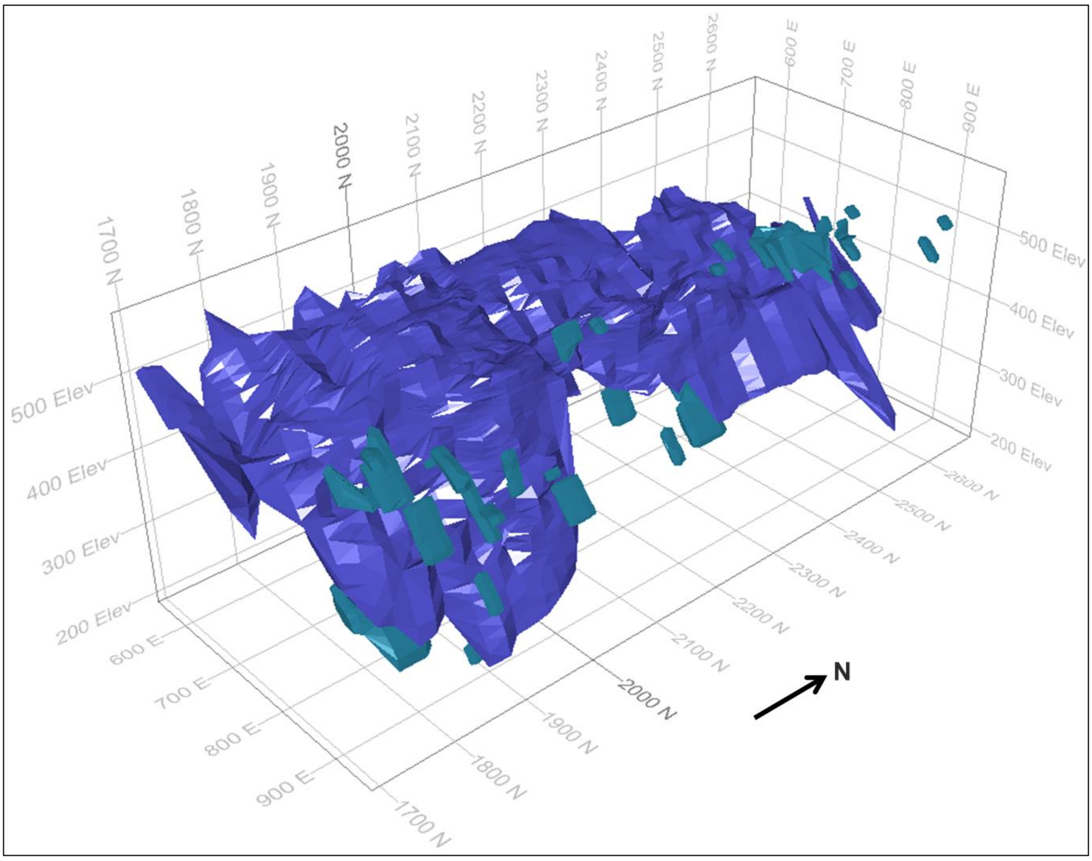
The sectional interpretation of the Selinsing gold mineralisation was completed by Monument based on east-west sections with a section spacing of 20 m. Logging information from RC and diamond drillholes, including vein intensity, alteration and structural information, along with the gold grades, were 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 m to 50 m thick (locally up to 80 m thick) in the southern half and narrows in the northern half to 10 m to 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 14.2 and an example cross section is shown in Figure 14.3.

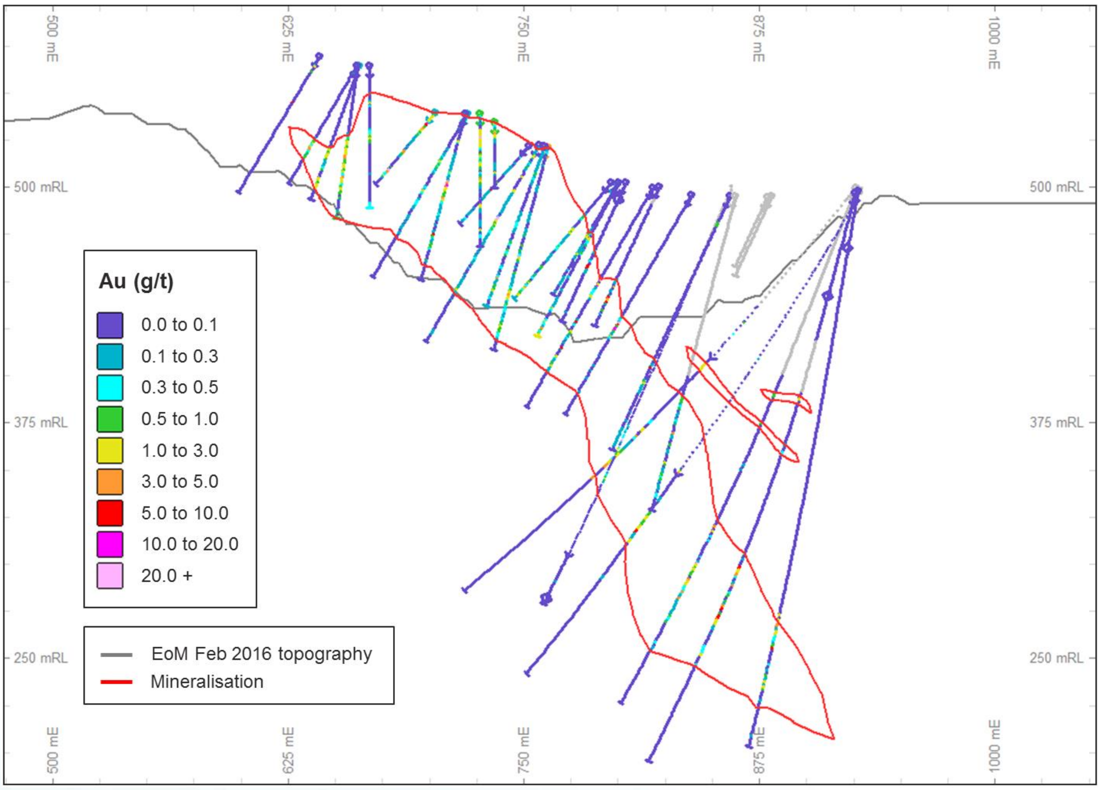
A minimum of three sample intervals (generally 3 m for RC holes and similar for diamond holes) was used for the interpreted mineralised intersections. External dilution was only included for isolated intervals with continuity from neighbouring intervals, either down-dip in the same section or along strike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

Figure 14.2 Oblique view (looking northwest) of the Selinsing mineralisation interpretation



Source: Snowden

Figure 14.3 Example west-east section (1920 mN) showing mineralisation interpretation



Source: Snowden

14.2.2 Drillhole data analysis

For the Selinsing model, 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 was used for the grade estimates in the mineralised domains.

The cut-off date for the data used in the model was for drilling assays received up to 24 February 2016.

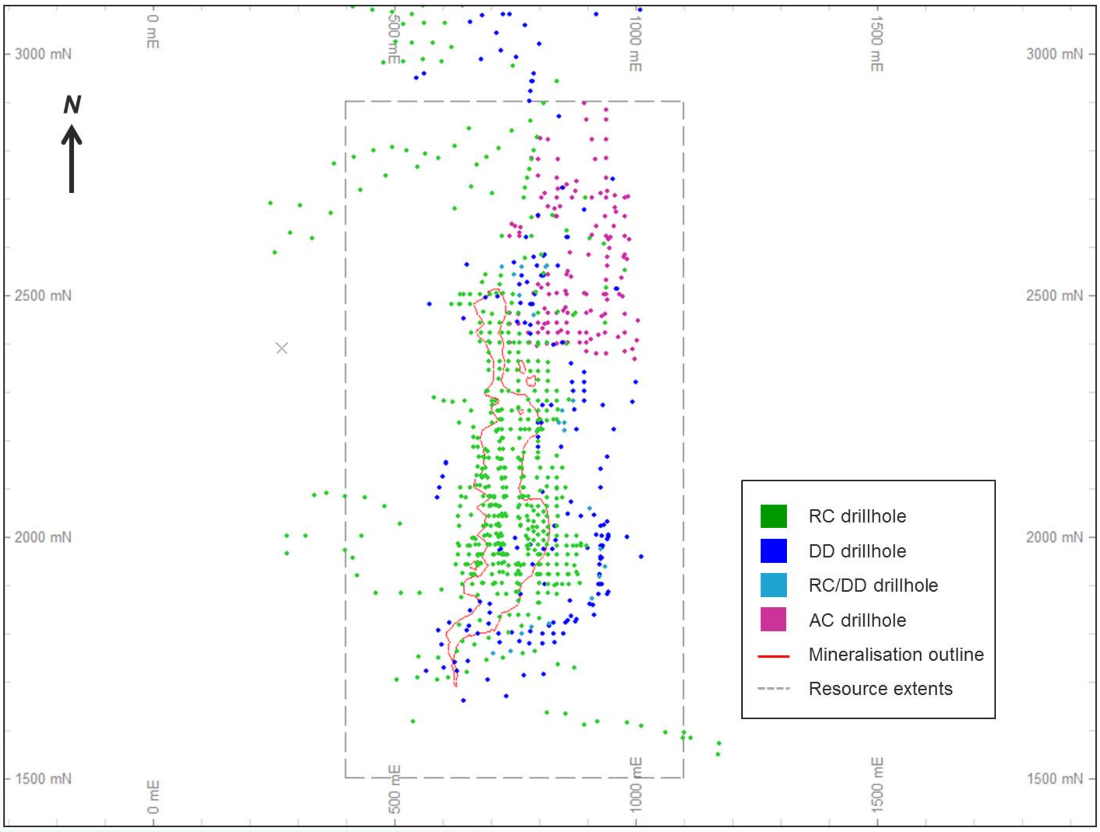
Table 14.1 Drilling data used for Selinsing model (south of 2900 mN)

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 historical tailings and were excluded from the resource estimation*

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.

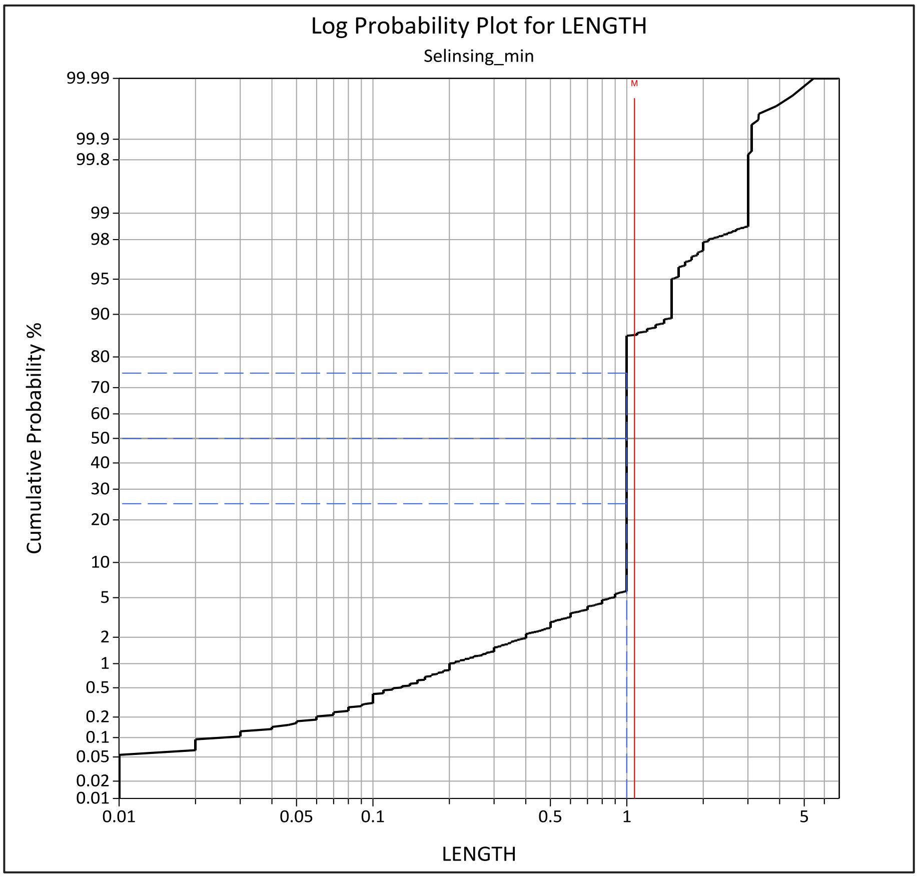
Figure 14.4 Collar location plan for Selinsing resource model area



Sample compositing

The drillhole data was composited downhole prior to running the estimation process using a 1.5 m compositing interval 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 14.5. The compositing interval of 1.5 m 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 14.5 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.

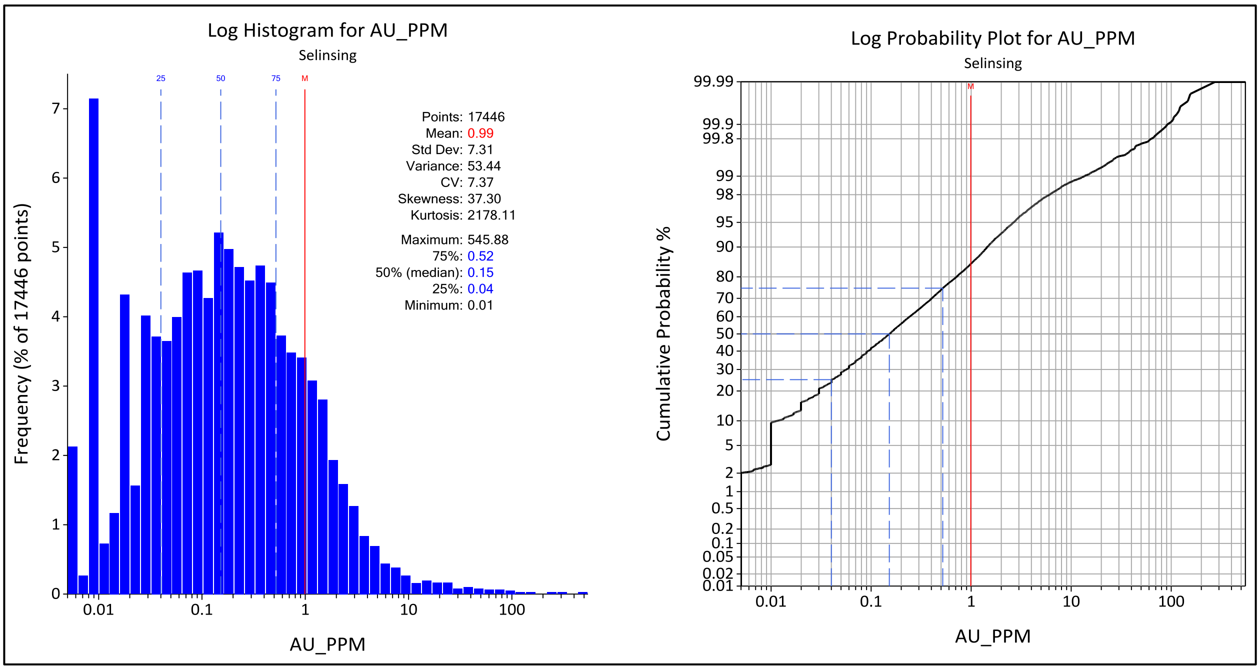
Summary statistics of composites for gold for the combined mineralised domain (i.e. MINZONE=3000) are presented in Table 14.2 and log histograms and log probability plots are presented in Figure 14.6. The statistics show that the Selinsing mineralisation has a strongly positively skewed gold grade distribution with a very high coefficient of variation ("CV" = ratio of the standard deviation to the mean).

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 MIK to estimate the block gold grades.

Table 14.2 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 14.6 Log histogram and log probability plot for gold for combined Selinsing mineralised domains



Source: Snowden

Top-cuts

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

14.2.3 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 14.3. 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 14.3 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

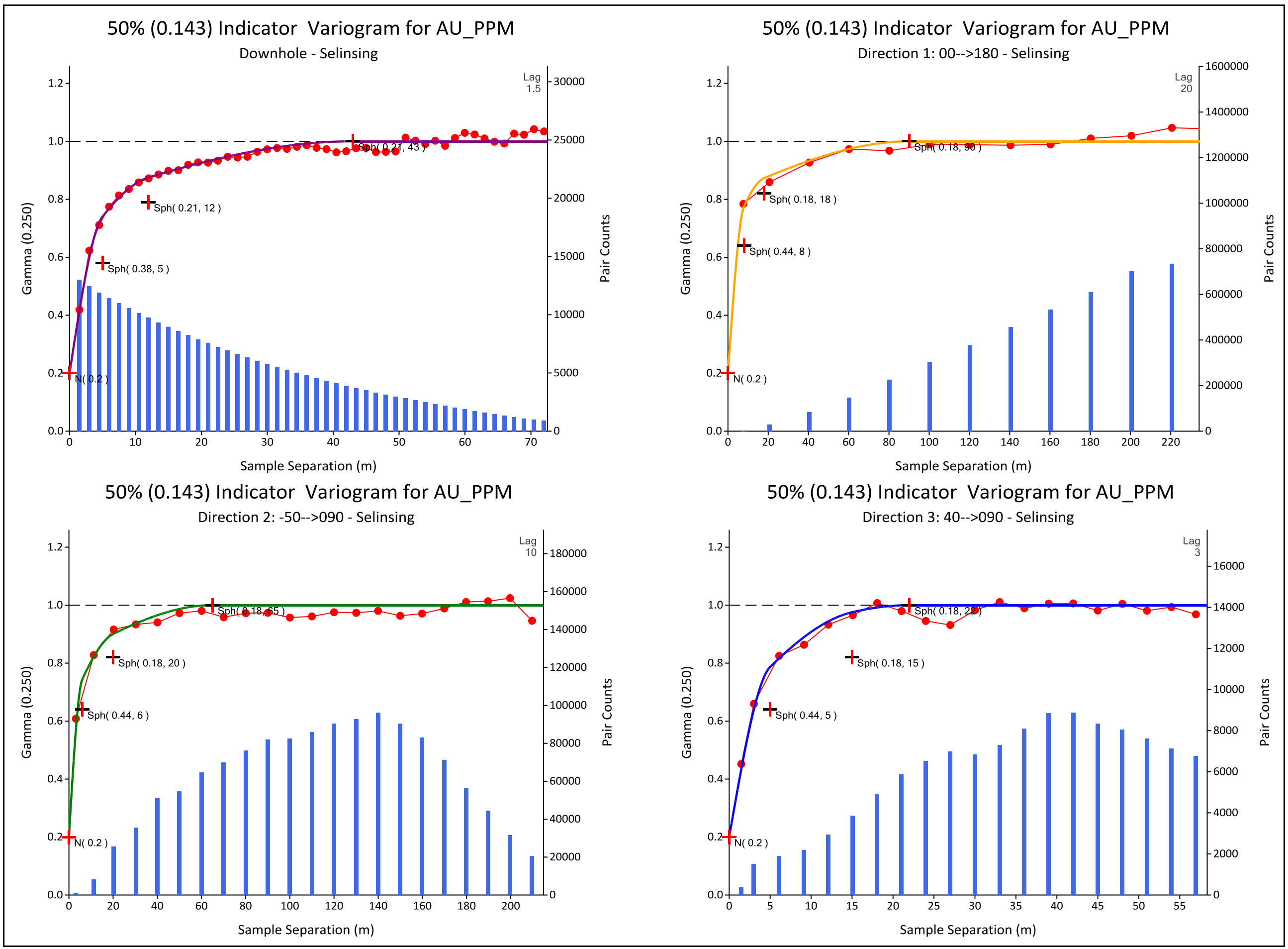
Variograms for the mineralised domain were modelled based on the following general approach:

- 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 14.5 and the median indicator (i.e. 50th percentile) variogram model is shown in Figure 14.7.

Figure 14.7 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 14.2. The upper and lower tail model parameters are summarised in Table 14.4 for each domain.

Table 14.4 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 14.5 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

14.2.4 Block model and grade estimation

Kriging neighbourhood analysis

A kriging neighbourhood analysis (KNA) was performed using Snowden Supervisor software to optimise and validate various kriging parameters, based on the median (50th percentile) indicator variogram for gold. 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 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 14.6.

Table 14.6 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 historical tailings material.

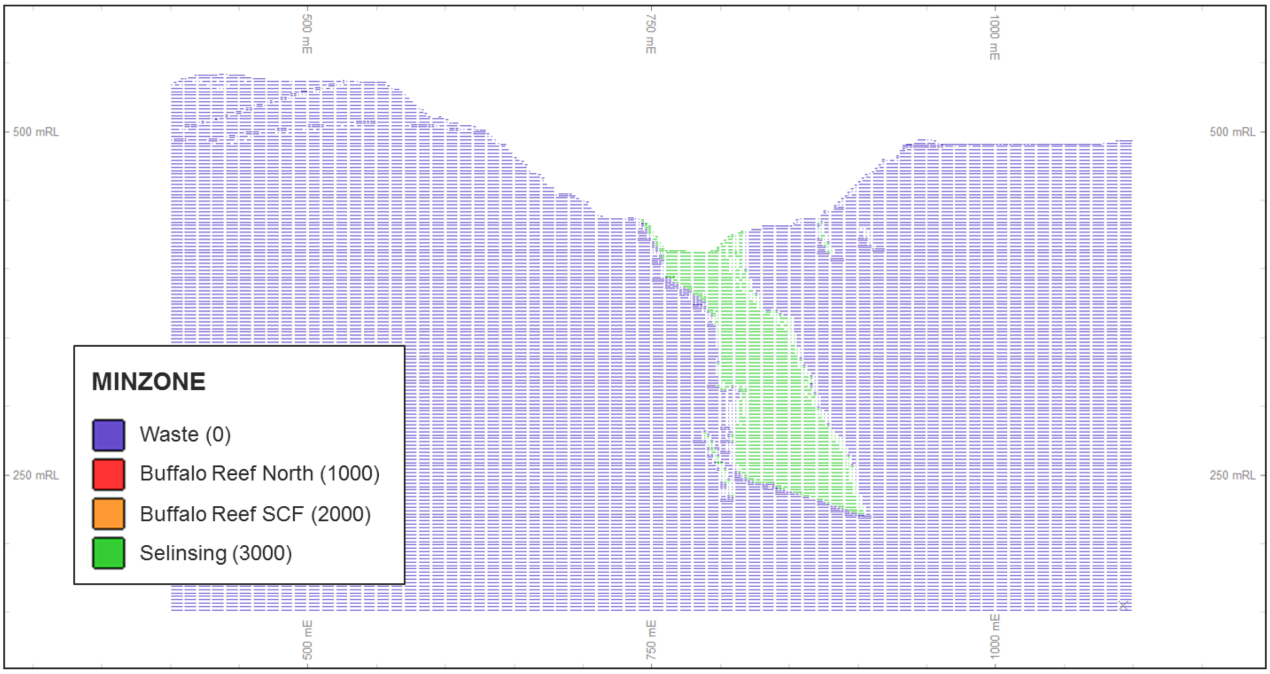
Mineralisation

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 14.7 and an example west-east cross section showing the MINZONE coding is shown in Figure 14.8.

Table 14.7 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	Historical tailings material

Figure 14.8 Example west-east section (1950 mN) showing block model MINZONE coding



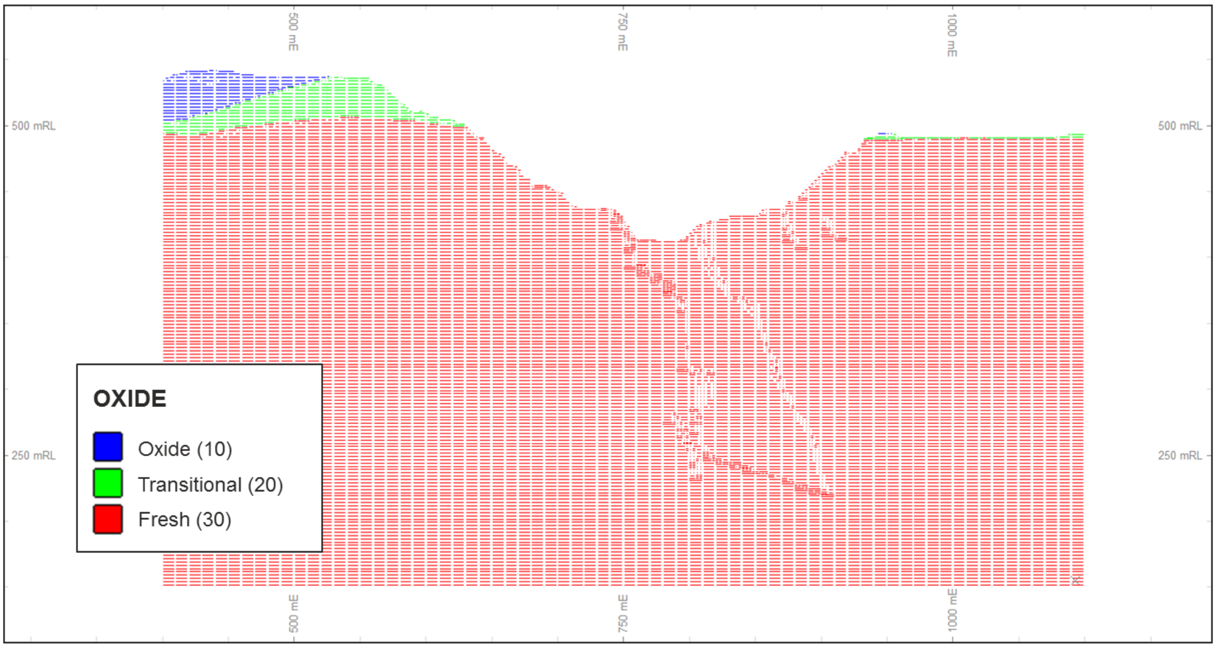
Oxidation

The oxidation zones were coded using a field called OXIDE. Field codes are defined in Table 14.8 and an example west-east cross section showing the OXIDE coding is shown in Figure 14.9.

Table 14.8 OXIDE field coding

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

Figure 14.9 Example west-east section (1950 mN) showing block model OXIDE coding



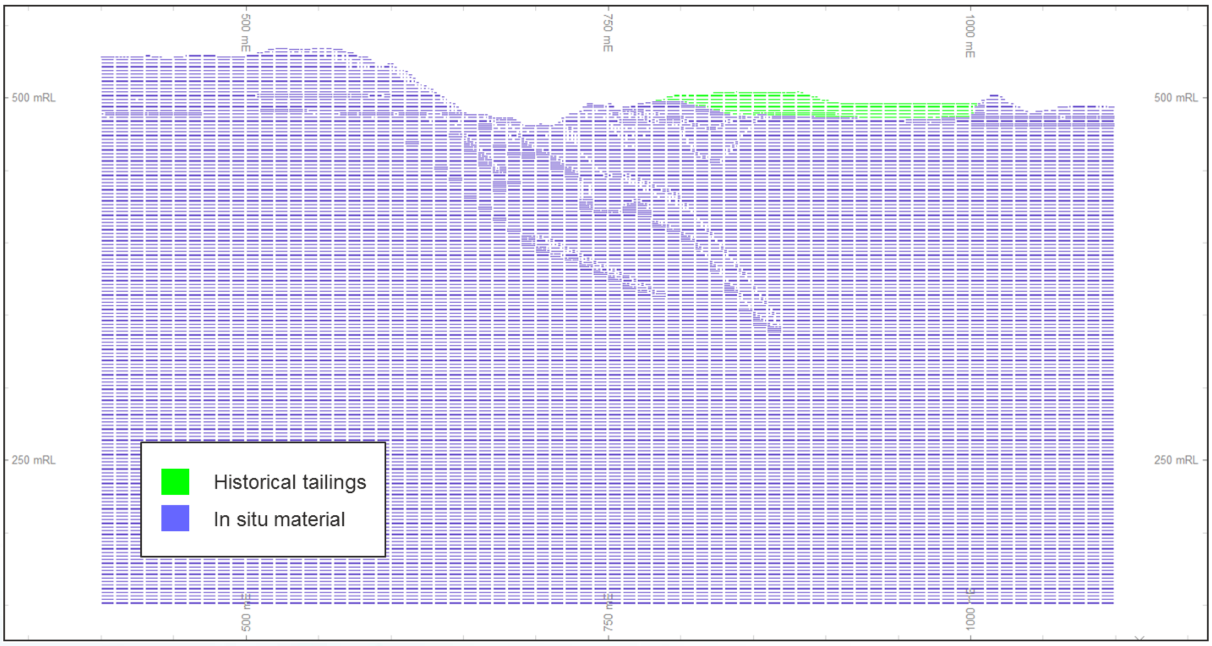
Tailings

The extent of the historical tailings material was coded using a field called TAILS, based on a wireframe solid provided by Monument. Field codes are defined in Table 14.9 and an example west-east cross section showing the TAILS coding is shown in Figure 14.10.

Table 14.9 TAILS field coding

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

Figure 14.10 Example west-east section (2400 mN) showing block model TAILS coding



Grade estimation methodology

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 14.4). 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 14.10. The median value was used for the default gold grade due to the skewed nature of the grade distributions.

Table 14.10 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

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 by 15 m by 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 14.11. 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 14.11 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**

Model validation

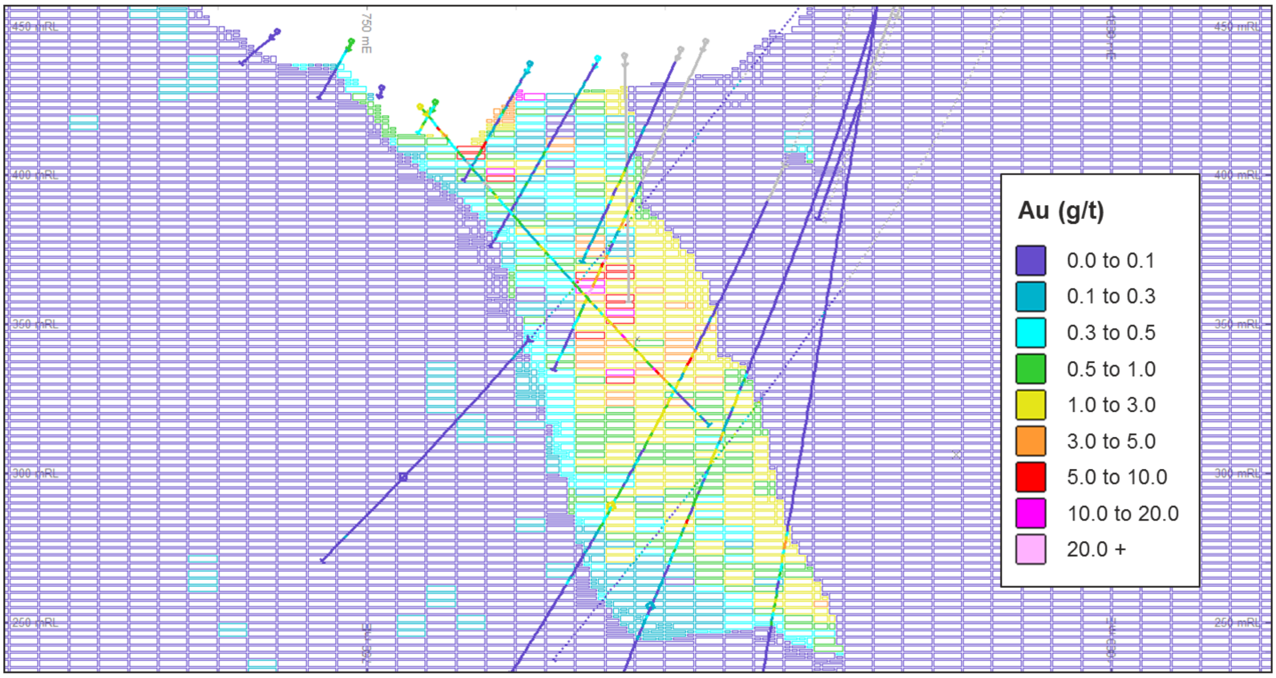
The block grade estimates were validated using:

- A visual comparison of block grade estimates and the input drillhole data
- A global comparison of the average composite (naïve and declustered) 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 14.11)
- 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 14.12)
- 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 14.12).

Figure 14.11 **Example west-east section (1990 mN; ±10 m) showing block grade estimates against the input drillhole composites**



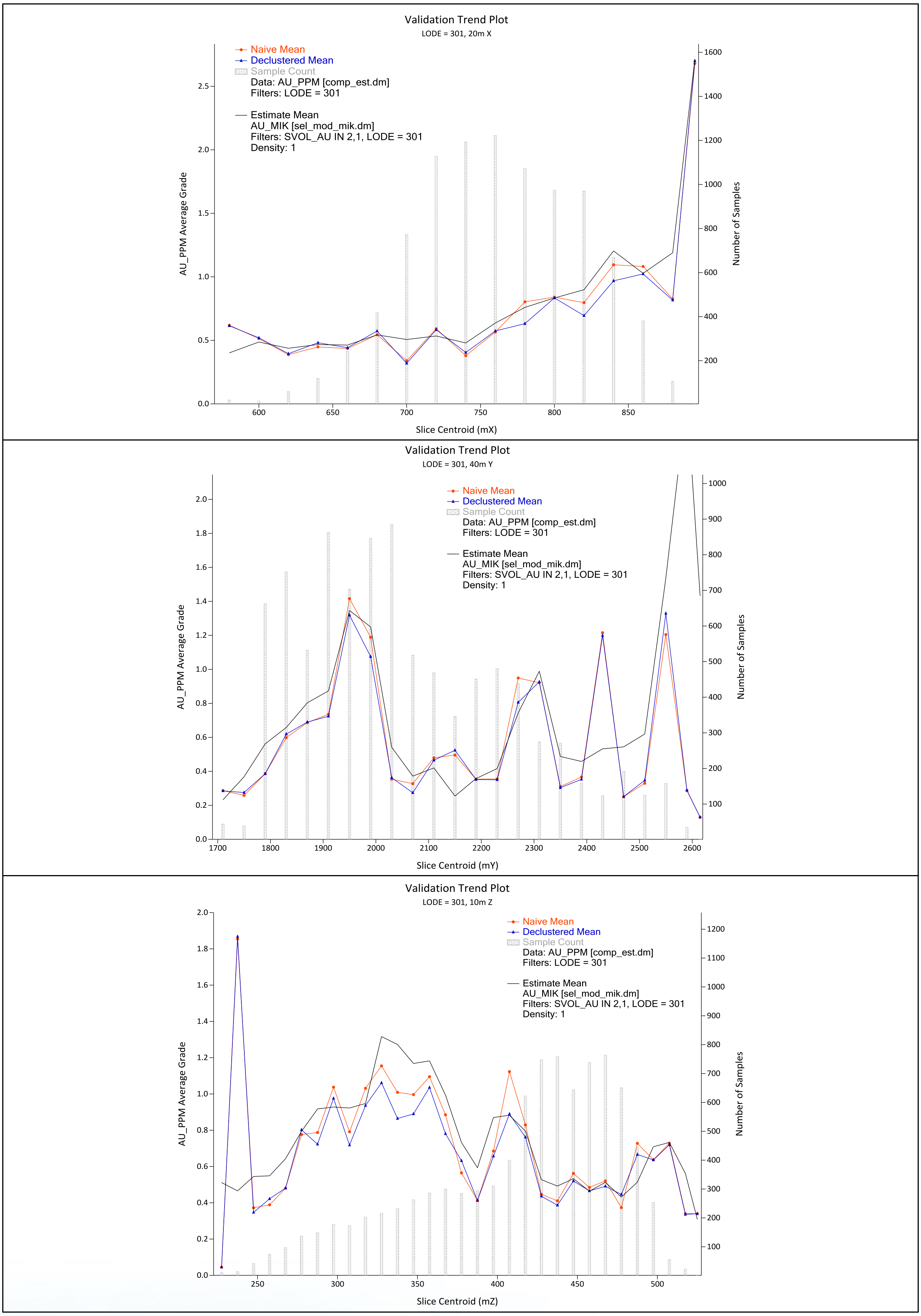
Source: Snowden

Table 14.12 **Selinsing model validation summary statistics**

Statistic	Input samples Naïve Au (g/t)	Input samples Declustered Au (g/t)	Block model estimate 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 14.12 Validation trend plots in the Y (top) and Z (bottom) directions



Source: Snowden

14.2.5 Bulk density analysis

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 kg to 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:

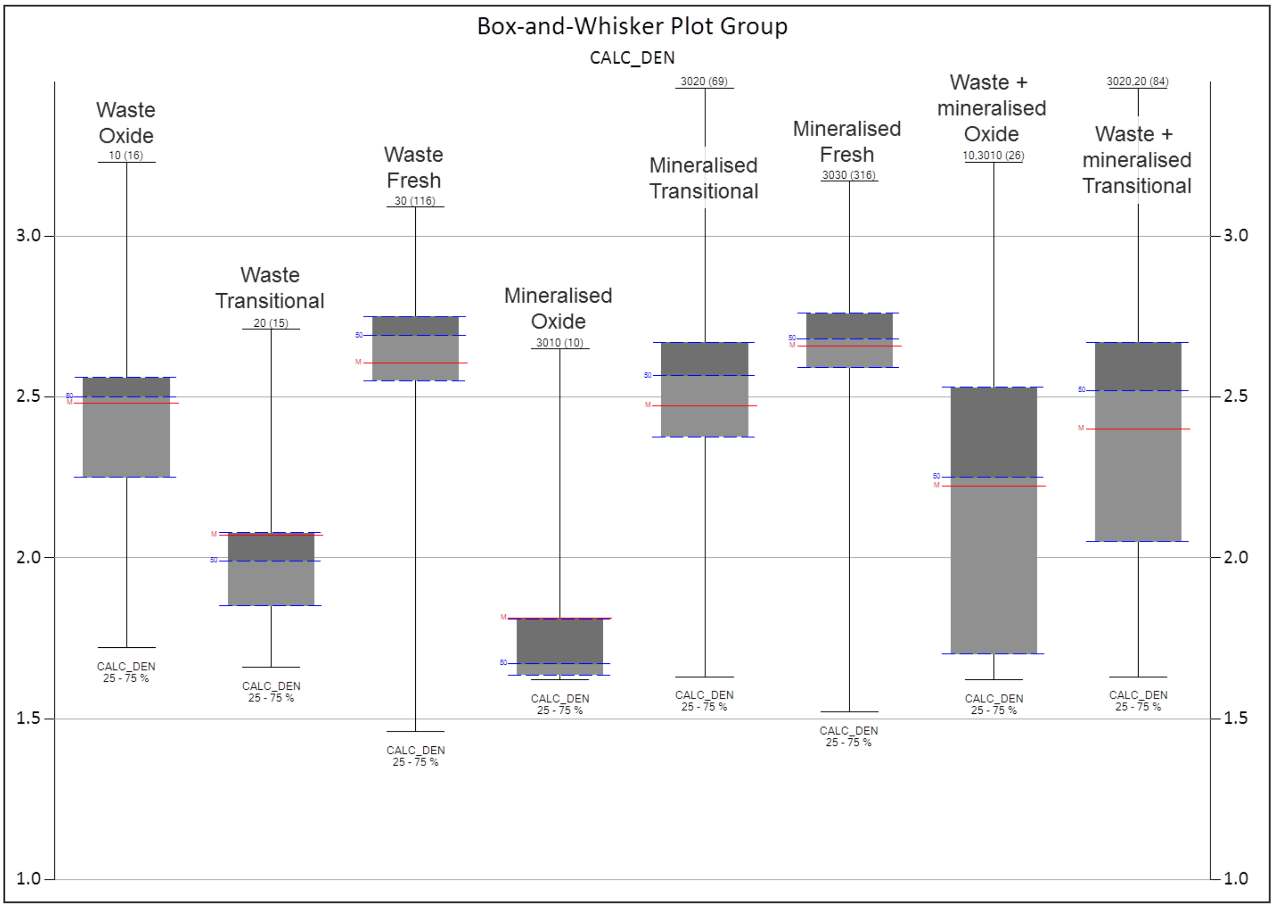
- The sample is dried in an oven at 105°C for 48 hours.
- Dry weight is determined.
- Sample is coated in wax and the dry weight with the wax is determined.
- The wax-coated sample is placed into a basket and submerged in water to determine the weight of the wax-coated sample suspended in water. Care 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 t/m^3 to 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 14.13.

Figure 14.13 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 14.13. 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 1.18 t/m^3 for the historical tailings material was provided by Monument.

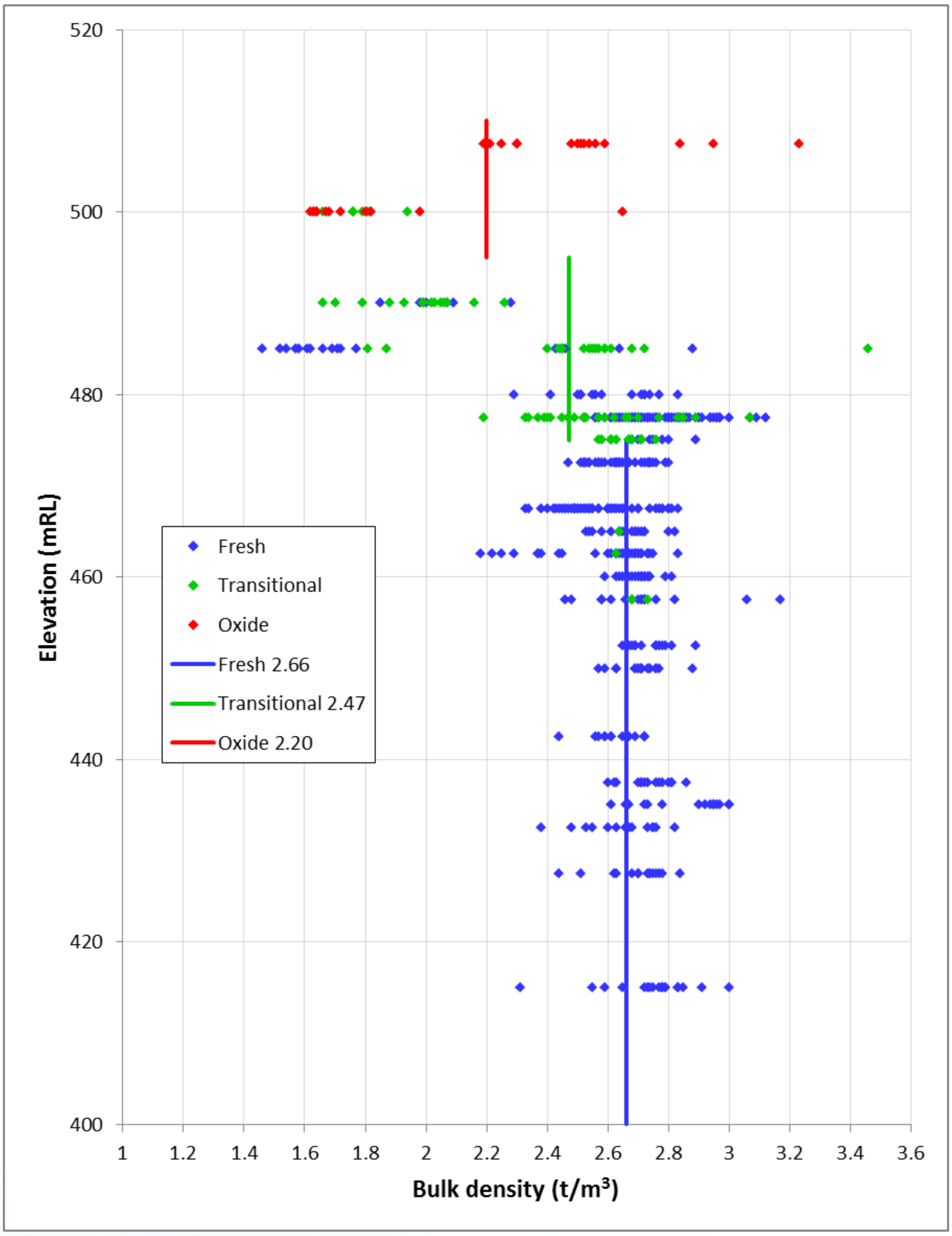
A plot of the bulk density values of all samples against the depth, with the assigned bulk density values for the mineralisation shown by solid lines, is presented in Figure 14.14 (the depth of the oxide and transitional boundaries in Figure 14.14 are indicative only). Snowden believes that the values 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 14.13 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
Historical tailings	-	1.18

Figure 14.14 Bulk density depth profile for Selinsing



Source: Snowden

14.2.6 Mineral Resource classification

The Selinsing 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 the company'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 Mineral Resource estimate is outlined as follows:

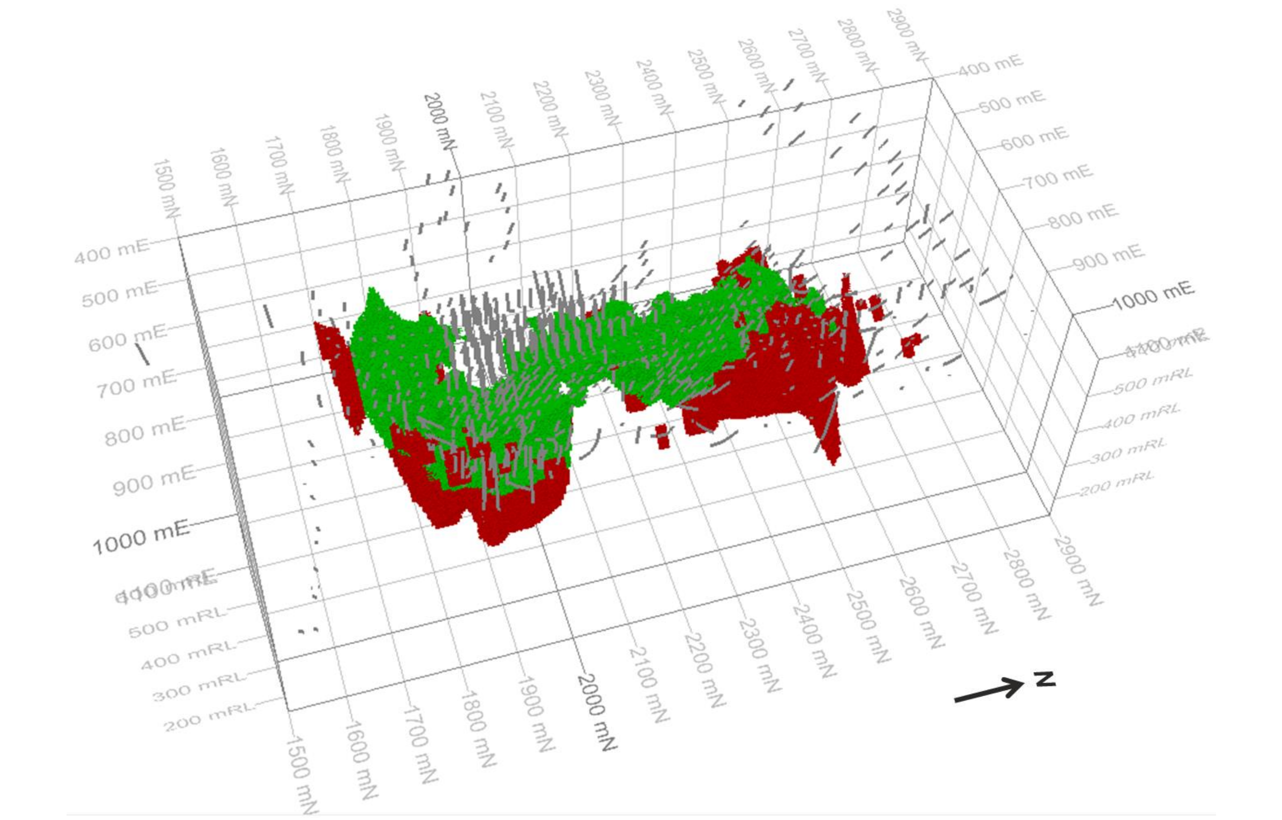
- Where the drilling density was approximately 40 m along strike by 20 m down dip (or less), mineralisation within the main mineralised lode (i.e. LODE = 301) was classified as an Indicated Resource.
- Where the drilling density was greater than 40 m x 20 m, the mineralisation was classified as an Inferred Resource.
- All minor lodes were classified as Inferred Resources due to the limited geological continuity within these domains.
- Only in situ mineralisation has been considered by Snowden. Historical tailings are discussed in Sections 14.2.4 and 14.3.4.

The classification was recorded in the model using a field called RESCAT as described in Table 14.14.

Table 14.14 Resource classification model field codes

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

Figure 14.15 Orthogonal view showing Selinsing resource classification (green = Indicated; red = Inferred)



Source: Snowden

14.2.7 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.

Past reconciliation indicates that mining typically does achieve lower tonnages and higher grades than indicated by the resource model due to more selective mining than has been modelled.

Cut-off grade

The Mineral Resource for the Selinsing deposit has been reported above a 0.3 g/t Au cut-off grade for oxide material and above a 0.7 g/t Au cut-off grade for transitional and sulphide material. The cut-off grades are based on the cost and metal price parameters detailed in Section 16.

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 June 2016.

Selinsing Mineral Resource statement

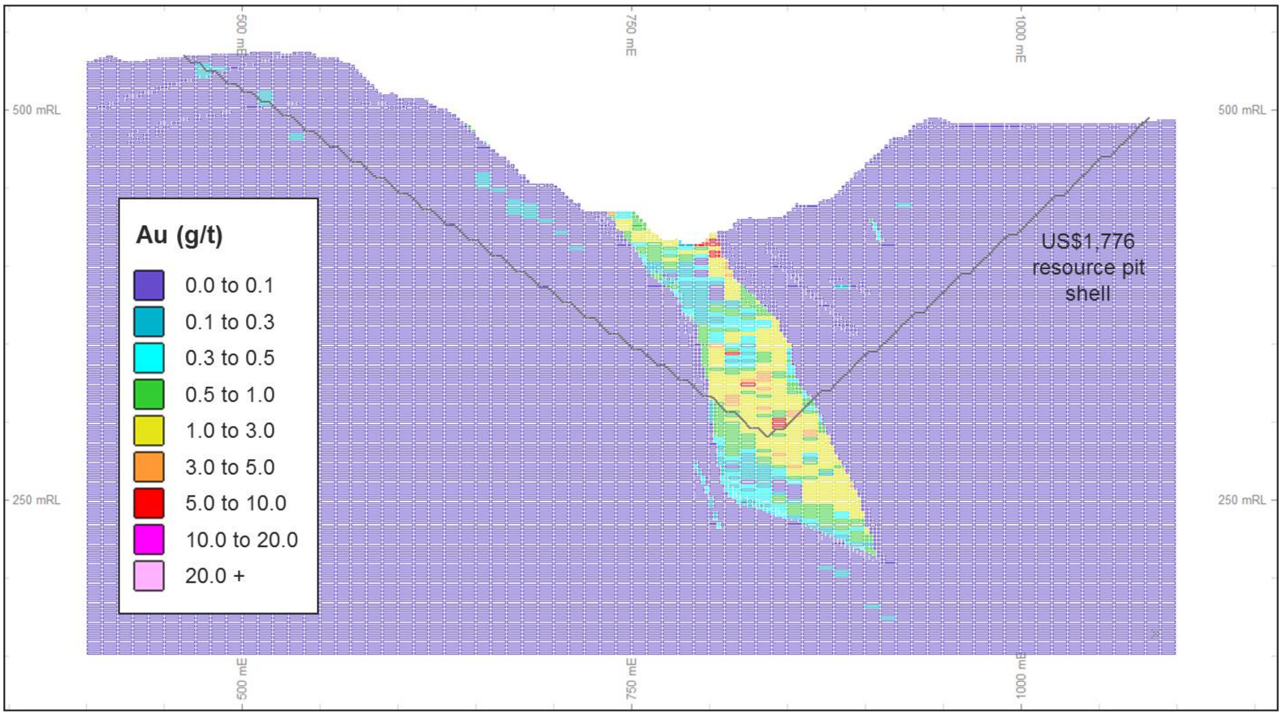
The Mineral Resource estimate for the Selinsing deposit is provided in Table 14.15. The Mineral Resource is limited to a pit shell provided by Monument based on a US$1,776/oz gold price. The pit shell was used by Snowden to define the likely limits of potential open pit mining. The mining and cost parameters used by Monument to generate the resource pit shell are not materially different to those described in Section 16. An example cross section showing the pit shell is presented in Figure 14.16.

Table 14.15 Selinsing Mineral Resource statement, depleted for mining to end of June 2016

Classification	Oxidation	Cut-off (g/t Au)	Tonnes (kt)	Au (g/t)	Ounces (koz)
Indicated	Oxide	0.3	90	0.67	2
	Transitional	0.7	90	1.42	4
	Fresh	0.7	3,040	1.98	193
Indicated Total			**3,220**	**1.93**	**200**
Inferred	Oxide	0.3	10	0.84	0.3
	Transitional	0.7	3	1.23	0.1
	Fresh	0.7	540	3.75	65
Inferred Total			**550**	**3.67**	**65**

Note: Small discrepancies may occur due to rounding

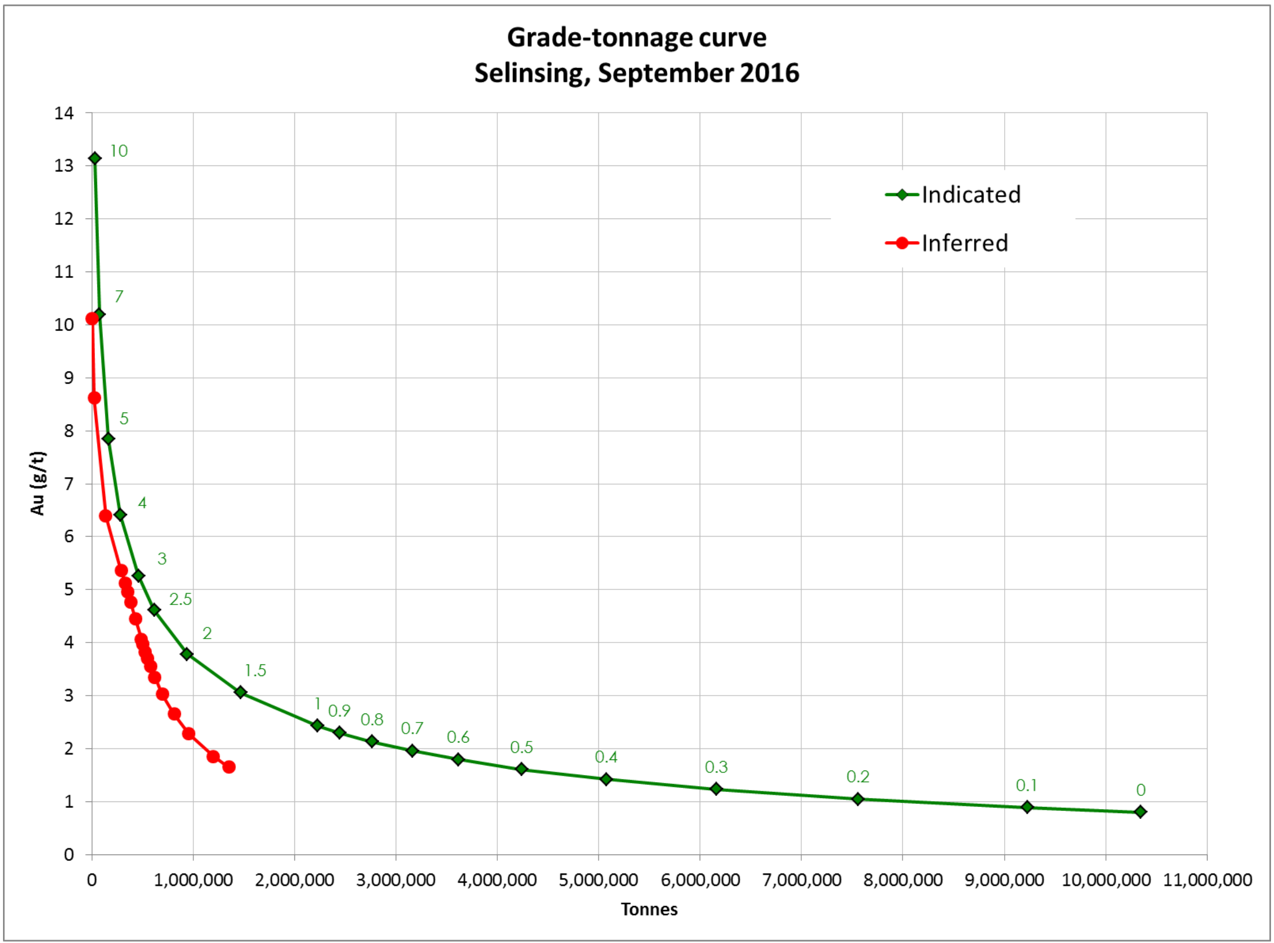
Figure 14.16 Example west-east cross section showing resource pit shell



Grade-tonnage curve

A grade-tonnage curve for the Selinsing Mineral Resource estimate, limited to the same US$1,776/oz pit shell, is presented in Figure 14.17.

Figure 14.17 Grade-tonnage curve for Selinsing



14.3 Buffalo Reef Mineral Resource estimate

14.3.1 Geological interpretation

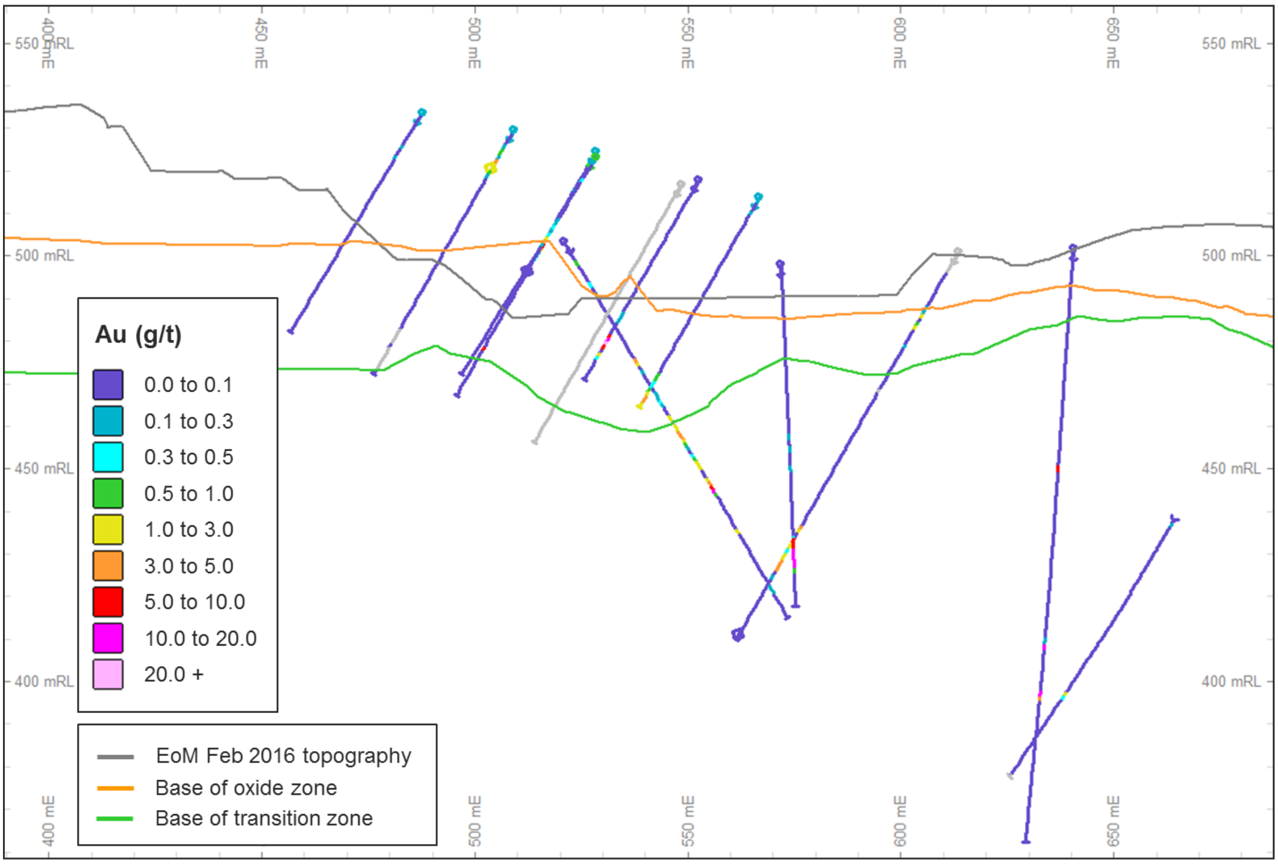
Oxidation

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 14.18.

Figure 14.18 Example west-east section (540 mN) showing interpretation of oxidation surfaces



Source: Snowden

Mineralisation

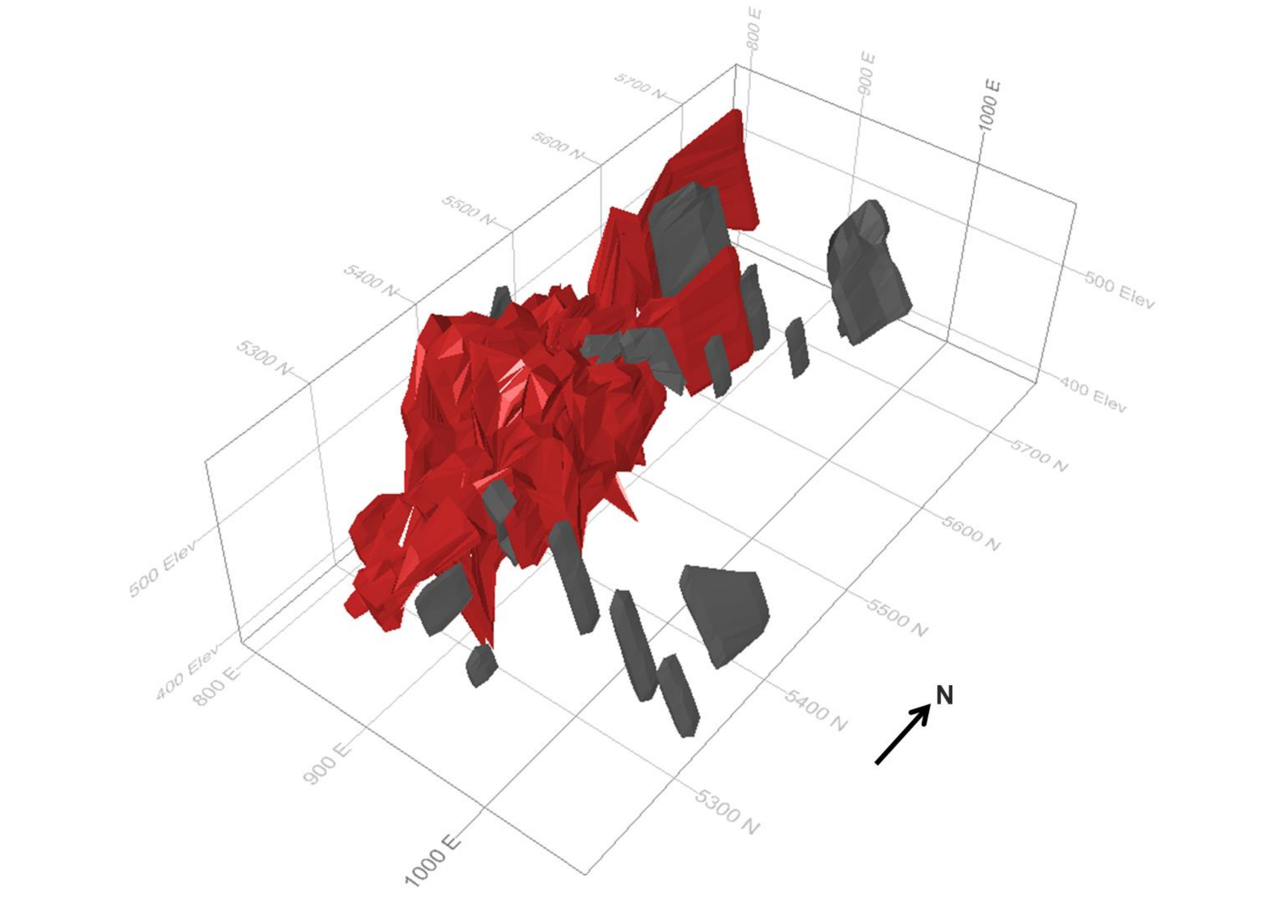
The sectional interpretation of the Buffalo Reef gold mineralisation was completed by Monument based on east-west sections with a section spacing of 20 m. Logging information from RC and diamond drillholes (along with a few surface trenches), including vein intensity, alteration and structural information, 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 m to 20 m thick, along with numerous minor sub-parallel mineralised structures. At BRN, the mineralisation is less continuous and narrower when compared to BRSCF. The interpreted mineralisation at Buffalo Reef North (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 14.19 and Figure 14.20 respectively for BRN and BRSCF. Example cross sections for BRN and South are shown in Figure 14.21 and Figure 14.22 respectively.

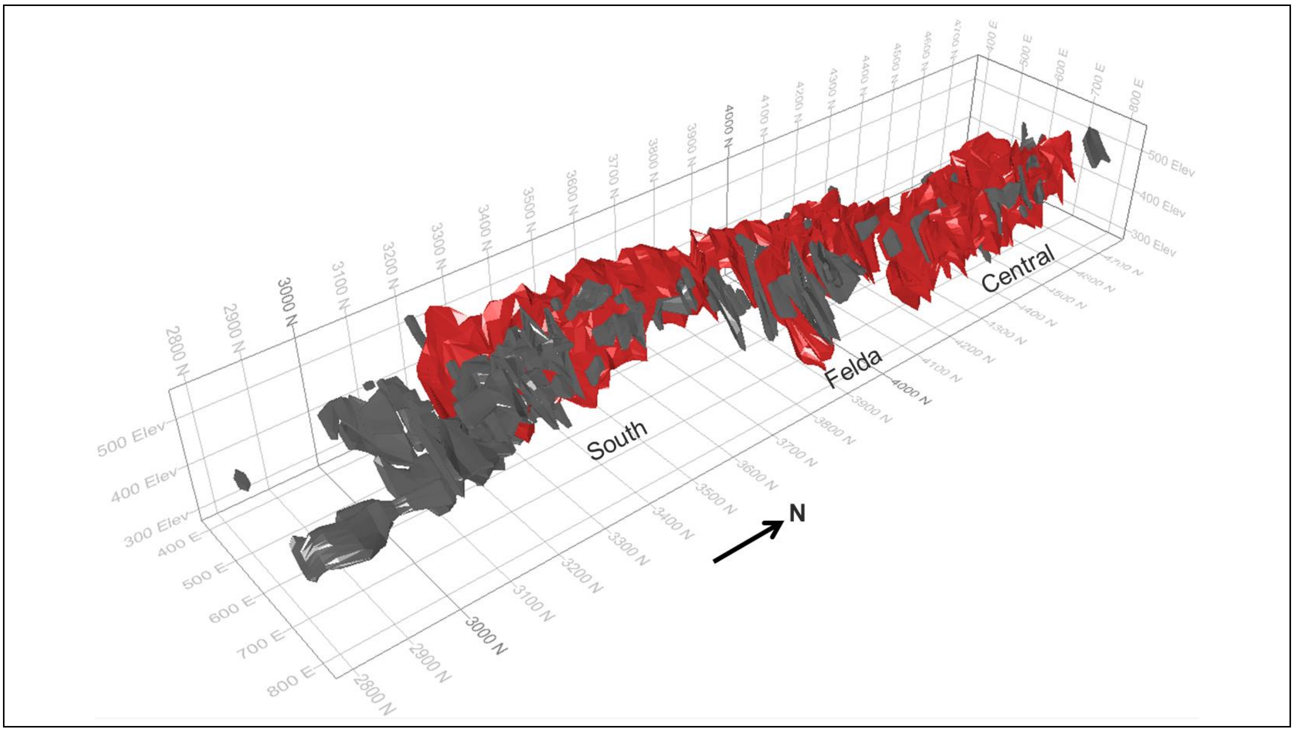
A minimum of three sample intervals (generally 3 m for RC holes and similar for diamond holes) was used for the interpreted mineralised intersections. External dilution was only included for isolated intervals with continuity from neighbouring intervals, either down dip in the same section or along strike 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 14.19 Oblique view (looking northwest) of the BRN mineralisation interpretation



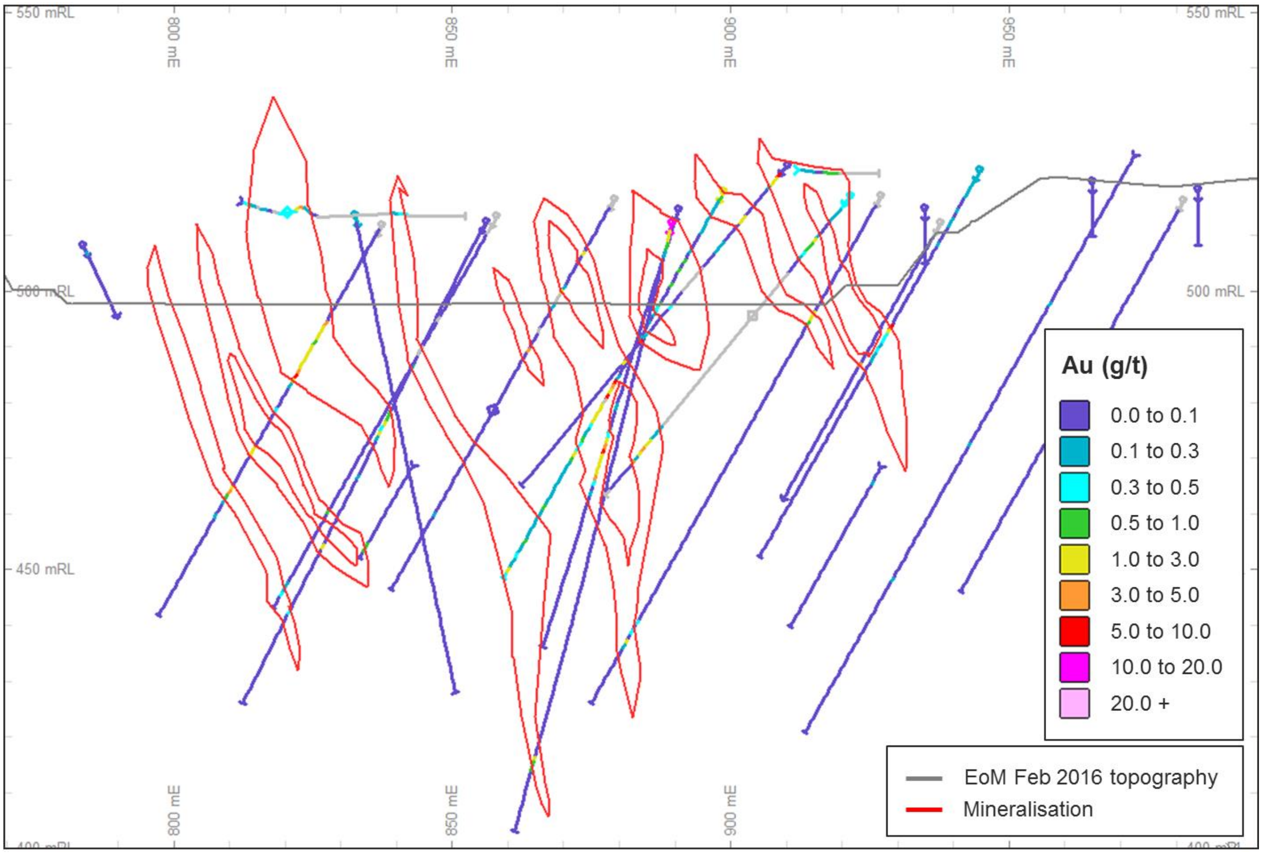
Source: Snowden

Figure 14.20 Oblique view (looking northwest) of the BRSCF mineralisation interpretation



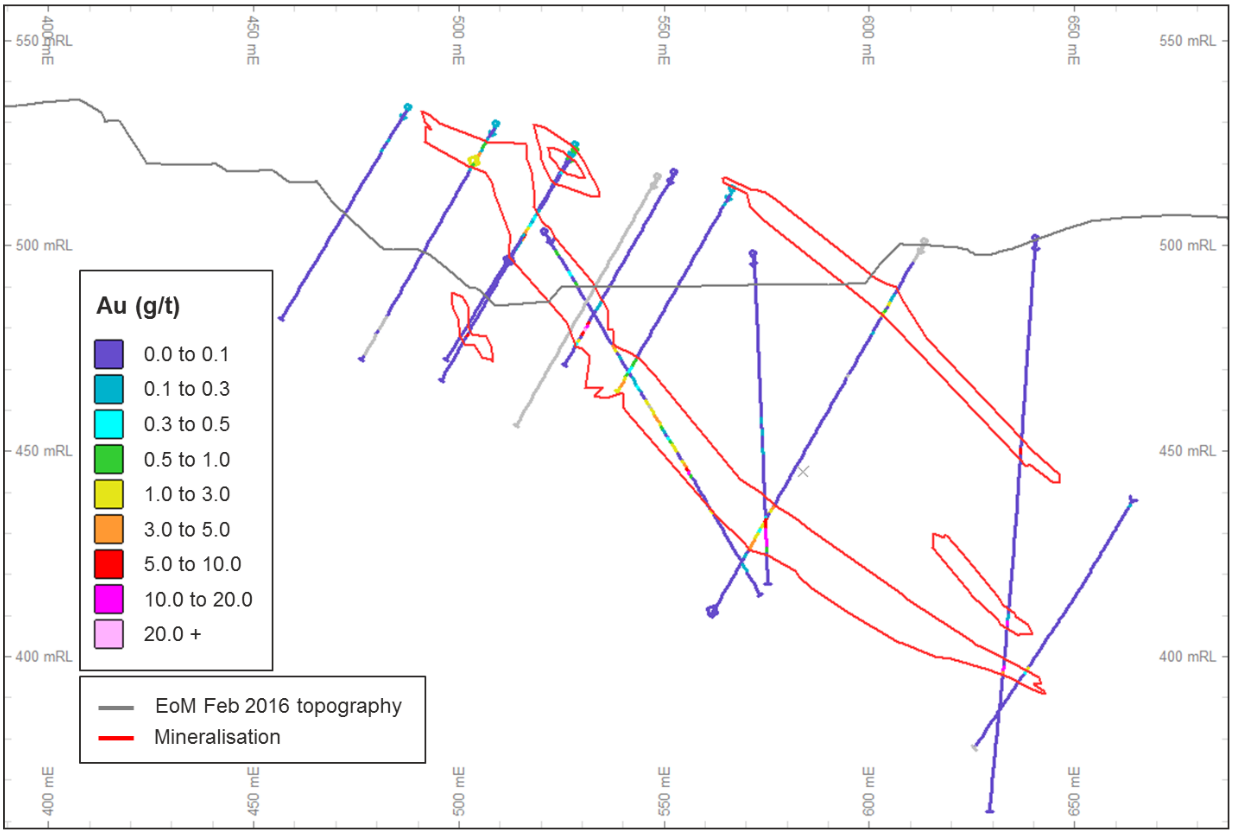
Source: Snowden

Figure 14.21 Example west-east section (870 mN, BRN) showing mineralisation interpretation



Source: Snowden

Figure 14.22 Example west-east section (540 mN, BRS) showing mineralisation interpretation



Source: Snowden

14.3.2 Drillhole data analysis

For the Buffalo Reef model, the drillhole dataset was limited to north of 2650 mN. Table 14.1 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 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 model was for drilling assays received up to 24 February 2016.

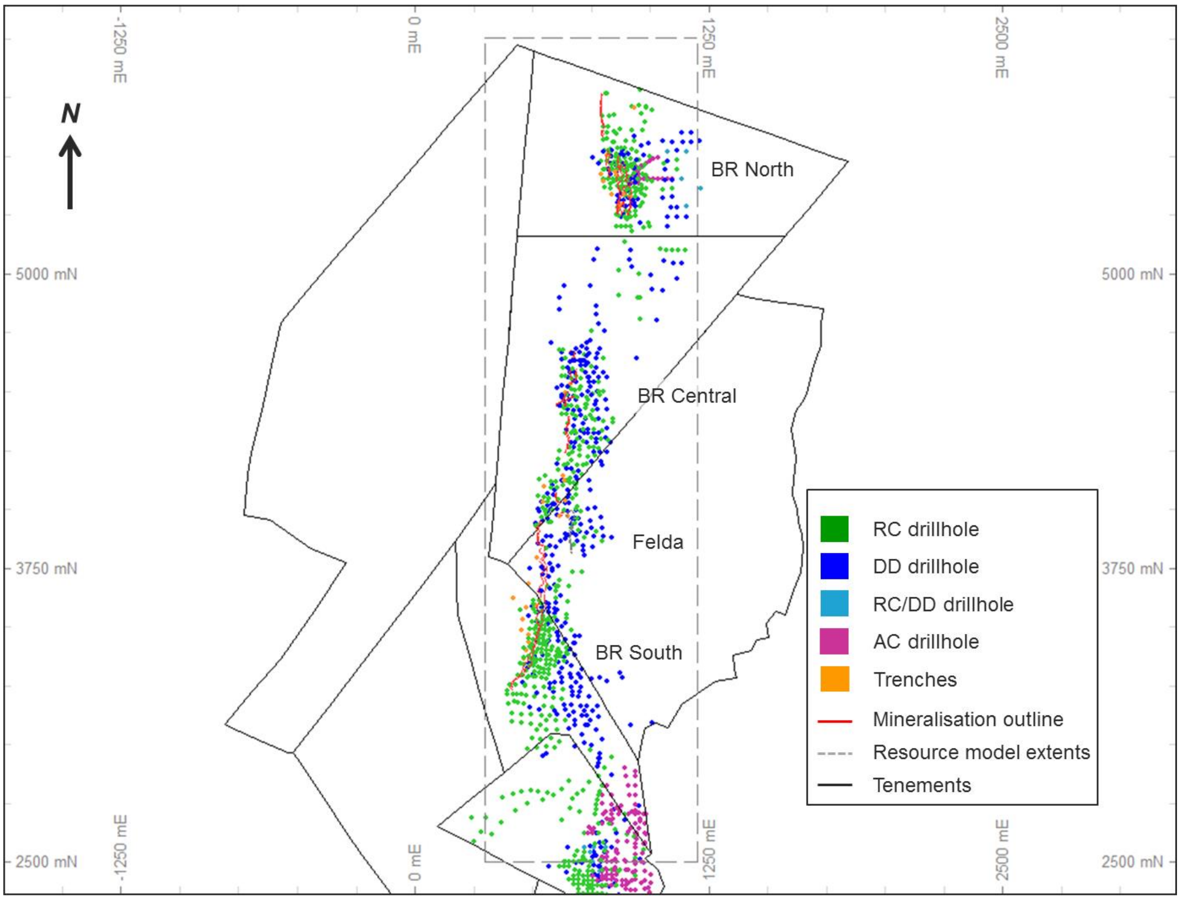
Table 14.16 Drilling data used for Buffalo Reef model (north of 2650 mN)

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*

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 Buffalo Reef North, 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.

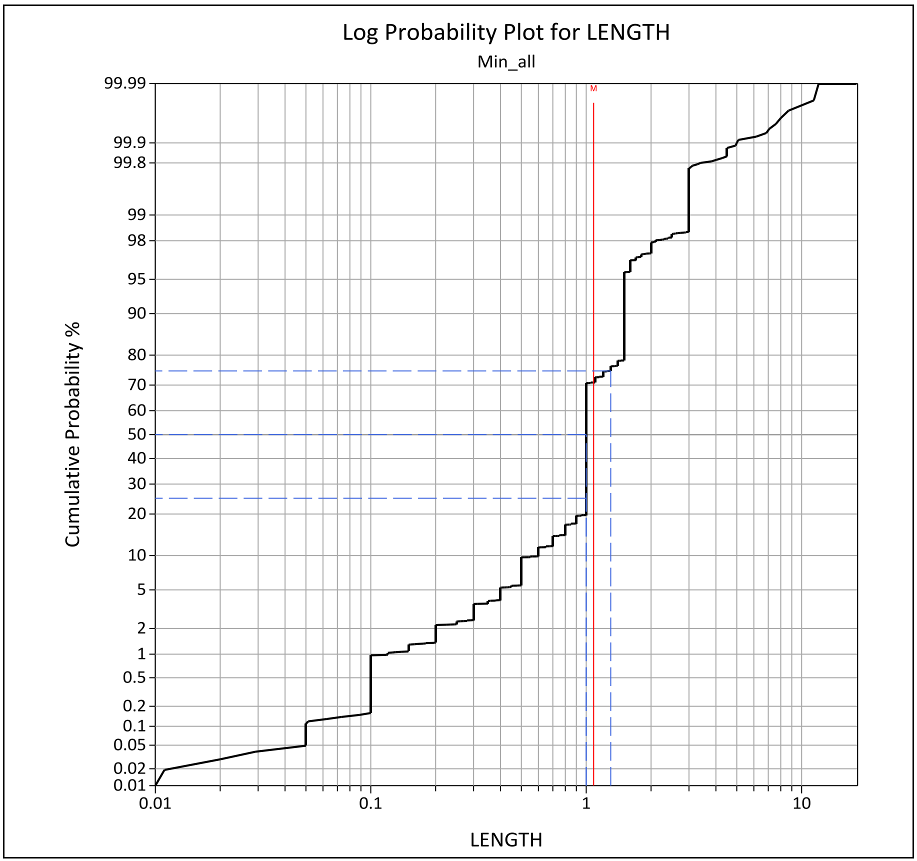
Figure 14.23 Collar location plan for Buffalo Reef resource model area



Sample compositing

The drillhole data was composited downhole prior to running the estimation process using a 1.5 m compositing interval 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 14.24. The compositing interval of 1.5 m 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 14.24 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.

No discrepancies were identified during the compositing process.

Statistical analysis

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

Gold

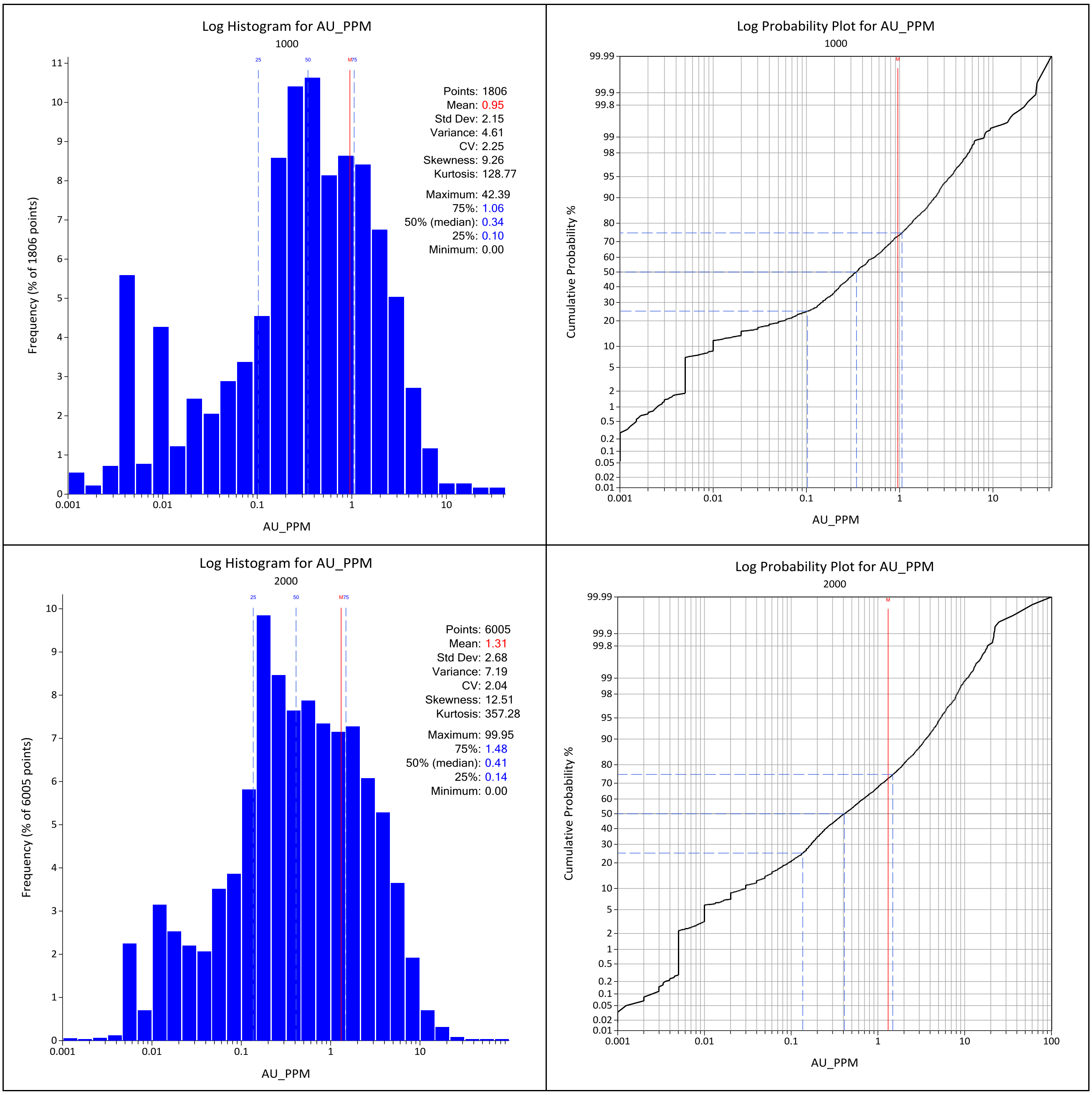
Summary statistics of composites for gold for Buffalo Reef North (BRN, MINZONE=1000) and Buffalo Reef South-Central-Felda (BRSCF, MINZONE=2000) are presented in Table 14.17 and log histograms and log probability plots are presented in Figure 14.25. The statistics show that the Buffalo Reef mineralisation 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 OK to estimate the block gold grades, with top-cuts applied to control the influence of extreme outliers.

Table 14.17 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 14.25 Log histogram and log probability plot for gold for BRN (top) and BRSCF (bottom) mineralised combined domains



Source: Snowden

Arsenic

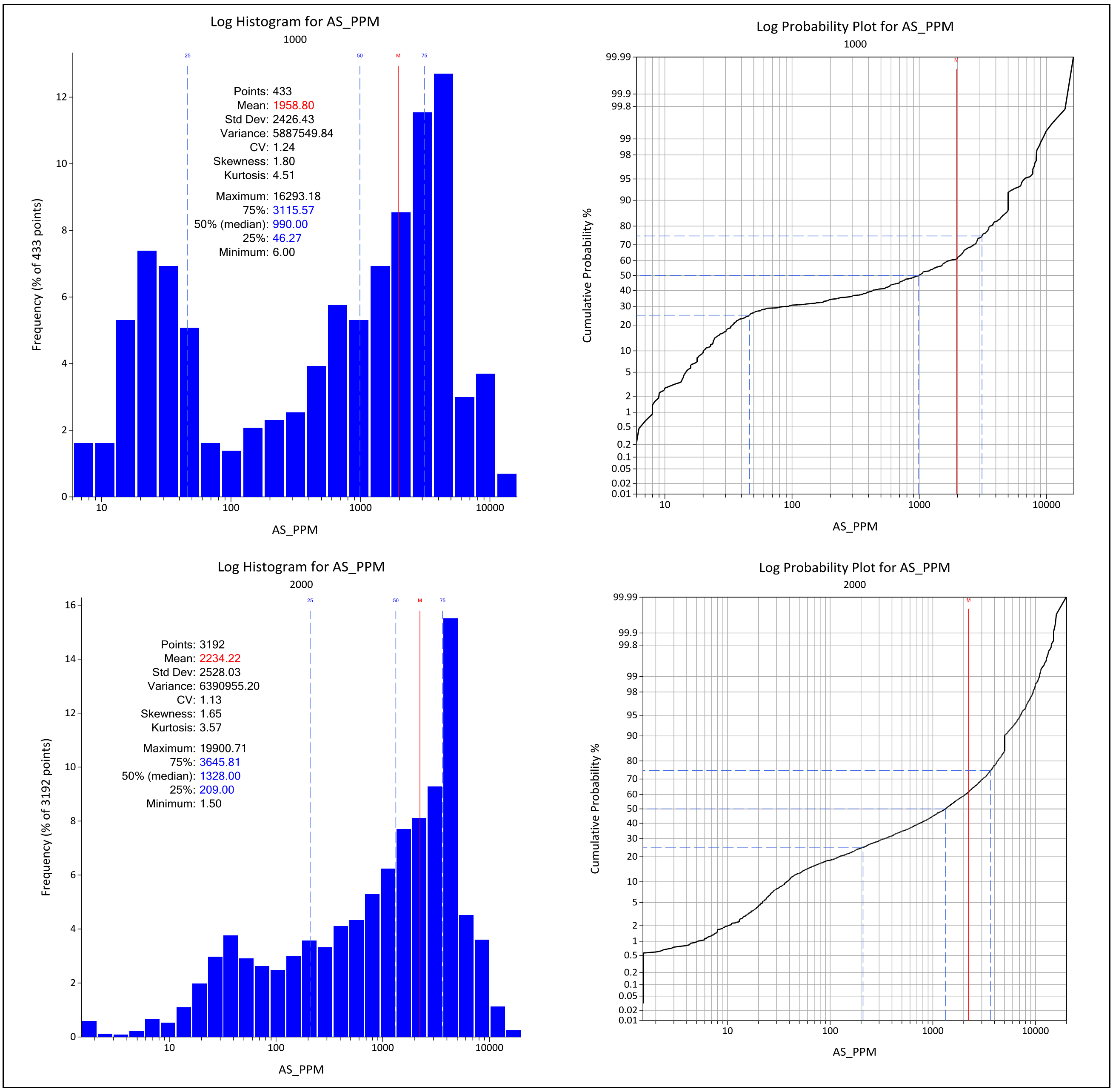
Summary statistics of composites for arsenic for Buffalo Reef North (BRN, MINZONE=1000) and Buffalo Reef South-Central-Felda (BRSCF, MINZONE=2000) are presented in Table 14.18 and log histograms and log probability plots are presented in Figure 14.26. The statistics show that the Buffalo Reef mineralisation 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 14.18 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 14.26 Log histogram and log probability plot for arsenic for BRN (top) and BRSCF (bottom) mineralised combined domains



Source: Snowden

Antimony

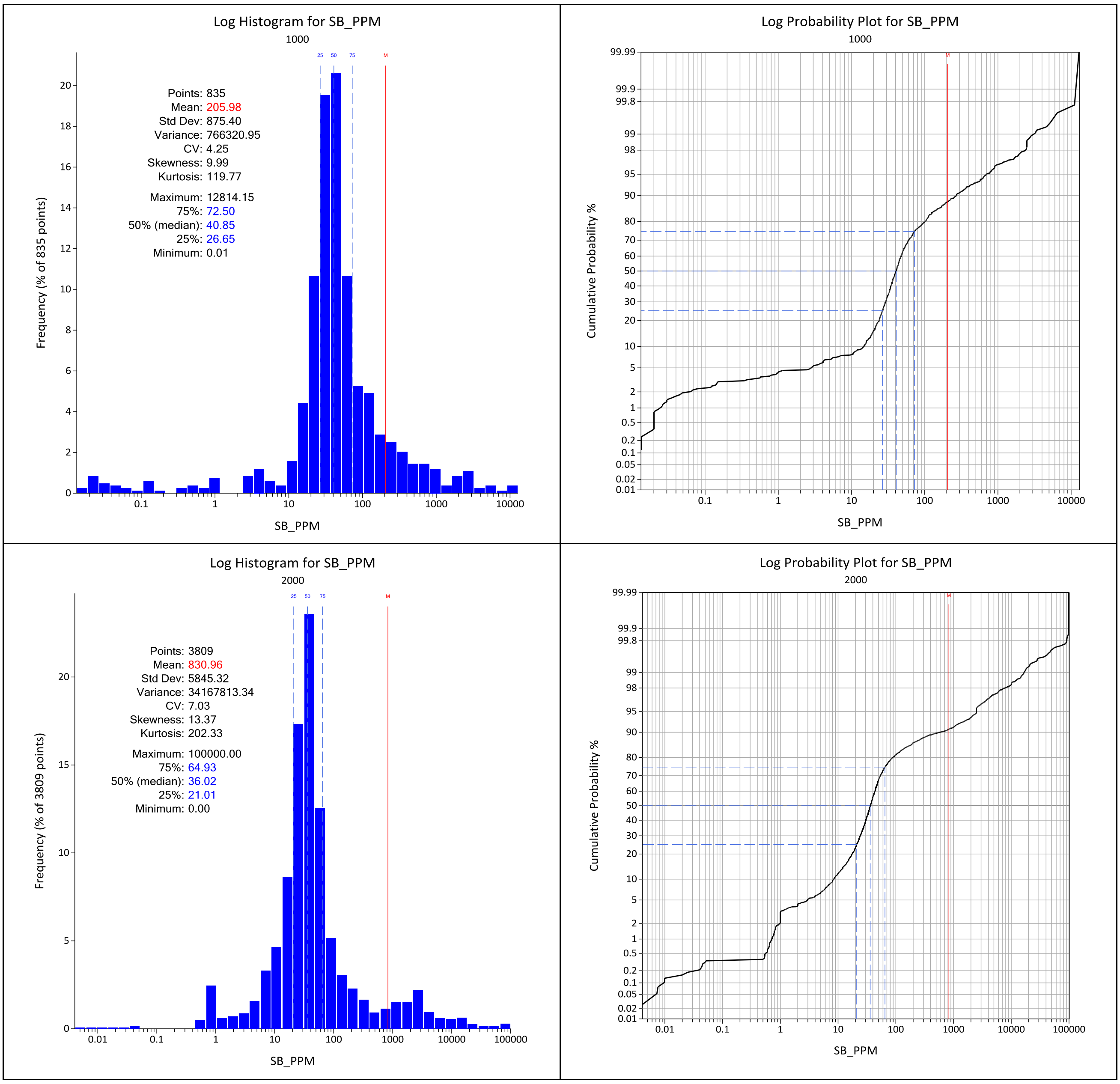
Summary statistics of composites for antimony for Buffalo Reef North (BRN, MINZONE=1000) and Buffalo Reef South-Central-Felda (BRSCF, MINZONE=2000) are presented in Table 14.19 and log histograms and log probability plots are presented in Figure 14.27. The statistics show that the Buffalo Reef mineralisation 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 14.19 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 14.27 Log histogram and log probability plot for antimony for BRN (top) and BRSCF (bottom) mineralised combined domains



Source: Snowden

Multivariate statistics

Correlation matrices for BRN and BRSCF are presented in Table 14.20 and Table 14.21 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 14.28. 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.

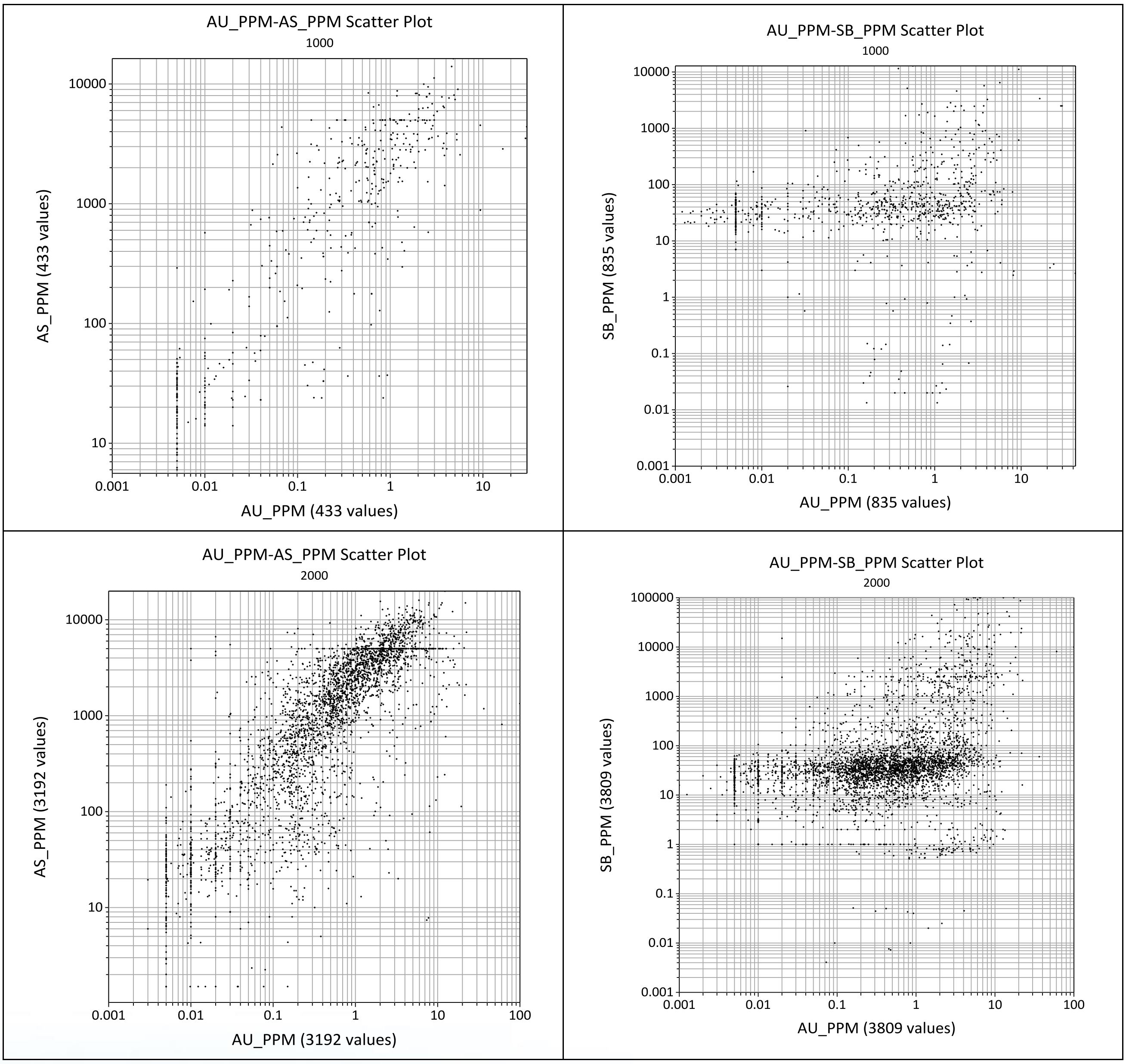
Table 14.20 Correlation matrix – BRN

Variable	Au	As	Sb
Au	1.00	0.37	0.20
As	0.37	1.00	0.08
Sb	0.20	0.08	1.00

Table 14.21 Correlation matrix – BRSCF

Variable	Au	As	Sb
Au	1.00	0.40	0.25
As	0.40	1.00	0.18
Sb	0.25	0.18	1.00

Figure 14.28 Scatterplots between Au-As and Au-Sb, for BRN (top) and BRSCF (bottom)



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 grades within the mineralised domains. The top-cuts are summarised in Table 14.22.

Table 14.22 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%

14.3.3 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 14.25 and the gold variogram for the main BRSCF lode (201) is provided in Figure 14.29. The median indicator gold variogram for the combined BRN domain is provided in Figure 14.30.

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 14.23. Additionally, the indicator variogram models from the BRSCF mineralised domain were applied to the waste domain.

Table 14.23 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 14.26.

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 14.19. The upper and lower tail model parameters are summarised in Table 14.24 for each domain.

Table 14.24 Distribution tail modelling – Buffalo Reef

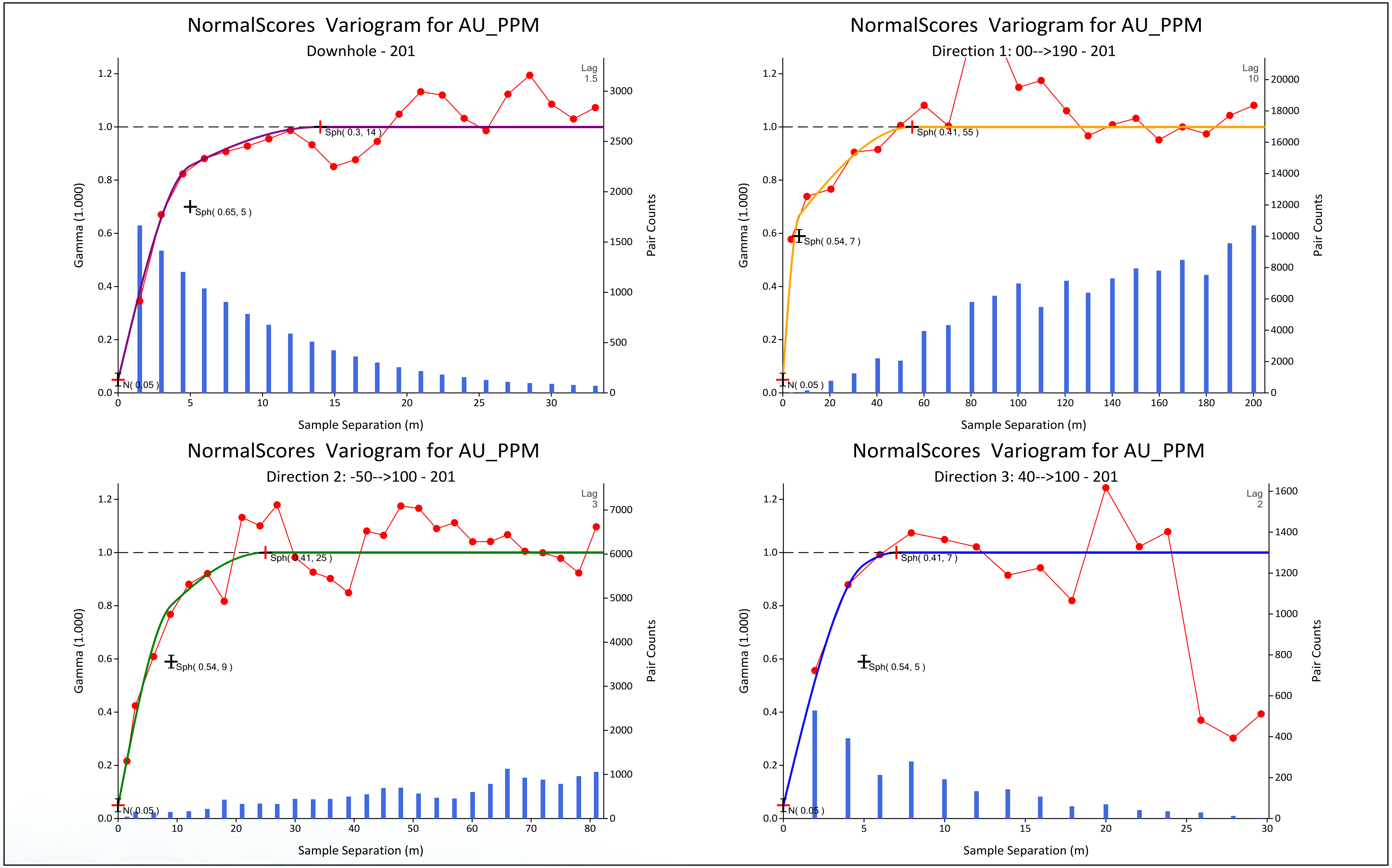
Domain (MINZONE)	Tail	Model type	Model parameter
0	Lower	Power	0.60
	Upper	Hyperbolic	1.10
1000	Lower	Power	0.40
	Upper	Hyperbolic	1.20
2000	Lower	Power	0.80
	Upper	Hyperbolic	1.50

Table 14.25 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	-	-	-	-

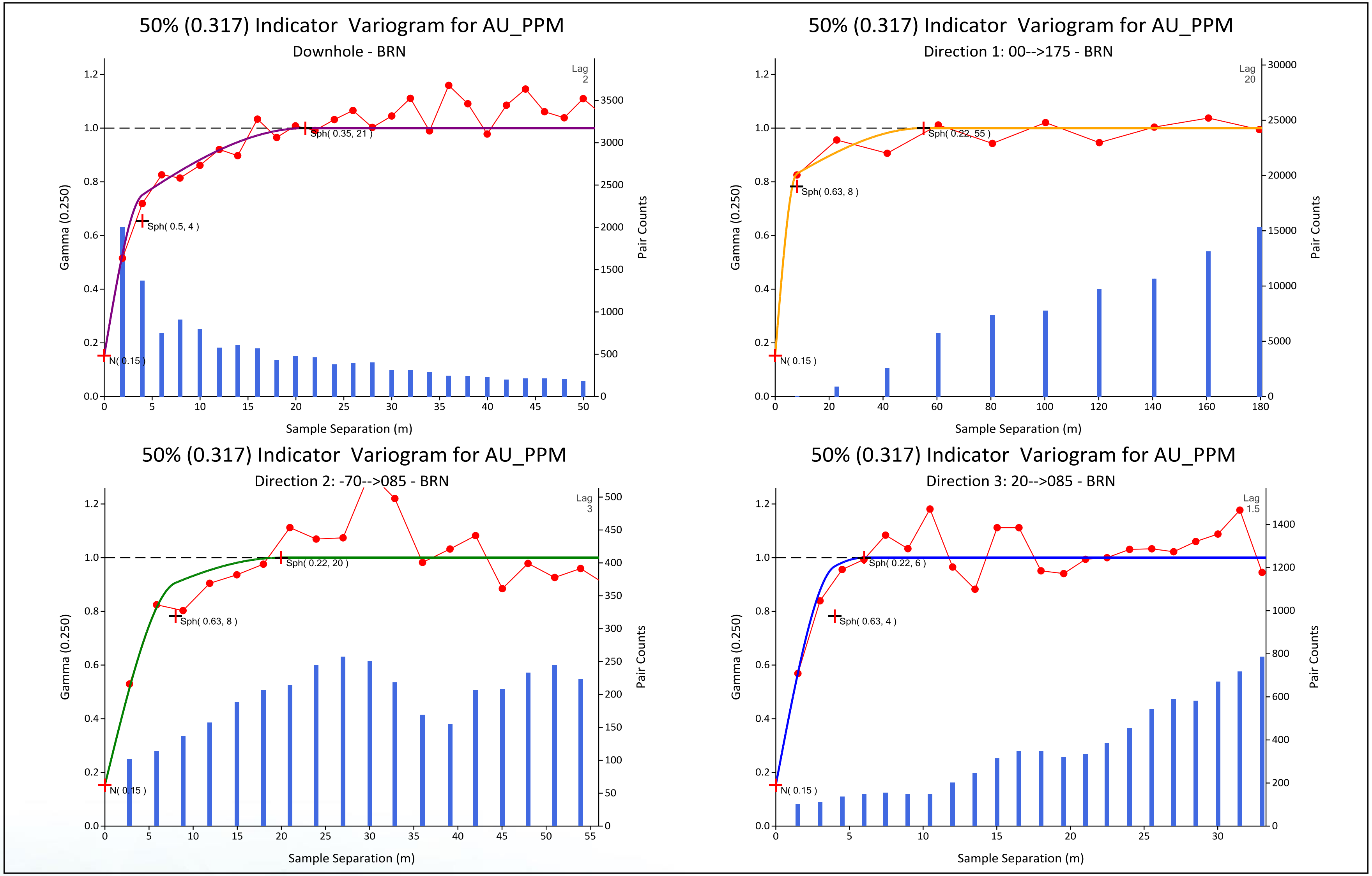
Note: Lode numbers 201 to 221 are from BRSCF; BRN Au variogram is median indicator, all others are back-transformed normal scores variograms

Figure 14.29 Normal scores variogram model for Au for main BRSCF mineralised domain



Source: Snowden

Figure 14.30 Median indicator variogram model for Au for combined BRN mineralised domains



Source: Snowden

Table 14.26 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

14.3.4 Block model and grade estimation

Kriging neighbourhood analysis

Similar to the Selinsing model, a KNA was performed 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 by 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 14.27.

Table 14.27 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 historical tailings material.

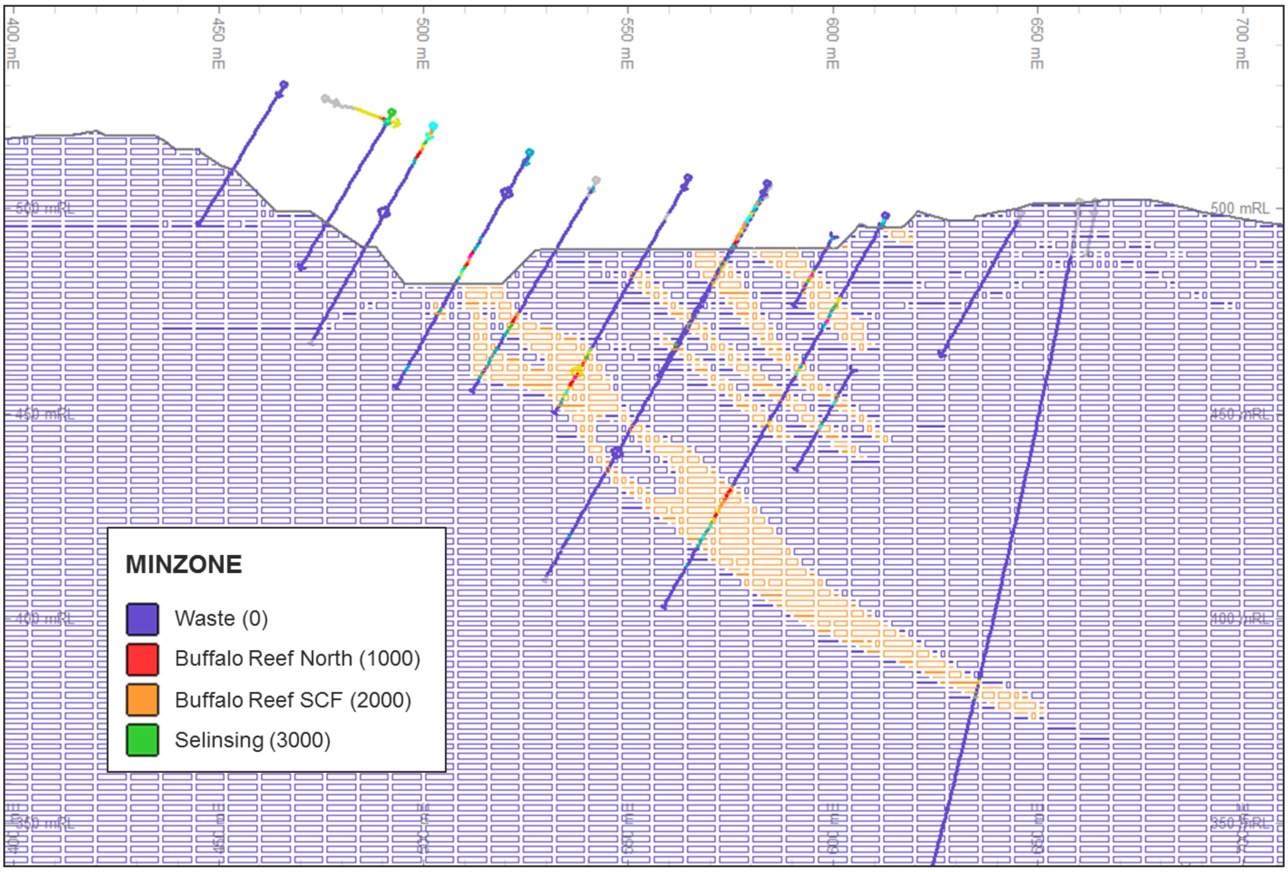
Mineralisation

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 14.28 and an example west-east cross section showing the MINZONE coding is shown in Figure 14.31.

Table 14.28 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	Historical tailings material

Figure 14.31 Example west-east section (3400 mN) showing block model MINZONE coding



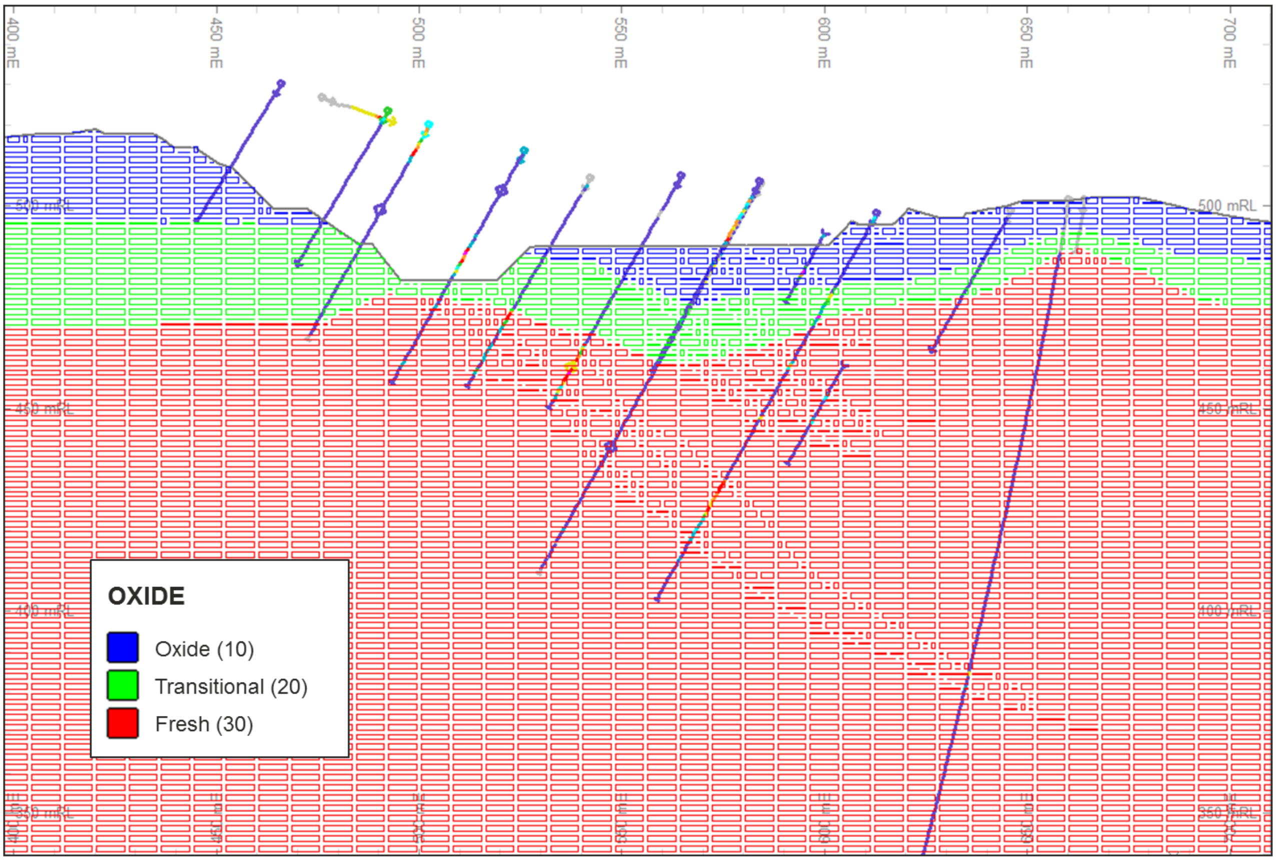
Oxidation

The oxidation zones were coded using a field called OXIDE. Field codes are defined in Table 14.29 and an example west-east cross section showing the OXIDE coding is shown in Figure 14.32.

Table 14.29 OXIDE field coding

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

Figure 14.32 Example west-east section (3400 mN) showing block model OXIDE coding



Tailings

The extent of the historical tailings material was coded as per Section 14.2.4. The tailings material only impacts the south-eastern region of the Buffalo Reef model.

Grade estimation methodology

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 14.24). 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 14.30. Either the median or mean values were used for the default grade depending on the nature of the grade distribution.

Table 14.30 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

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 14.31. 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 14.31 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**

Model validation

The block grade estimates were validated using:

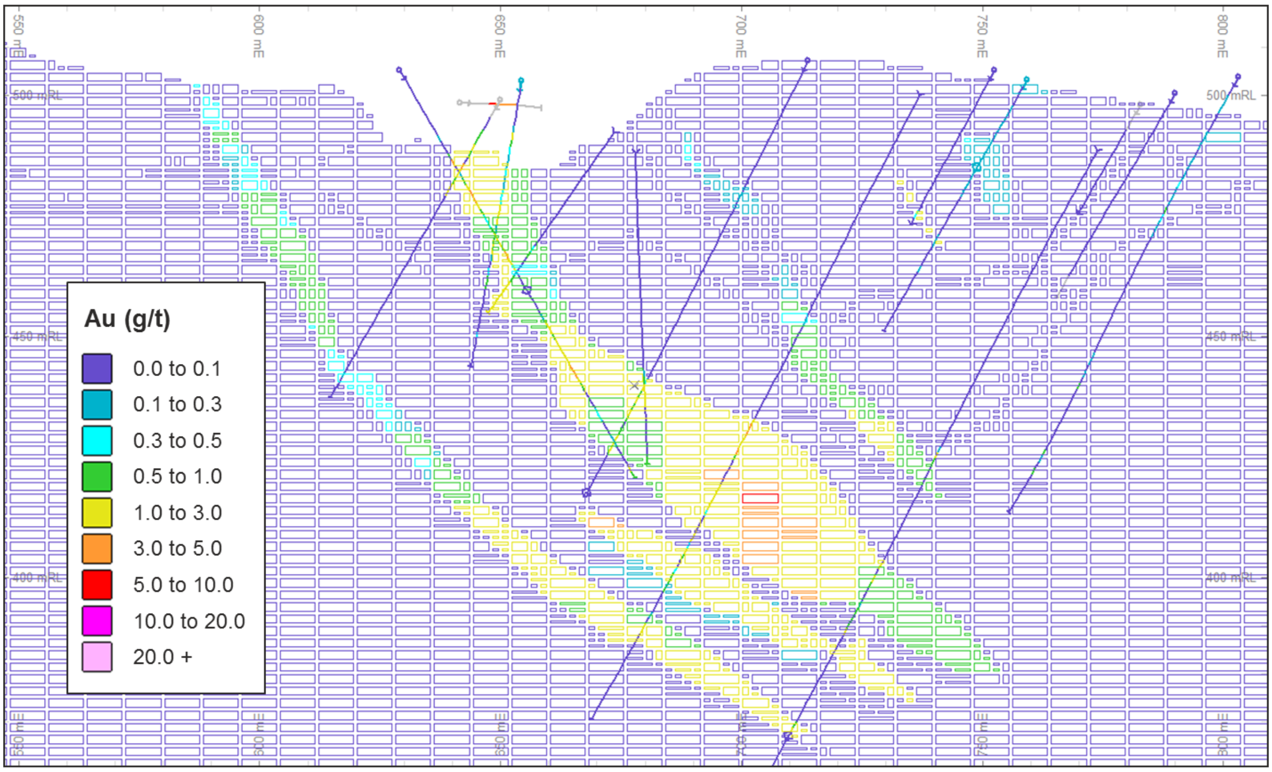
- A visual comparison of block grade estimates and the input drillhole data
- A global comparison of the average composite (naïve and declustered) and estimated block grades
- Moving window averages comparing the mean block grades to the composites.

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 14.33).
- 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 14.32 to Table 14.34).

- 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 14.34 and Figure 14.35).

Figure 14.33 Example west-east section (4490 mN; ±10 m) showing block grade estimates at BRSCF against the input drillhole composites



Source: Snowden

Table 14.32 Buffalo Reef model validation summary statistics – gold

Statistic	Input composites		Block model estimate Au (g/t)
	Naïve Au (g/t)	Declustered 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 14.33 Buffalo Reef model validation summary statistics – arsenic

Statistic	Input composites		Block model estimate As (ppm)
	Naïve As (ppm)	Declustered 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

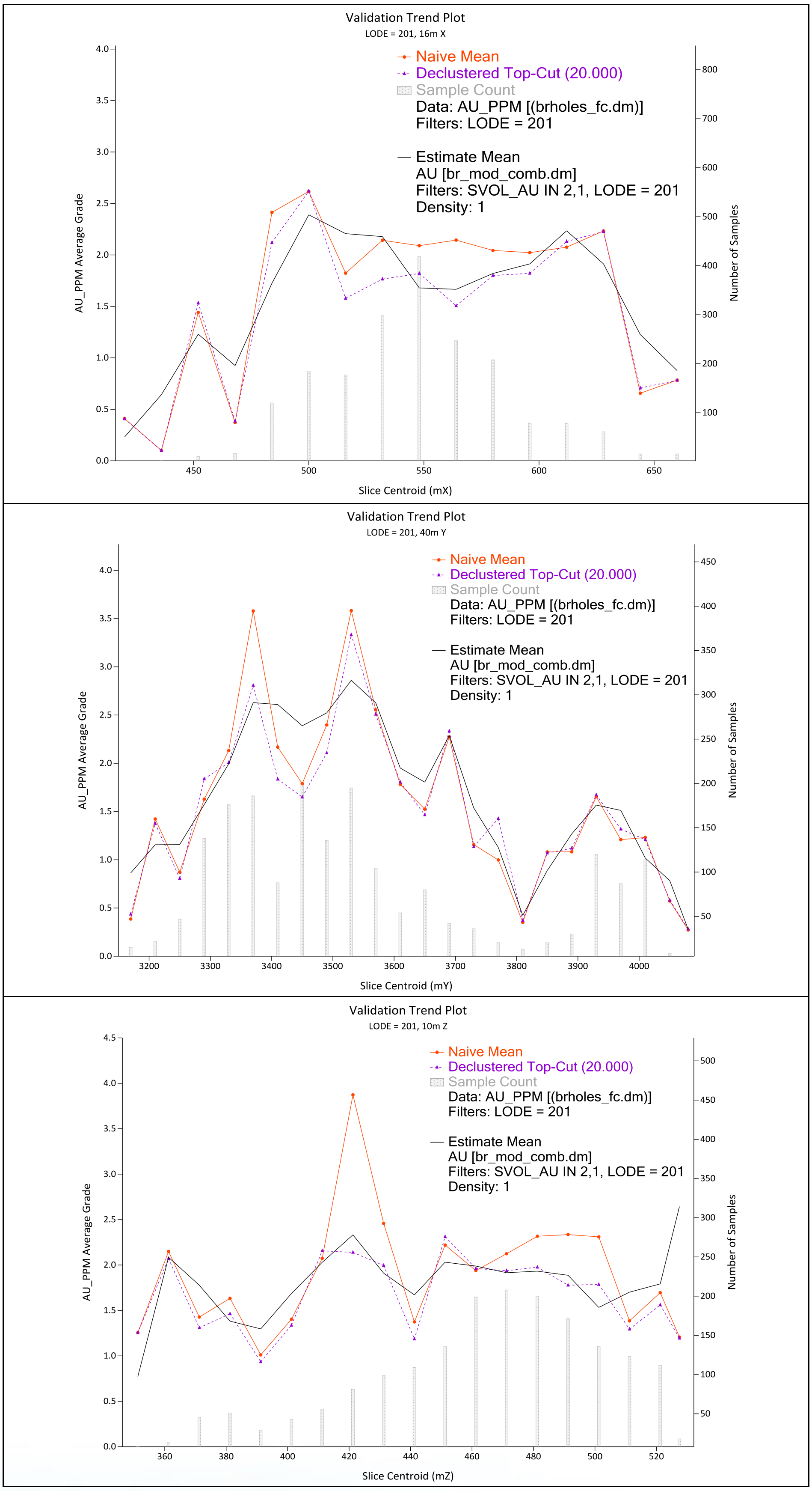
Notes: Model restricted to blocks estimated in the first and second search passes only

Table 14.34 Buffalo Reef model validation summary statistics – antimony

Statistic	Input composites		Block model estimate Sb (ppm)
	Naïve Sb (ppm)	Declustered 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

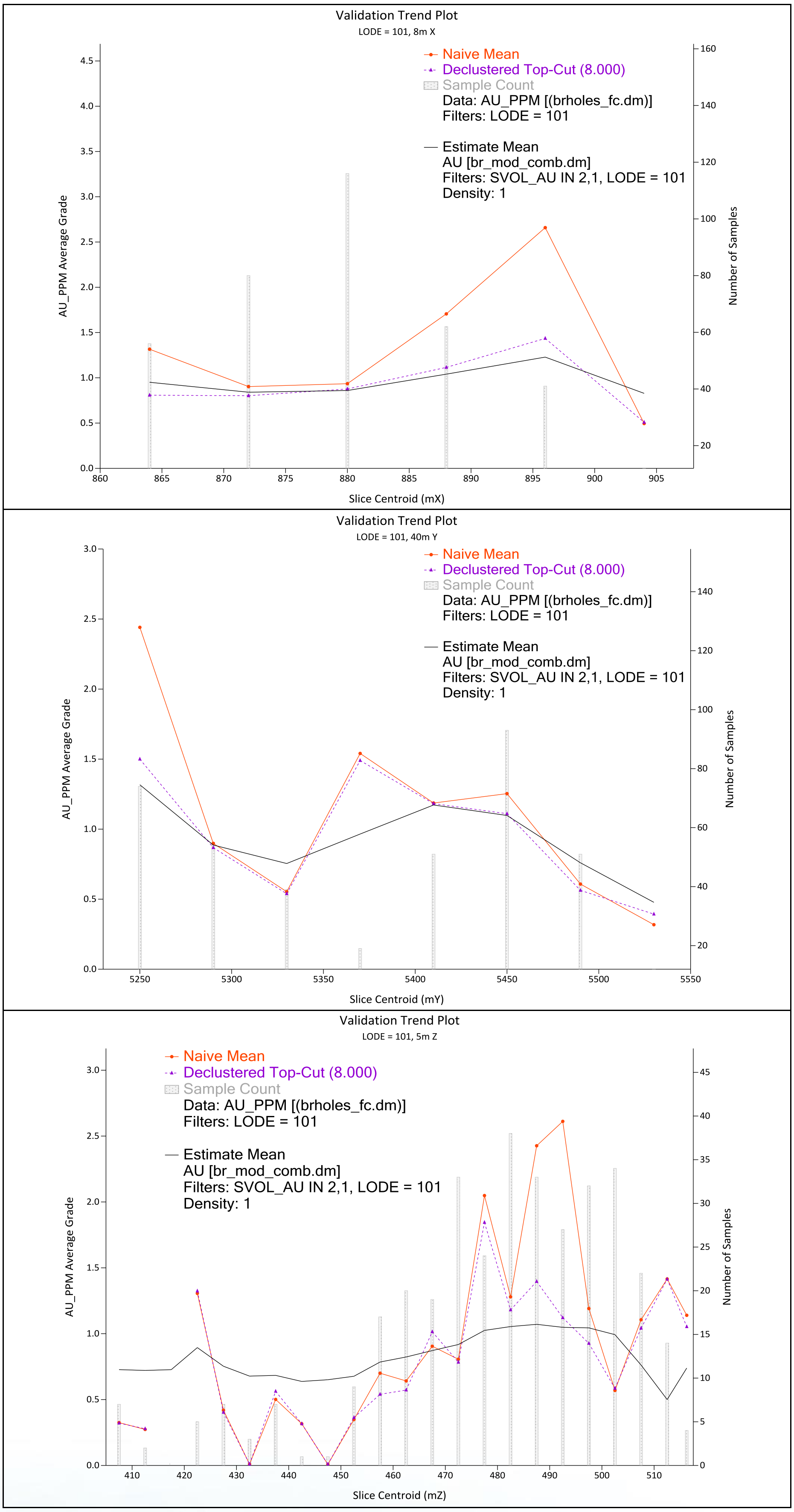
Notes: Model restricted to blocks estimated in the first and second search passes only

Figure 14.34 Validation trend plots – BRSCF main lode (LODE=201), gold



Source: Snowden

Figure 14.35 Validation trend plots – BRN main lode (LODE=101), gold



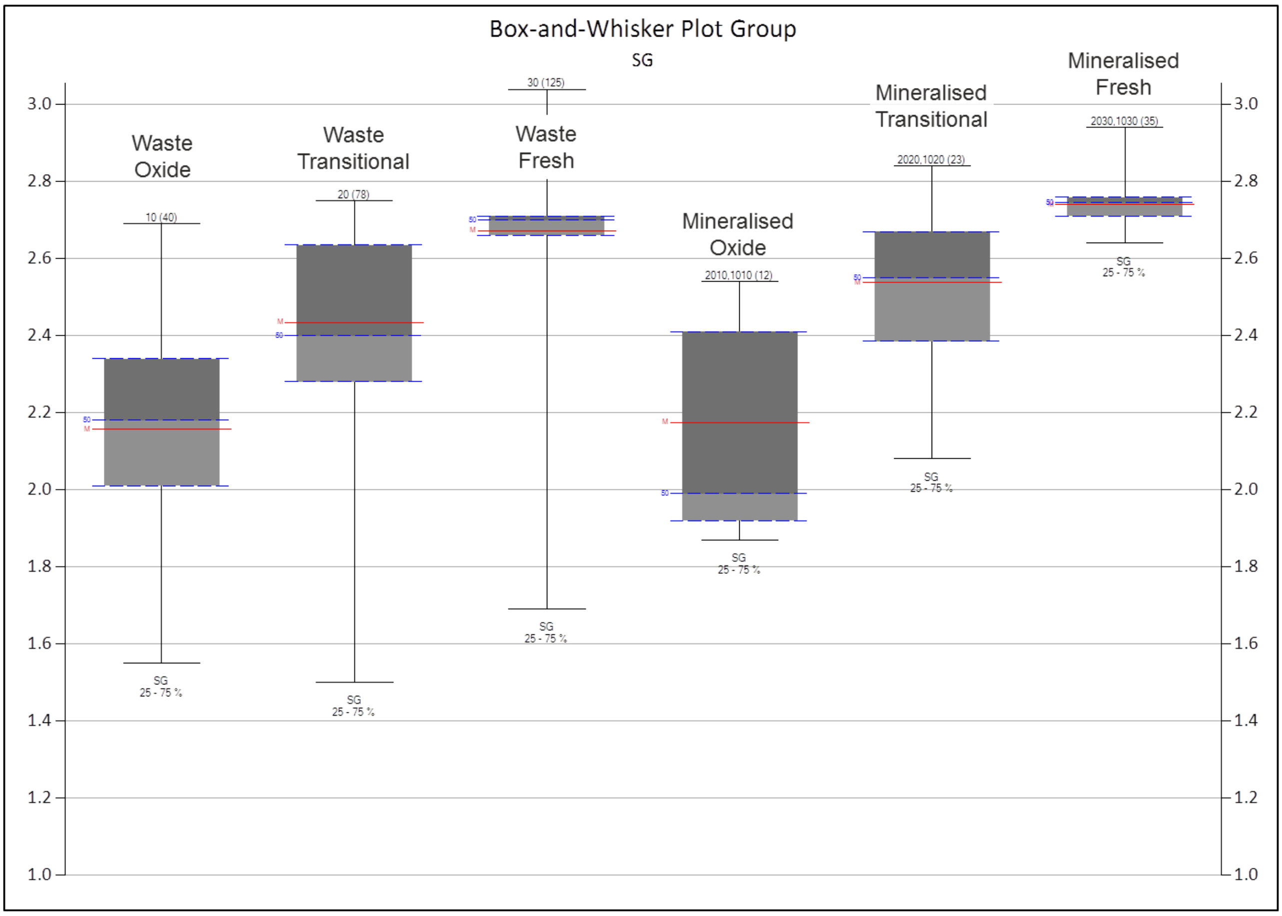
Source: Snowden

14.3.5 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 14.2.5.

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 14.36. Snowden notes that there is no correlation between the density and the gold, arsenic or antimony content.

Figure 14.36 Box-and-whisker plot of bulk density samples from Buffalo Reef



Source: Snowden

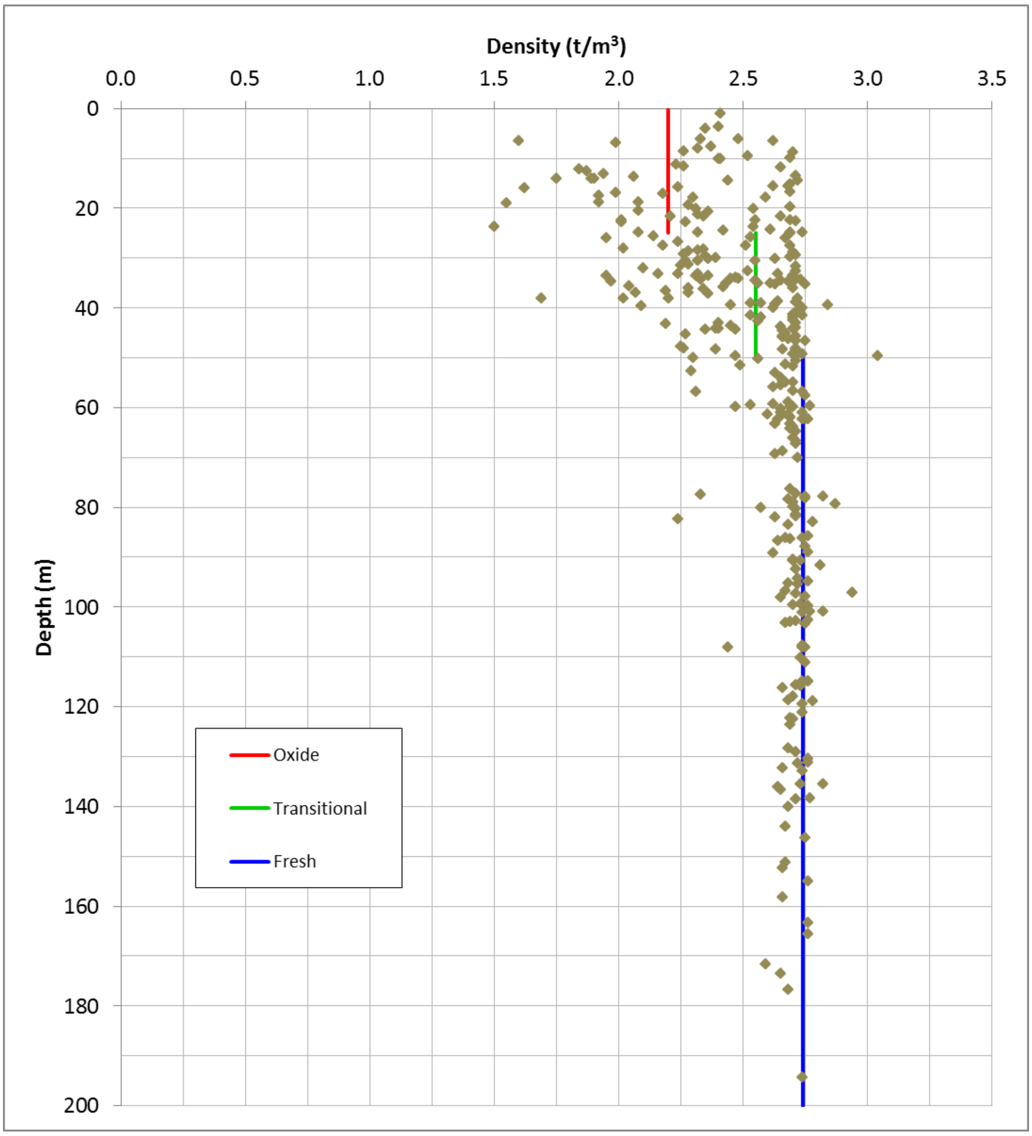
Based on the analysis, default bulk density values were assigned to the model blocks based on the OXIDE and MINZONE coding, as per Table 14.13. Due to the low number of BRN samples, the BRN and BRSCF domains were combined to assess the bulk density of the combined mineralised domains. A default bulk density of 1.18 t/m^3 for the historical tailings material was provided by Monument.

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 14.37 (the depth of the oxide and transitional boundaries in Figure 14.37 are indicative only). Snowden believes that the values assigned to the model are reasonable for the Buffalo Reef mineralisation.

Table 14.35 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
Historical tailings	-	1.18

Figure 14.37 Bulk density depth profile for Buffalo Reef



Source: Snowden

14.3.6 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 the company'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 in situ mineralisation has been considered by Snowden. Historical tailings are discussed in Sections 14.2.4 and 14.3.4.

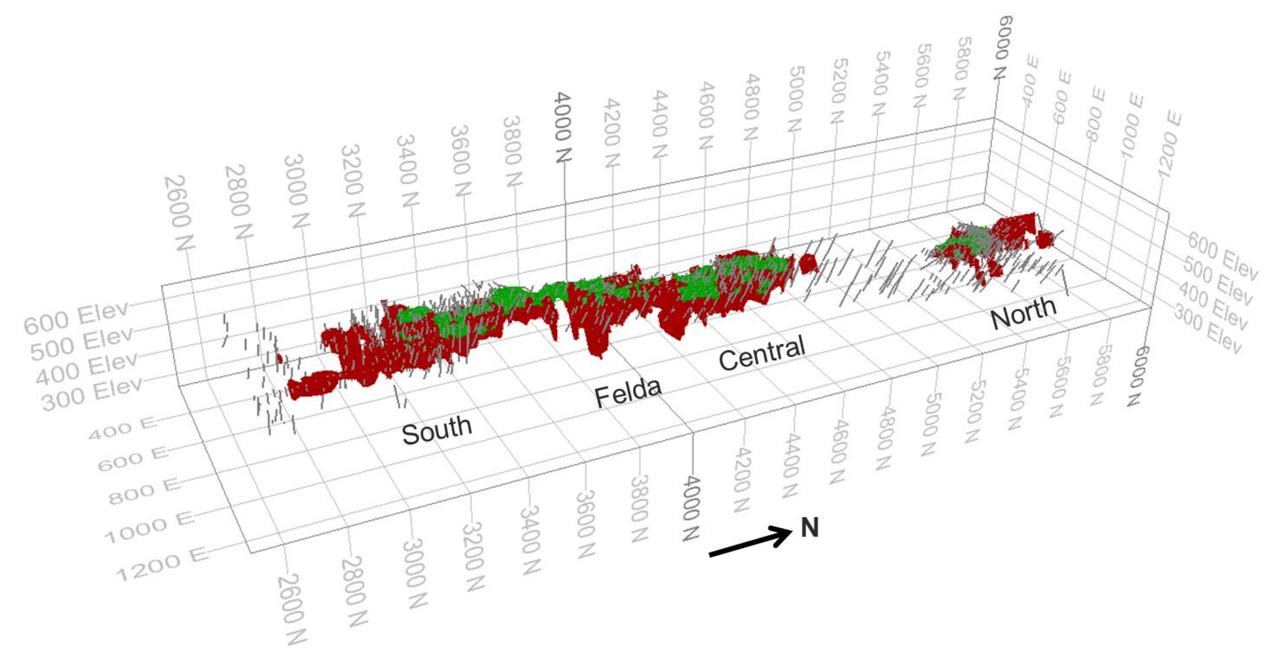
For Buffalo Reef, arsenic and antimony were estimated, in addition to gold, as these elements are considered by Monument to be important for the processing. Snowden notes that not all samples contain assays for these elements and as such, estimates of these variables should be considered to be of low 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 14.36.

Table 14.36 Resource classification model field codes

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

Figure 14.38 Orthogonal view showing Buffalo Reef resource classification (green = Indicated; red = Inferred)



Source: Snowden

14.3.7 Mineral Resource reporting

Cut-off grade

The Mineral Resource for the Buffalo Reef deposit has been reported above a 0.3 g/t Au cut-off grade for oxide material and above a 0.7 g/t Au cut-off grade for transitional and sulphide material. The cut-off grades are based on the cost and metal price parameters detailed in Section 16.

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 June 2016.

Buffalo Reef Mineral Resource statement

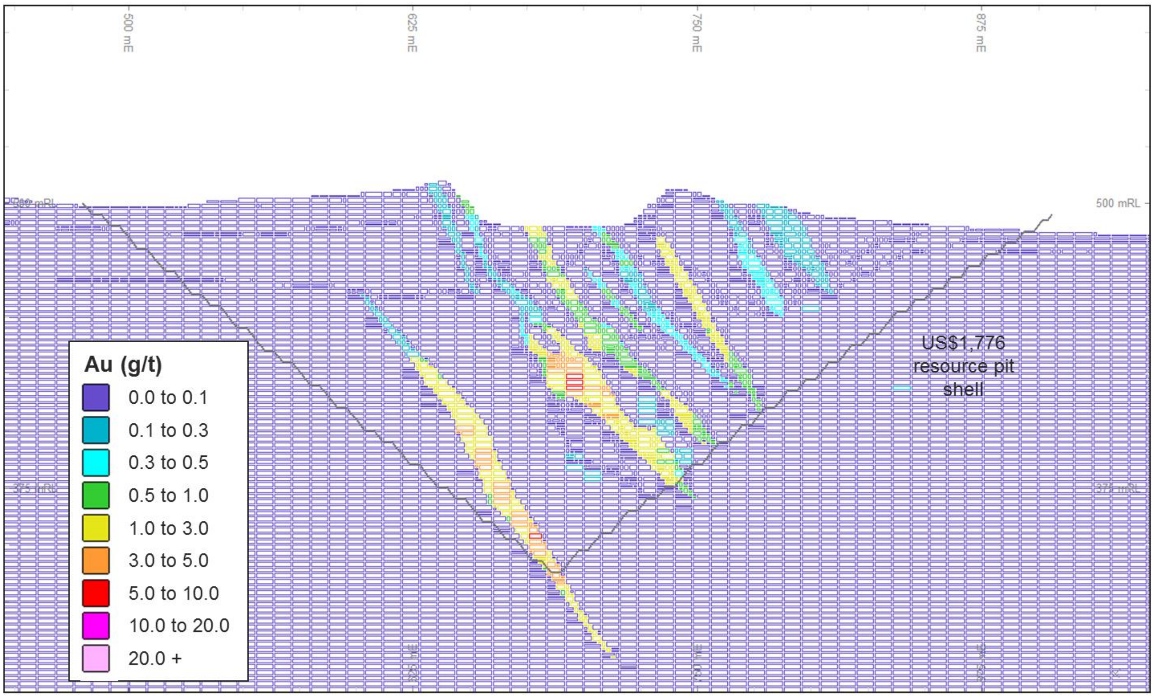
The Mineral Resource estimate for the Buffalo Reef deposit is provided in Table 14.37. The Mineral Resource is limited to a pit shell provided by Monument based on a US1,776/oz gold price. The pit shell was used by Snowden to define the likely limits of potential open-pit mining. The mining and cost parameters used by Monument to generate the resource pit shell are not materially different to those described in Section 16. An example cross section showing the pit shell is presented in Figure 14.39.

Table 14.37 Buffalo Reef Mineral Resource statement, depleted for mining to end of June 2016

Classification	Oxidation	Zone	Cut-off (g/t Au)	Tonnes (kt)	Au (g/t)	As (ppm)	Sb (ppm)	Ounces (koz)
Indicated	Oxide	BRN	0.3	180	0.99	1,900	270	6
		BRC	0.3	170	0.83	1,600	140	4
		Felda	0.3	260	1.33	2,700	230	11
		BRS	0.3	100	2.10	3,200	560	7
	Oxide total			**700**	**1.23**	**2,300**	**270**	**27**
	Transitional	BRN	0.7	150	1.26	2,200	230	6
		BRC	0.7	310	1.19	2,300	110	12
		Felda	0.7	190	1.64	3,000	330	10
		BRS	0.7	230	2.65	3,000	3,250	19
	Transitional total			**860**	**1.68**	**2,600**	**1,010**	**46**
	Fresh	BRN	0.7	70	1.18	2,300	100	2
		BRC	0.7	990	1.67	3,400	1,990	53
		Felda	0.7	620	1.78	2,900	960	35
		BRS	0.7	1,130	2.12	2,800	1,150	77
	Fresh total			**2,790**	**1.87**	**3,000**	**1,380**	**167**
Indicated Total				**4,330**	**1.73**	**2,800**	**1,130**	**240**
Inferred	Oxide	BRN	0.3	100	0.81	1,700	120	2
		BRC	0.3	120	1.15	1,600	60	4
		Felda	0.3	70	1.03	1,500	150	2
		BRS	0.3	90	1.14	1,400	190	3
	Oxide total			**370**	**1.04**	**1,500**	**120**	**12**
	Transitional	BRN	0.7	90	1.34	2,300	110	4
		BRC	0.7	140	1.40	2,100	170	6
		Felda	0.7	50	1.54	1,900	150	2
		BRS	0.7	90	1.62	1,700	760	4
	Transitional total			**350**	**1.46**	**2,000**	**290**	**16**
	Fresh	BRN	0.7	30	1.61	2,300	60	1
		BRC	0.7	1,500	1.86	2,800	1,980	89
		Felda	0.7	1,040	1.98	3,300	1,190	66
		BRS	0.7	530	1.59	2,500	630	27
	Fresh total			**3,100**	**1.85**	**2,900**	**1,470**	**184**
Inferred Total				**3,810**	**1.74**	**2,700**	**1,230**	**212**

Notes: Small discrepancies may occur due to rounding. The classification applies to the Au grades only; As and Sb are considered indicative only.

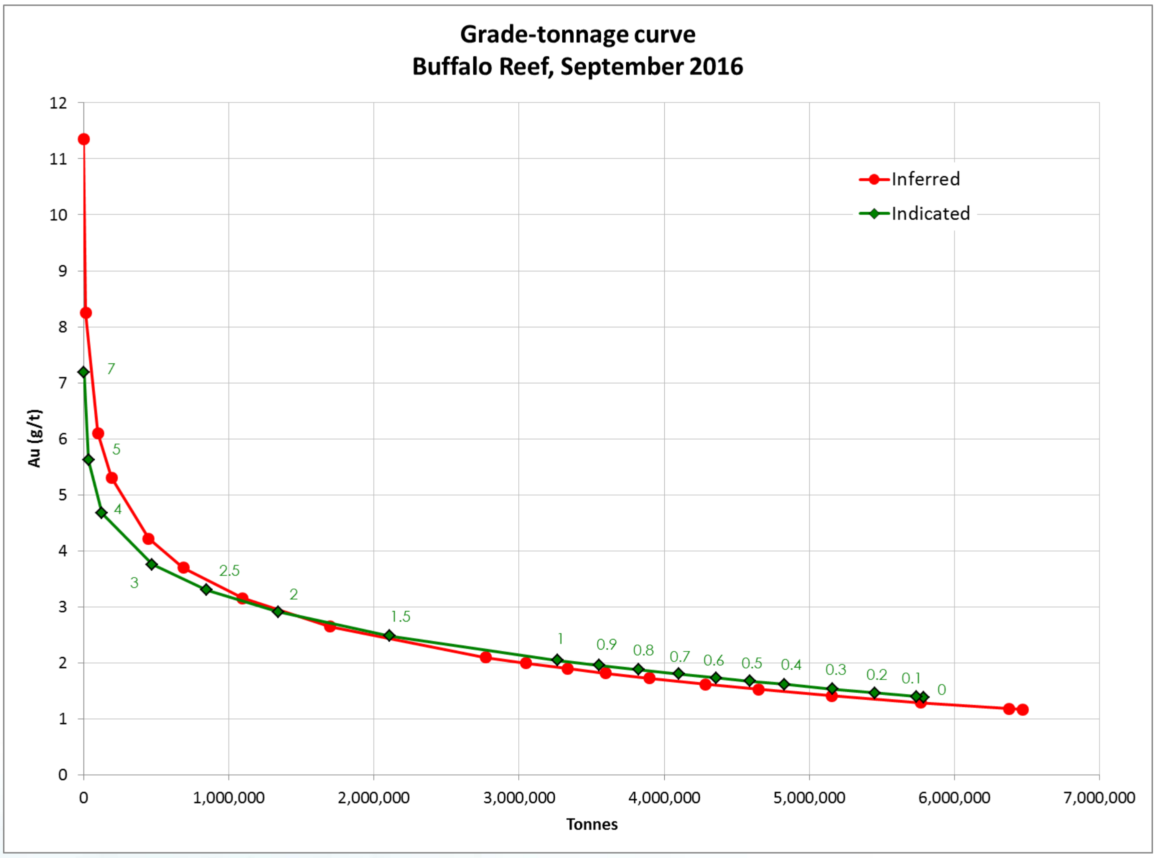
Figure 14.39 Example west-east cross section (BRC) showing resource pit shell



Grade-tonnage curve

A grade-tonnage curve for the Buffalo Reef Mineral Resource estimate, limited to the same US$1,776/oz pit shell, is presented in Figure 14.40.

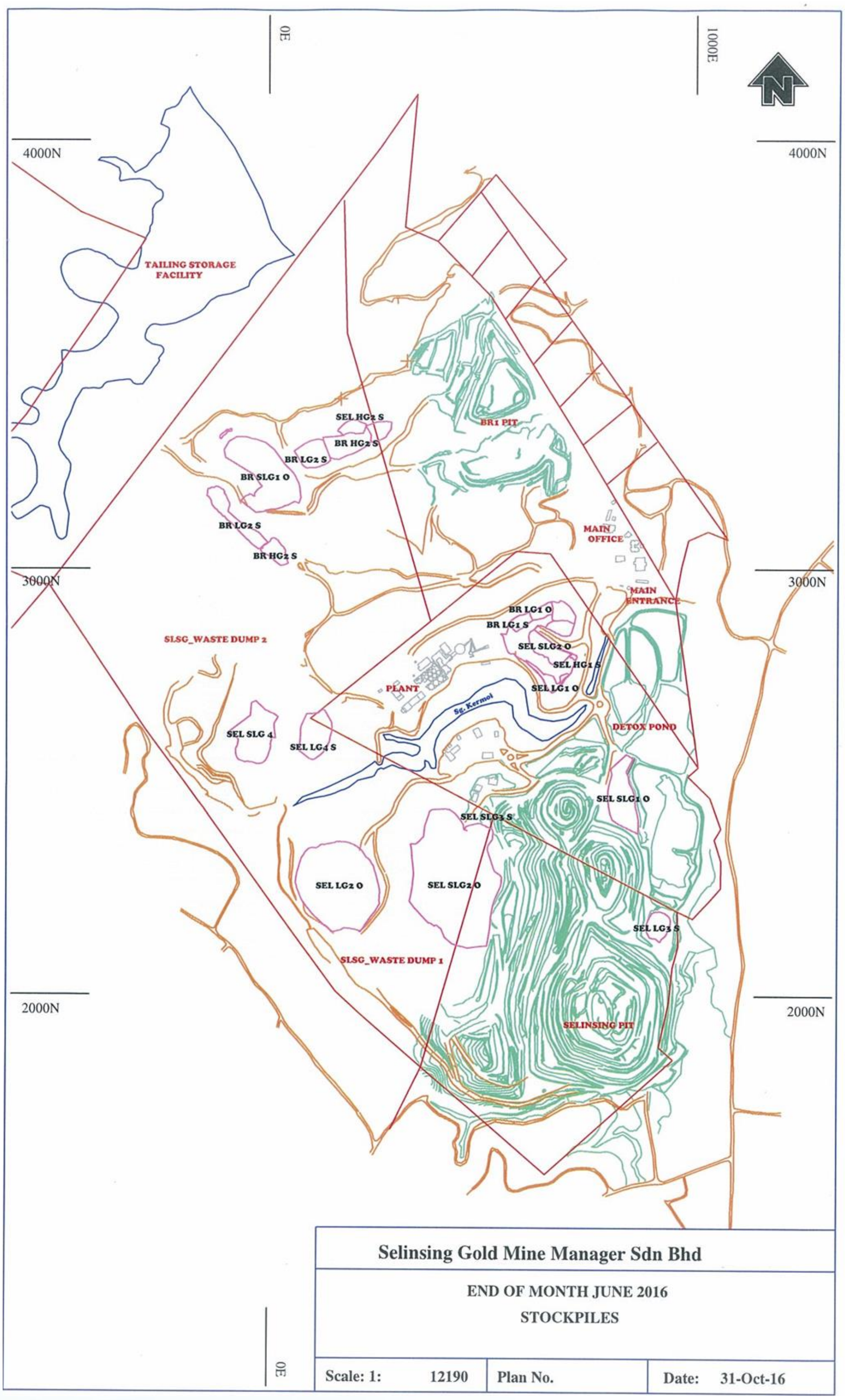
Figure 14.40 Grade-tonnage curve for Buffalo Reef



14.4 Stockpile Mineral Resources

Stockpiles at the Selinsing project include ore mined from the Selinsing and Buffalo Reef pits. The current location of the stockpiles is shown in Figure 14.41.

Figure 14.41 Stockpile location plan, as at end June 2016



Source: Monument

14.4.1 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 currently).

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.

14.4.2 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 done using Surpac software.

14.4.3 Stockpile bulk density

The bulk density of the stockpiles is based on applying a 25% swell factor to the in situ density. The in-situ density determination procedure is described in Section 14.2.5. Approximately 10 to 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.

14.4.4 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.

14.4.5 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.

14.4.6 Stockpile Mineral Resource report

Mineral Resources for the stockpiles at the Selinsing Project, as at the end of June 2016, are summarised in Table 14.38.

Table 14.38 Stockpile Mineral Resources, as at end of June 2016

Stockpile name	Stockpile ID	Volume (lcm)	Tonnes (t)	Au (g/t)	Contained gold (oz)
Oxide stockpiles					
Selinsing					
Low grade 1 (oxide)	SEL LG1 O	6,885	14,075	1.03	467
Low grade 2 (oxide)	SEL LG2 O	3,189	6,442	0.73	152
Super low grade 1 (oxide)	SEL SLG1 O	2,845	5,349	0.44	76
Super low grade 2 (oxide)	SEL SLG2 O	907,006	1,859,251	0.51	30,747
Super low grade 4 (oxide)	SEL SLG 4	31,378	67,776	0.50	1,090
Buffalo Reef					
Low grade 1 (oxide)	BR LG1 O	186	353	0.33	4
Super low grade 1 (oxide)	BR SLG1 O	111,268	217,422	0.53	3,739
Oxide subtotal		**1,062,757**	**2,170,668**	**0.52**	**36,275**
Leachable sulphide stockpiles					
Selinsing					
High grade 1 (leachable sulphide)	SEL HG1 S	81	175	6.41	36
Low grade 1 (leachable sulphide)	SEL LG1 S	88	190	0.98	6
Buffalo Reef					
Low grade 1 (leachable sulphide)	BR LG1 S	82	166	0.32	2
Leachable sulphide subtotal		**251**	**531**	**2.56**	**44**
Non-leachable sulphide stockpiles					
Selinsing					
High grade 2 (non-leachable sulphide)	SEL HG2 S	5,065	10,940	2.71	953
Low grade 3 (non-leachable sulphide)	SEL LG2 S	8,402	16,983	0.97	529
Low grade 4 (non-leachable sulphide)	SEL LG4 S	25,331	54,715	0.95	1,663
Super low grade 3 (non-leachable sulphide)	SEL SLG3 S	748	1,511	0.60	29
Buffalo Reef					
High grade 2 (non-leachable sulphide)	BR HG2 S	18,536	36,695	2.58	3,045
Low grade 2 (non-leachable sulphide)	BR LG2 S	22,444	43,206	1.03	1,429
Non-leachable sulphide subtotal		**80,526**	**164,051**	**1.45**	**7,648**
TOTAL		**1,143,534**	**2,335,250**	**0.59**	**43,966**

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)*

15 MINERAL RESERVE ESTIMATES

Mineral Reserve estimates are currently reported for the mining operations at the Selinsing property, comprising in situ reserves for the Selinsing and Buffalo Reef deposits and stockpiles for Selinsing and Buffalo Reef.

15.1 Disclosure

Mineral Reserves reported in Section 15 were based on prefeasibility studies supervised by Frank Blanchfield who is a Qualified Person for this report.

15.1.1 Known issues that materially affect Mineral Reserves

Snowden is unaware of any issues that materially affect the Mineral Reserves in a detrimental sense.

Snowden estimated gold Mineral Resources and Mineral Reserve estimates for Monument's Selinsing gold deposit. Snowden identified an updated mining inventory based on the new Mineral Resource estimates from June 2016. The Selinsing and Buffalo Reef 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.

While exercising all reasonable due diligence in checking and confirming the data validity, Snowden has relied largely on the data as supplied by Monument to estimate and classify the Mineral Reserve. As such, Snowden accepts responsibility for the geotechnical design configuration, pit design, production schedule, direct mining costs and the reserve estimate and classification, while Monument has assumed responsibility for the accuracy and quality of the metallurgical data.

The key Modifying Factors used to estimate the Mineral Reserve are based on the experience of Snowden and Monument employees in this type of deposit and style of mineralisation. Table 15.1 summarises the status of material aspects of the September 2016 Selinsing Mineral Reserve estimate, against the items listed in the table.

The information in this report that relates to Selinsing Mineral Reserve is based on information reviewed or work undertaken by Mr Frank Blanchfield, FAusIMM, an employee of Snowden. Mr Blanchfield has sufficient experience relevant to the style of mineralisation and type of deposit under consideration and to the preparation of mining studies to qualify as a Qualified Person as defined by NI 43-101.

The scientific and technical information in this report that relates to process metallurgy is based on information reviewed by Dr Leon Lorenzen, FAusIMM, who is an employee of Snowden. Dr Lorenzen 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.

Table 15.1 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	Snowden prepared the updated Selinsing Mineral Resource estimate in June 2016. 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, Buffalo Reef and stockpiles and are inclusive of Mineral Reserves.

Buffalo Reef

Classification	Oxidation	Zone	Cut-off (g/t Au)	Tonnes (kt)	Au (g/t)	Ounces (koz)
Indicated	Oxide	BRN	0.3	180	0.99	6
		BRC	0.3	170	0.83	4
		Felda	0.3	260	1.33	11
		BRS	0.3	100	2.1	7
		Oxide Total		**700**	**1.23**	**27**
	Transitional	BRN	0.7	150	1.26	6
		BRC	0.7	310	1.19	12
		Felda	0.7	190	1.64	10
		BRS	0.7	230	2.65	19
		Transitional Total		**860**	**1.68**	**46**
	Fresh	BRN	0.7	70	1.18	2
		BRC	0.7	990	1.67	53
		Felda	0.7	620	1.78	35
		BRS	0.7	1,130	2.12	77
		Fresh Total		**2,790**	**1.87**	**167**
	Indicated Total			**4,330**	**1.73**	**240**

Selinsing

Classification	Oxidation	Cut-off (g/t Au)	Tonnes (kt)	Au (g/t)	Ounces (koz)
Indicated	Oxide	0.3	90	0.67	2
	Transitional	0.7	90	1.42	4
	Fresh	0.7	3,040	1.98	193
Indicated Total			**3,220**	**1.93**	**200**

Stockpiles

Stockpile name	Volume (klcm)	Tonnes (kt)	Au (g/t)	Contained gold (koz)
Oxide total	**1,063**	**2,171**	**0.52**	**36**
Leachable sulphide total	**0.3**	**0.5**	**2.56**	**0.04**
Non-leachable sulphide total	**81**	**164**	**1.45**	**8**
GRAND TOTAL	**1,144**	**2,335**	**0.59**	**44**

All stockpiles are classified using CIM guidelines as Proven Mineral Reserves.

The Selinsing Mineral Resources are inclusive of Mineral Reserves.

Item	Comment
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.
Study status	The current NI 43-101 technical report if a PFS to establish the viability of sulphide ore extraction through the extension of the existing oxide plant to incorporate additional sulphide ore extraction. Another study was completed by Practical Mining LLC in 2012 for the extraction of sulphides from the Selinsing and Buffalo Reef deposits. Snowden has re-evaluated this work using the reports from Lycopodium that updated the metallurgy costs and recoveries in 2016. Snowden considers that most of the mining work completed is of a prefeasibility level of accuracy.
Cut-off parameters	A nominal cut-off grade of 0.35 g/t Au was applied to oxides and 0.75 g/t for sulphides when developing the Ore Reserve estimate.

<table>
<tr><th>Item</th><th>Comment</th></tr>
<tr><td>Mining factors and assumptions</td><td>
To identify the Selinsing and Buffalo Reef Ore Reserve, a process of Whittle pit optimisation, staged pit design, production scheduling and mine cost modelling was undertaken by Snowden.
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, block dilution has reduced the recovered ounces by approximately 2% and marginally increased the ore tonnage processed by 5%.
</td></tr>
<tr><td>Metallurgical factors and assumptions</td><td>
The Selinsing Gold Mine (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 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 t/d gold treatment plant in three phases.
During 2011, Monument Mining Limited (Monument) engaged Inspectorate Exploration and Mining Services Ltd (Inspectorate) of Vancouver, Canada, to carry out a metallurgical test program on a selection of diamond drill core material collected from the Buffalo Reef deposit at its Selinsing operation in Malaysia. The extraction for sulphides at Selinsing has been assessed in the engineering study ultimately prepared for Monument by Lycopodium of Brisbane, Australia, and reported by Lycopodium in “Selinsing Phase IV Study”, February 2013.
The mineralisation modelling and metallurgical testwork indicate that conventional carbon-in-leach (CIL) extraction from oxide ores and bio-oxidation leach for transition and fresh ores can be used to produce gold as Dore.
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.

Lycopodium applied industry standard methods to prepare this estimate by developing the following components:

Process design criteria – based on the Inspectorate report (ibid)
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 Lycopodium and reviewed by Snowden. 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)</th><th>Recovery (%)</th></tr>
<tr><td rowspan="4">Oxide#</td><td rowspan="4">Both</td><td><1.0</td><td>66</td></tr>
<tr><td>1.0 to 1.5</td><td>75</td></tr>
<tr><td>1.5 to 2.5</td><td>83</td></tr>
<tr><td>>2.5</td><td>87</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>87</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>87</td></tr>
</table>
</td></tr>
</table>

<table>
<tr><th>Item</th><th>Comment</th></tr>
<tr><td></td><td>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 PFS level of accuracy.</td></tr>
<tr><td>Environmental</td><td>Rock characterisation was completed in Malaysia and potentially acid-forming (PAF) 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.</td></tr>
<tr><td>Infrastructure</td><td>Monument is negotiating with the local authority for power purchase from the electricity grid.

Monument has indicated the bio-oxidation plant build will be a EPC execution with Monument providing the management

Accommodation will be in surrounding communities.</td></tr>
<tr><td>Cost and revenue factors</td><td>Process costs were used from historical oxide costs from site and developed by Lycopodium in 2016 for sulphide as:
<table>
<tr><th>Material treated</th><th>Deposit</th><th>Gold grade (g/t)</th><th>Process operating cost (US$/t)</th></tr>
<tr><td rowspan="4">Oxide[#]</td><td rowspan="4"></td><td><1.0</td><td>7.11[^]</td></tr>
<tr><td>1.0 to 1.5</td><td>7.11[^]</td></tr>
<tr><td>1.5 to 2.5</td><td>7.11[^]</td></tr>
<tr><td>>2.5</td><td>7.11[^]</td></tr>
<tr><td rowspan="2">Transition*</td><td>Selinsing</td><td>All</td><td>20.00</td></tr>
<tr><td>Buffalo Reef</td><td>All</td><td>20.00</td></tr>
<tr><td rowspan="2">Fresh/Sulphides</td><td>Selinsing</td><td>All</td><td>19.56[^]</td></tr>
<tr><td>Buffalo Reef</td><td>All</td><td>19.56[^]</td></tr>
</table>
Mining costs were supplied by Monument and developed from the existing contract.

The all-up mining operating cost was estimated to be US$1.90/t mined.

The mining capital cost was absorbed by contract mining.

Monument provided other capital costs, that were estimated by Lycopodium and others as follows:

Process capital costs: US$38.9 million
Closure costs: US$7.8 million
Sustaining costs: US$0.5 million per annum.

Closure costs are included in the valuation model.

All costs were supplied in US$.

Refining costs of US$5.00/t and royalties of 5% were applied to all gold produced, except Felda (that was 7%)</td></tr>
<tr><td>Revenue factors</td><td>Monument supplied a gold price of US$1,255/oz. This was applied as real and flat forward in the financial model.</td></tr>
<tr><td>Market assessment</td><td>Monument supplied a gold price of US$1,255/oz.

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:</td></tr>
</table>

Item	Comment
	<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$/oz produced</td><td>830.1</td></tr><tr><td>IRR ungeared</td><td>%</td><td>34.8</td></tr><tr><td>NPV (at 8%)</td><td>US$ M</td><td>23.1</td></tr><tr><td>Net cashflow</td><td>US$ M</td><td>37.2</td></tr><tr><td>Initial capital cost[a]</td><td>US$ M</td><td>39.5</td></tr></table> [a] *Excludes working capital*
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 that there are no community or social encumbrances that could obstruct the provision of a MLA from the Malaysian government.
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 in situ. No Inferred Resources is included in the Ore Reserve estimate.
Audits or reviews	Snowden has completed an internal peer review of the Ore Reserve estimate.
Relative accuracy/ confidence	It is Snowden’s opinion that the Ore 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 Mineral Reserve reporting

15.2.1 Selinsing and Buffalo Reef deposits – Mineral Reserve statement

The Mineral Reserve estimate for the Selinsing and Buffalo Reef deposits, as at the end of June 2016, is provided in Table 15.2. Note that tonnes and ounces have been rounded and this may have resulted in minor discrepancies.

Table 15.2 Selinsing and Buffalo Reef in situ Mineral Reserve estimate as at June 2016

Classification	Oxidation	Zone	Approximate cut-off (g/t Au)	Tonnes (kt)	Au (g/t)	Ounces (koz)
Probable	Oxide	Selinsing	0.3	8	1.2	0.3
		BRN	0.3	105	1.05	3.5
		BRC	0.3	114	0.91	3.3
		Felda	0.3	234	1.34	10.1
		BRS	0.3	103	1.95	6.5
	Oxide total			**565**	**1.31**	**23.8**
	Transitional	Selinsing	0.7	25	2.02	1.7
		BRN	0.7	69	1.29	2.9
		BRC	0.7	214	1.26	8.6
		Felda	0.7	158	1.66	8.5
		BRS	0.7	232	2.52	18.5
	Transitional total			**698**	**1.80**	**40.4**
	Fresh	Selinsing	0.7	551	2.33	41.2
		BRN	0.7	14	1.25	0.6
		BRC	0.7	719	1.76	40.6
		Felda	0.7	474	1.75	26.7
		BRS	0.7	862	2.22	61.4
	Fresh total			**2,619**	**2.03**	**170.6**
PROBABLE TOTAL				**3,882**	**1.88**	**235.4**

15.2.2 Selinsing property Stockpile Mineral Reserves statement

Mineral Reserves for the stockpiles at the Selinsing Project, as at the end of June 2016, 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 June 2016

Stockpile name	Stockpile ID	Volume (lcm)	Metric tonnes (t)	Au (g/t)	Contained gold (Troy oz)
Oxide stockpiles					
Selinsing					
Low grade 1 (oxide)	SEL LG1 O	6,885	14,075	1.03	467
Low grade 2 (oxide)	SEL LG2 O	3,189	6,442	0.73	152
Super low grade 1 (oxide)	SEL SLG1 O	2,845	5,349	0.44	76
Super low grade 2 (oxide)	SEL SLG2 O	907,006	1,859,251	0.51	30,747
Super low grade 4 (oxide)	SEL SLG 4	31,378	67,776	0.50	1,090
Buffalo Reef					
Low grade 1 (oxide)	BR LG1 O	186	353	0.33	4
Super low grade 1 (oxide)	BR SLG1 O	111,268	217,422	0.53	3,739
Oxide subtotal		**1,062,757**	**2,170,668**	**0.52**	**36,275**
Leachable sulphide stockpiles					
Selinsing					
High grade 1 (leachable sulphide)	SEL HG1 S	81	175	6.41	36
Low grade 1 (leachable sulphide)	SEL LG1 S	88	190	0.98	6
Buffalo Reef					
Low grade 1 (leachable sulphide)	BR LG1 S	82	166	0.32	2
Leachable sulphide subtotal		**251**	**531**	**2.56**	**44**
Non-leachable sulphide stockpiles					
Selinsing					
High grade 2 (non-leachable sulphide)	SEL HG2 S	5,065	10,940	2.71	953
Low grade 3 (non-leachable sulphide)	SEL LG2 S	8,402	16,983	0.97	529
Low grade 4 (non-leachable sulphide)	SEL LG4 S	25,331	54,715	0.95	1,663
Super low grade 3 (non-leachable sulphide)	SEL SLG3 S	748	1,511	0.60	29
Buffalo Reef					
High grade 2 (non-leachable sulphide)	BR HG2 S	18,536	36,695	2.58	3,045
Low grade 2 (non-leachable sulphide)	BR LG2 S	22,444	43,206	1.03	1,429
Non-leachable sulphide subtotal		**80,526**	**164,051**	**1.45**	**7,648**
TOTAL		**1,143,534**	**2,335,250**	**0.59**	**43,966**

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

16 MINING METHODS

16.1 Geotechnical investigation

16.1.1 Previous geotechnical studies

Golder Associates

Golder Associates (Golder) undertook a geotechnical study for the Selinsing project in 1997 (Golder Associates, 1997). Open Pit Geotechnical Assessment, Selinsing Gold Project, Malaysia. Draft technical report to Target Resources Australia NL, May 1997. This included:

- 10 geotechnical logged boreholes with collar and survey information (897 m of geotechnical logs)
- Associated lithology logs
- UCS test results on 16 samples
- Direct shear test results on seven samples.

The pit slope design parameters recommended by Golder are summarised in Table 16.1.

Table 16.1 Golder Associates pit design recommendations for Selinsing

Wall	Maximum face angle	Maximum face height	Berm width	Maximum overall slope angle
West above 490 mRL, west of 740 mE	40° or parallel to shear fabric (whichever is flatter)	10 m (above 530 mRL) 20 m (below 530 mRL)	5 m	37°
West below 490 mRL, east of 740 mE	55° or parallel to shear fabric (whichever is flatter)	20 m	5 m	45°
East	60°	20 m	5 m	55°
North	60°	20 m	5 m	55°
South	60°	20 m	5 m	55°

However, Golder concluded that:

- Their recommended overall wall angles may be aggressive in weaker, weathered or sheared zones
- There was insufficient data to determine position and thickness of material types, including the weak zones
- That such zones daylighting in the pit walls should be identified and appropriate slope angles for these zones reassessed.

Snowden

Snowden undertook a geotechnical assessment of the Selinsing and Buffalo Reef deposits in 2013 (Geotechnical Assessment of Selinsing and Buffalo Reef, May 2013). This work included a site visit to inspect and assess the performance of the pit slopes, and log selected drill core, and to undertake slope stability analyses in order to provide revised slope design parameters. The review identified the following geotechnical issues in the Selinsing pits:

- Batter angles on west wall too steep, causing localised batter failure and spalling
- Catch berms full or inadequate width on the south and west walls

- Sub-parallel and circular cracking behind the west wall
- Sub-vertical cracking on the north and west wall
- Circular cracking behind the east wall
- Toppling failure developing in east wall (grey limestone).

The slope design angles recommended based on the 2013 review are summarised in Table 16.2.

Table 16.2 2013 Snowden slope design recommendations for Selinsing and Buffalo Reef

Wall	Maximum face angle	Maximum face height	Berm width	Maximum overall slope angle
West above 490 mRL, west of 740 mE	40° or parallel to shear fabric (whichever is flatter)	10 m (above 530 mRL) 20 m (below 530 mRL)	5 m	37°
West below 490 mRL, east of 740 mE	55° or parallel to shear fabric (whichever is flatter)	20 m	5 m	45°
East	60°	20 m	5 m	55°
North	60°	20 m	5 m	55°
South	60°	20 m	5 m	55°

16.1.2 Latest geotechnical study

Snowden

Snowden undertook a geotechnical review of the Selinsing and Buffalo Reef mines in 2016 (Memo to Monument - Geotechnical Review of Selinsing and Buffalo Reef, 2016).

16.1.3 Findings and recommendations

Selinsing

Information on the geological and geotechnical characteristics of the units forming the west wall and any pit-scale structures underlying the slopes is very limited. Should deepening of the pit be considered, this wall will require cutting back and the dump material moved. Further mining is likely to destabilise this wall. A substantial step-in buttress is required if the pit is deepened without a cutback.

The east wall has sufficiently competent conditions to increase the inter-ramp angle by at least 5°. Use of pre-splitting and lower batter heights would help control of batter-scale stability issues.

The north and south walls have adequate stability at present, but need monitoring if the pit is deepened. Potential issues include groundwater pressures within the shear fault structures; failure of these zones may lead to progressive failure along the slopes.

In general, the successful application of 20 m high batters requires more competent ground conditions than those observed at Selinsing. As a rule of thumb, the size of a slope failure such as a wedge or rotational slip is proportional to the square of the height, hence 10 m batters usually provide much better outcomes in moderate to poor ground conditions.

It is also essential to control/reduce groundwater pressures in the pit walls and to prevent ingress of surface water run-off, and to assess potential recharge from adjacent streams/ponds. Depressurisation of slopes is probably more efficiently achieved by horizontal drain holes on each bench rather than de-watering bores – many more potentially water-bearing structures are intersected by a substantial number of drain holes than by a small number of de-watering bores. Drain holes are recommended to be spaced at 10 m intervals along each berm, and be a minimum length 30 m (preferably 50 m).

The waste and low-grade dumps are sufficiently close to the west wall crest in some areas to be applying surcharge loading. For a wall with marginal stability this loading may be sufficient to trigger failure.

Buffalo Reef

Geotechnical information is very limited for the Buffalo Reef deposits; however, the mine has undertaken detailed interpretation and modelling of pit-scale structural features.

Current pit walls are excavated largely in completely to highly weathered rock, and above the local groundwater levels. Exposures of moderately weathered rock below the oxide boundary are limited to the south deposit pit only. Only the footwall ground conditions were observed in the pits; hangingwall conditions are assumed to be the similar, although may be slightly better as the bedding dips eastwards, into the wall.

The main host rock material appears to be similar in nature to the Selinsing Pit 4 footwall argillite hence is expected to behave in similar manner. These have been subjected to shearing along bedding, resulting in low shear strength of the argillite parallel to the bedding orientation. In all areas, there are at least three structure sets resulting in fissile slabby/blocky rock mass conditions.

Two walls in the central deposit pit with heights of 30 m to 35 m have failed due to a combination of very weak material and groundwater pressures. These factors will clearly influence stability of the other pits, in combination with the low shear strength parallel to bedding within the argillite.

16.1.4 Recommended batter/berm dimensions

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

- Maximum recommended batter height: 10 m
- Minimum berm width: 4.0 m (increase where necessary to suit maximum inter-ramp angle).

Recommendations for pit wall design parameters have been developed using the current slope stability outcomes and the estimated GSI for the main units; these are summarised in Table 16.3.

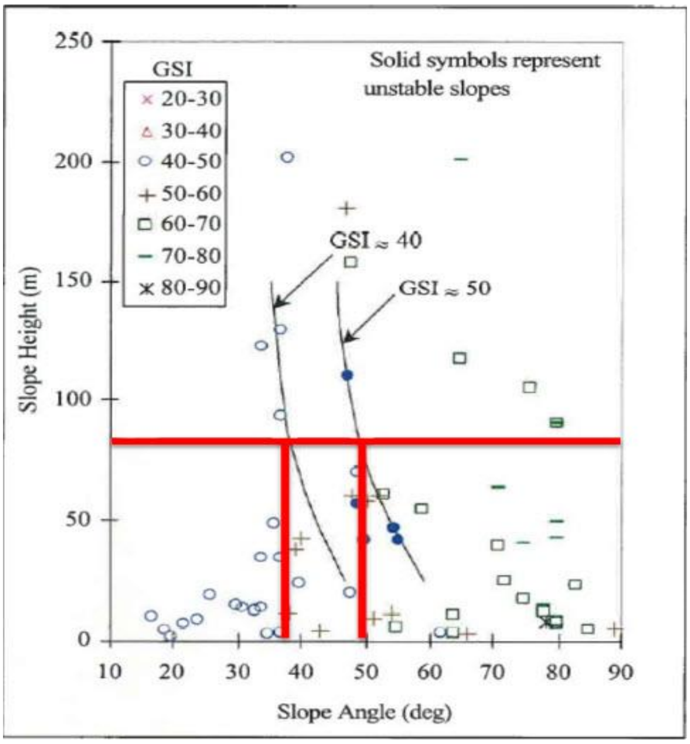
Table 16.3 Snowden slope design recommendations for Selinsing and Buffalo Reef

Deposit	Geotechnical domain	Batter face angle	Inter-ramp slopes	
			Face angle	Maximum bench stack height
Selinsing	Saprolite/highly weathered Rock	45°#	30°	30 m
	Footwall	50°#	35°	80 m
	Ore zone	60°	40°	80 m
	Hangingwall	65°	45°	80 m
Buffalo Reef	Saprolite/highly weathered rock	45°#	30°	30 m
	Footwall	50°#	40°*	80 m
	Hangingwall^	50°	40°*	80 m

 # Or parallel to bedding/shear fabric (whichever is flatter)
** May be increased for bench stacks with lower heights – see Figure 16.1*
^ Conditions not observed, assumed similar to “Footwall”

The empirical design chart used to develop the recommended slope angles is presented in Figure 16.1. Adjustments to slope angles obtained for Selinsing from the chart have been made according to Snowden’s assessment of the influence of structural and groundwater conditions.

Figure 16.1 Empirical slope design chart (80 m bench stack height)



The maximum bench stack height is the maximum vertical interval between ramps. In the absence of a ramp at this interval, a 10 m geotechnical berm should be included in the pit design.

Given the slabby, blocky nature of the rock mass throughout the deposits, the mine should consider smooth blasting techniques such as pre-splitting to limit blast damage when forming the final pit walls.

16.2 Mining concept

The Selinsing and Buffalo Reef deposits have been mined using open pit methods for seven years and the mining contractor for the mining activities is Minetech Construction Sdn Bhd.

16.2.1 Bench height

The height of the mining benches is usually determined according to physical characteristics of the mineralisation. However, the need for flexibility in the mining fleet to be able to adequately mine and blend from a number of different ore faces within the pit to achieve the forecast ore feed requirements has meant that the maximum bench height and therefore the largest mining fleet configuration has not been selected for the optimal extraction of the ore.

Ore is drilled and blasted on standard 10 m benches, and then mined by excavators nominally on 2.5 m lifts, taking into account blast-induced swell, into rear dump, off-highway, haul trucks. A 10 m bench has been selected for this study for relatively selective mining of the ore benches, but larger benches and their impact on grade control, product quality and mining cost and productivity should be reviewed in any further study work.

Waste is drilled and blasted on 10 m benches, and then mined by excavators nominally on a 2.5 m high lift, not taking into account blast-induced swell. The final bench height will need to be optimised in any future study to the mining unit selected for waste mining.

16.2.2 Waste dumps

The waste dump design is based on the waste material types generated through mining. Both PAF (potentially acid forming) and NAF (non-acid forming) waste have been identified from the testwork undertaken and modelling to generate a 3D block model that was in turn used to generate the mining waste schedule for the expected life of mine. The waste dump design was based on the mining waste schedule and involves selective mining and placement of PAF and NAF waste within the waste rock precinct to form a waste rock dump that encapsulates the PAF waste.

The waste dump design involves placement of a base layer of NAF material over the entire dump footprint using paddock dumping technique. The base is then flattened out by dozing to allow passage of dump trucks for placement of the PAF waste. The PAF waste is then placed on the prepared NAF base by paddock dumping within the designated encapsulation cell precinct. The outer walls of the PAF cell will be NAF waste. Each layer of PAF waste will be smoothed. The design of the dump will be compacted by directed haul truck movement. An interim NAF cover compacted by haul truck movement will be placed on the PAF waste prior to the wet season to further limit net percolation.

16.2.3 Dump height

The waste rock stockpile is placed away from ridges and its height does not exceed 10 m so as to not cause a visual impact.

16.2.4 Dump profile

Key objectives are to ensure geotechnical and erosional stability (both aided by the placement of coarse-textured waste rock on outer batter slope, and the placement of waste rock at its angle of repose: 1:1.5 or 34º) while facilitating revegetation, rather than to set an unrealistic final stockpile outer slope.

16.2.5 Roads, dumps and stockpiles

All haul roads, dumps and stockpiles that are required for the life of mine will be constructed during pre-production.

The waste rock dump associated with mining operations is constructed conforming to the guidelines set out by the Minerals & Geoscience Department Malaysia in the approved Operations Mining Scheme.

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 run-off is controlled and directed down to sediment traps.

The waste dump design is to incorporate features to minimise the effect of leaching of contaminants.

16.2.6 Topsoil stripping and spreading

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.

16.2.7 Haul road design and traffic management

A detailed set of designs for haul road construction were reviewed by the Qualified Person during the site visit. Haulage is the largest mining activity cost and the design, construction and maintenance of haul roads is considered in the haulage costs.

Light vehicle traffic is restricted to only production personnel and selected technical personnel being able to drive in pit and mix with the heavy traffic.

Light vehicle travel speeds are limited to 40 km/hr and heavy equipment roads at 30 km/hr. Pit floors, dumps and ramps are limited to 15 km/hr.

Major two-way all-vehicle roads where heavy traffic occurs have PVC pipe demarcation on both sides of the road, with reflectors to increase safely at night.

National roads rules are followed on site, and all intersections use standard road signage.

16.2.8 Stockpiling and reclaiming

Due to the waste-to-ore strip ratio, scheduling issues, ore presentation and equipment availability, it is not possible to direct tip ore all the time. The difference between the mobile and fixed operating hours creates a requirement to rehandle a minimum of between 10% and 15% of the ore at stockpiles including the ROM stockpile at the crushing facility.

Ore rehandle is required to ensure the following:

- Good effective utilisation of the mining fleet allowing excavators to move from ore into their allocated waste zones, without relocating to another ore zone upon completion of their ore block.
- There will regularly be times where there is an oversupply of ore which is stockpiled for future use. This ensures maximum effective utilisation of the select mining fleet and less disruption to the short-term mine plan.
- The ROM stockpile is used to blend appropriate grade and texture to feed ore grade to optimise process plant throughput and recovery.
- The mine will build up very large stockpiles of low grade material, which cannot be used for ROM feed in any significant volume. Whenever the blend can be absorbed it is fed, ensuring maximum mill feed production.

During normal operations, the ore feed is achieved by a combination of ore direct tipped from the pit into the ROM bin by the haul trucks with the ROM loader adding the makeup from ROM stockpiles. Approximately 40% of all ore sent directly to the ROM pad is stockpiled and rehandled due to ore blending and scheduling requirements.

16.2.9 Pit dewatering and drainage

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.

16.2.10 Site layout

Mining related infrastructure includes offices, preparation laboratory, core and sample storage, and mining contractor hard stand.

There are no treatment facilities at Buffalo Reef. Ore from Buffalo Reef would be processed at the nearby SGM processing plant.

The site layout with location of all open pit deposits and main infrastructure layout are shown below in Figure 16.2 and Figure 16.3.

Figure 16.2 Site layout

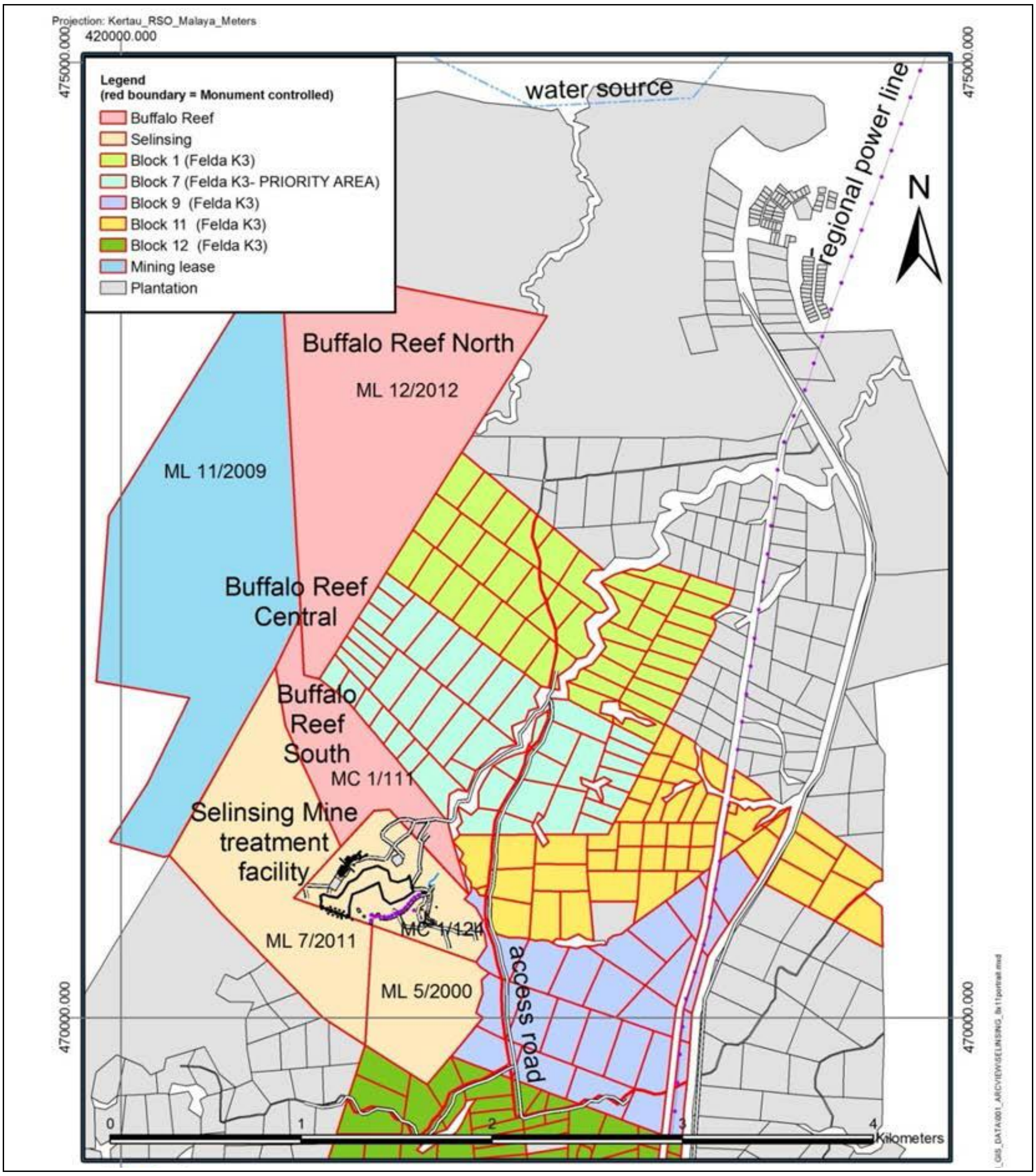
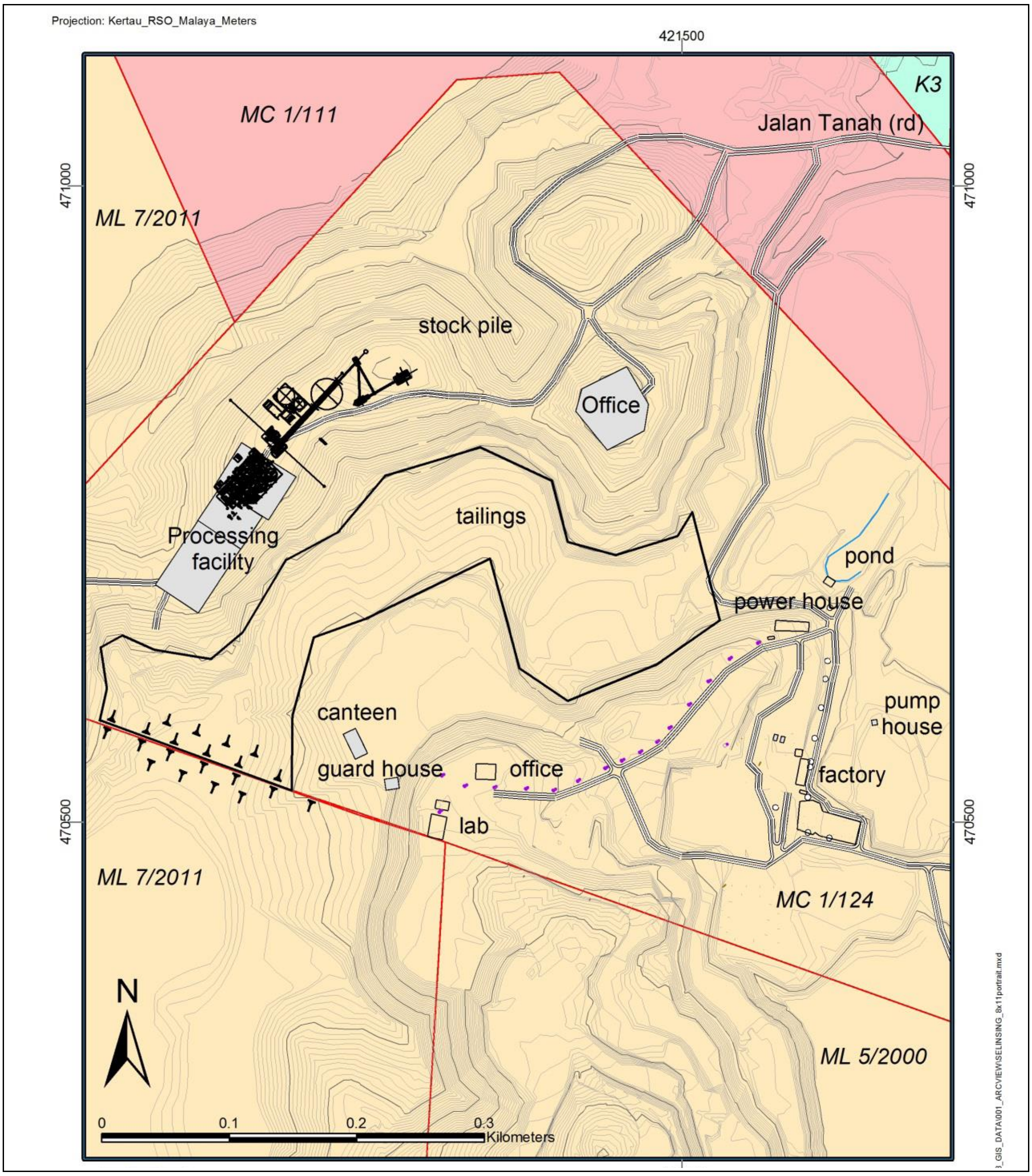


Figure 16.3 Site layout of main infrastructure



16.2.11 Drill and blast

The preliminary geotechnical investigation determined that the rock mass is altered to varying degrees. The alteration indicates that not all material will require blasting. It is assumed for the PFS that 90% of the ore will require blasting and 92% of the waste. The entire rock mass requires drilling for grade control purposes. Below 500 mRL, most blast holes have some groundwater present. The explosive is in the form of Bulk Emulsion. The variability of rock strength varies the powder factor throughout the pit. The forecast average powder factor is 0.22 kg of explosives per tonne of rock. It is proposed to use non-electric initiation for blasts.

Drilling is executed using rotary blast hole drills with a hole diameter of 89 mm. The drilling pattern is dependent on the powder factor assigned for each drill bench. Hole drilling is 10 m deep and sub-drilling is 0.7 m to 0.8 m.

16.2.12 Load and haul

The mining fleet is owned and operated by Minetech Construction Sdn Bhd. The load and haul fleet was selected to achieve close to 8,000 t/d to 10,000 t/d movement. Loading operations are carried out using a fleet of up to four excavators. The excavator has a bucket capacity of 1.5 BCM to 3.2 BCM. The excavator loads a fleet of up to 28 x 30 t and 2 x 80 t capacity haul trucks. The number of haul trucks was based on an average haul distance.

Ore is to be hauled to the ROM pad adjacent to the primary crusher or to designated grade and oxidation level related stockpiles.

The load/haul fleet is supported by ancillary equipment including dozers, graders, water trucks and service trucks. In addition, the ancillary equipment is used to construct and maintain roads, stockpiles and waste dumps.

Waste rock is hauled to separate waste dumps and also to retention walls for water control and tailings storage and where required for construction. Selective dumping and encapsulation of sulphide waste rock is required to maximise neutralisation and minimise any mobilisation of acid.

16.2.13 Grade control

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.

16.2.14 Other mining activities

Haul roads are constructed by the contractor when required and as suitable material becomes available, to meet pit access and ore release production schedules.

The main function of mine services is to provide assistance to the production units. The work performed includes, road maintenance, pit dewatering, rainfall run-off management and waste dump preparation and maintenance.

The Selinsing Gold Project ROM pad works dayshifts and nightshifts. Portable lighting towers are required where there is a lack of permanent lighting. Permanent lighting is installed close to main power supplies such as at the crusher and road intersections adjacent to the crusher pad.

16.2.15 Open pit work roster

The mining operations are scheduled to work throughout the year, less public holidays and unscheduled delays such as high rainfall events which may cause mining operations to be temporarily suspended.

The mining operation operates six days a week, in 10-hour working shifts with the equipment services scheduled as required. The crushing plant is scheduled to operate continuously except for planned maintenance periods.

16.3 Mine design

16.3.1 Methodology

A mining model was prepared from the resource model using Datamine Studio 3.

The mining model was imported into GEOVIA's Whittle Four-X™ (Whittle). Whittle uses the Lerch-Grossman algorithm to produce a number of incremental pit shells (nested) based on varying the input price. For a given set of costs, a revenue factor of one equates to the input price (i.e. results in a marginal undiscounted cash flow of $0/t ore). These nested pit shells are used in selecting the "optimum" pit shell, guiding the location of pushbacks/stages and determining resource envelopes.

The selected pit shells were used to design pits in GEOVIA's Surpac ®.

16.3.2 Pit optimisation

Mining model

Snowden used the following resource models for mine planning:

- Selinsing – sel1605v1.dm
- Buffalo Reef – br1604v2.dm.

Snowden adjusted this model to account for mining (and waste dumping) conducted up until 30 June 2016.

A report of each of these models (at a 0.5 g/t Au cut-off grade) is provided in Table 16.4.

Table 16.4 Resource model report (Indicated Resource at 0.5 g/t Au cut-off)

Deposit/oxidation	Mass (kt)	Grade (g/t Au)	Contained gold (koz)
Selinsing	**5,466**	**1.46**	**256.5**
Oxide	61	0.80	1.6
Transition	149	1.12	5.4
Fresh	5,256	1.48	249.6
Buffalo Reef North	**425**	**1.10**	**15.0**
Oxide	151	1.11	5.4
Transition	171	1.15	6.3
Fresh	103	0.99	3.3
Buffalo Reef South	**4,220**	**1.72**	**233.0**
Oxide	399	1.58	20.2
Transition	843	1.59	42.9
Fresh	2,978	1.77	169.8
Total	**10,111**	**1.55**	**504.5**
Oxide	611	1.38	27.2
Transition	1,163	1.46	54.6
Fresh	8,337	1.58	422.7

To prepare the models for optimisation, Snowden applied a 0.5 m dilution skin around the orebody (as defined as blocks with gold grade of above 0.1 g/t Au) in all dimensions. "Mineralised" blocks were then aggregated into regular blocks of the following dimensions:

- Selinsing – 5 mE x 5 mN x 5 mRL
- Buffalo Reef – 4 mE x 5 mN x 5 mRL.

The difference in block size selection is due to differences in the resource model parent block size.

The final mining model contains the following additional fields:

- PORE – the proportion of the parent block that is mineralised
- AU – the average grade of the mineralised portion of each block.

The categorical fields (RESCAT, MINZONE, OXIDE) were assigned to each parent block based on the dominant category (as defined by gold content).

A report of the mining model is provided in Table 16.4.

Table 16.5 Summary of mining model (Indicated Resources at 0.5 g/t Au cut-off grade)

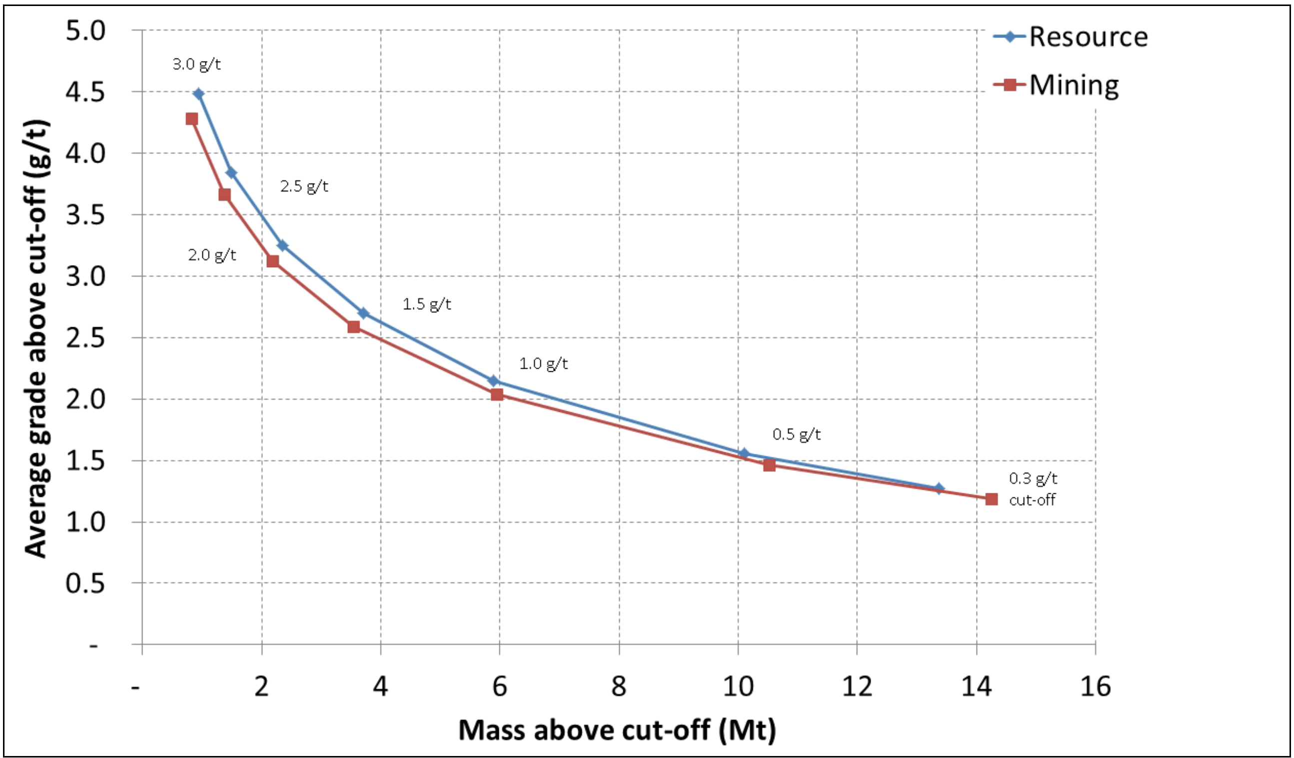
Deposit/oxidation	Mass (kt)	Grade (g/t Au)	Contained gold (koz)
Selinsing	**5,564**	**1.41**	**251.9**
Oxide	61	0.77	1.5
Transition	155	1.05	5.2
Fresh	5,348	1.43	245.2
Buffalo Reef North	**444**	**1.00**	**14.3**
Oxide	157	1.00	5.0
Transition	182	1.05	6.1
Fresh	105	0.92	3.1
Buffalo Reef South	**4,515**	**1.59**	**230.1**
Oxide	420	1.45	19.6
Transition	887	1.47	42.0
Fresh	3,208	1.63	168.5
TOTAL	**10,523**	**1.47**	**496.3**
Oxide	638	1.28	26.2
Transition	1,224	1.35	53.3
Fresh	8,661	1.50	416.8

A comparison of the resource model and mining model is provided in Table 16.6 and a comparison of grade-tonnage curves is shown in Figure 16.4.

Table 16.6 Relative change between resource model and mining model (Indicated Resources at 0.5 g/t Au cut-off grade)

Deposit/oxidation	Mass (kt)	Grade (g/t Au)	Contained gold (koz)
Selinsing	**1.8%**	**-3.4%**	**-1.8%**
Oxide	0.0%	-3.8%	-6.3%
Transition	4.0%	-6.3%	-3.7%
Fresh	1.8%	-3.4%	-1.8%
Buffalo Reef North	**4.5%**	**-9.1%**	**-4.7%**
Oxide	4.0%	-9.9%	-7.4%
Transition	6.4%	-8.7%	-3.2%
Fresh	1.9%	-7.1%	-6.1%
Buffalo Reef South	**7.0%**	**-7.6%**	**-1.2%**
Oxide	5.3%	-8.2%	-3.0%
Transition	5.2%	-7.5%	-2.1%
Fresh	7.7%	-7.9%	-0.8%
TOTAL	**4.1%**	**-5.2%**	**-1.6%**
Oxide	4.4%	-7.2%	-3.7%
Transition	5.2%	-7.5%	-2.4%
Fresh	3.9%	-5.1%	-1.4%

Figure 16.4 Grade-tonnage curves for resource block model and diluted mining block model (Indicated Resources)



No further mining recovery and dilution were applied for mine planning.

Parameters and modifying factors

The pit optimisations were based upon the assumption of bio-oxidation processing of fresh ore.

Modifying factors

Slope angles were recommended by Snowden as per Table 16.7.

Table 16.7 Slope angle assumptions

Deposit	Geotechnical domain	Bench height (m)	Minimum berm (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

Metallurgical recoveries were applied by rock type and gold grade. A summary is provided in Table 16.8. The relationships are the same for each deposit.

Table 16.8 Metallurgical recovery assumptions

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

Operating costs

A summary of operating costs is provided in Table 16.9.

Table 16.9 Operating cost assumptions

Item	Selinsing	Buffalo Reef North	Buffalo Reef South	Comments
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	

Sales

A gold price of $1,255/oz was applied.

Cut-off grade

Cut-off grades were calculated to be approximately 0.4 g/t Au for oxide and approximately 0.8 g/t Au for transition/fresh.

Results

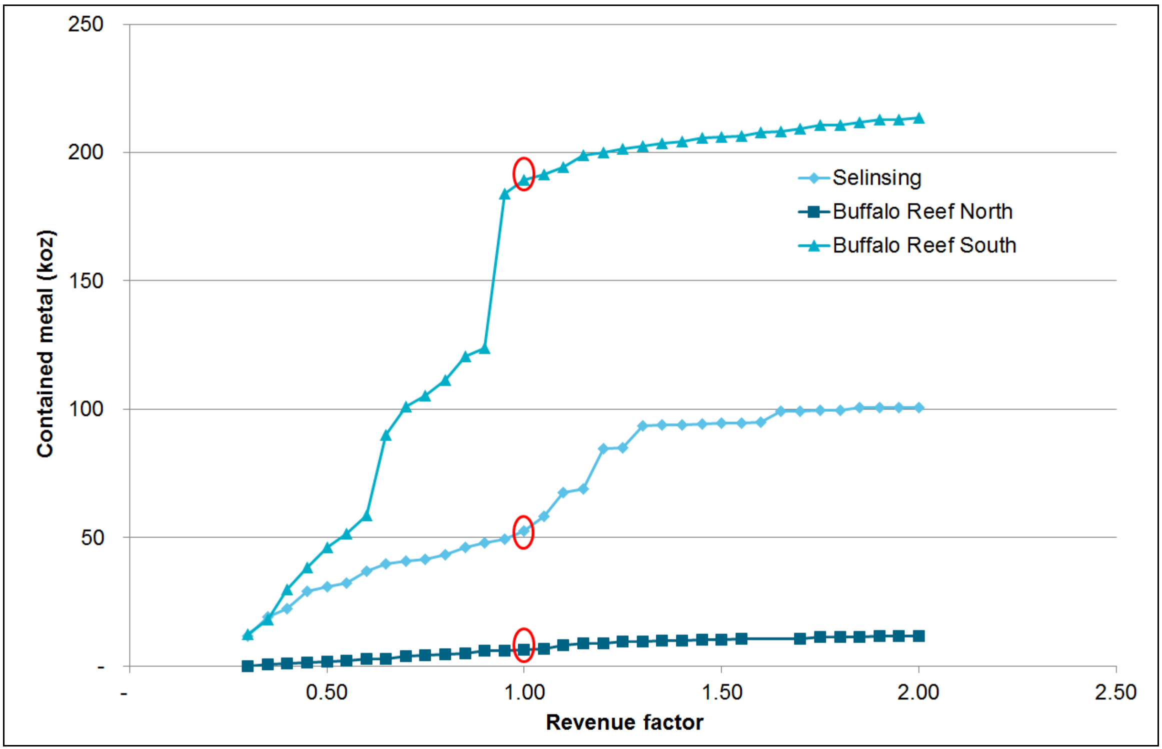
Pit optimisation was completed using Whittle 4X software. Optimisations were completed at a range of revenue factors. A summary of selected pit shells is shown in Table 16.10.

Table 16.10 Selected pit shells

Item	Selinsing	Buffalo Reef North	Buffalo Reef South	Total
Revenue factor	1.00	1.00	1.00	
Physicals				
Pit size (kt)	4,594	666	30,418	35,678
Waste (kt)	3,784	500	27,176	31,460
Strip ratio (w:o)	4.67	3.00	8.38	7.46
Ore – total (kt)	810	166	3,242	4,219
Au grade – total (g/t)	2.03	1.21	1.82	1.83
Contained gold – total (koz)	53	6	189	249
Recovered metal – total (koz)	46	5	159	210
Ore – oxide (kt)	54	90	468	612
Au grade – oxide (g/t)	0.77	1.11	1.34	1.26
Ore – sulphide (kt)	756	77	2,775	3,607
Au grade – sulphide (g/t)	2.12	1.32	1.90	1.93
Economics				
Revenue ($ M)	57.3	6.5	199.7	263.5
Selling cost ($ M)	3.0	0.5	14.5	17.9
Oxide processing cost ($ M)	0.5	0.9	4.5	6.0
Sulphide processing cost ($ M)	18.6	1.9	68.1	88.6
Mining cost ($ M)	8.8	1.1	50.9	60.9
Unit cost ($/oz)	675	849	868	825
Operating cash flow ($ M)	26	2	62	90

Based on the Indicated Resources, there is only minor upside associated with price (approximately 70 koz in total), mainly in Selinsing (Figure 16.5).

Figure 16.5 Contained metal sensitivity analysis to gold price



Additionally, Snowden considered the inclusion of Inferred Resources to determine potential future drilling opportunities. A comparison of pit optimisation results between Indicated Resources and Indicated and Inferred Resources is shown in Table 16.11. This shows the potential for an increase in contained metal of 50% and increase in cash flow of 51%. An increase in strip ratio of 22% is offset by an 8% increase in mined grade. Most of the potential increase is at Selinsing and Buffalo Reef Central and South (Figure 16.6 and Figure 16.7). At Selinsing, the opportunity is drilling to the north of the pit. At Buffalo South, there are some areas within the Indicated pit that could be infilled to improve strip ratio. Additionally, there is a zone between Buffalo Reef South and Central that provides some opportunity. Snowden recommends further drilling to improve the confidence in these areas.

Table 16.11 Comparison of pit optimisations including Inferred Resources

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)	1.83	1.98	8%
Contained gold – total (koz)	249	374	50%
Recovered metal – total (koz)	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 ($/oz)	825	827	0%
Operating cash flow ($ M)	90.0	135.5	51%

Figure 16.6 Selinsing pit shell comparison[1]

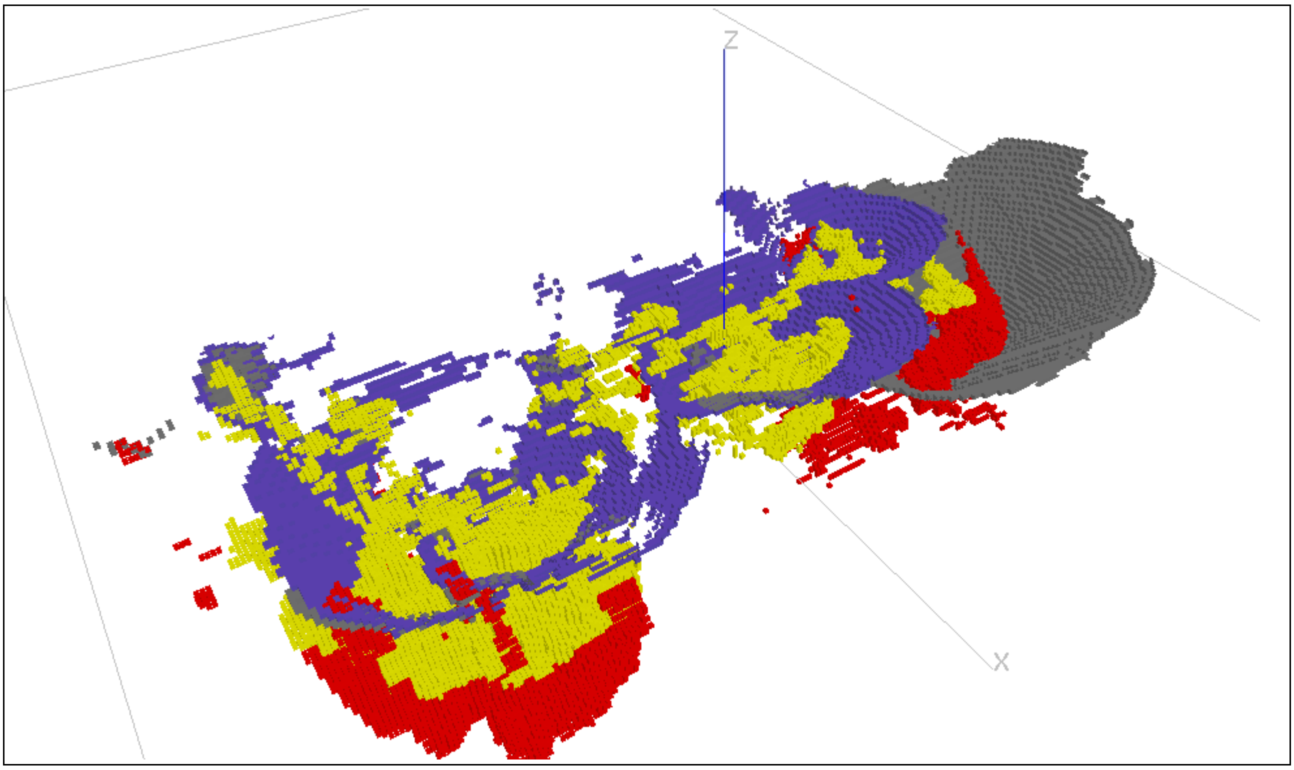
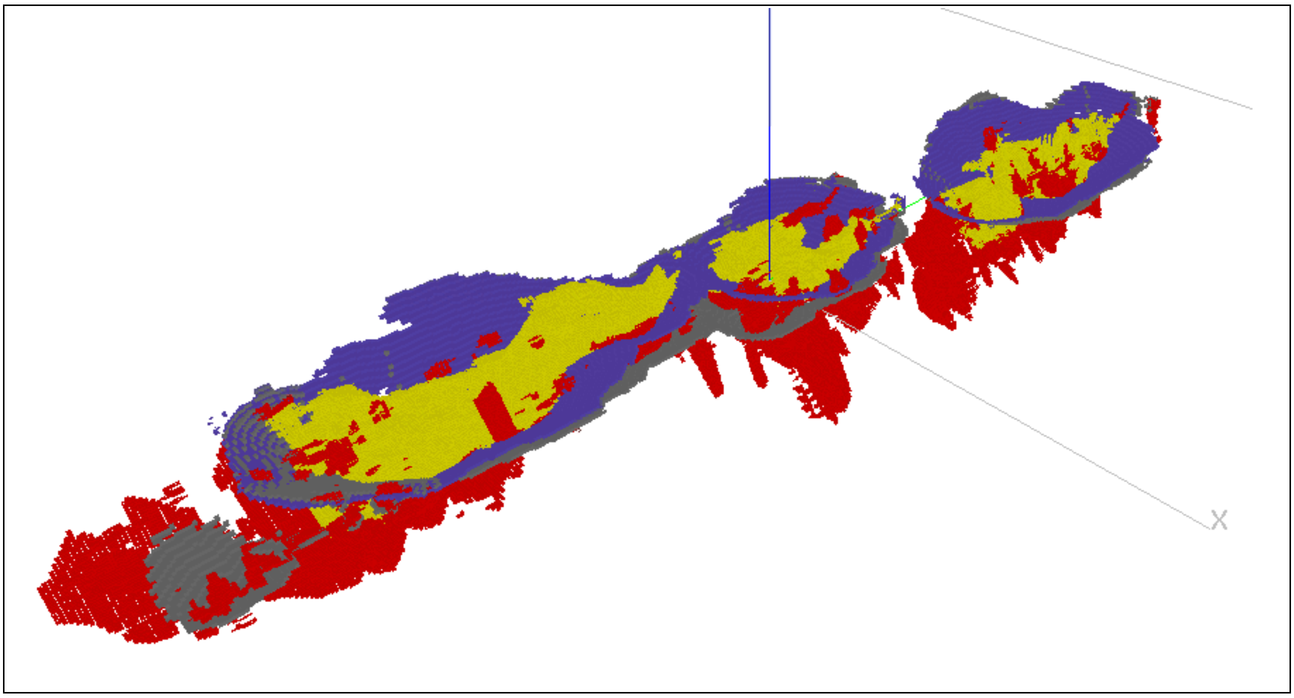


Figure 16.7 Buffalo Reef South, Central and Felda and North pit shell comparison



[1] Yellow blocks – Indicated Resource, Red blocks – Inferred Resource, Purple surface –Indicated Resource pit shell, Grey surface – Indicated and Inferred pit shell

16.3.3 Pit design

Pit design parameters are provided in Table 16.7.

Selinsing is an existing pit. The pit optimisation identified two areas for potential mining. One is a deepening of the existing main southern pit (S1). This pit has recently experienced a wall failure in the oxide, and requires some mitigation prior to ongoing mining. The Snowden design for S1 removes a switchback on the eastern wall and includes a ramp with a switchback on the western wall reducing the overall slope angle of the pit, and minimises geotechnical risk.

Selinsing Stage S2 pit is a narrow cutback of the northern part of the existing pit. This cutback was designed primarily based on the Whittle optimisation to maximise cash flow for the pit.

Both pit designs are shown in Figure 16.8 and Figure 16.9. In all figures, current haulage routes are shown as red lines whilst proposed routes are shown as blue lines and base of pit and ramp exit elevations annotated.

Figure 16.8 Selinsing pit designs

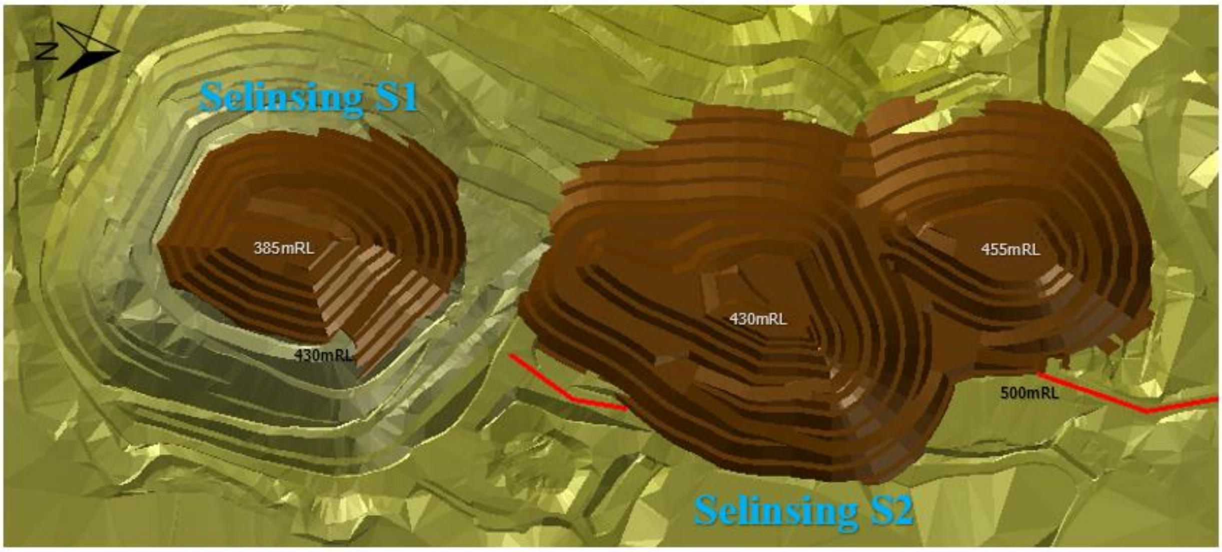
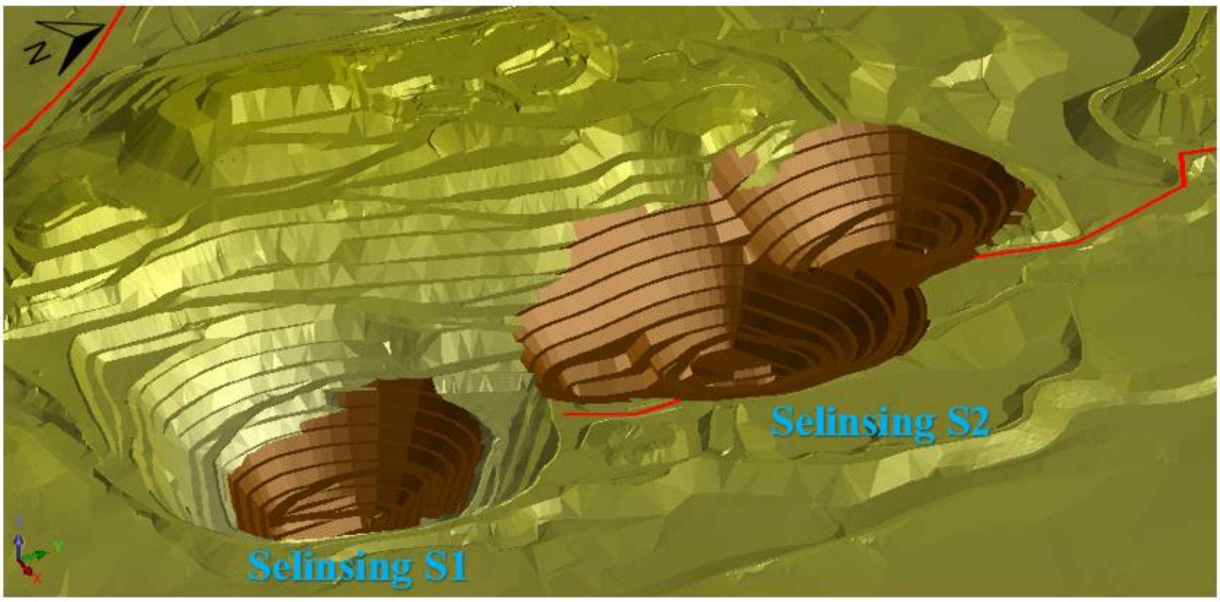


Figure 16.9 Selinsing pit designs asymmetric view



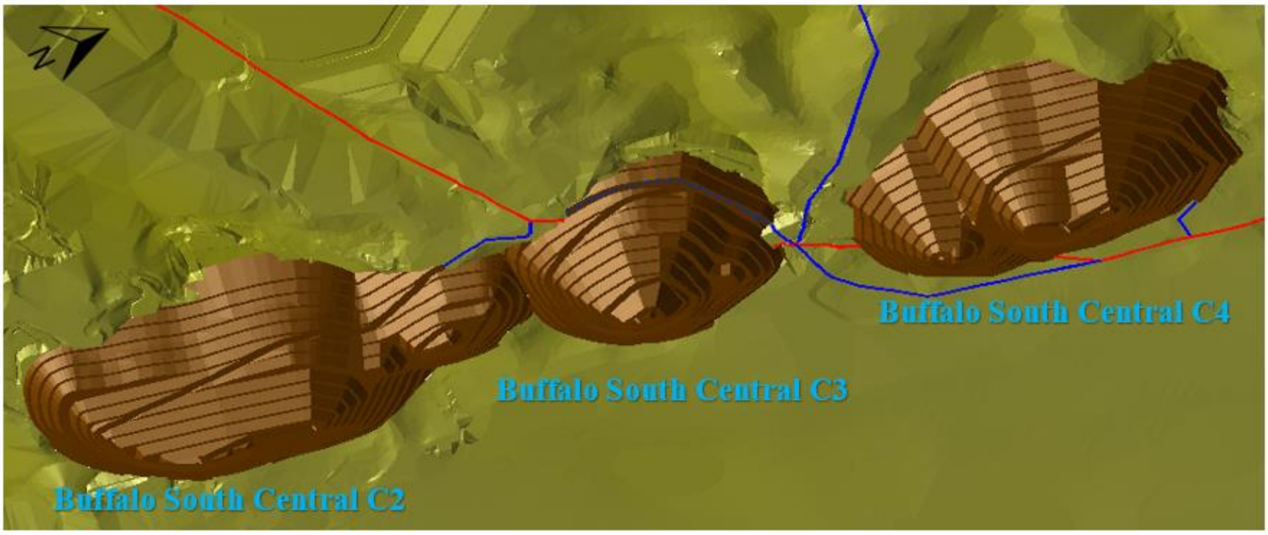
Buffalo Reef South/Central has a number of small existing oxide pits. The pit optimisations identified three distinct pits for design. These are named C2, C3, C4, from south to north. A small initial stage (C1) is a continuation of the existing small pit within C2. Each of the larger pits follow the pit optimisation and aim to provide exits at positions complementary to the existing haul road network (Figure 16.10 and Figure 16.11).

The C3 pit cuts off the existing haul road to the ROM. As such, the pit design contains a diversion around the pit wall at the 495 mRL level. There may be a small amount of fill required to access the diversion, which can be included in short term planning.

The C4 pit cuts off a haul road on the eastern side, but this can be easily diverted around the pit.

Figure 16.10 Buffalo Reef South, Central and Felda and North pit designs

Figure 16.11 Buffalo Reef South, Central and Felda asymmetric view



Buffalo Reef North forms two small pits which expand on existing pit voids (Figure 16.12 and Figure 16.13). The design creates access from the south to the north of the pit to minimise haulage distances for ore and waste.

Figure 16.12 Buffalo Reef North pit designs

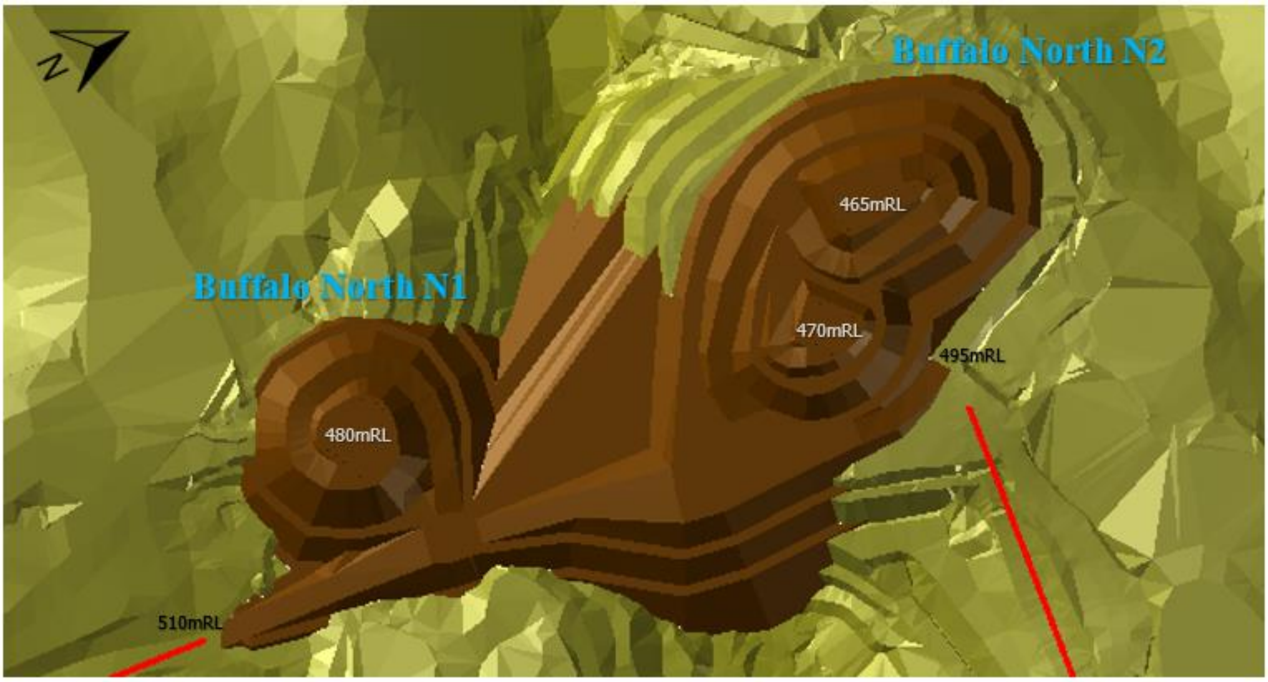


Figure 16.13 Buffalo Reef North pit designs asymmetric view

A comparison of pit optimisation and pit design volumes and indicative economics is provided in Table 16.12.

Table 16.12 Pit design reconciliations to optimisation results

Item	Optimisation	Design	Difference
Selinsing			
Total movement (kt)	4,594	4,019	(13%)
Ore (kt)	810	608	(25%)
Cash flow ($ M)	26.5	21.6	(18%)
Buffalo Reef South/Central			
Total movement (kt)	30,418	31,932	+5%
Ore (kt)	3,242	3,201	(1%)
Cash flow ($ M)	61.7	57.3	(7%)
Buffalo Reef North			
Total movement (kt)	666	903	+36%
Ore (kt)	166	190	+14%
Cash flow ($ M)	2.1	1.8	(14%)

Snowden considers these variances to be reasonable for the level of study. The differences can be explained as:

- Selinsing – The Selinsing pit optimisation takes thin slices on the existing wall that are not practical to design around (particularly on the southern wall). Additionally, the pit pushes against the mining lease boundary, restricting access to some ore deep in the pit to the east.
- Buffalo Reef South/Felda/Central – These differences are within the expected tolerance.
- Buffalo Reef North – These pits are very small and hence additional waste is expected. The designed pit is still viable.

16.3.4 Mining inventory

The total mining inventory includes all the pit designs, as well as existing stockpiles, as at 1 July 2016 (Table 16.13).

Table 16.13 Total mining inventory for scheduling (all Indicated Resources)

Item	Selinsing	Buffalo Reef South/Central	Buffalo Reef North	Stockpiles	Total
Oxide ore (kt)	8	453	105	2,206	2,773
Au (g/t)	1.20	1.37	1.05	0.52	0.68
Trans ore (kt)	26	623	72	0	721
Au (g/t)	1.98	1.82	1.28	3.58	1.77
Fresh ore (kt)	573	2,125	14	163	2,875
Au (g/t)	2.27	1.91	1.25	1.46	1.96
Total ore (kt)	608	3,201	190	2,369	6,368
Au (g/t)	2.24	1.82	1.15	0.58	1.38
Waste (kt)	3,377	28,252	675	-	32,304
Strip ratio (w:o)	5.6	8.8	3.6		5.1

The majority of the oxide feed to supply the plant until the bio-oxidation circuit is installed to treat sulphides, is from the existing low grade stockpiles. Buffalo Reef provides some higher grade feed (approximately six months' supply). Selinsing contains very little oxide resource.

There is approximately three and a half years of transitional/fresh ore supply to the bio-oxidation plant. The highest grades come from the Selinsing pits.

16.3.5 Site layout

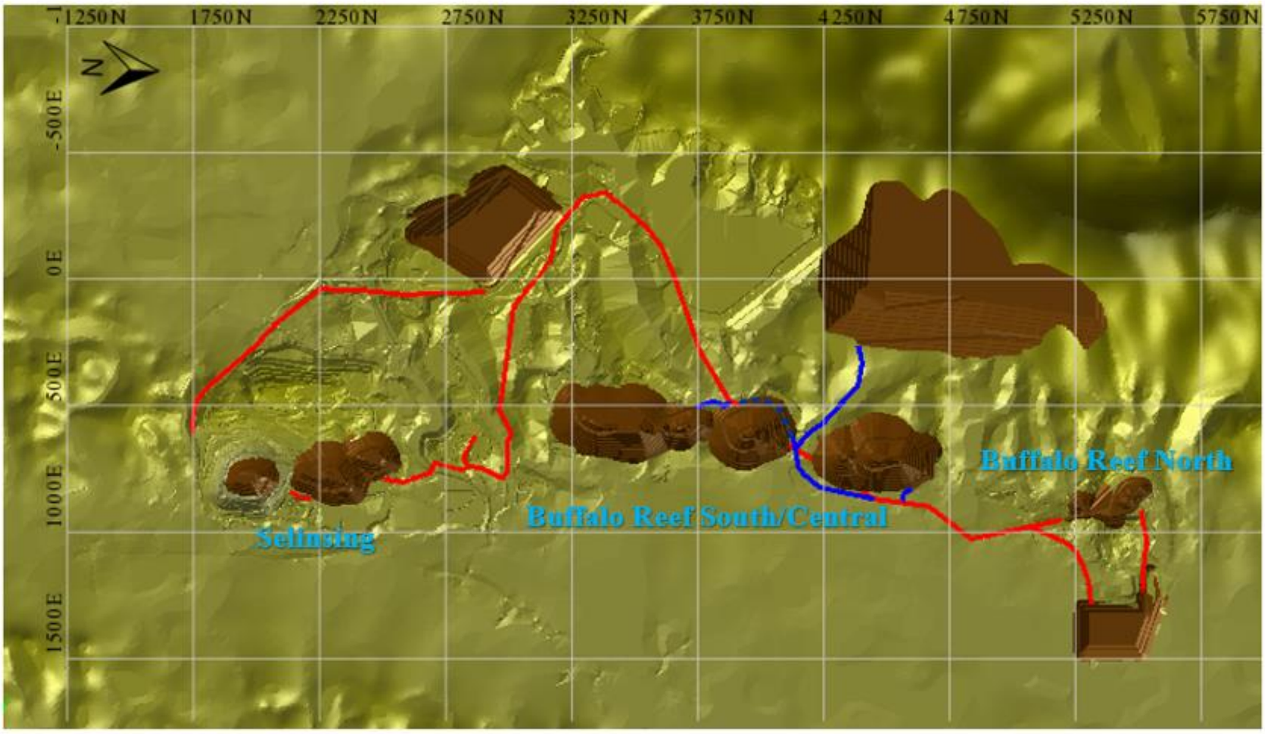
Waste dump designs based on guidelines provided by Monument; these are shown in Table 16.14.

Table 16.14 Waste dump parameters

Parameter	Value
Dump lift (m)	10
Berm width (m)	5
Face slope (degrees)	33.7
Ramp gradient (%)	10
Ramp width (m)	12

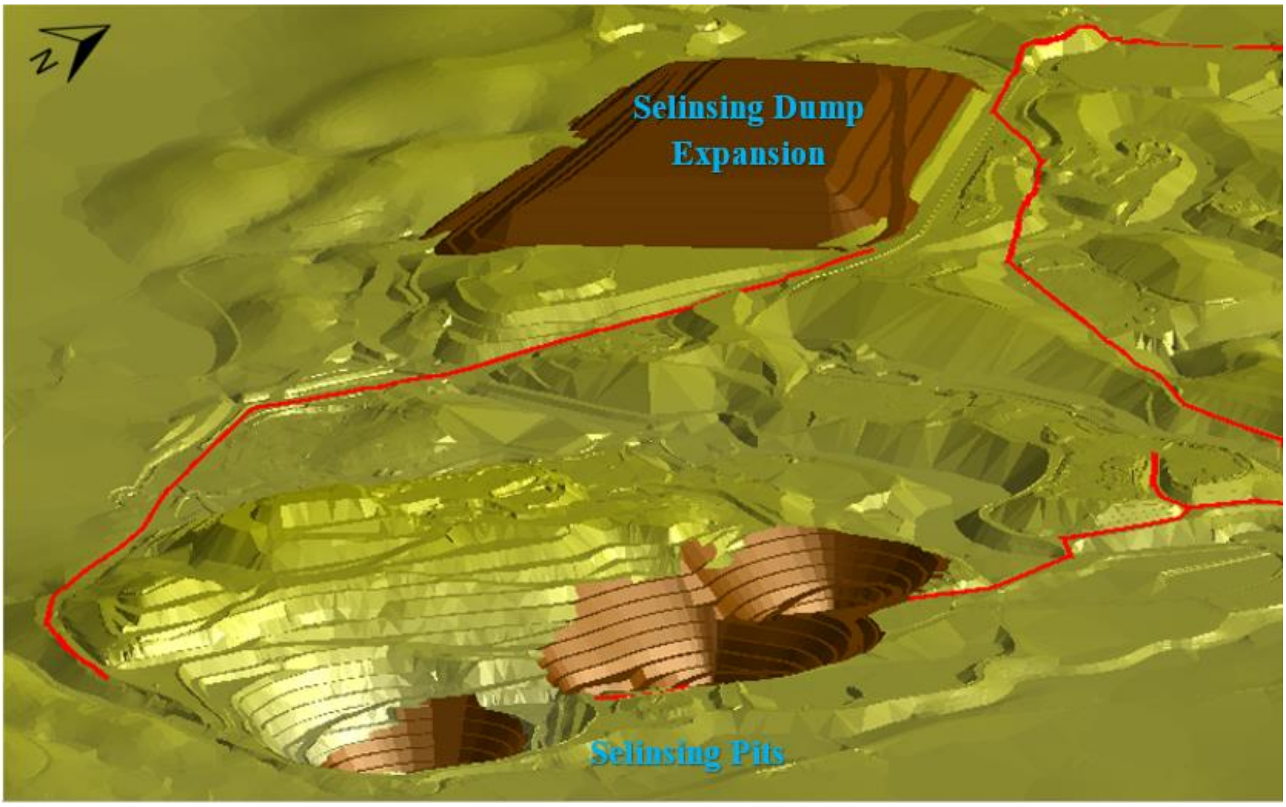
The overall mine layout, with haulage routes and waste dumps, is shown in Figure 16.14.

Figure 16.14 Overall mining layout



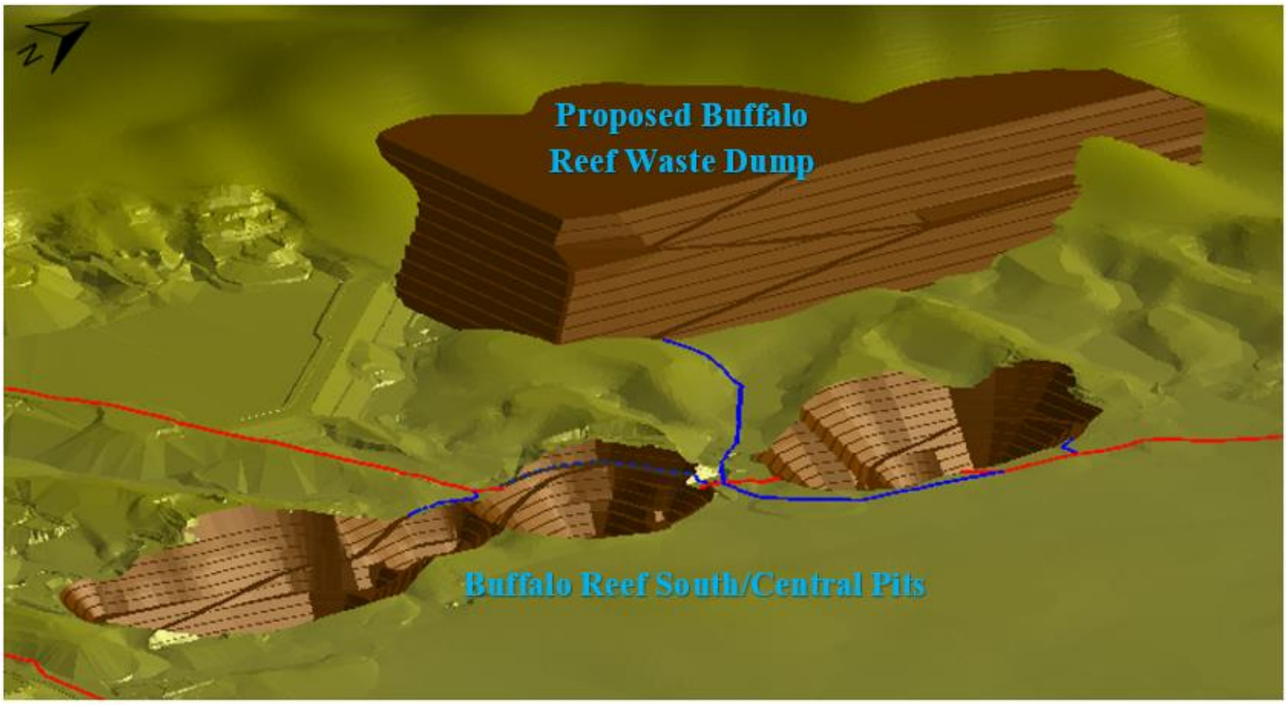
The Selinsing waste dump is located approximately 750 m west of the pit (Figure 16.15). Snowden designed an expansion of the dump to store the additional waste to a maximum elevation of 570 mRL. The total capacity of this dump is 4.6 Mlcm. The remaining Selinsing waste requirement is approximately 1.5 Mlcm. Hence, this dump can also provide storage to the Buffalo Reef South/Central pits if required.

Figure 16.15 Selinsing mine layout



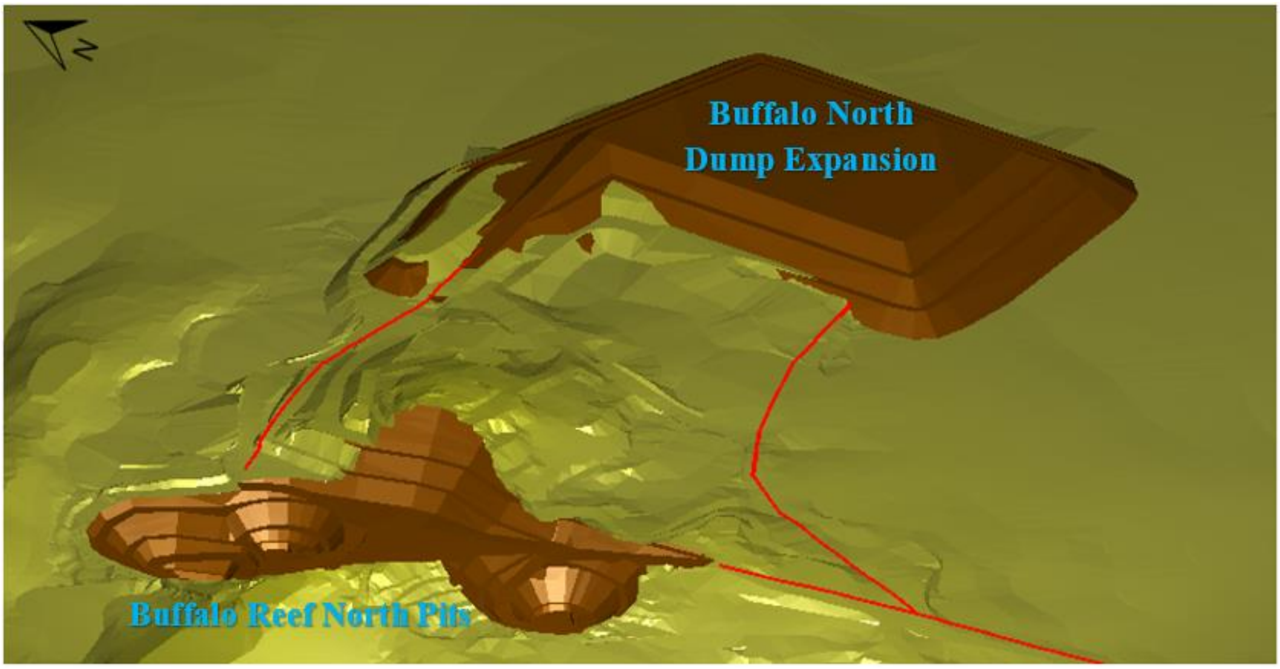
Buffalo Reef South/Central contains the majority of the waste remaining for the project. The Selinsing waste dump is insufficient to store all the Buffalo Reef waste. A new dump waste was designed to the northwest of these pits (by approximately 500 m to 750 m) to store additional waste (Figure 16.16). The lack of area for the dump means it can become relatively high if fully filled. The total capacity to 570 mRL is 6.8 Mlcm, and to 610 mRL is 15.6 Mlcm. The total Buffalo Reef South/Central waste requirement is approximately 14.6 Mlcm, meaning there is a small amount of spare capacity between the Selinsing and Buffalo Reef Central waste dumps for minor pit expansions.

Figure 16.16 Buffalo Reef South/Felda/Central mine layout



The Buffalo North pits provide a small amount of waste (0.4 Mlcm) which can be easily accommodated by the existing waste dump designs to the east of these pits (Figure 16.17).

Figure 16.17 Buffalo North mine layout



16.3.6 Mine Waste Management Plan and final landform dumps

The purpose of the Mine Waste Management Plan (MWMP) is to provide a framework for the management of mine waste rock at Buffalo Reef operation. This management plan outlines how environmental issues associated with the potentially acid forming sulphide waste rock identification, storage and handling of this waste rock at Selinsing and Buffalo Reef operation will be managed for the duration of mining operations and through to decommissioning and closure.

Waste rock management objectives

Waste rock management is an integral component of successful rehabilitation and closure at the Selinsing and Buffalo Reef operation. The design and construction of waste rock stockpiles must consider potential environmental impacts that may threaten the success of future rehabilitation. The following key issues directly related to mine waste:

- There is potential for erosion during torrential rainfall; this is particularly the case for skeletal soils on steep terrain and in areas disturbed by previous mining operations
- Geochemical characterisation studies have identified PAF material in samples collected from the pits; when exposed, this material oxidises and acidifies rapidly, contaminating water that comes into contact with it
- Waste rock stockpiles may not fit in with the surrounding natural landscape, and can erode, particularly when rainfall run-off is concentrated and/or erodible fine-grained materials are applied to the surface.

The broad objective of mine closure to prevent or minimise adverse long-term environmental impacts, and to create a self-sustaining natural ecosystem or alternate land use based on an agreed set of objectives.

Thus, the general closure objectives of the operation are based on successful rehabilitation strategies that ensure the post-mining condition of the landscape is:

- Safe, stable and minimises long-term environmental impact
- Without any future liability to the stakeholders
- In conformance with the agreed post-mining land use.

Waste rock dump design criteria

The criteria used for waste rock stockpile design is governed by the Minerals and Geoscience Department Malaysia. The following management strategies are generic and are based on Industry best practice and have been derived from a number of active mine sites throughout Australia. Some of the strategies may not be directly applicable but the principles can be adapted to the Selinsing and Buffalo Reef operation, subject to topographic and hydrologic constraints, and material availability. Key design criteria are summarised in the following sections.

Waste dump design

The waste dump design is based on the waste material types generated through mining. Both PAF and NAF waste have been identified from the testwork undertaken and modelling to generate a 3D block model that was in turn used to generate the mining waste schedule for the expected life of mine. The waste dump design was based on the mining waste schedule and involves selective mining and placement of PAF and NAF waste within the waste rock precinct to form a waste rock dump that encapsulates the PAF waste as detailed in Section 16.2.2.

Dump height

The waste rock stockpile should be placed away from ridges and its height should not exceed 10 m above existing ridges so it does not cause a visual impact.

Dump profile

Key objectives are to ensure geotechnical, and erosional stability (both aided by the placement of coarse-textured waste rock on the outer batter slope, and the placement of waste rock at its angle of repose: 1:1.5 or 34º), while facilitating revegetation, rather than to set an unrealistic final stockpile outer slope.

This would involve leaving slopes for low stockpile heights at the angle of repose. For higher stockpiles, the preferred design retains upper slopes at the angle of repose, retaining intermediate benches, and pushing out the lower bench only, to create a nominal lower batter slope of half the angle of repose or 18° (1 in 3). This will mimic a concave profile consistent with surrounding natural landforms. Figure 16.18 shows schematically the construction of the final concave slope for waste rock stockpiles. Figure 16.19 shows the preferred dump design with the encapsulated PAF cell and concaved final batter slope.

Figure 16.18 Simply constructed concave slopes to mimic natural slopes

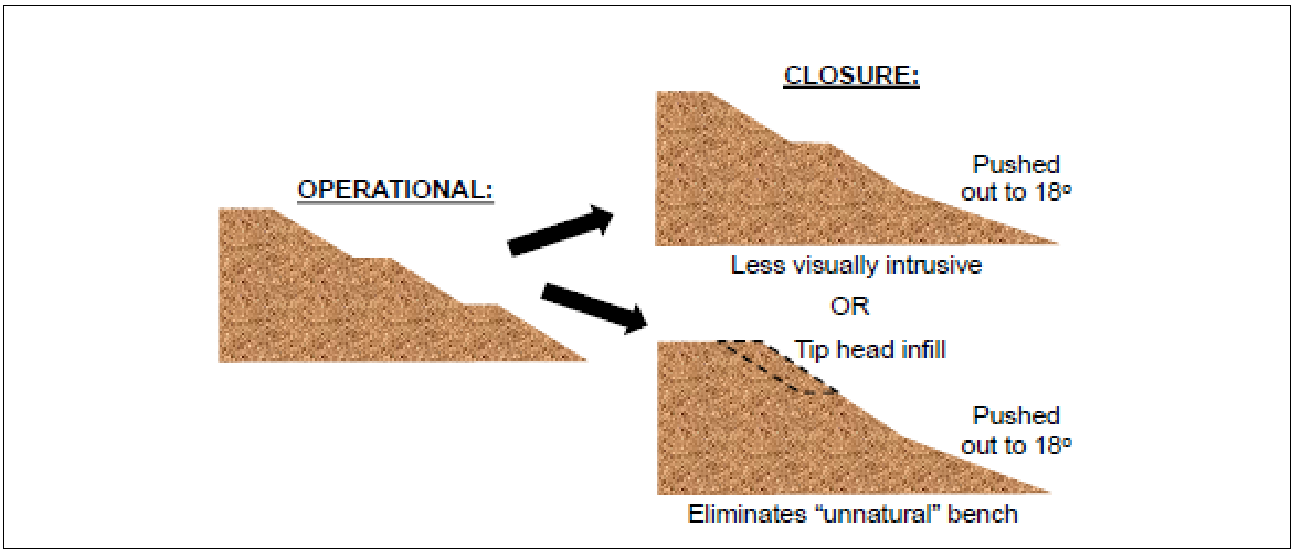
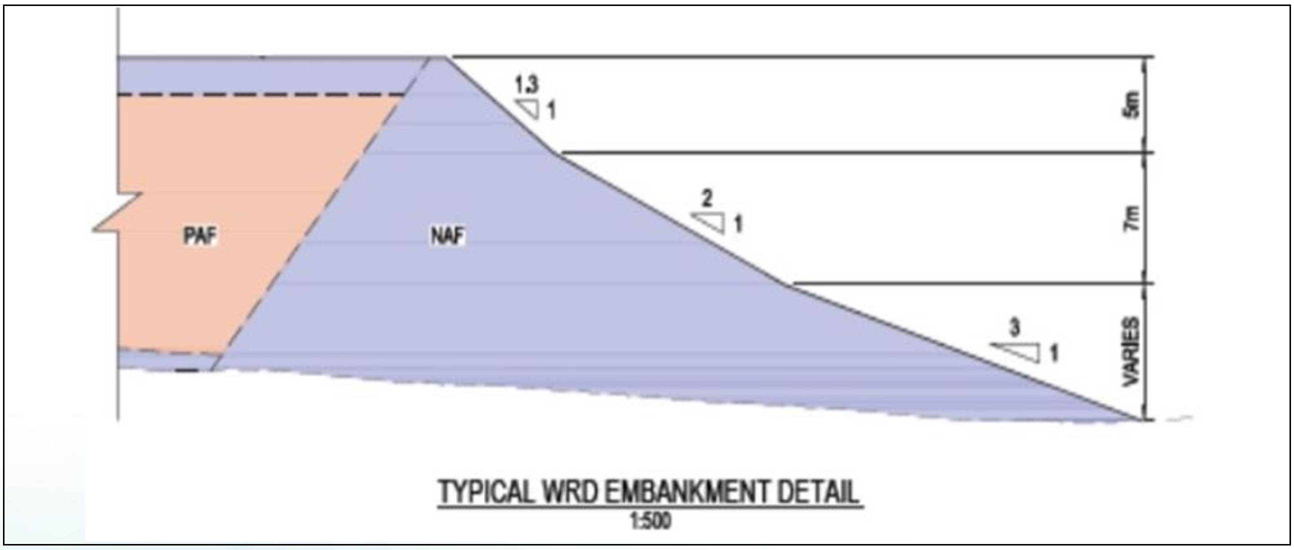


Figure 16.19 Conceptual waste rock dump design showing encapsulation of PAF waste



Management of PAF waste rock

The key objective is to identify PAF waste rock so as to develop a suitable management plan in order to prevent or minimise the potential environmental impacts associated with the formation of acid rock drainage and other geochemical hazards. Management options for PAF waste rock involves selective placement of the PAF waste rock within a PAF encapsulation cell of NAF waste rock in surface waste rock dump, deep and centrally (not under side slopes or benches), with NAF waste rock below to carry any base flow-through and a low net percolation top cover to limit oxygen ingress and water transport.

Management of rainfall run-off and water quality

Management of rainfall run-off and water quality will require the following activities to be undertaken:

- Divert clean water around stockpiles and pits
- Divert mine water into completed pits
- Avoid end-dumping waste rock down valley slopes, which:
  - interrupts stream flows and may contaminate streams
  - results in long and wide waste rock slopes that are difficult to rehabilitate.
- Infill waste rock across valley to:
  - ultimately store or shed run-off
  - limit waste rock slopes, enabling attention to be directed at stabilising the end slope to blend back into the valley.

Waste rock dump slopes

- Concave slope profiles in nature are most stable.
- Slopes of NAF stockpiles should not shed run-off from the top, to limit erosion (up to 100-fold magnification of run-off can occur due to overtopping and concentration of run-off in drains).
- PAF waste rock should not be placed within the waste rock dump slopes or placed on outer surfaces of tailings embankments.
- Angle of repose stockpile slopes present the smallest catchment and roughest surface texture, and hence minimise erosion.
- Slope flattening increases the catchment for a given height and reduces surface texture, increasing run-off and erosion, and is detrimental for all but short slopes.
- While contour ripping of flattened slopes aids water capture, they overtop or pipe, allowing gullying, and hence should only be used on a temporary basis.

Waste rock dump tops

The tops of the NAF protective layer can be paddock-dumped and allowed to pond rainfall, from which it will infiltrate and evapotranspire; or infiltrate rainfall and discharge through the drainage layer above the low permeability layer.

Stockpiling materials for rehabilitation

Identification and stockpiling of materials for rehabilitation purposes is subject to the following rock management activities to be implemented.

NAF waste rock

Each waste rock stockpile design is unique due to differing land access regulations and the regional topography. Waste rock stockpile designs are typically based on two to three lifts of varying heights and sizes. The design is based on allocating the total volume of waste rock within the available area. The following sections describe the basic guiding principles used in the development of each waste rock stockpile design.

Base lift

The base lift is generally comprised of NAF material. It may be constructed in a series of lifts by end-dumping, which does not include benches on the outer batter slopes. The foundation of the base lift is likely to be geotechnically stable. The base lift is compacted during construction by haul truck trafficking.

Encapsulating wall

The purpose for an encapsulating wall is to contain any placed PAF material. A reconsideration of volumes of PAF and NAF material generated from the mining the ore has identified an alternative disposal strategy for generated PAF waste rock. All PAF waste – waste rock with an average sulphur concentration >0.2% will be disposed within the TSF. A separate encapsulation wall will not be required for the Selinsing and Buffalo Reef operation waste rock dump.

PAF waste rock

PAF waste rock will be selectively placed within the waste rock dump and encapsulated with NAF waste. The PAF will be dumped in layers and each layer will be compacted using dump truck compaction to limit net percolation of incidental rainfall. Prior to each wet season a temporary cover of NAF waste will be placed over the PAF cell to further protect the PAF waste from oxidation.

Final landform

The final waste rock stockpile and cover design seeks to maximise the volume available for waste rock storage for any given height of stockpile, minimise rehandling and earthworks requirements, and minimise the amount of rocky soil mulch required on the surface of the final mined landform.

Flattening the side slopes of the waste rock stockpile for rehabilitation purposes would increase the slope footprint increasing the slope catchment, reduce the surface roughness and increase run-off coefficient, and hence increase the erosion potential. The addition of fine-grained topsoil to facilitate vegetation of the slope exacerbates erosion in a dry or seasonally dry climate due to the inadequacy of the vegetation cover to limit erosion.

Slope flattening is often intended to address concern over the geotechnical stability of angle of repose slopes. However, the friction angle of reasonably durable waste rock under unsaturated conditions is several degrees higher than the angle of repose of the material (this representing the loosest possible state), providing a factor of safety against geotechnical slope instability of about 1.3. Any slope instability would be surficial given the frictional nature of waste rock. The final surface will be seeded with a mix of native species.

The angle of repose batter slopes will be over-dumped by topsoil or other appropriate growth medium. The placed topsoil will progressively wash into the coarse-grained slope and down the slope with successive rainfall. The angle of repose slopes will require seeding and possibly fertilising. Some of these rehabilitation activities have been in practice since commercial production in 2010.

Drainage management

Drainage off the waste rock stockpiles is an important design consideration. While rainfall infiltration into the dump cannot be completely eliminated, it is minimised by the following design principles. The drainage layer above the sealing layer of a store and release cover is a key feature in managing drainage.

<u>Intermediate covers for the waste rock</u>

On a regular basis, but particularly prior to each rainy season, the top surface is sloped at a nominal grade of 1:100, subject to site configuration and geometry constraints, to rapidly shed rainfall run-off.

Rainfall run-off from the top surface of the dump drains towards the dump wall where it will be directed into the base lift, and thence towards silt traps. This minimises the amount of water ponding within the waste rock dump.

<u>Silt traps</u>

Silt traps and stormwater ponds are used to settle suspended particles from the water column. They also serve to act as a control mechanism to reduce downstream erosion by reducing flow velocity.

Closure plan

At closure, the waste dump design will approximate the post closure configuration to facilitate re-contouring at closure. Sufficient bench setbacks will be provided during construction for regrading activities.

Prior to closure, the waste dumps will be evaluated to confirm mass stability including stabilising slopes, ensuring limiting infiltration, re-contouring on rounding edges to minimise visual impacts, revegetation, and erosion control. Reclamation of the waste dumps should be conducted concurrently with regular mine operations whenever possible. As sections of the waste dump reach the ultimate configuration and become inactive, the slopes should will be re-graded in order to minimise surface water run-off velocities and associated erosion as well as to provide variable topography. Upon completion re-grading of a portion of the waste dump, growth media (topsoil and grass) will be efficiently placed over the re-contoured surface to avoid top soil erosion which will contribute to slope instability. Final design will be determined as part of the final permanent closure plan.

Closure aspects associated with the waste dumps are:

- Neutralisation of sulphide generated acid run off. Limestone for acid neutralisation is available on site. Each mine site has a unique geological setting, although similar in occurrences. Due to the presence of limestone at the mine site, it is feasible to be utilised as part of the acid mine drainage (AMD) mitigation components to reduce and control contaminations of metals at the mine.
- Slope stabilisation and re-contouring. All final slopes and barren land will be rehabilitated and revegetated. Some of this is achievable at the current stage as certain areas will not be disturbed once mining operations commence. Certain slopes can be re-vegetated to further enhance stability and reduced congestion of scope of work during closure.
- Top soil cover and revegetation. Areas with abundant topsoil coverage will be exploited to obtain topsoil for the revegetation activity leading up to, and at mine closure.

16.3.7 Costs for mine waste management plan and final landform dumps

The costs for rehabilitation management and final landform dumps are provided in detail in Section 21.

16.4 Production schedule

Snowden completed a quarterly mining schedule in its Evaluator optimisation software. Evaluator is based on a Mixed Integer Programming formulation which seeks to maximise NPV for a given inventory, economics and set of constraints.

16.4.1 Parameters

The following constraints were applied for mine scheduling:

- Mining:
  - Maximum of six 5 m benches per year of vertical advance
  - 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
  - No in situ mining in quarter 1 (only stockpile depletion).
- Processing:
  - 1 Mt/a throughput for oxide circuit
  - 950 kt/a throughput for sulphide circuit
  - 75% ramp-up of throughput in the first quarter of sulphide production (Q7)
  - Maximum grade of 2 g/t Au in the ramp-up period
  - No simultaneous processing of oxide and sulphides.

16.4.2 Mining schedule

The mining schedule is shown in Figure 16.20 and Figure 16.21. The first year of mining, prior to the sulphide circuit commencing, mines only a small amount of in situ material, sourced mostly from Buffalo Reef surface deposits. The total mining rate in this period is the equivalent of 5 Mt/a. Just prior to the sulphide circuit commencing the mining rate ramps up to 9 Mt/a to 10 Mt/a, with the initial focus of mining 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.

Figure 16.20 Total movement schedule by pit stage

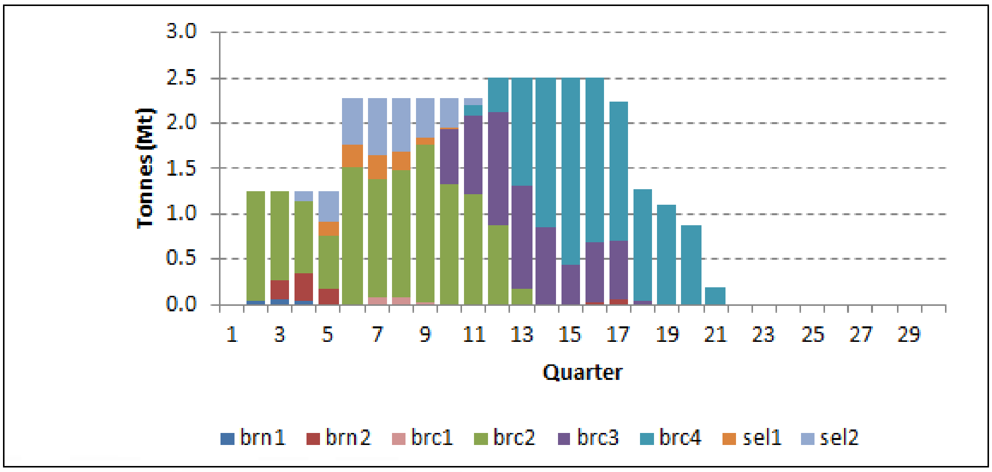
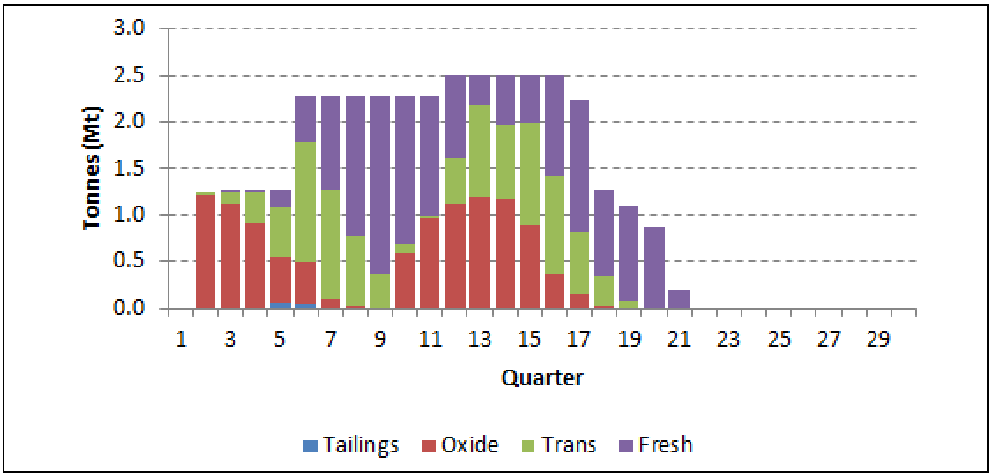
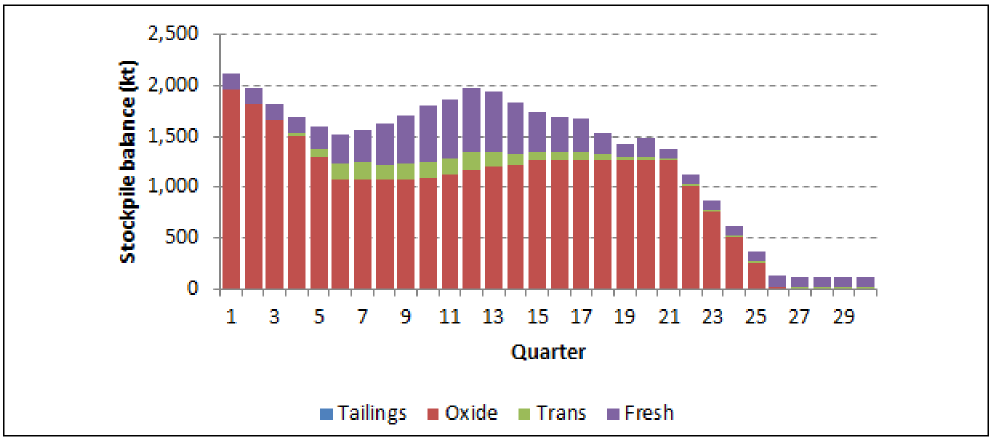


Figure 16.21 Total movement schedule by rock type



The initial stockpile as at 30 June 2016 was approximately 2.4 Mt. The higher-grade oxide stockpiles are depleted in the first 1.5 years of processing, making up for the shortfall of in situ oxide resources (Figure 16.22). During this time, transition and fresh ore mined is stockpiled ahead of the sulphide plant commencement. In order to target high grade, lower grade fresh ore is stockpiled and depleted towards the end of sulphide production. After mining is completed in Q21 there is approximately 1.5 years of stockpile depletion. The total stockpile size never exceeds the initial levels, meaning there is no additional footprint required to store this material.

Figure 16.22 Stockpile balance schedule



16.4.3 Processing schedule

The process feed schedule is shown in Figure 16.23 (tonnes) and Figure 16.24 (grade). The first 1.5 years of production mines lower grade oxide resources from in situ and existing stockpiles. The sulphide circuit is commissioned and ramped up in Q7, leading to a dip in production. 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 (>2.5 g/t Au) is achieved for about five 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 on the remaining stockpiles.

Figure 16.23 Process feed schedule

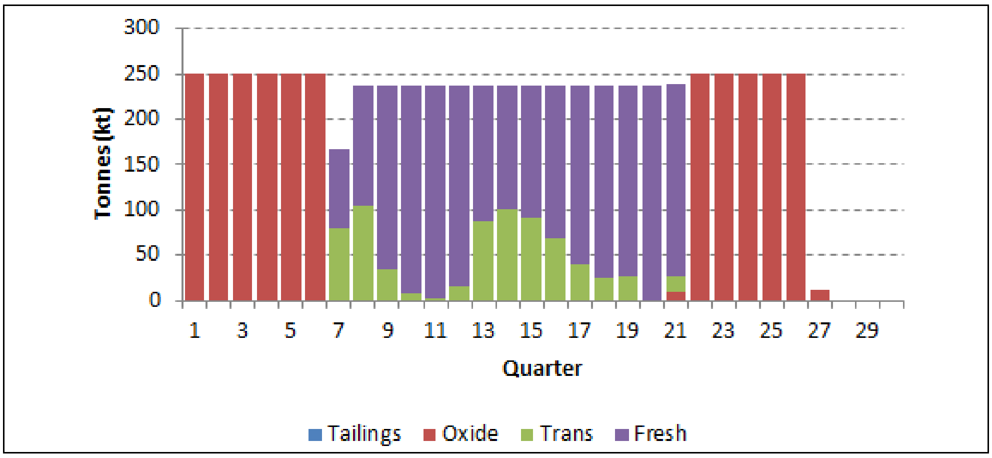
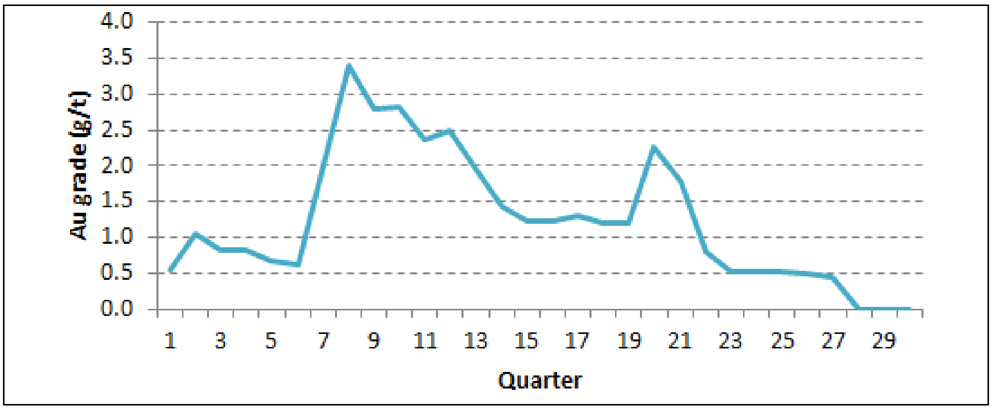


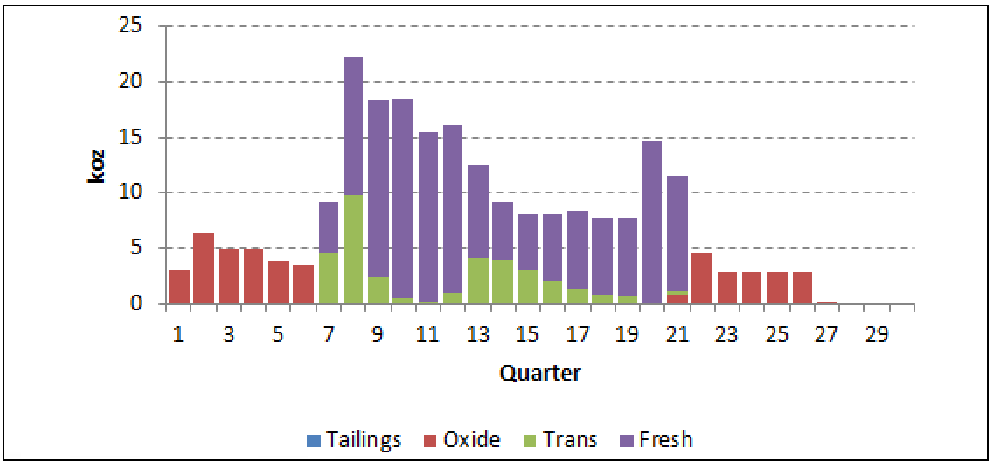
Figure 16.24 Process grade schedule



16.4.4 Production schedule

The gold production schedule is quite variable. During initial oxide processing the production rate is approximately 15 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 10 koz/a.

Figure 16.25 Recovered gold schedule



16.5 Mine requirements

16.5.1 Equipment requirements

Summary

All mining and associated activities are executed by a contractor, Minetech, and compensated by units of work completed.

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.

16.5.2 Hauling

Based on LOM ore release:

- Selinsing and Buffalo Reef pits: From six to eighteen 80 t trucks.

Ancillary

There is a fleet of bulldozers, graders and three water trucks for dust suppression provided by the contractor.

16.5.3 Manning

Current manning levels are provided in the following categories.

Administration

There are 13 employees in administration which includes Site Manager, Site Engineer, Safety Officer, Site Supervisor (Mining) and Site Supervisor.

Maintenance

There are eight employees for Maintenance consisting of Chief Mechanic, Mechanic, Mechanic Helper, Diesel Clerk and Tyre Man.

Operators

There are 55 operators which consist of Trip Recorder, General Worker, Excavator Operator and Tamrock Operator.

16.5.4 Consumables

Fuel

Average monthly consumption for diesel usage is 131,974 litres based on July 2015 to September 2016 usage.

Power

Electrical power is derived from SGM TNB Power Supply.

Tyres

Average monthly usage of tyres is 20/month based on July 2015 to September 2016 usage.

Explosives

Average monthly usage of explosive is 34,677 kg/month based on July 2015 to September 2016 usage.

Samples

Average monthly grade control samples produced is 3,430 samples based on July 2015 to September 2016 usage.

17 RECOVERY METHODS

17.1 Existing Selinsing oxide ore process treatment plant

The results of the Selinsing Gold Mine's recent operating procedures and processing facilities, including production costs, are presented in this section. Details of the gold processing are described below with a flowsheet diagram portraying the processes of gold recovery.

The existing processing plant is designed to effectively treat oxidised ore. As material from the oxidised level of ore transitions with depth into sulphide ores, the gold recovery in the existing process circuit begins to decrease. This has been verified by previous metallurgical tests and gold recoveries reported in recent mill production.

The existing plant consists of conventional processing of the ore by means of crushing and grinding to approximately 80% passing 74 µm to the cyclone overflow. The ball mill operates in closed circuit with a hydro-cyclone. A split of the cyclone underflow is subjected to gravity recovery methods using a Knelson centrifugal concentrator. The Knelson concentrate is then subjected to intense cyanidation. The cyclone overflow is forwarded to a 36-hour CIL cyanidation leach process. CIL slurry is disposed into the TSF, from which effluent is recycled back to the process plant. Excess water from the TSF is detoxified with hydrogen peroxide (H_2O_2) and ferric sulphate ($Fe_2(SO_4)_3$) to destroy the cyanide and precipitate the arsenic prior to discharge the water to the environmental pond.

The loaded carbon from the CIL is stripped and forwarded with the precious metal pregnant solution from intense cyanidation to electrowinning station, from which the final precious metal is recovered by smelting to Dore.

The current unit operations at the Selinsing processing plant comprise the following circuits:

- 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.

Table 17.1 summarises the operating parameters for the processing plant based on Monument plans for fiscal year 2013 starting 1 July 2012. A simplified flowsheet of the mineral processing operation is shown in Figure 17.1. The capital cost for Phases I and II were in the region of US$18.2 million.

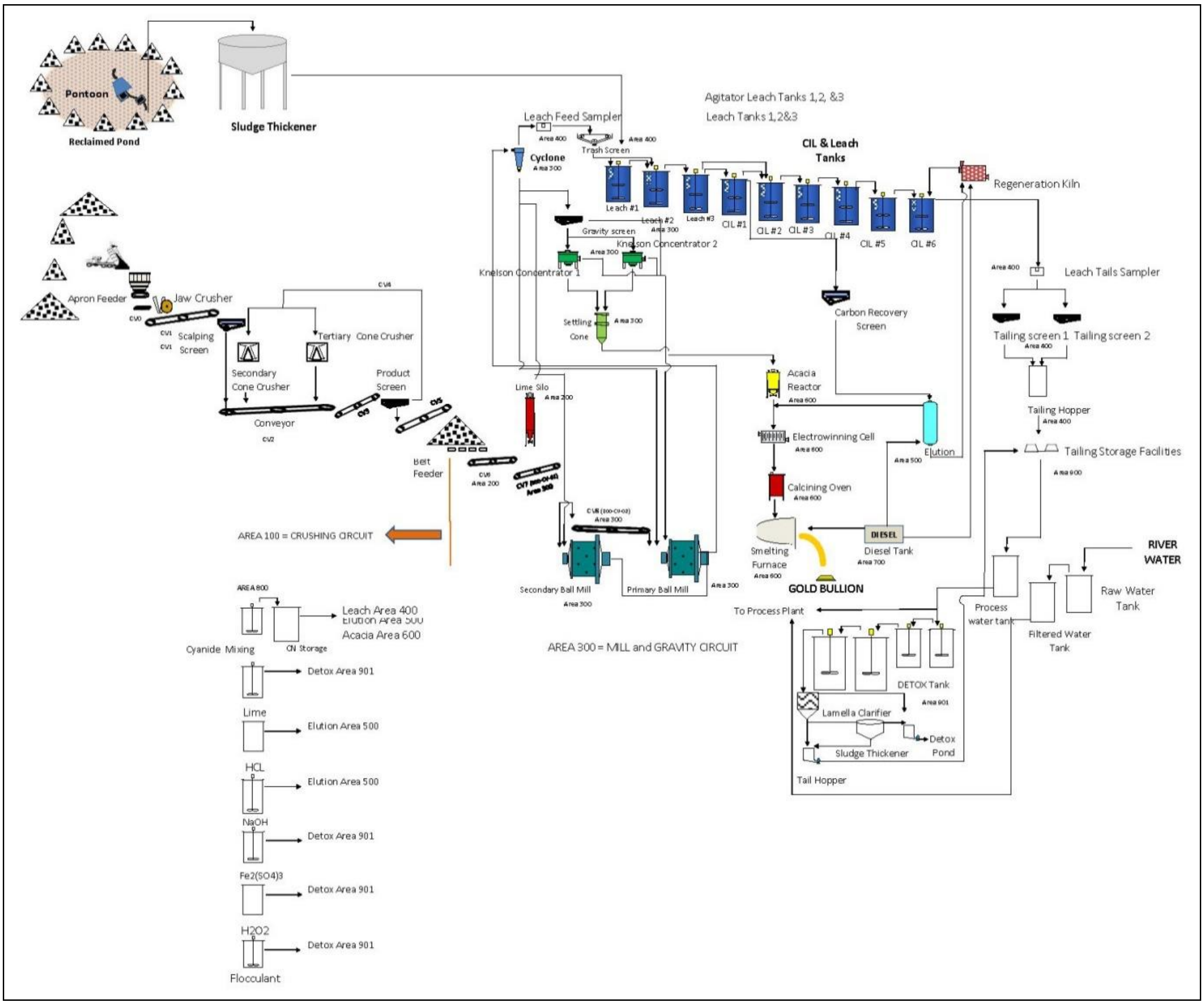
Table 17.1 Planned operating parameters for the Selinsing process plant (2013 fiscal year)

Operating parameter	Units	Value
Processing capacity rate	t/year	1,057,125
Processing capacity rate (based on 350 days per year)	t/day	3,012
Ore grade	g/t Au	1.99
Gold recovery (1)	%	85%

Notes:

1. *The average gold recovery for oxide ore is 92%; the lower overall gold recovery assigned to the fiscal 2013 budget is a result of sulfide and oxide ore blending to the mill*
2. *Fiscal year 2013 is from 1 July 2012 to 30 June 2013*

Figure 17.1 Simplified Selinsing Gold Mine oxide ore process flowsheet



17.1.1 Phase III expansion

The Phase III plant expansion (designed by Metallurgical Design and constructed by SGM) began on 6 September 2011 with a budget of $8.1 million and was completed in June 2012 on time and at a cost of $8.6 million. The Phase III expansion increased the capacity of the plant throughput from 400,000 t/a to approximately 1,000,000 t/a, and allows greater operating flexibility for processing blended feed.

The following changes were incorporated as part of the Phase III plant expansion:

- Installation of an additional crusher
- Installation of additional ball mill
- Installation of three additional leach tanks
- Improvements to the gold room, detox circuit, tailings pipelines, and pumping system.

The initial Selinsing TSF was designed and constructed by Knight Piesold in late 2009. The TSF is a compacted earth-filled dam with a clay blanket on the upstream slope. It contains a filter core layer and compacted structural fill on the downstream slope. The TSF was constructed using a "downstream construction" method. The initial Stage 1 tailings became operational in July 2010 and provided storage capacity for 16 months of tailings discharged from the production plant. It was designed to allow for expansion in lifts over a five-year project life.

In conjunction with the Phase III plant expansion, an additional US$1.7 million was invested to enlarge the TSF capacity to accommodate an increased discharge from the upgraded plant for ten years of production. This current design has a dam top elevation of 530 mRL with a capacity of 6.7 Mt of tailings. The final design of the TSF provides further extension for an additional 10 years, subject to an increase in embankment height, bringing the total TSF life to 20 years. The final tailings design covers 45 ha and can hold a maximum of 11 Mt of tailings with the highest dam top planned at an elevation of 540 mRL.

17.1.2 Production and cost details

The mill operates 24 hours per day, seven days per week with an expected 92% availability.

During the initial mine start-up in September 2009, only the gravity circuit was in operation. The CIL plant was being developed to full capacity. In February 2010, the CIL circuit was commissioned for operation. Tailings generated under Monument management from mine start-up until July 2010 were stored in Pond A (85% of the material) and Pond B (15% of the material). The Monument-age tailings were stored on top of the old processed tailings. Tailings from this early phase of gravity mill processing are currently being processed through the CIL circuit in combination with crushed and milled Selinsing pit or Selinsing ROM stockpile material. Gold head grades from the reprocessed gravity tailings are ranging in average from 0.50 g/t Au to 1.00 g/t Au based on recent processing performance estimations.

Table 17.2 provides the operating results and some financial results for the years 2010 to 2016. Table 17.3 provides the processing operating costs for the period 2011 to 2016.

Table 17.2 Selinsing historical process production and cost data (FY2010 to FY2016)

	Unit	30 June 2016	30 June 2015	30 June 2014	30 June 2013	30 June 2012	30 June 2011	30 June 2010
Operating results								
Ore mined	t	423,011	421,845	494,141	882,159	501,881	740,909	662,330
Waste removed	t	2,489,500	3,196,553	4,245,156	2,831,755	2,770,491	2,707,598	2,326,502
Stripping ratio		5.89	7.58	8.59	3.21	5.52	3.65	3.51
Ore stockpiled	t	2,335,250	2,755,981	558,252	947,794	920,633	773,432	387,545
Ore processed	t	992,070	954,165	1,018,972	938,598	364,680	351,999	272,120
Average ore head grade	g/t Au	0.88	1.45	1.31	2.07	4.24	4.31	3.08
Process recovery rate	%	67.40	82.40	75.87	86.97	93.70	92.90	58.70
Gold production	oz	18,155	36,473	35,983	52,982	44,585	44,438	13,793
Gold sold	oz	23,150	36,500	37,670	57,905	36,938	40,438	13,793
Financial results								
Gold sales	US$'000	23,595	44,838	48,583	91,275	61,709	56,627	16,316
Processing cost	US$/t	7.00	7.31	6.75	7.40	10.68	8.80	
Cash costs[(a)]								
Mining	US$/oz	114	214	219	112	54	53	64
Processing	US$/oz	444	313	326	207	140	120	90
Royalties	US$/oz	51	63	66	78	107	69	62
Operations, net of silver recovery	US$/oz	3	(3)	2	3	5	-	-
Total cash cost per ounce	**US$/oz**	**612**	**587**	**613**	**400**	**306**	**242**	**216**

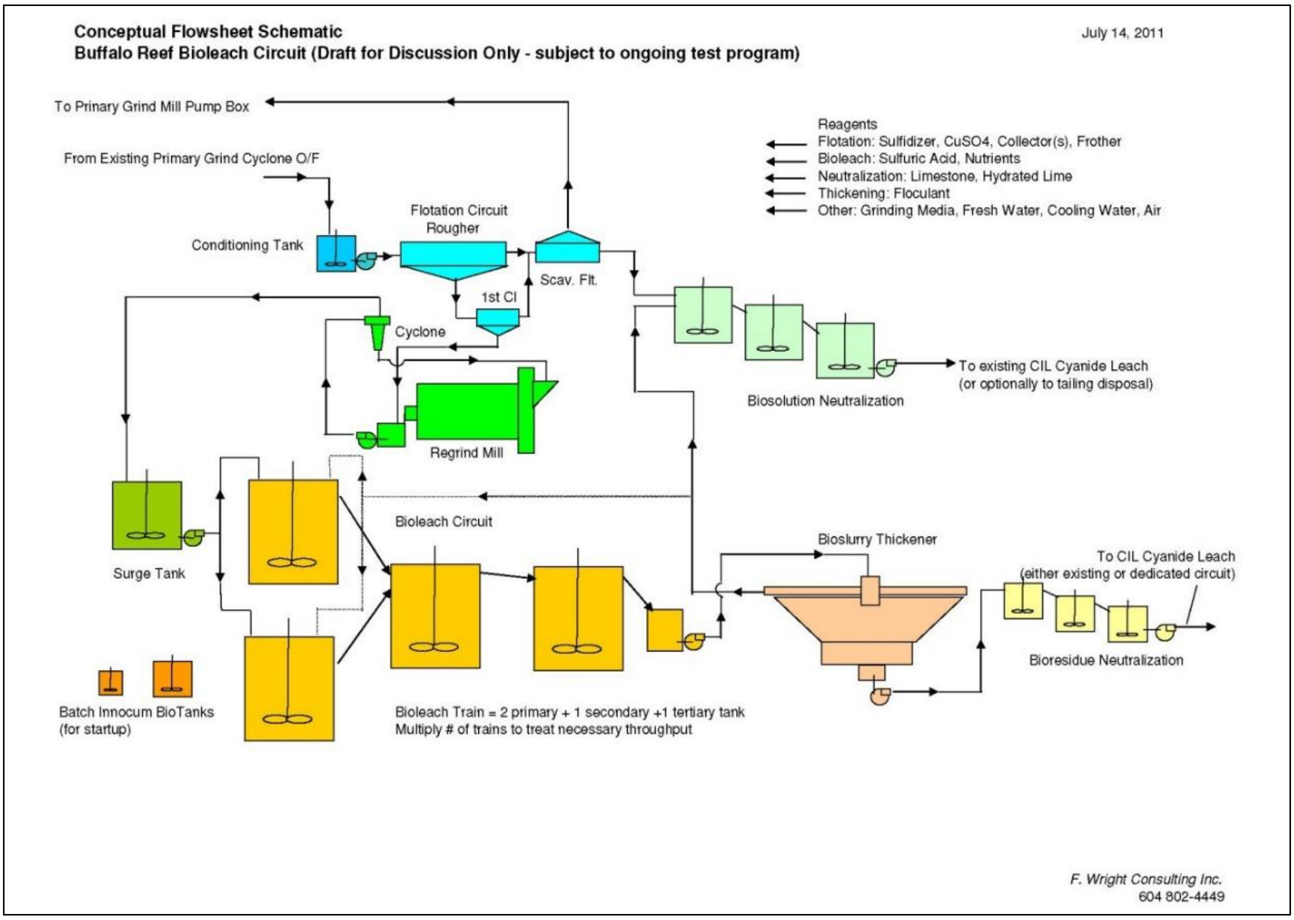
Table 17.3 Selinsing historical process costs (FY2011 to FY2016)

	Year ending					
	June 2016	**June 2015**	**June 2014**	**June 2013**	**June 2012**	**June 2011**
Ore processed (t)	992,070	954,165	1,018,972	938,598	364,680	351,999
Average gold grade (g/t Au)	0.88	1.45	1.31	2.07	4.24	4.31
Gold production (oz)	18,155	36,473	35,983	52,982	44,585	44,438
Gold sold (oz)	23,150	36,500	37,670	57,905	36,938	40,438
Process Plant operating cost (US$ M)	3.097	3.893	6.9456	6.881	6.974	6.947
Operating cost (US$/t ore processed)	7.00	7.31	6.75	7.40	10.68	8.80

17.2 Proposed Phase IV expansion

An initial conceptual flowsheet diagram for processing sulphide ores from Selinsing and Buffalo Reef was developed and is presented in Figure 17.2. This process is based on tests from 2010 to 2013 carried out by Inspectorate. The concept for processing Buffalo Reef sulphide ore is to produce gold-bearing sulphide concentrate using a froth flotation circuit that could be readily adapted into the existing plant. This flotation concentrate would then be reground and then bacterially leached. The resulting slurry would be neutralised and then forwarded to the CIL tanks. Overall recovery would take into account losses from both the flotation and leaching circuits. Based on initial response to this process in tests performed on sulphide ore material, bio-oxidation has shown to be the best pre-treatment method for processing the sulphide ore material at both Selinsing and Buffalo Reef and is therefore being planned as the Phase IV plant expansion project.

Figure 17.2 Simplified conceptual flowsheet diagram for processing refractory ore from Buffalo Reef and Selinsing



MML engaged Lycopodium in 2013 to prepare a PFS level process design with capital and operating cost estimates for a bioleach plant to service a 1 Mt/a ore milling facility. Lycopodium applied industry standard methods to prepare this estimate by developing the following components:

- Process design criteria – based on the Inspectorate report *(ibid)*
- Process design and flow diagrams
- Engineering design criteria
- Mechanical and electrical equipment lists
- Process plant layout.

17.2.1 Process design criteria

The process design criteria developed by Lycopodium forms the basis for the design of the bioleach process plant and required site services. A summary of the design criteria used for design purposes and to determine equipment sizing is provided in Table 17.4.

The Lycopodium process design criteria seem to be sufficiently detailed for PFS purposes. However, additional detail on limestone and lime requirements will be required for final process design as limited data is available from test work and it is a major contributor to the operating costs.

17.2.2 Process selection

Inspectorate tested other processes that maybe employed to liberate gold from sulphide minerals as described in Section 13. Bioleaching of Buffalo Reef sulphides produced superior gold extraction results and Inspectorate concluded *"bulk flotation and bioleaching of the float concentrate prior to CIL appears to offer the most promising response in treating the Buffalo Reef sulphide samples that were received"*. They also concluded from testwork and similarity of the ore characteristics from sulphide ore at Selinsing Deeps, that the metallurgical response from the proposed Phase IV process plant expansion would be similar to those for Buffalo Reef. These expectations are based on the mineralisation at Buffalo Reef occurring in close spatial relationship to Selinsing and being interpreted as sharing the same genetic origin related to the regional Raub-Bentong Suture zone.

Other geological characteristics in common to both deposits include structural control of orebodies in moderately southwest dipping cataclasite + mylonite quartz vein gold mineralisation morphology. Buffalo Reef and Selinsing share similar ore mineralogy and spatial proximity between orebodies. Results from testing the bio-oxidation and gravity/CIL gold recovery processes indicates that processing of material from either ore deposits will respond well to sulphide flotation followed by bio-oxidation pre-treatment. A design criterion for the bioleach process was proposed by Lycopodium as summarised in Table 17.4.

17.2.3 Proposed process description

The final process flow diagram for the proposed flotation and bioleach Phase IV expansion is depicted in Figure 17.3. This Phase IV expansion involves the addition of four distinct sections: the flotation circuit, the bioleach circuit, reagents/nutrients and utilities/services (see Figure 13.8), which were described by Lycopodium as:

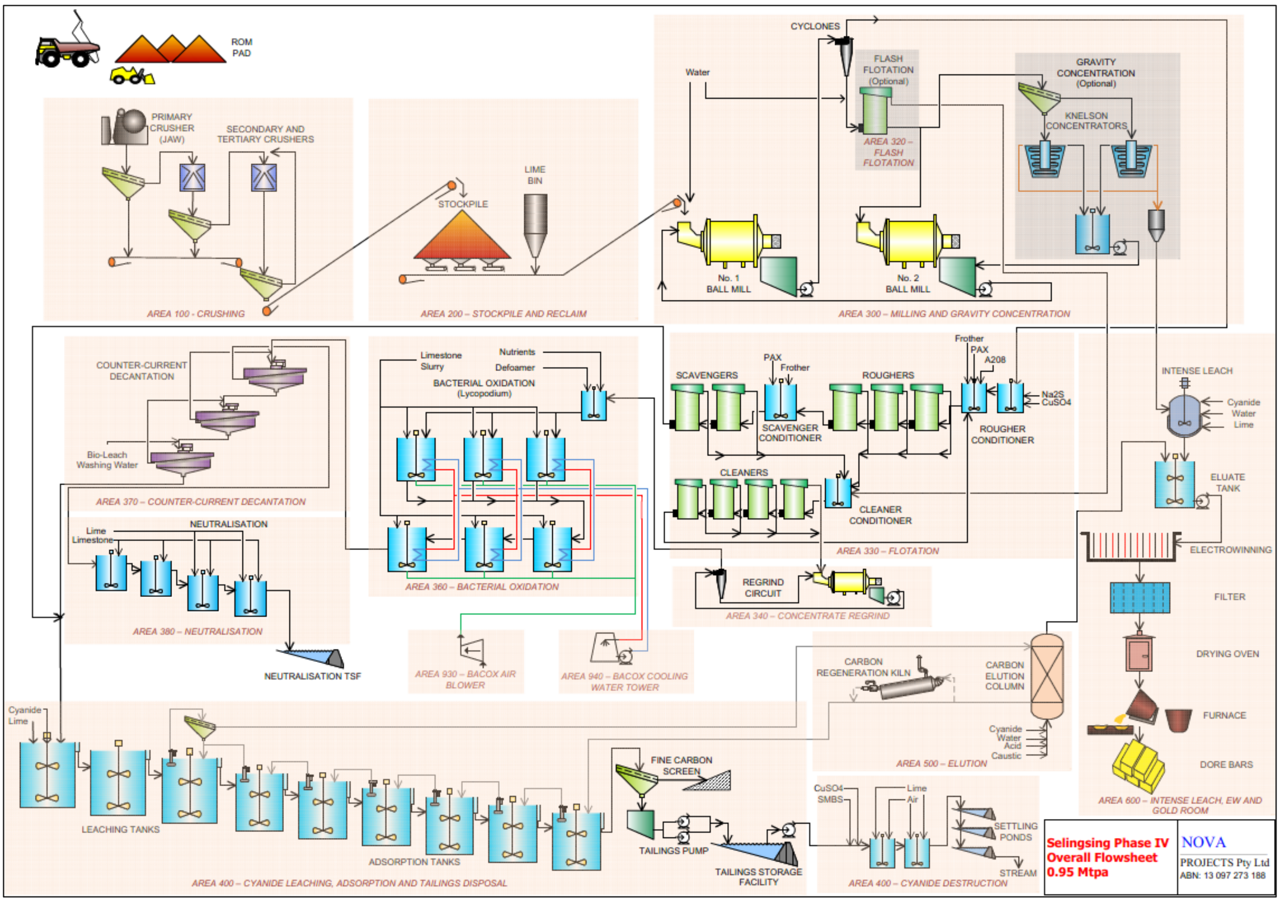
- **Flotation:** The flotation circuit comprises flash flotation on the secondary mill cyclone underflow, rougher flotation of the cyclone overflow, scavenging flotation of rougher tails and one stage of cleaning of rougher and scavenger concentrate. Also included in this circuit is the regrind mill circuit.
- **Bacterial oxidation:** The bioleach circuit will consist of a single bioleach train with three primary reactors in parallel and three secondary reactors in series followed by a three-stage counter-current decantation (CCD) wash circuit and liquor neutralisation circuit and neutralisation effluent tailings system.
- **Reagents and nutrients:** The reagents area includes the inoculum system, the nutrient make-up system, flotation circuit reagents (PAX, Na_2S, $CuSO_4$, Frother and A208), defoamer, hydrated lime, limestone, flocculant and cooling water treatment chemicals (anti-scalant, inhibitor and biocide).
- **Utilities and services:** The utilities area includes instrument air, LP air for the oxidation section, LP air for the flotation circuit, gland water, potable water, process water, tailings reclaim water, cooling water, the heated water system, the diesel fuel system and the electrical distribution system.

Table 17.4 Summary of bioleach process design criteria

Criteria	Units	Value	Source
Ore			
Ore throughput	t/year	950,000	SGM
Sulphur grade – nominal	% S	1.2	SGM
Sulphur grade – maximum	% S	1.37	SGM
Grinding			
Primary product size – P80	microns	75	SGM
Regrind product size – P80	microns	45	Testwork
Flotation			
Mass pull to concentrate	% feed	10	Testwork
Concentrate sulphur grade – design	% S	11.3	Testwork
Bacterial oxidation			
Capacity – nominal	t/a	100,558	SGM
Plant availability – design	%	95	Lycopodium
Sulphur oxidation	%	91.2	Lycopodium
CCD			
Number of CCD stages	number	3	Lycopodium
Neutralisation			
Flotation tails used for neutralisation	t/t concentrate	3.2	Lycopodium
pH stage 1		4 - 5	Lycopodium
pH stage 2		6 - 8	Lycopodium
Neutralisation reagent		flotation tails/limestone/lime	SGM
Medium pressure process air			
Air requirement – bacterial oxidation reactors	Nm^3/h	46,500	Lycopodium
Neutralisation	Nm^3/h	11,731	Vendor
Reagents			
Limestone neutralisation	kg/m^3	50	Lycopodium
Nutrient addition – $(NH_4)_2SO_4$	kg/t ore	3.23	Lycopodium
- $(NH_4)H_2PO_4$	kg/t ore	0.6	Lycopodium
- K_2SO_4	kg/t ore	0.72	Lycopodium
Gold recovery			
Overall gold recovery	%	86	Testwork

Source: Lycopodium, 2016

Figure 17.3 Flowsheet diagram for processing sulphide ore from Buffalo Reef and Selinsing



The process flowsheet developed by Lycopodium comprises the following key components:

- A sulphide flotation circuit (including a flash flotation cell in the existing milling circuit)
- A concentrate regrind mill
- Flotation roughing and cleaning cells
- A bioleach circuit
- A bioleach residue CCD washing circuit
- A bioleach effluent solution neutralisation circuit
- Effluent disposal to a TSF
- Associated services and utilities (blower air, cooling water)
- Reagent and nutrient systems associated with scope.

Flotation

Slurry from the grinding circuit, at 30% solids, will report to a conditioning tank where flotation reagents will be added. The conditioned slurry will gravitate to the first of four rougher flotation cells.

Recovered sulphides from roughing and scavenging flotation will report to the concentrate, which will be pumped to the cleaner cell feed conditioning tank where supplementary reagents and dilution water are added.

The cleaned sulphides will report to the concentrate launder while the cleaner tails will return to the rougher feed tank.

The final rougher tails will be pumped to the final tails hopper or to the neutralisation circuit as required using separate variable speed pumps.

The cleaner concentrate will be reground and pumped via a surge tank of 19 hours' capacity to the bioleach circuit.

Bioleach circuit

The bioleach circuit will operate 24 hours per day with a concentrate throughput of 12.1 t/hr (at design availability of 95%). Maintaining a high availability is essential for bioleach circuit operation as frequent interruptions can significantly reduce bacterial activity and overall oxidation performance.

The bioleach circuit will be configured as three primary reactors in parallel, and three secondary reactors in series. Stored concentrate will be pumped from the surge tank to a feed splitter box above the primary bioleach reactors by means of variable speed pumps. Dilution water sourced from a cyanide-free water system will be injected into the pump discharge line to control the concentrate slurry density.

Nutrient solution will be dosed to the feed splitter box to maintain the correct levels of nitrogen (N), potassium (K) and phosphorous (P) in the bioleach reactors for optimum bacterial activity. The primary bioleach reactors will overflow into launders, which will deliver the partially oxidised concentrate to the first of three secondary bioleach reactors in series.

The bioleach culture will be kept active in the reactors by controlling the level of nutrients, temperature, oxygen level and pH characteristics of the slurry.

The oxidation reactions are highly exothermic and it will be necessary to constantly cool the slurry. The reactors will be equipped with cooling coil baffles through which cooling water will be circulated to control the slurry temperature at 42°C in each reactor.

Oxygen requirements for sulphide oxidation are significant and medium pressure air will be injected into each of the reactors by sparge rings installed below the agitator impeller. The slurry pH in each of the reactors will be controlled between 1.0 and 1.6 by the addition of limestone slurry from a ring main system.

The oxidised product discharging from the final secondary bioleach reactor will gravitate via a launder to CCD thickeners.

CCD wash circuit

During the bioleach of flotation concentrate, iron, sulphur and arsenic are solubilised. These will be washed from the oxidised residue in a series of three 15 m diameter CCD thickeners. Lycopodium selected a three-stage CCD circuit and a wash ratio of 7.8 to reduce soluble cyanicides and acid to acceptable levels in the solution associated with the oxidised residue reporting to the CIL process.

The oxidised bioleach residue will gravitate to the agitated feed tank of the first CCD thickener, where it will be mixed with overflow solution flowing by gravity from the second CCD thickener. Diluted flocculant solution will be added to the feed box to flocculate the slurry prior to the feed well of the first CCD thickener. The overflow solution from the first CCD thickener will gravitate to the neutralisation circuit. The underflow from the last CCD thickener will be pumped by the thickener underflow pumps to the CIL circuit.

Raw water will be used as wash water in the CCD circuit and will be added to the feed tank ahead of the last CCD thickener.

Lycopodium recommends thickening and rheological testwork be conducted during the next phase of the project to identify the limiting thickener underflow percent solids.

Neutralisation circuit

In the liquor neutralisation circuit, the majority of the sulphuric acid will be neutralised and precipitated as calcium sulphate (gypsum) and the soluble arsenic and iron precipitated as stable basic ferric arsenate using flotation tailings, limestone and lime. Milk of lime will also continue to be required by the CIL circuit and allowance has been made in the design for lime slurry preparation and addition to the CIL.

The neutralisation circuit will consist of six aerated and agitated 289 m^3 tanks in series and the solution will flow from tank to tank via overflow launders. As gypsum scaling can be expected in the tanks, bypass launders will allow tanks to be taken offline for cleaning and maintenance.

The oxidation liquor will be neutralised in two stages. In the first stage flotation tails slurry and limestone slurry will be combined with the acidic solution feeding the first tank to raise the pH of the solution above pH 3 where iron and arsenic are co-precipitated as ferric hydroxide and stable basic ferric arsenate. In the second step the pH will be raised to between 6 and 8 in the remaining tanks using lime to neutralise the remaining sulphuric acid as gypsum precipitate.

Precipitation of basic ferric arsenate under controlled pH conditions ensures optimum arsenic stability in the effluent solids. The neutralised effluent will gravitate to the neutralised effluent hopper and be pumped to the neutralisation storage dam.

Tailings disposal

Tailings streams from the plant will be segregated for disposal to enable separate recycle of cyanide free decant water. The two tailings systems will be as follows.

The existing tailings disposal system:

- Flotation tailings (excluding possible bleed to neutralisation)
- CIL tailings
- Miscellaneous waste streams from elution and sump pumps.

Neutralised CCD effluent:

- Neutralised CCD slurry (including the float tails bleed)
- Miscellaneous cyanide free waste streams from sump pumps.

The flotation and CIL tailings will be pumped to the existing TSF. Decant water will be returned to the plant as cyanide contaminated water for reuse in some existing process water applications and the existing decant detoxification system. Neutralisation tails will be pumped to a new tailings facility. The cyanide-free water in the new tailings facility will be decanted and returned to the Process Plant for reuse in milling, flotation and bioleaching.

Reagents and services

<u>Air services</u>

Compressed air will be supplied by duty/standby compressors and all air will be instrument air quality. Medium pressure air for bioleach and neutralisation will be supplied from medium pressure blowers in a common manifold system and flotation air by low pressure air blowers.

Water services

The existing process water tank will continue to be used to supply the process water requirements in the existing process where cyanide contaminated water is acceptable, including leaching, CIL, elution, reagent makeup and cyanide detoxification. Raw water and decant return water from the neutralised effluent tailings facility will return to a flotation process water tank and from there will be distributed to the non-cyanide water users including milling, flotation, bioleach/CCD/neutralisation, reagents and nutrients.

Treated raw water, abstracted from the local river will be used for gland water, cooling tower make-up and in safety showers.

The cooling water circuit will consist of two cooling tower vendor packages, equipped with biocide, anti-scalant and corrosion inhibitor dosing facilities. Side stream filtration and UV sterilisation will also be provided to maintain the cooling water quality and inhibit the growth of algae and bacteria in the cooling water circuit.

Flotation reagents

A conventional approach to flotation reagents will be taken. PAX will be supplied in one-tonne bags and mixed to 15% solution prior to use. Frother will be supplied as a liquid and added undiluted. Copper sulphate will be supplied in 750 kg bags and mixed to 15% solution prior to use, A208 will be supplied in bulk liquid containers and will be transferred to a bulk storage tank for dosing to the process. Sodium sulphide will be supplied in bulk bags and mixed with water prior to transfer to a bulk storage tank for dosing to the process.

Limestone and hydrated lime

Powdered limestone will be delivered in bulk by truck and stored in a powdered limestone silo prior to mixing with water and circulated to end use points in the oxidation circuit via a limestone ring main. Hydrated lime will be delivered by bulk tanker and stored in a lime silo prior to mixing with water and circulated to end use points in the existing CIL and to the neutralisation circuit by a lime slurry ring main.

Bacterial oxidation nutrients

Pre-mixed nitrogen, phosphorous and potassium (NPK) nutrients will be supplied in 100 kg bulk bags and dissolved on site for addition to bacterial oxidation feed.

Inoculum build-up

The bacterial culture will be transferred to site once the onsite laboratory is ready to receive it. During the pilot program the bacterial culture inventory will gradually be built up in staged reactor tanks as follows:

$$10\ \text{L} \rightarrow 150\ \text{L} \rightarrow 1\ \text{m}^3 \rightarrow 10\ \text{m}^3 \rightarrow 100\ \text{m}^3 \rightarrow 1{,}000\ \text{m}^3$$

With each step the inoculum volume will increase tenfold. The 10 L and 150 L build-ups are usually done in an on-site laboratory. The process will continue until enough inoculum has been produced to inoculate the first primary reactor.

The 1 m^3 inoculum build up tank will be equipped with agitators, air spargers, heating and cooling coils. The 10 m^3 tank will ultimately be reused to mix nutrients in the full-scale plant. The 100 m^3 tank will ultimately be reused in the full-scale plant as the last neutralisation tank.

Emergency power requirements

It is essential that immediately a power outage is identified as anything but transient, action is taken to sustain a viable bacterial culture in a minimum of one primary bioleach reactor. Lycopodium has estimated an emergency power requirement of 1.3 MW will be required to sustain the bioleach facility during grid power outages.

Manpower requirements

Wages and salaries used for the study were provided by SGM based on the current Selinsing CIL operation. These salaries were provided inclusive of bonus payments and overhead costs. Allowance has been made for an additional 22 operators and supervisory personnel and three additional technical personnel on top of the current 56. So, total staffing of process plant (including flotation and bio-oxidation sections) will be 78 which are more than adequate for such an operation.

17.2.4 Bioleach operating cost estimate

Lycopodium revised its 2013 operating cost estimate in its report *“3193-STY-001_0_ OPEX_CAPEX_Update Sept., 2016”*. Table 17.5 summarises the outcome of this revision.

Table 17.5 Summary of process operating cost estimate (US$, 3Q 2016, ±25%)

Cost centre	US$/year	US$/t
Power	4,417,043	4.65
Labour	108,871	0.11
Consumables	5,728,532	6.03
Maintenance materials	963,763	1.01
General and administration	618,136	0.65
Total	**11,836,345**	**12.46**

Source: Lycopodium, 2016

The operating cost estimate is only for the bioleach process of sulphides, exclude crushing, grinding, gravity and cyanidation circuit. The operating cost of US$12.46/t is an additional cost to existing operating cost for the current Process Plant. The total operating cost for the Process Plant including the bioleach process will be US$23.50/t.

The estimated operating cost for the bioleach process has been significantly reduced from the 2013 estimate (US$301.54/oz compared to the revised estimate of US$207.27/oz). The main reasons for this are:

- The labour cost has been reduced by rationalising the number of operators actually assigned to the bio-oxidation area (better sharing with the remainder of the operation) and better reflecting the actual shift rates at Selinsing for shift personnel. In addition, the cost of expatriate supervision and additional personnel during the commissioning and ramp-up phases has been deleted from the operating cost estimate and included in the capital estimate.
- The operating cost of light vehicles has been deleted on the basis of advice from Selinsing that sufficient light vehicles are available from the existing operation.
- The Selinsing site provided revised costs for operating consumables; in particular, the cost of limestone was significantly reduced from US$97/t to US$38/t.
- The cost of maintenance is based on the direct installed equipment cost, and in line with the reduced plant capital estimate, the maintenance cost has been reduced accordingly.

- The Selinsing site power cost has been reduced to US$0.061/kWh compared to the value of US$0.07/kWh used in the 2013 estimate.

17.2.5 Bioleach capital cost estimate

Lycopodium revised its 2013 capital cost estimate in its report *"3193-STY-001_0_ OPEX_ CAPEX_Update Sept., 2016"*. Table 17.6 and Table 17.7 summarises the outcome of this revision.

Table 17.6 Summary of capital cost estimate (US$, 3Q 2016, 15%+25%)

Primary discipline	Subtotal cost (US$)	Project contingency cost (US$)	Grand total (US$)
A General	292	29	321
B Earthworks	343,344	44,635	387,979
C Concretelwork	638,319	95,748	734,067
D Steelwork	1,024,445	102,445	1,126,890
E Platework	3,793,971	531,156	4,325,127
F Mechanical	12,762,131	1,659,048	14,421,179
G Piping	5,011,106	1,002,221	6,013,327
H Electrical	7,052,417	705,242	7,767,658
J Instrumentation and Control	1,001,130	100,113	1,101,243
M Buildings and Architectural	848,690	127,160	975,849
O Owners[1]	1,164,333	116,433	1,280,766
P EPCM	1,006,396	100,640	1,107,036
Total	**34,646,574**	**4,584,870**	**39,231,444**

Source: Lycopodium, 2016

Notes: 1 Owners and EPCM cost estimates have been provided by Monument Mining Limited, Selinsing Gold Mine Management SDN BHD and have not been vetted by Lycopodium. Consequently, Lycopodium makes no comment as to their veracity.

Table 17.7 Detail of capital cost summary estimate (US$, 3Q 2016, -15%+25%)

Main area	US$
300 Flash flotation	335,360
400 Treatment Plant costs	26,581,929
500 Reagents and services	5,558,555
500 Management costs	1,006,396
700 Owner's project costs	1,164,333
Subtotal	**34,646,574**
Contingency	4,584,870
Duties/taxes	-
Escalation	-
Total	**39,231,444**

Source: Lycopodium 2016

The estimated capital cost has also been significantly reduced from the 2013 estimate (US$58.1 million compared to the revised estimate of US$39.2 million). The main reasons for this are:

- Foreign exchange rate change since 2013. In 2013, the A$/US$ exchange rate was 1:1.04 and the revised rate used is A$1 = US$0.73, which represents a 30% reduction in all costs with an Australian dollar basis, including equipment prices and engineering costs.
- Concrete supply reduced from a supply rate of US$384/$m^3$ to current estimate of US$180/$m^3$ (including rebar).
- Revised budget quotations were solicited from the major equipment vendors and equipment costs were revised based on recent quotation and current level of competition in the marketplace.
- In 2013, the EPCM was estimated using a flat 26% of the estimated direct cost for the EPCM. This has been revised to 15% based on current level of competition in the marketplace.
- The capital cost was further reduced to US$39.2 million as SGM indicated that they will do the procurement and construction management in-house and only the detail design (engineering) would be outsourced. That will reduce the EPCM and owner's costs to about US$2.66 million from about US$6 million estimated by Lycopodium in Table 17.6.

17.3 Overall opinion of the adequacy of the proposed flowsheet and cost estimation

The Qualified Person has provided the summary and write-up for this section based on previous NI 43-101 reports and recent testwork completed by Inspectorate, PFS costing by Lycopodium who developed a process flow diagram and operating and capital cost for the flotation and bioleach project.

The Qualified Person's opinion is that the proposed flowsheet is applicable and correctly designed for the proposed sulphide material to be treated.

The Qualified Person's opinion is that the operating cost provided in Table 17.5 is somewhat optimistic but achievable when the plant is at steady state. However, it should be stated that during commissioning and ramp-up this operating cost will not be achieved and will be substantially higher than proposed.

The capital cost provided in Table 17.6 and Table 17.7 is within the PFS range and achievable. The biggest risk with the capital cost estimation is the exchange rate fluctuations as can be seen from discussion above.

Production schedule going forward will stockpile oxide material and treat oxide and sulphide material separately, thus the oxide material will be campaign treated as indicated in the schedule. It is important to note that the proposed expansion is to treat sulphide material and not a blend of sulphide and oxide material, thus the plant should either be fed sulphide material or oxide material in campaign mode and not a blend.

Sustaining capital of ~$2 million for the processing plant (mainly for oxide plant) should be sufficient for LOM.

Operating costs for campaign treatment of oxide material going forward will be dependent on gold grade. Historical data as provided in Table 17.2 show that operating cost is dependent on gold grade as expected. Going forward for oxides and sulphides the following operating costs and recoveries provided in Table 17.8 are recommended by Snowden.

Table 17.8 Process operating costs and recoveries for oxide and sulphide material treated (after Phase IV expansion)

Material treated	Deposit	Gold grade (g/t)	Recovery (%)	Process operating cost (US$/t)
Oxide[#]	Both	<1.0	66	7.11[^]
	Both	1.0 to 1.5	75	7.11[^]
	Both	1.5 to 2.5	83	7.11[^]
	Both	>2.5	87	7.11[^]
Transition*	Selinsing	All	87	20.00
	Buffalo Reef	All	85	20.00
Fresh/Sulphides	Selinsing	All	87	19.56[^]
	Buffalo Reef	All	85	19.56[^]

** Reagent costs should be slightly higher for transitional material especially during flotation*

[#] From Figure 13.5 in Section 13

[^] Taking into account historical data (about 1 Mt/a processed) from Table 17.3 as well as current 2016 year to date oxide cost of $6.83/t

18 PROJECT INFRASTRUCTURE

18.1 Road access

The Selinsing Gold Mine is situated approximately two hour's 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.

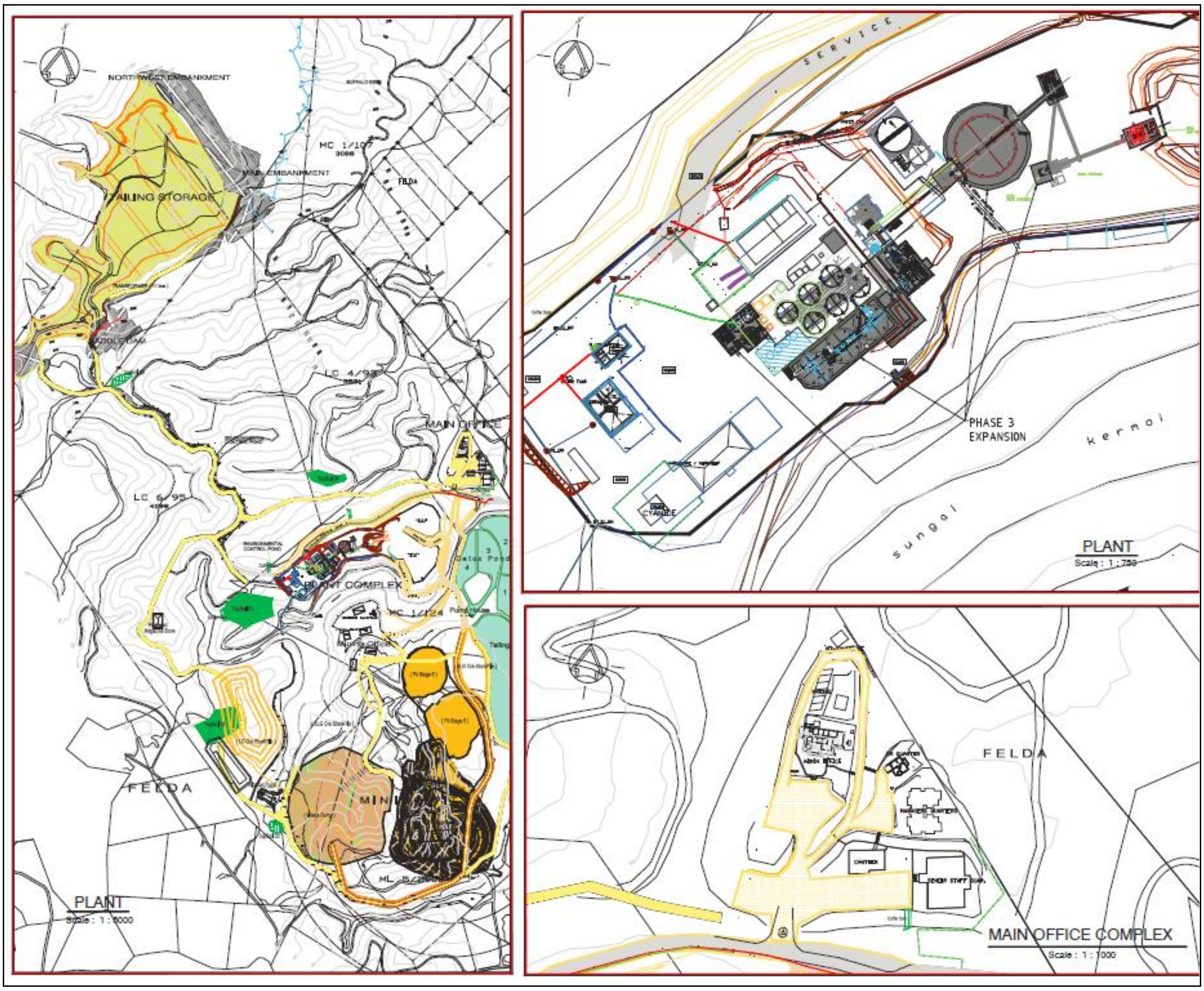
There are very few restrictions for transporting plant components provided they are nominal "wide-load" in nature. At Genting Highlands, 40 km from Bentong, a two-lane one-way 1,000 m tunnel has to be considered. Of note, a hydroelectric power plant has just been commissioned where all equipment came through Sg Koyan to Cameron Highlands.

Police or professional escort is required at nominal cost.

18.2 Existing plant

The 3,000 t/d oxide plant consists of three-stage crushing, a 5,000 t surge capacity COS stockpile reclaimed to a 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.

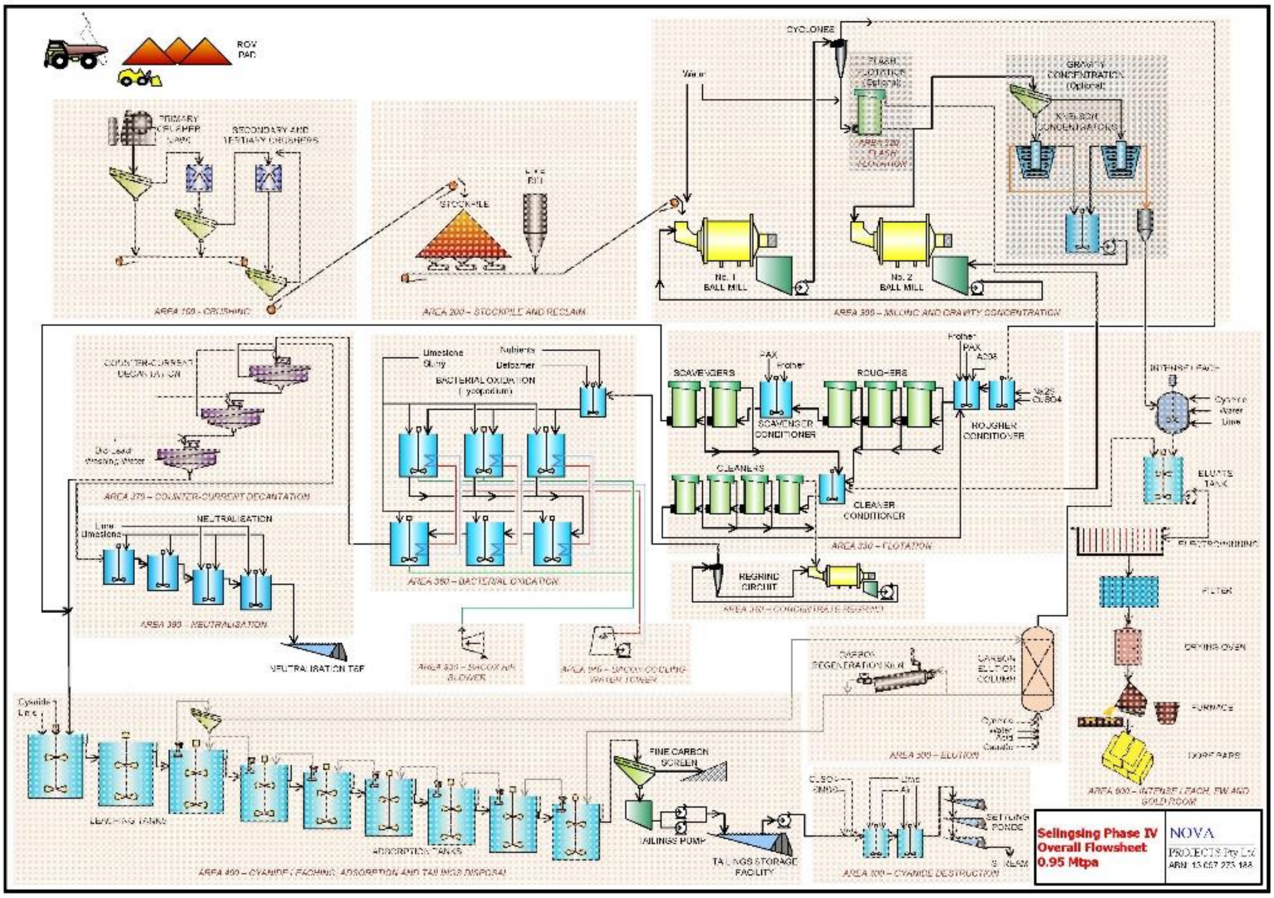
Figure 18.1 Left: site overview; Top right: existing plant; Lower right: office and accommodation



18.3 New bioleach plant

The proposed bioleach plant additions to the existing oxide plant are provided in Figure 18.2.

Figure 18.2 Integrated flowsheet including Phase IV flotation and bioleach circuit in relation to existing oxide plant



18.4 Power

The existing power reticulation at Selinsing is provided in Figure 18.3.

Figure 18.3 Existing Phase 3 electrical power reticulation

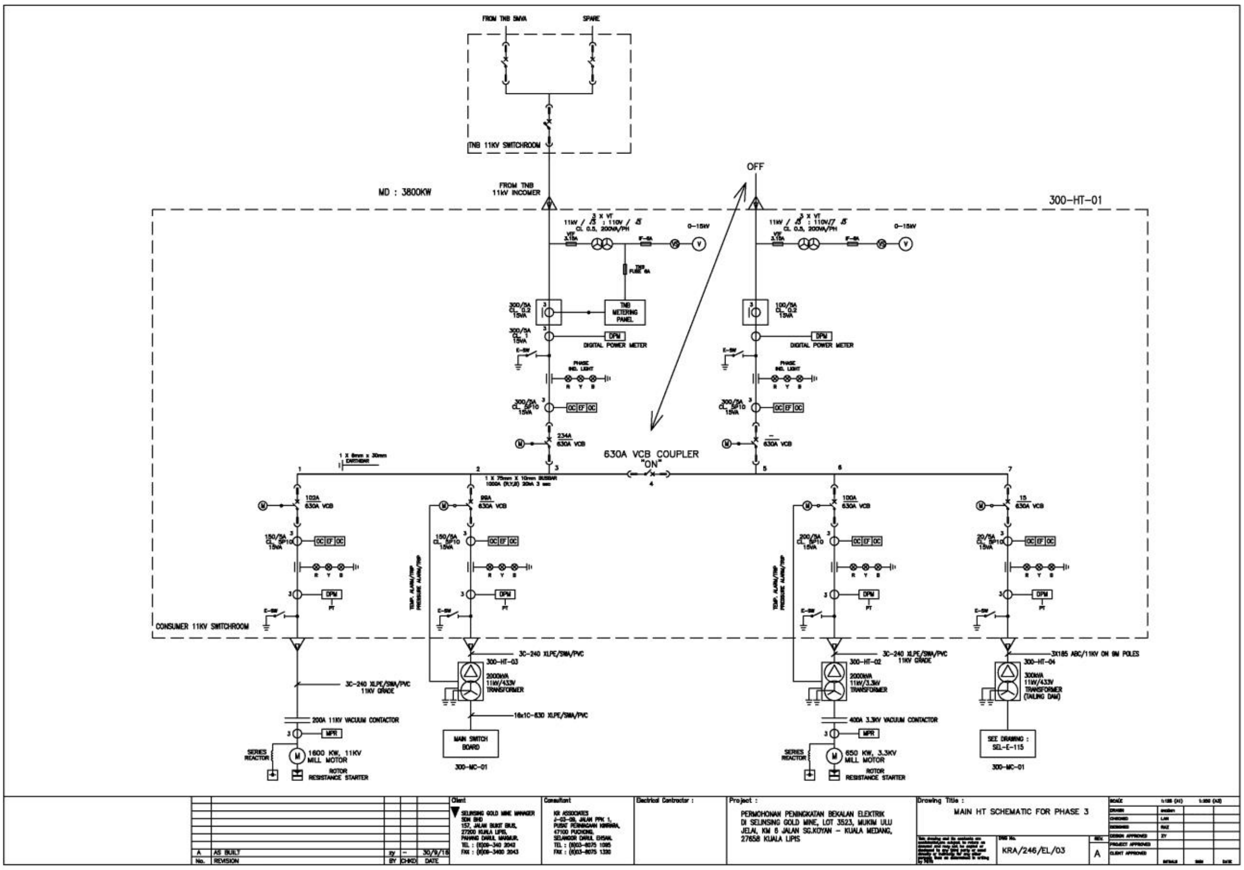


Figure 18.4 TNB 33 kVA incomer from left to transformers in foreground (SGM transformers sit behind)

A 33 kVA line from the Tempoyang substation in Kuala Lipis terminates at TNB's substation located within the footprint of the plant to a 33/11 kV stepdown transformer. SGM then takes the power to its own substation and the stepdown distribution is as follows:

- 3,000 kVA transformer to plant
- 300 kVA transformer to TSF
- Direct feed to primary mill without stepdown
- 11/2.3 2,000 kVA to secondary mill.

Current draw of 3.8 MW is the maximum the national power company can provide the Selinsing site.

Additional line(s) would have to be brought in from the Tempoyang substation in Kuala Lipis for the bio-oxidation plant.

Draw after steady state of bioleach would be an additional 6.9 MW for a total of 10.7 MW.

18.5 Water storage and usage

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, from which there is no abstraction limit.

Water reticulation consists of a raw water tank for the plant fed by the River Kermoi, and a process recycle water tank.

Domestic water for the camp is pumped from a bore to a 20,000 litre tank.

18.6 Fuel storage and usage

Diesel fuel is contained in certified tanks located inside berms (Figure 18.5). Spillage is stored temporarily and removed as waste oils by a certified scheduled waste contractor. The tanks shown have a 20,000 litre capacity and are licensed through the Ministry of Domestic Trade and Consumerism.

Figure 18.5 Left: plant diesel storage; right: lab and light vehicle storage

18.7 Explosives magazine and storage

No explosives are stored on site. All explosive brought on site are consumed in the blasts.

18.8 Contractor facilities

A mining contractor 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 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 Accommodations

There are accommodations on site for 42 people (Figure 18.6). Mining contractor personnel reside in rented premises in the adjacent communities.

Figure 18.6 Accommodations and canteen complex (office is the building at the rear)

18.10 Offices

The administration office is adjacent to the accommodations and canteen (Figure 18.7).

Figure 18.7 Administration office

18.11 Security

Security is outsourced to a local provider consisting of 13 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 formally with the Western Australia Gold Squad.

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.

19 MARKET STUDIES AND CONTRACTS

19.1 Marketing

Monument has indicated that gold from Selinsing has, for a number of 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 Dore 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 dollars out of New York USA through to Maybank (Malaysia’s National Bank) in KL, where it is converted to RM.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Summary

Snowden assessed key items as part of an environmental review of the Selinsing Gold Mine Project, based on information that was 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.

Based on the information provided for this environmental review, it is not anticipated the Selinsing 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.

Future environmental work programs will need to address the implementation of environmental studies that consider all items needed to gain approval for the environmental approvals. The points below summarise the key environmental aspects identified during the review that, if not effectively managed, could impact on future project approval and development:

- Finalise process approval for the bioleach method of gold extraction.
- Finalising the material characterisation program currently being undertaken is a high priority to ensure robust mine plans can be developed, particularly in relation to the sulphidic materials identified at the Buffalo Reef project. This will reduce the potential for significant risk and costs associated with AMD management.
- There have been no significant TSF issues identified to date. The continued use of a reputable engineering specialist to design and regularly review TSF performance will significantly reduce risks associated with geotechnical failure and impacts to ground and surface water as the project progresses. This reduces the risk of significant environmental commercial consequences.
- To adequately protect waterways and resources, drainage and water protection measures will require definition, implementation and ongoing monitoring to prevent any future non-compliance in line with the project expansion and Buffalo Reef project.

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 targeting the mine or expansion project.

20.2 Project environmental approvals and permitting

20.2.1 Malaysian mining legislation framework

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 are located.

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 20.2.2).

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.

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.

20.2.2 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.

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 hectares • 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	

20.2.3 Permit and approval status

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 ML 5/2000, MC 1/124, MC 1/107, MC 1/111, LC 6/95 and LC 4/93; 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	04/05/2016	Conditional approval	Renewal should be submitted one month before the expiry date
Water Permit	SGMM	DMG	11/06/2015	04/05/2016	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

Licence description	Company	Issuing office	Issue date	Expiry date	Permit status	Comments
Diesel Storage Permit (50,000 litres)	SGM	Department of Trade and Consumers Affairs	23/09/2015	22/09/2016	Approved	Permit granted on the annual basis and to be renew one month before expiry date; storage address at MC1/124, Lot 3253, Mukim Ulu Jelai, Km 6 Jln Sg. Koyan-K.Medang
Registration of High Voltage transformer 11KV/3440kW	SGM	Energy Commission, Pahang	20/05/2015	19/05/2016	Approved	Permit granted on the annual basis and to be renew 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	03/02/2017	Approved	Granted for one year and to be renews two months before the expiry date
Permit for buy, store and use sodium hydroxide (NaOH)	SGM	Pharmacy Department, Pahang	01/01/2016	31/12/2016	Approved	Granted for one year and to be renews two months before the expiry date
Erosion, Soil and Sedimentation Control Plan (ESCP) Report	SGM	DOE and Department of Irrigation and Drainage (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

20.3 Environmental and social risks

20.3.1 Environmental approvals

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 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.

20.3.2 Audits

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.

Recommendation

The EMP, as required by the EIA, was approved by the DOE in March 2016. The EMP will include environmental compliance for Phase IV technology. If not already in place, it is 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.

20.3.3 Mine waste management

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.

Recommendations

Appoint an AMD specialist to:

- Assess the results of the material characterisation
- Scope out all PAF management requirements in the mine planning phase
- Input the PAF handling information and requirements into the next operational mining scheme
- Provide input to the Mine Closure and Rehabilitation Plan.

20.3.4 Tailings storage facility

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.

Recommendations

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.

20.4 Project water management

20.4.1 Water quality

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.

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

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.

20.4.2 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

Review and update the site water balance model to ensure adequate water is secured for the project expansion and Buffalo Reef project development.

20.5 Mine closure, remediation and reclamation and decommissioning

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.

Recommendation

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.

20.6 Social

Broad-based community support is essential for the long-term sustainability of the Selinsing Gold Mine and mining in the country in a broader sense. It is important that community members are involved in decisions that affect them. This builds trust and support between both parties.

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 regards 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 basic 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 battery limits

The operating and capital costs for the Project are based on the mining, processing and sale of gold Dore. The process rates of 1.00 Mt/a for oxide and 0.95 Mt/a for sulphide after an initial ramp-up period for the bio-oxidation sulphide process are used. All costs in this section are in US$.

21.1.1 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 Lycopodium in 2016. Administration costs were provided by Monument from the 2017 budget.

21.1.2 Capital cost sources

The oxide processing plant will continue processing stockpiled oxide materials until the bio-oxidation 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 Lycopodium, with the EPCM cost excluded and replaced with Monument’s internal estimate of the owner costs for management of execution engineering, construction and procurement. The sulphide plant pre-production capital costs in the cash flow model have been allocated 30% in Year 1 and 70% in Year 2 of the two years of construction.

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

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

The total estimated pre-production capital cost, including cost to engineer, procure, construct and manage all the works required for sulphide production is $39.2 million. All costs were estimated by Lycopodium except the EPCM cost estimate of $1.12 million of that was made by Monument.

The objective of the Selinsing PFS is to investigate the viability of Selinsing as a sulphide operation by considering:

- The basis of a detailed economic evaluation of the Project

- The basis of further studies to optimise the return on the investment
- A framework to increase confidence in the investment
- The basis for future studies.

21.2 Exclusions

The following items are not included in the capital and operating estimations:

- Foreign currency exchange rate fluctuations although these have been reviewed in the sensitivity analysis
- Cost effect of risks due to potential government policy changes, labour disputes or permitting delays.

21.3 Capital costs

The capital costs have been estimated following a series of indicative prices received from major equipment manufacturers and suppliers. The capital costs have been provided in detail and for this report are summarised in Table 21.1 and Table 21.2. All capital costs have been reviewed by Snowden.

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 is applied to the capital cost estimate.

21.3.1 Bioleach capital cost estimate

In 2016, Lycopodium revised its 2013 capital cost estimate provided to Monument in its report *"3193-STY-001_0_ OPEX_CAPEX_Update Sept., 2016"*. Table 21.1 and Table 21.2 summarises the outcome of this revision.

Table 21.1 Summary of capital cost estimate (US$, 3Q 2016, 15%+25%)

Primary discipline	Subtotal cost (US$)	Project contingency cost (US$)	Grand total (US$)
A General	292	29	321
B Earthworks	343,344	44,635	387,979
C Concretelwork	638,319	95,748	734,067
D Steelwork	1,024,445	102,445	1,126,890
E Platework	3,793,971	531,156	4,325,127
F Mechanical	12,762,131	1,659,048	14,421,179
G Piping	5,011,106	1,002,221	6,013,327
H Electrical	7,052,417	705,242	7,767,658
J Instrumentation and Control	1,001,130	100,113	1,101,243
M Buildings and Architectural	848,690	127,160	975,849
O Owners	1,164,333	116,433	1,280,766
P EPCM	1,006,396	100,640	1,107,036
Total	**34,646,574**	**4,584,870**	**39,231,444**

Source: Lycopodium, 2016

Table 21.2 Monument capital cost summary estimate (RM, -15%+25%)

Phase IV Engineering, Design and Procurement Budget		
Activity	**RM**	**Comment**
Metallurgical review		
2010 test program design	21,000	Review test program design
Process design criteria		
Develop process design	48,000	Incorporate all metallurgical data and SGMM project requirements
Process flowsheet and mass balance		
Ore flows	18,000	
Water balance	18,000	
Flotation and bioleaching circuit	16,500	
Reagent handling	16,500	Process Engineering Consultant
Metallurgical balance	60,000	
Fuel requirements	12,000	
Air requirements	18,000	
Drawing supervision and sign-off	54,000	Crushing; milling; CIL; leach; flotation; bioleaching; carbon handling; services; water
P&IDs and control philosophy		
Develop final piping and instrumentation diagrams	132,000	Based on review of 2007 work
Develop control philosophy documentation	54,000	In consultation with SGMM and Project Electrical Engineers
Equipment list and datasheets		
Review and update 2007 documents	54,000	Will require extension of list
Project design review		
Regular design review meetings with Project Team	270,000	Site meetings: input to plant design, 6 x visit 10 days
Remote review of project design drawings	54,000	4 x hours per 16 weeks
Review final TSF design with consultant	34,318	includes site water balance
Subtotal – Metallurgical design	880,318	
Civil and structural design		
Geotechnical testwork and analysis	72,000	PIII quotation
Foundation design	70,000	PIII quotation
Structural design	80,000	PIII quotation
Subtotal – Civil and structural	**222,000**	
Mechanical and electrical design		
Mechanical and electrical engineer consultants	300,000	Based on previous costs
Subtotal – Electrical	**300,000**	
Design and drafting		
Gravity tails, ball mill 2, CIL and piping	393,000	Overflow Design Services (ODS); estimated 900 hrs
Subtotal – Design and drafting	**393,000**	
Procurement and management		
Salary	1,048,000	RM 75,000 for 12 months; SGMM staff time allocated to Phase IV
SGMM allocated admin expenses	650,000	5% of total admin cost
Subtotal – Procurement and management	**1,698,000**	
Expenses		
Metallurgical Consultant travel (Perth-Selinsing)	72,000	Estimated: 8 trips x A$3,000
Design Engineer Consultant travel (Perth-Selinsing)	108,000	Estimated: 12 trips x A$3,000
Local Civil and Structural consultants travel	20,000	Estimated: 2 x 10 trips x RM1,000
QAQC from vendor – Specialist Consultant	500,000	Inspections and QAQC test on equipment prior to acceptance
Subtotal – Expenses	**700,000**	
Allowance		
Contingency @ 15%	419,332	
	4,612,650	
US$	**1,107,036**	Forex RM:US$ 1:0.24

Source: Lycopodium, 2016

The estimated capital cost has also been significantly reduced from the 2013 estimate (US$58.1 million compared to the revised estimate of US$39.5 million). The main reasons for this are:

- Foreign exchange rate change since 2013. In 2013, the A$/US$ exchange rate was 1:1.04 and the revised rate used is 1 A$= 0.73 US$ which represents a 30% reduction in all costs with an Australian dollar basis, including equipment prices and engineering costs
- Concrete supply reduced from a supply rate of US$384/m^3 to current estimate of US$180/m^3 (including rebar)
- Revised budget quotations were solicited from the major equipment vendors and equipment costs were revised based on recent quotation and current level of competition in the marketplace
- Re-estimation of the EPCM costs by Monument.

In 2013, the EPCM was estimated using a flat 26% of the estimated direct cost for the EPCM. This has been revised to 15% based on current level of competition in the market place.

21.3.2 Other capital expenditure

Monument estimated the capital in *"FY2017_BudgetSummary_V4.xlsx"* as sustaining capital for the process and operation for the 2017 budget as summarised in Table 21.3.

Table 21.3 Other capital expenditure

2017 budget item	RM
Processing equipment	
Crushing	310,200
Mill and gravity	540,000
CIL	1,045,000
Operation equipment	
Laboratory	35,000
Workshop	18,500
Total processing capital	**1,948,700**

Snowden used a forex conversion of RM:US$ as 1.00:0.24 to estimate this cost as $0.5 million per annum.

Comprehensive closure and rehabilitation costs were provided by Monument. These were provided in a Microsoft Excel file "SGMM ARO FY 2016_KJW_V1.xlsx" and considered:

- Neutralisation sulphide generated acid run-off
- Slope stabilisation
- Re-contouring
- Topsoiling
- Revegetation
- Run-off drainage control (ditches and sed. ponds)
- Water sample analysis
- Restrict access fencing
- Monitoring period.

Total costs were RM25,414,066 and Snowden estimated this as $6.1 million using a forex conversion of RM:US$ as 1.00:0.24. The final costs were split as:

- Closure cost: $3.7 million
- Rehabilitation cost: $2.3 million.

No other capital expenditure was provided by Monument for the Selinsing operation.

21.4 Operating costs

21.4.1 Process operating costs

Operating costs for campaign treatment of oxide material going forward will be heavily dependent on the stockpile gold grade. Historical cost data review as provided in Section 17 shows that operating cost is dependent on gold grade as expected. For the oxides and sulphides the following operating costs provided in Table 21.4 are recommended by Snowden.

Table 21.4 Operating costs and recoveries for oxide and sulphide material treated (after Phase IV expansion)

Material treated	Deposit	Gold grade (g/t Au)	Recovery (%)	Process operating cost (US$/t)
Oxide#	Both	<1.0	66	7.11^
		1.0 to 1.5	75	7.11^
		1.5 to 2.5	83	7.11^
		>2.5	87	7.11^
Transition*	Selinsing	All	87	20.00
	Buffalo Reef	All	85	20.00
Fresh/Sulphides	Selinsing	All	87	19.56^
	Buffalo Reef	All	85	19.56^

** Reagent costs should be slightly higher for transitional material especially during flotation*

 # From Figure 13.5 in Section 13

^ Taken into account historical data (about 1 Mt/a processed) from Table 17.2 as well as current 2016 year to date oxide cost of $6.83/t

21.4.2 Mining operating costs

The mining operating costs were provided by Monument for the project from the mining contract and these were compared to the budget figures *"Y2017_BudgetSummary_V4.xlsx"* to build up the total mining cost.

Table 21.5 2017 Selinsing budget figures

Category	Item	Total cost MR	Total US$ unit cost	% of cost
Services	Drill, blast and haulage	22,797,105	1.61	89.2
	Assay and laboratory	861,559	0.06	3.4
Mine maintenance	Supplies	86,215	0.01	0.3
	Pit maintenance	591,359	0.04	2.3
Labour		1,214,215	0.09	4.8
Total mining cost		**25,550,453**		**100.0**
Cost per tonne mined		7.51	**1.80**	

The LOM mining services cost provided by Monument is expected to average:

- $1.89/t moved for Selinsing
- $1.71/t moved for Buffalo Reef.

There is provision in the contractor schedule of rates for pit depth and horizontal haulage incremental cost change as a trucking cost increment cost of RM0.03 per BCM per 5 m depth increment, and RM0.0024/t per horizontal metre as in current contract with Minetech. In addition to this, there is a provision for cost of $0.13/t moved to cover grade control, supplies pit maintenance (pumping) and owner labour. The total mining cost is estimated by Snowden to be:

- $2.08/t moved for Selinsing
- $1.90/t moved for Buffalo Reef.

A stockpile cost of $0.79/t moved was considered by Snowden for stockpile extraction.

21.4.3 Administration operating costs

Selinsing provided costs from the 2017 budget (*"FY2017_BudgetSummary_V4.xlsx"*) for the administration of the operation.

Table 21.6 Administration operating costs

Item	Total cost MR	Total US$ unit cost	% of cost
Site maintenance and development	3,374,121	0.73	25.6
Security	1,872,327	0.40	14.2
Health and safety	488,908	0.11	3.7
Environment	431,314	0.09	3.3
Site support	7,011,435	1.51	53.2
Total administration cost	**13,178,105**		**100.0**
Cost per tonne processed	13.14	2.84	

A cost of $3.15/t of ore processed was used for the administration costs. This cost does not consider corporate overheads.

21.5 Purchase of the Felda leases

Monument advised that there is a purchase cost $1.5 million to acquire the mining lease.

21.6 Taxation

The Malaysian Government tax holiday was granted for the construction of the Selinsing Mine in 2010. The tax holiday expired 1 December 2015, when the income tax rate became 25% of taxable income.

Depreciation is considered in the Section 22 (Economic Analysis), as written off at the rate of reducing ore tonnes.

21.7 Royalties

All mining leases in Malaysia carry a 5% royalty payable to the Malaysian 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).

22 ECONOMIC ANALYSIS

All dollar ($) values in this section are US$.

22.1 Project economic headline results

The following Table 22.1 and Table 22.2 provide the Project headline results before and after taxation for a gold price of $1,255/oz of gold (base case).

Table 22.1 Project economic model headline results after taxation

Item	Unit	Value at $1,255 /oz Au
Net cash flow	$ M	37.2
NPV_8	$ M	23.1
IRR	%	34.8

Table 22.2 Project economic model headline results before taxation

Item	Unit	Value at $1,255 /oz Au
Net cash flow	$ M	47.7
NPV_8	$ M	31.4
IRR	%	43.5

22.2 General criteria

Snowden prepared an economic cash flow and financial analysis model based on inputs derived from mining and processing schedules as well as capital and operating cost estimates including royalties for the Project. The model was prepared from construction and mining schedules estimated on a quarterly basis for project life. All inputs are consolidated annually in this report. The cash flow model was based on the following:

- 100% equity ownership by Monument
- Costing from July 2016
- Two-year production period for plant construction
- No cost escalation
- All costs reported in US$ and where costs were estimated in Malay Ringgit (MR), the exchange rate used was US$0.24 to the MR.

The objective of preparing the cash flow model was to:

- Collate all the inputs for the following disciplines into a single model:
  - Mining
  - Processing
  - Metallurgical
  - Metal pricing
  - Sulphide production capital costs
  - Production sustaining capital
  - Operating costs

  - Rehabilitation, salvage and closure costs
  - Royalties
  - Taxation.
- Provide sufficient information to management so that they are supported in any decision-making process.
- Provide the basis for future studies.

The economic cash flow model was then interrogated to determine the following values after taxation:

- Headline values:
  - Net cash flow
  - NPV at 8% discount rate (NPV8)
  - IRR
  - Breakeven (NPV8) nickel price
  - C1 cost per ounce of gold produced (Brooke Hunt Methodology)
  - Production year payback.
- KPIs:
  - Operating costs
  - Total costs
  - Production payback years.

22.3 Economic model inputs

Table 22.3 shows the inputs were used in the economic cash flow model.

Table 22.3 Selinsing Project economic model inputs

Item	Unit	Value
Life of Project production	years	6.74
LOM ore mined	kt	3,990
LOM ore processed	kt	6,252
LOM waste mined	kt	32,864
LOM average Au grade	g/t Au	1.10
LOM average Au recovery	%	79.0
LOM recovered ounces	koz	227
Plant throughput	Mt/a	0.95
LOM Au price	$/oz	1.225

22.4 Production summary

The physical production of the Project is based on the mining, processing and recovery of metal reported in Section 16.

22.5 Key performance indicators

The Project LOM KPIs after taxation are presented in Table 22.4 below.

Table 22.4 Selinsing Project KPIs after taxation

Item	Unit	Value
Total value of product sold	$ M	285.2
Cash cost	$/oz	830.1
Total cost	$/oz	1,099.0
Production year payback	year	2.6
Brooke Hunt methodology C1 cost	$/oz	767.4

The cash costs include all direct operating costs plus royalties, the total costs include the cash costs plus capital costs and taxation. The Brooke Hunt methodology C1 costs include all direct operating expenses but do not include royalties.

22.6 Sensitivity analysis

The economic cash flow model was used to prepare a sensitivity analysis for the NPV_8 for the Project after taxation. The sensitivity analysis was completed on the following variables:

- Grade of gold
- Recovery of gold
- Price of gold
- Plant capital
- Production capital
- Mining cost
- Processing cost.

The sensitivity analysis determines how the NPV_8 is affected with changes to one variable at a time while holding the other variables constant. The results of the sensitivity analysis are presented in Table 22.5 below. In this table, "B/E" represents the breakeven and it indicates the change in the variable that will bring the project NPV_8 to $0.000. Elasticity is a measure of sensitivity that indicates for a 1% change in the variable what change in the NPV_8 will occur. A value greater than one indicates that the change in the variable will have a higher value change in the NPV_8 than the change in the variable and indicates a higher sensitivity to change.

Table 22.5 Sensitivity table for the base case ($12,000/t) NPV_8 – after taxation

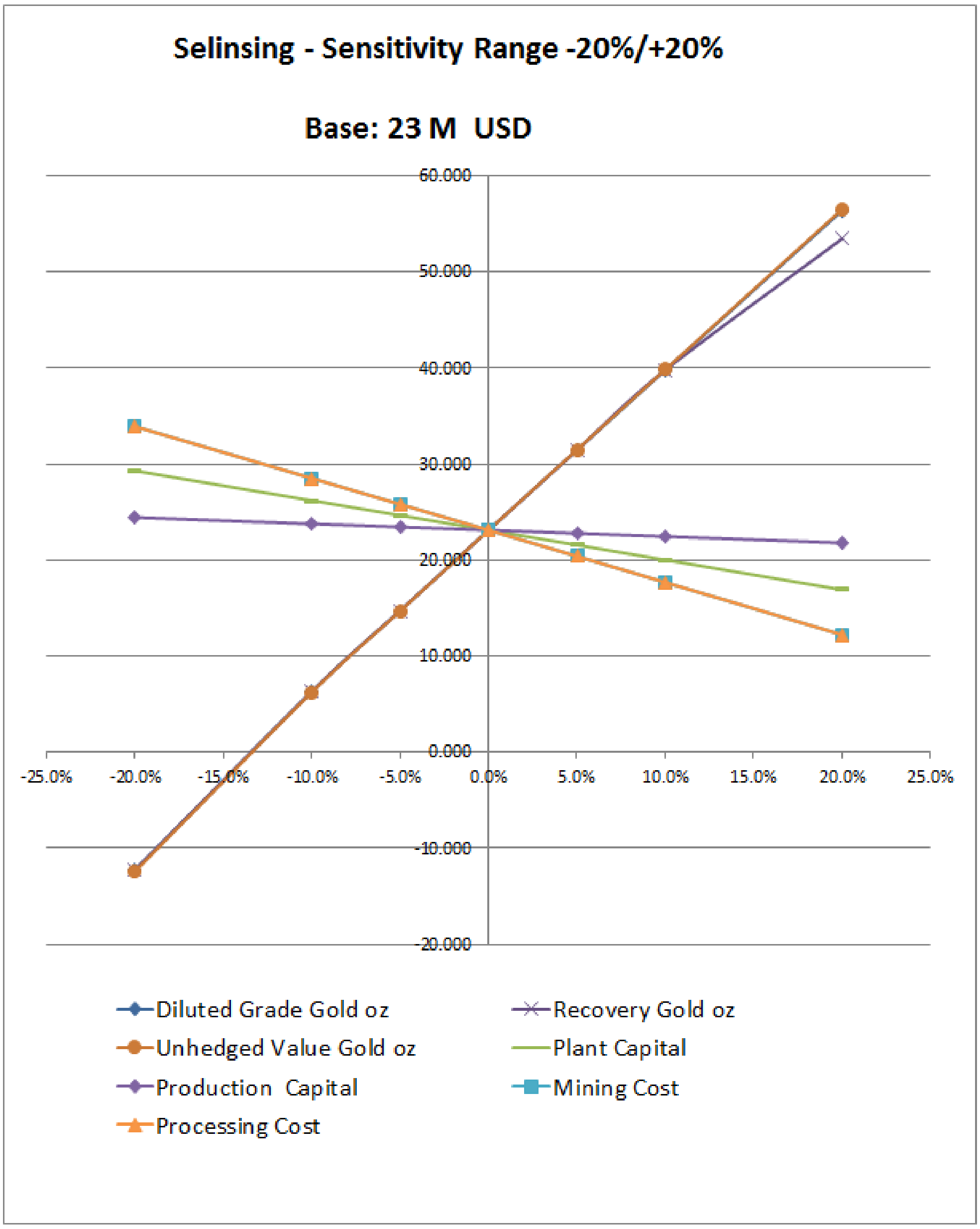
	-20%	-10%	-5%	0%	5%	10%	20%	B/E	Elasticity
Grade Au	-12	6	15	23	31	40	56	-14%	7.3
Recovery Au	-12	6	15	23	31	40	56	-14%	7.3
Price Au	-12	6	15	23	31	40	56	-14%	7.4
Plant capital	29	26	25	23	22	20	17	75%	1.3
Production capital	24	24	23	23	23	24	24	-	0.3
Mining cost	34	29	26	23	20	18	12	43%	2.3
Processing cost	34	29	26	23	20	18	12	43%	2.3

The B/E on the right of Table 22.5 indicates the breakeven change for the variable if all other variables remain unchanged. For example, if the grade of gold grade reduces by 14% the Project will break even on NPV_8.

The Elasticity on the right of Table 22.5 indicates the change in the Project NPV_8 that is induced by a 1% change of the variable if all other variables remain unchanged. For example, if the grade of Au changes positively or negatively by 1% it will induce a 7.3% change of the NPV_8. A change of greater than 1.0 indicates that the project is sensitive to a change of the variable as the NPV_8 will change by a greater amount than the change of the variable.

The sensitivity chart is shown in Figure 22.1 below and it covers a range of variable changes from -20% to +20%. As the variable line increases towards the vertical it indicates the project is more sensitive to changes of the variable.

Figure 22.1 Cash flow sensitivity graph



22.7 Breakeven analysis

A breakeven analysis after taxation was undertaken on the gold price and gold grade for net cash flow. This analysis was conducted on the sensitivity analysis data and provides the gold price which will bring either the net cash flow to $0. The results of this analysis are presented in Table 22.6.

Table 22.6 Breakeven analysis after taxation

Item	Unit	Breakeven
Gold price	$/oz Au	1,082
Gold grade	g/t Au	1.20

22.8 Conditional simulation

A conventional cash flow model provides a single point analysis based on the values of variables that have been used. A simple cash flow model will use the same variables for the life of the mine. A more advanced cash flow model will schedule the variables on a period basis. A sensitivity analysis is undertaken to determine the influence that a change in any variable will have on the project while holding all the other variables constant. The scenario analysis is a more advanced sensitivity analysis where each of the variables may be changed at the same time by different sensitivity change conditions and the results are updated in real time. This is a comprehensive form of a "What if" analysis and is used to stress test a project to see what changes to various variables are required to "break" the project. Whether the cashflow model is a simple form or scheduled, the values applied are most unlikely to occur in a real-life operational project.

The conditional simulation is applied to simulate all the possible ranges, within reasonable limits, that the variables may be within for each of the annual periods for the life of the project. To set the model up, risk limits (positive and negative) to the anticipated input variables are set. These may be skewed and they can have different ranges for each variable. A random number generator then selects a value within the range for each variable and for each period separately and applies that value to the period in the cash flow model. For each iteration, the conditional simulation will generate a large number of new variable values across the whole spreadsheet (all variables) and will provide a single output result, e.g. the NPV for that set of data. This process is repeated a large number of times, generally between 100 and 10,000 times, and all of the output NPV values are collected. The results for the conditional simulation are provided in Table 22.7.

Table 22.7 Conditional simulation for the Selinsing Project

Variable	Unit	Risk limits		Value ranges		
		+ve%	-ve%	High	Base	Low
Grade Au	g/t	2	4	1.42	1.39	1.33
Tonnes ore	Mt	10	0	4.39	3.99	3.99
Recovery Au	%	1	2	79.8	79.0	77.4
Price Au	$/oz	30	20	1,632	1,225	1,004
Costs						
Plant production	$M	20	5	47.4	39.5	37.5
Production	$M	20	5	12.7	10.6	10.1
Mining	$/t	10	5	23.2	21.1	20.0
Processing	$/t	10	5	15.67	14.25	13.53

22.8.1 Statistical analysis

A statistical analysis was carried out on the collected conditional simulation values to determine a real-life NPV. In addition, because there are a large number of values collected, a probability analysis can also be applied which gives the range of possible outcomes for the Project at different confidence levels.

A statistical analysis for a sample of a population is valid for a sample greater than 80 values. The conditional simulation was run for 10,000 iterations and the results for the statistical analysis are provided in Table 22.8.

Table 22.8 Statistical and probability analysis for the Selinsing Project

Item	Unit	Value	Value
Mean NPV_8	$M		20.7
Standard error	$M		0.115
Standard deviation	$M		11.5
Minimum value	$M		-21.3
Maximum value	$M		61.7
Risk index (coefficient of variation)	%		56%
Range at 99.7% confidence	$M	-13.3	54.6
Range at 95.0% confidence	$M	-1.9	43.2
Probability of > value	$M	-2.0	98%
Probability of > value	$M	13.0	75%
Probability of > value	$M	20.7	50%
Probability of > value	$M	28.0	26%
Probability of > value	$M	43.0	3%

23 ADJACENT PROPERTIES

There is no adjacent properties information to disclose.

24 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data and information to disclose that makes the technical report not misleading.

25 INTERPRETATION AND CONCLUSIONS

Monument provided data to the authors for interpretation and modelling of geology and Mineral Resources and Mineral Reserves for the Selinsing Mine and the Buffalo Reef gold deposits. A process of mine planning was used to apply modifying factors to the Mineral Resource and identify a viable Mineral Reserve after the application of reasonable modifying factors. Mine planning included:

- Pit optimisation studies on the resource estimates
- Designs for an open pit mine to access the resources
- Generation of a mine production schedule for exploitation of the reserve over a period of six years
- Identification of the bio-oxidation process method as economic for the Phase IV plant expansion and mining provides a US$37.2 million NPV and 23% rate of return.

Monument has demonstrated through metallurgical testing that processing refractory ore through a CIL plant with bio-oxidation pre-treatment as a viable recovery method.

The area has been explored over a number of years with predominately RC drilling and some diamond drilling and 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 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 confidence and size of the Buffalo Reef resource through additional drilling.

The authors of this report concluded that a viable project processing the remaining oxide and sulphide ore is identified at the PFS level and that future studies as feasibility studies be commenced to identify the final process for extraction of gold from the sulphide ore and increase the accuracy of the economic estimate of the Project.

26 RECOMMENDATIONS

The 2016 PFS for the Selinsing Gold Mine Project has identified key areas to optimise the project and increase the confidence of the Project by:

- Sourcing more sulphide ore to utilise the life of the plant that is currently not optimal
- Investigate metallurgical testwork data from the INTEC process to identify suitable gold recovery and process costs
- Progression of environmental licence permits for the Phase IV expansion
- Update capital and operating cost inputs to a feasibility study level.

26.1 Resource exploration programs

Monument plans to follow-up with diamond drilling programs at the Selinsing and Buffalo Reef deposits focused on defining preferentially sulphide mineralisation at depth below and around the existing pits within gap zones in between the known resources that contain little drillhole information, and to convert Inferred Resources into Indicated and/or Measured Resources (proposed "Deep Sulphide Holes"). The main programs are listed below:

- Infill/resource definition and down dip extension sulphide drilling program for BRC and Felda
- Selinsing Pit 4 sulphide gold high grade confirmation/extension drilling
- Sampling geometallurgical program.

The sulphide drilling resource definition and down dip extension program for Buffalo Reef Central Deposit (Figure 26.1) comprises 22 holes for a total of 1,635 m, with an average depth per hole of around 75 m and holes oriented dipping 60^{o} to the west (azimuth 270^{o}). The objective of this drilling is to check deeper down dip extensions of the identified quartz-stibnite high grade mineralisation, aiming to increase the total resource of sulphide material, while also promoting current Inferred oxide and transition Resources into Indicated Resources from shallower intercepts.

The sulphide drilling resource definition down dip extension and subordinately exploration drilling program for the Felda Block 7 Deposit (Figure 26.1) also comprises 22 holes for a total of 2,800 m, with an average depth per hole of around 130 m and the same hole orientation as for BRC. The objective for this drilling is the same as for BRC. In addition, there is an aim to add new resources and promote current resources from Inferred to Indicated Mineral Resources. The drilling program at Felda will enable the Buffalo Reef Central and Buffalo Reef South open pits to be extended at depth.

The Selinsing Pit 4 high grade confirmation, extension and infill drilling program (collar locations shown at Figure 26.1) comprises 13 holes for a total of 796 m, with an average depth per hole of around 60 m and holes oriented dipping 60^{o} to 70^{o} to the west (azimuth 270^{o}). The objective of this drilling is to confirm high grade gold mineralisation intercepted with older TRA holes underneath the current pit floor. In addition, the program includes infill drilling and extension drilling, verifying the down dip extension of the sulphide mineralisation.

Metallurgical drilling associated to all drilling programs is also planned, with an aim to provide sulphide material to be used in metallurgical testwork.

The sampling geometallurgical program to be conducted in BRN, BRC, Felda, BRS and Selinsing deposits (sample hole locations shown in Figure 26.2) aims to define leachable mining blocks to improve mining and plant production, with a total of 66 composited samples to be submitted to and tested in the on-site laboratory. Coarse reject, quarter-core and coarse-retain samples from already drilled holes will be selected, collected and composited under a defined criteria, and will be analysed by the on-site laboratory for intensive CIL testing, sulphur, organic carbon, and gravity analyses to identify leachable mining blocks. Detailed geological mapping is expected to be conducted in parallel, aiming to delimitate geological domains with distinct mineralisation styles, which will help in identifying the geological controls for the delineation of the leachable mining blocks.

Figure 26.1 Deep sulphide drilling program for BRC, Felda and Selinsing Pit 4

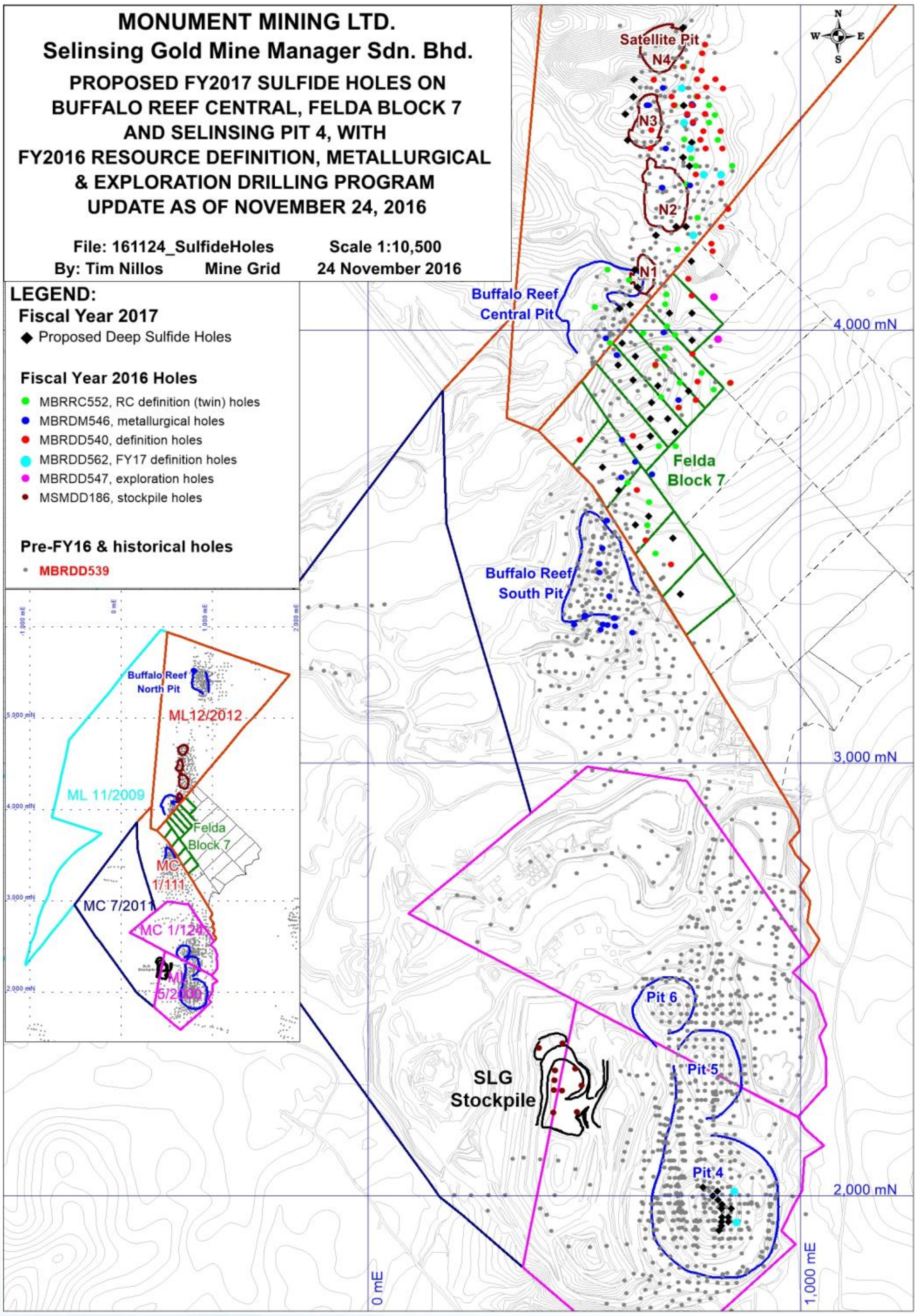
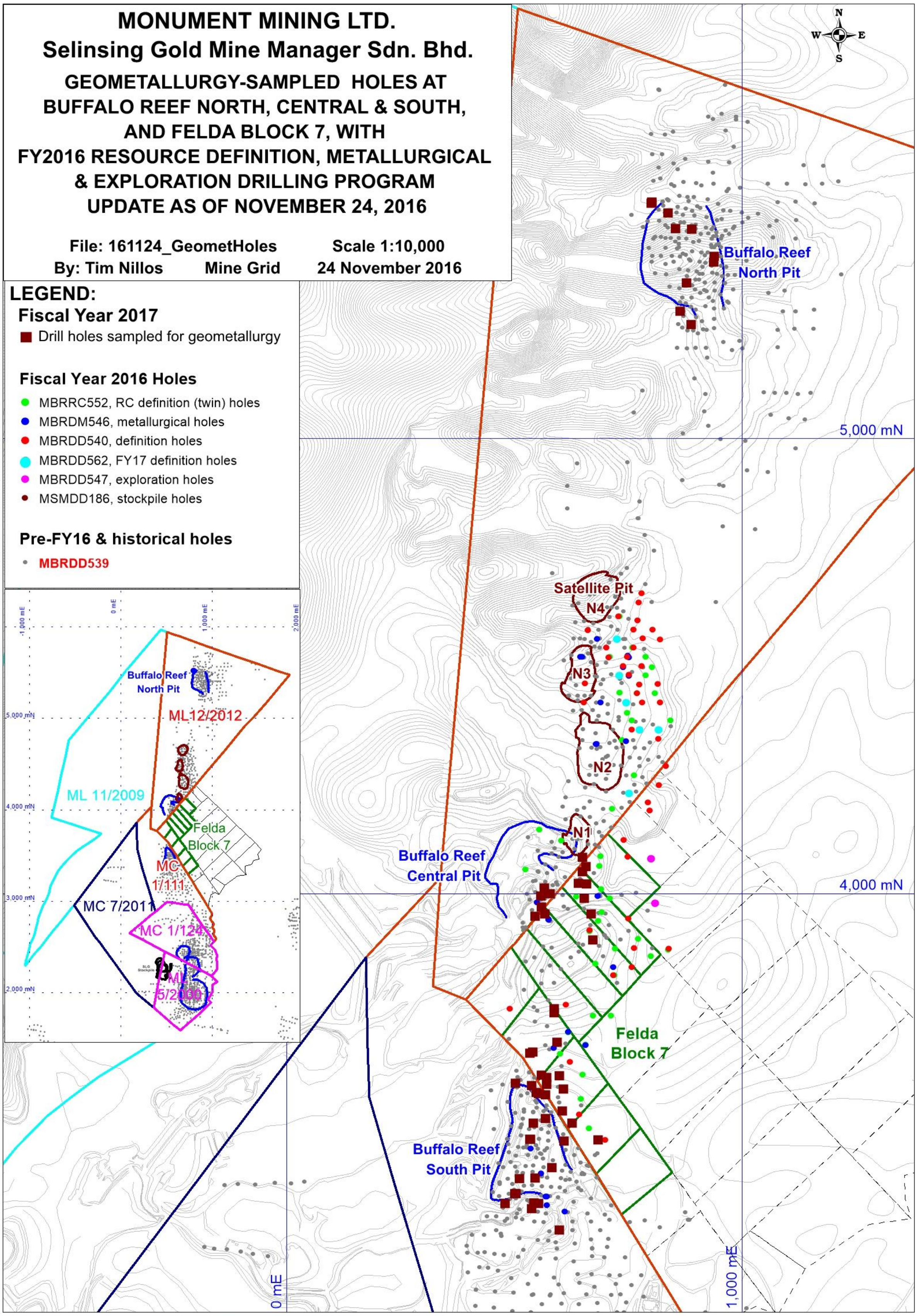


Figure 26.2 Location map of holes for geometallurgical samples



26.2 Geotechnical studies

As noted in the geotechnical site visit early in 2016, the level of geotechnical information for the Buffalo Reef and Selinsing deposits is very limited, and investigations are advisable for the deeper pits. The current lack of information results in significant risk for the pits in terms of potentially not being able to extract to the design slopes due to adverse conditions linked to weak rock and groundwater pressures. Conversely, the investigations could show better than expected conditions, opening the possibility of steepening the slopes. The fresh unweathered rock hosting the sulphides should have much better geotechnical conditions than the weathered material.

Similarly, the poor condition of the Selinsing west wall poses major risks to any deepening of this pit, and investigations are strongly advised if any cutback is contemplated to access the deeper resources.

To reduce the geotechnical risk, the following is recommended:

- A small number of NQ or HQ holes recovering orientated core from the wall rock units
- Detailed geotechnical and structure core logging
- Some basic strength testwork (pointload tests).

Mapping the persistent structures in current pit walls and assessing their potential influence as pits develop would also be a key component of a continuous improvement program.

26.3 Mining

A 10 m bench has been selected for this study for relatively selective mining of the ore benches, but larger benches and their impact on grade control, product quality and mining cost and productivity should be reviewed in any further study work.

26.4 Metallurgy and process design

Monument has commenced work on an alternative process method using the proprietary INTEC process. The current status of the investigation of the alternative INTEC process is that the testwork, is suspended, pending the outcomes of alternative approaches to Buffalo Reef sulphide treatment.

27 REFERENCES

Author	Title
Isaaks, E.H., and Srivastava, R.M., 1989	An Introduction to Applied Geostatistics. Oxford University Press (New York) 561pp.
CIM, 2010	CIM Definition Standards – For Mineral Resources and Mineral Reserves. Prepared by the CIM Standing Committee on Reserve Definitions. Adopted by CIM Council on 27 November 2010
JORC, 2012	The Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves. Prepared by the Joint Ore Reserves Committee of The Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Minerals Council of Australia (JORC)
CIM, 2003	CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines, adopted by CIM Council on 23 November 2003
Journel, A.G., and Huijbregts, Ch.J., 1978	Mining Geostatistics. Academic Press (London), 695pp.
David, M., 1977	Geostatistical Ore Reserve Estimation. Developments in Geomathematics 2. Elsevier (Amsterdam), 364pp.
Damar, 2002	*Buffalo Reef Prospect – PL 4/93, Exploration Licence Work Report: 1997-2001*, unpublished internal report for Damar Consolidated Exploration Sdn Bhd. File dated 16/05/2002.
Flindell, P., Chye, L.T., and Mohamed, S. 2003	Buffalo Reef Project, Project Review, unpublished internal report for Avocet Mining PLC. March 2003.
Harun, Z.B. 2011	Brief Report Buffalo Reef Drilling Programs Oct 2007 – Sept 2008, unpublished internal report for Monument Mining Limited, 2011.
Makoundi, C. 2011	Geology, Geochemistry and Metallogenesis of Selected Sediment-hosted Gold Deposits in the Central Gold Belt, Peninsular Malaysia, MSc Thesis; School of Earth Sciences, CODES Centre of Excellence in Ore Deposits, University of Tasmania, December 2011.
Naidu, K.V.G. 2005	Buffalo Reef Prospect, First Interim Report May 2005, unpublished internal report prepared for Avocet Gold PLC. May 2005
Odell, M., Swanson, K., White, M., and Fox, J. 2013	Selinsing Gold Mine and Buffalo Reef Project Expansion, Pahang State, Malaysia, NI 43-101 Technical Report prepared by Practical Mining LLC for Monument Mining Limited, dated 23 May 2013.
Yeap, E.B. 1993	Tin and gold mineralisation in Peninsular Malaysia and their relationships to the tectonic development, Journal of Southeast Asian Earth Sciences, 8 No 1-4, p 329-348.
Lycopodium Minerals Pty Ltd 2016	3126-STY-001 C IFR Pre-Feasibility Study Report.
Fry, R 2016	Geotechnical Review of Selinsing and Buffalo Reef Mines, Internal memo to Monument.

28 CERTIFICATES

28.1 John Graindorge

CERTIFICATE of QUALIFIED PERSON

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 Mine and Buffalo Reef Project dated 14 December 2016 (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 14 December 2016

Signed and sealed John Graindorge

John Graindorge, BSc (Hons), Grad. Cert. Geostatistics, MAusIMM(CP)

28.2 Frank Blanchfield

CERTIFICATE of QUALIFIED PERSON

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 Mine and Buffalo Reef Project dated 14 December 2016 (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 14 December 2016

Signed and sealed Francis James Blanchfield

Frank Blanchfield, B.Eng, FAusIMM

28.3 Dr Leon Lorenzen

CERTIFICATE of QUALIFIED PERSON

I, Dr Leendert (Leon) Lorenzen, an Executive Metallurgical Consultant, 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 Mine and Buffalo Reef Project dated 14 December 2016 (the "Technical Report") prepared for Monument Mining Limited.

b) I graduated with a University of Stellenbosch, Stellenbosch, South Africa and hold a Bachelor of Engineering (Chemical), Master of Science in Engineering (Metallurgy), cum laude and Doctor of Philosophy (Metallurgical Engineering) from Stellenbosch University, Stellenbosch, South Africa (1993). I am a Fellow of the Australasian Institute of Mining and Metallurgy (FAusIMM #304479), Fellow of the Southern African Institute of Mining and Metallurgy (FSAIMM #20258), Fellow of the Institute for Chemical Engineers (FIChemE #20029470), Fellow of the Institute of Engineers Australia (FIEAust #3671379) as well as a Chartered Professional Engineer (Australia), Chartered Engineer (UK) and Professional Engineer (South Africa). I have practiced my profession continuously since 1984. I have worked as a chemical and metallurgical engineer for a total of 33 years. I have been involved in international metallurgical engineering consulting work for the last 25 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.

c) I have not undertaken a site visit to the Selinsing property.

d) I have prepared sections 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 scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Perth, Western Australia, on 14 December 2016

Signed and sealed Leendert (Leon) Lorenzen

Dr Leendert (Leon) Lorenzen, CPEng, CEng, PrEng, FIEAust, FAusIMM, FIChemE, FSAIMM, GAICD